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USBM H0242007

STUDY OF METHODS TO IMPROVE PILLAR EXTRACTION PRACTICES IN UNDERGROUND COAL MINES

VOLUME 1

PILLAR SAFETY AND PRODUCTION PRACTICES

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

USBM CONTRACT REPORT (H0242007)
DECEMBER 1, 1975

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ASSOCIATES, INC.
MANAGEMENT ENGINEERS

USBM H0242007

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FOREWORD

This report was prepared by J. J. DAVIS ASSOCIATES, McLean, Virginia under USBM Contract Number H0242007. The contract was initiated under the Coal Mine Health and Safety Program. It was administered under the technical direction of SMRC, with Mr. Paul McWilliams acting as the Technical Project Officer. Mr. David Askins was the contract administrator for the Bureau of Mines.

This report is a summary of the work recently completed as part of this contract during the period January, 1974, to July, 1975. This report was submitted by the authors on December 1, 1975.

J. J. DAVIS ASSOCIATES wants to acknowledge our special appreciation to The John T. Boyd Company for their assistance as technical consultant and Mr. Paul McWilliams for his guidance as TPO.

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I. INTRODUCTION

I. INTRODUCTION

The following pages discuss the objectives, the background, the scope and a statement of the problem of the study of methods to improve the pillar extraction practices in underground coal mines.

1. OBJECTIVES

The objectives of this project were:

- to determine the principal factors contributing to high accident frequency and severity in both conventional and continuous mining operations;
- to compare and evaluate the safety aspects of the "open end" and "split and fender" systems of pillar mining;
- to establish basic data required to decide what constitutes an adequate pillaring plan that will optimize efficiency, recovery, and safety;
- to determine what measurements or information should be collected, (e.g. roof sag, pillar stress,) to make on-the-spot judgment of impending roof or pillar collapse;
- to make recommendations to improve existing methods or to insure proper techniques are implemented, and
- to suggest areas where additional studies or new research may be of value.

In addition it was necessary to gather data on factors influencing productivity of pillar extraction practices for room and pillar coal mining in the United States, using these data to perform an industrial engineering study of these pillar extraction practices. This industrial engineering study suggests ways to increase pillar extraction ratios within the restrictions imposed by subsidence, safety, and law. It will also suggest ways of increasing coal pillar efficiency through the development of improved techniques, procedures, and equipment design.

2. BACKGROUND

Pillar extraction, or pillar robbing, in underground U.S. coal mines is conducted using two basic systems: the open end system, whereby cuts are mined from one or two sides of the pillar adjacent to the gob; and the split and wing system, whereby cuts are mined in the same way as in the open end method, except that a fender of coal or a series of small coal stumps are left adjacent to the gob to help support the roof. These fenders or stumps may be later blasted and the coal may be partially recovered. Both systems are equally adaptable to conventional or continuous mining operations with minor modifications. It is estimated that 40 to 70 percent of production will come from pillar robbing.

Artificial supports used in pillar extraction consist of timber breaker props, wood crossbars or wood cribs. Hydraulic props and yielding steel props are also used successfully since they can be tripped and pulled to a safe area when pillaring is complete.

Caving of the roof is required following pillar extraction in order to relieve weight that would otherwise be transferred to the timbers or solid coal along the pillar line. Large unmined blocks of coal, pillars, or excessively stiff supports may prevent caving and cause the excess roof weight to be transmitted to the pillar line. The result can be bottom heaving, squeezing, or other unfavorable mining conditions. In extreme cases, the pillar line may have to be abandoned. Sometimes an entire section may be lost. This can create difficult and hazardous conditions in adjacent panels and greatly reduce overall recovery.

Pillar robbing is accomplished best when both development and pillar mining are planned in advance and executed systematically.

Regularity and speed of retreat are conducive to an efficient and safe operation.

An analysis of fatal coal mine accidents between 1966 and 1973 showed that from a sample of 1000 fatalities, 291 occurred at the face area during retreat or pillar recovery work. There is very little information available detailing the specific operations and mining conditions contributing to these fatalities. However, significant factors seem to be: use of the wrong system for roof conditions; use of improper methods of recovery, such as inadequate or wrong type of supports; and improper implementation of safe, established procedures.

MESA is now faced with the responsibility for improving the roof control plans submitted by the mining companies. A plan for pillar extractions is included in this requirement when applicable. The job of deciding what constitutes an adequate roof control plan requires knowledge of the following:

- What constitutes a safe system for different roof conditions;
- What methods of excavation, support, and haulage are optimum in terms of safety; and
- What factors can be used as a basis for judging unsafe conditions?

3. SCOPE OF WORK

In order to meet the objectives of the project a detailed analysis was made of normalized accident data on pillar extraction. Roof control plans for pillar extraction submitted by mining companies were analyzed. Extensive discussions were held with MESA Health and Safety Inspectors and mine operators. In addition, detailed underground industrial engineering studies were made of a selected cross-section of pillar robbing operations, (both conventional and continuous.)

In total 25 mines were selected for the conduct of the field studies. The selection formed a representative cross section of mines in terms of normalized accident rates and different mining conditions. The mines selection list was approved by the U.S. Bureau of Mines. During the field analysis, a work element breakdown was prepared for each operation in sufficient detail to permit the study of all variables which would affect safety and productivity. The study and observation forms prepared included the following:

- the name of the mine;
- the daily production and production by man shift;
- the number of men employed and man-hour employment;
- the particular coal bed being mined;
- the mining system;
- the mining method;
- the pillaring system;
- the depth and dimensions of the mine;
- the average seam thickness;
- the general lithology of the roof, floor and coal;
- the accident data as it may pertain to pillaring operations;
- job classifications of the workers;
- the work being performed;
- the location of individual workers in the mine;
- the position of the worker with respect to support and equipment;
- the dimensions of the work place;
- any geological abnormalities;
- the length of time the roof was exposed, with or without support;

- the type of support used;
- the type of equipment used;
- the location of support and equipment with respect to the work place;
- the roof support plan;
- the subsidence considerations; and
- any general terrain considerations which may affect productivity or safety.

In each of the mines selected for study all parts of all functions pertinent to the pillar extraction process were analyzed. This included the installation and removal of supports and blasting of stumps or fenders when appropriate. Both conventional and continuous operations, and both open end and split and fender systems were analyzed. The pillar robbing activities were treated as a total system comprised of such factors as mining pattern, mining method, roof control, methane control, dust control, ventilation, and subsidence control.

During the entire study, in order to prevent inadvertant public disclosure of proprietary information, all findings and data were coded. In this manner the identities of companies and particular mines involved in the study was kept confidential.

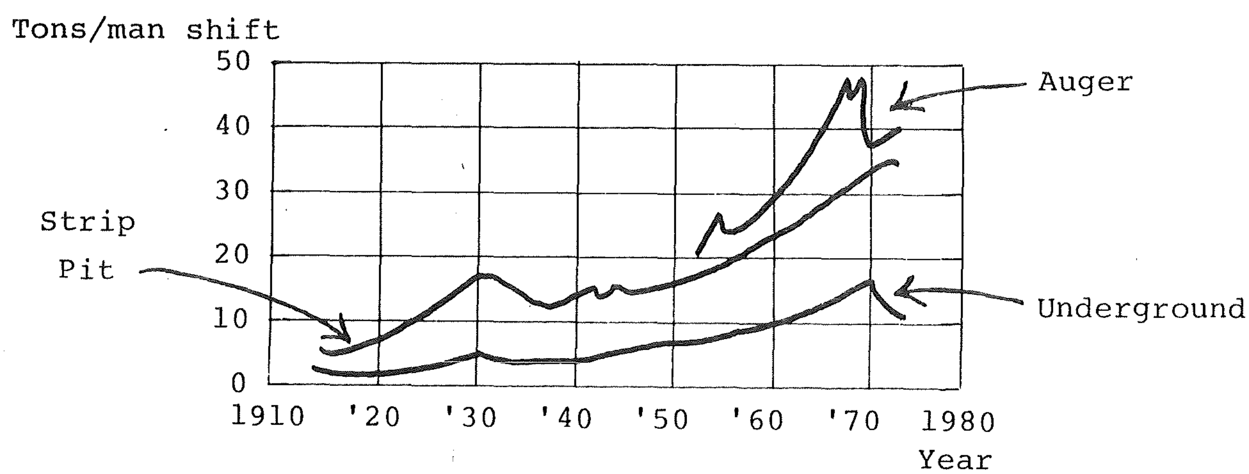
Methods were explored to change the work elements in a way which would reduce the accident exposure during pillar robbing. The measurements and other information required for deciding what constitutes an adequate pillaring plan were determined and applied to presently used pillaring systems. Based upon this information, recommendations were made for making changes in work practices, mining methods, equipment, and safety operations which would improve overall safety.

In addition, the economic impact of increasing the extraction ratio was analyzed and, where possible, estimates were made of the amount of coal lost under current practices and how this loss could be minimized.

Finally, areas of additional research were noted which could potentially reduce hazards, improve safety, and increase extraction ratios and production in retreat mining.

4. STATEMENT OF THE PROBLEM

Production efficiency (expressed in tons/man shift) in underground coal mining has enjoyed a nearly uninterrupted increase in rate throughout this century. Figure I-1 delineates the growth rate of underground mining and contrasts this rate to strip and auger mining. From 1900-1950 underground production increased rather slowly. This is primarily due to lack of automation. Hand and conventional mining contributed nearly 100% of underground production during that period. During the late 1940's and early 1950's, however, conventional mining systems became more automated;



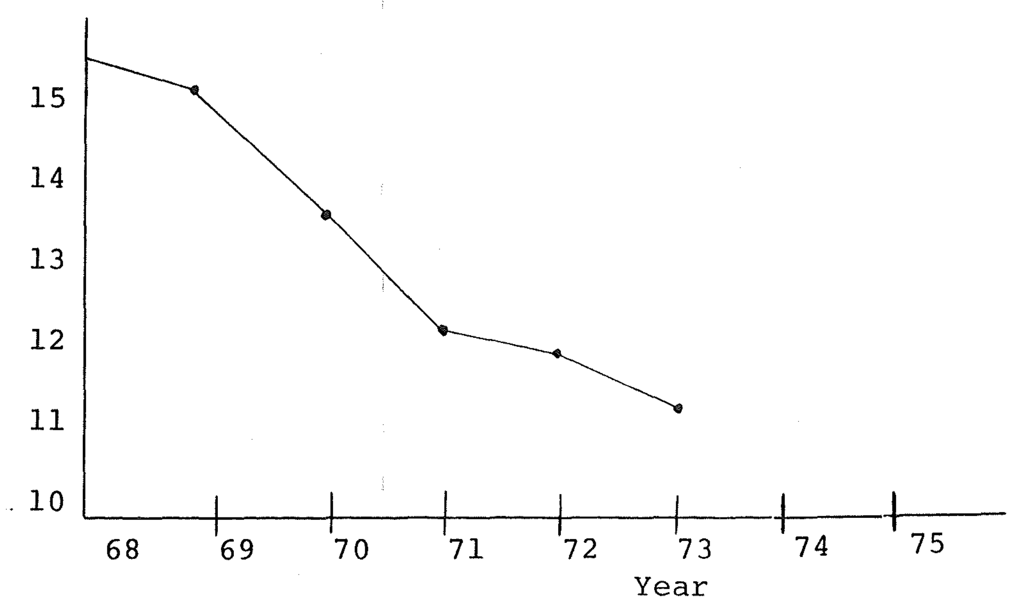
U.S. Soft Coal Productivity by Mine Type

Figure I-1

and, at the same time, the continuous miner was introduced. Both systems greatly improved production efficiency in underground soft coal mines and led to dramatic increases in tons/man shift.

The peak of underground production efficiency in this country was attained by the end of 1968, at a rate of approximately 15.7 tons/man shift. The advent of the Coal Mine Safety Act of 1969 and other factors caused coal production efficiency to decline during the five years following the peak from 15.7 to 11.2 tons/man shift.

Figure I-2 magnifies this dramatic decrease in production efficiency.

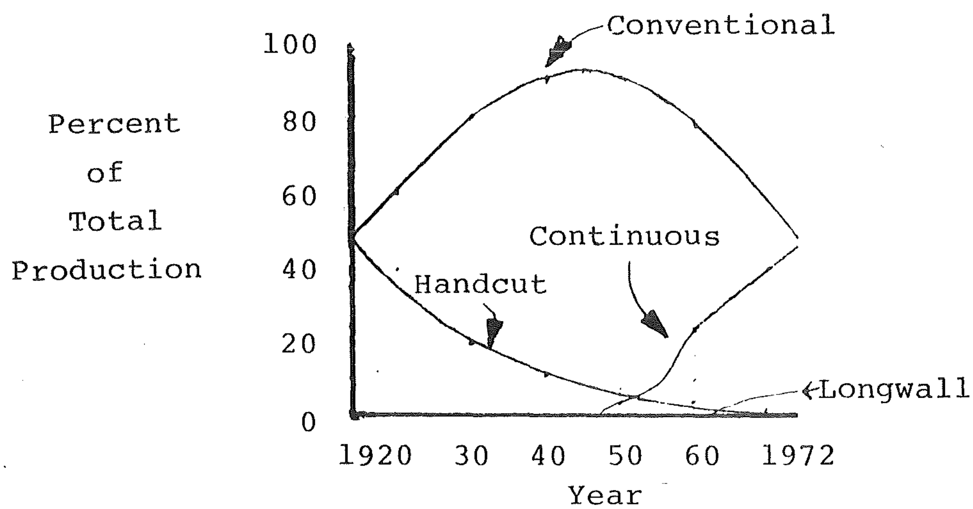


Tons Per Manshift

Figure I-2

There are many variables affecting this change. However, we will dwell only on pillar related factors affecting production. One major factor, in our judgment, is the increasing rate of retirement of experienced "pillar" men. For a period of time young men did not enter mining. This phenomenon has created a vacuum of experienced men who can extract coal in an efficient manner. These skills often take a lifetime to acquire, and the absence of middle-aged workers has contributed to this shortage of skill.

Figure I-3 depicts the rapid increase in continuous mining as a percent of total production. Continuous mining and remote controlled continuous mining drastically changed the techniques and reduced the hazards of retreat mining.

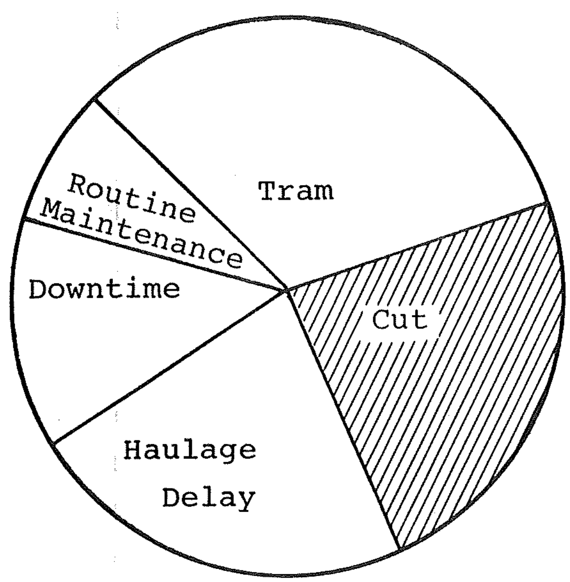


Methods of Underground Mining
Figure I-3

Because of the high cost of continuous mining equipment, many "efficient" pillaring systems were altered or changed because of managements' fear of covering up the equipment. Many other variables influence the adoption of one system rather than another; however, for illustration purposes, this highlights a major problem in pillar recovery.

There have been articles in recent newspapers and publications relating to coal mine production efficiency. Many suggest that the U. S. mining industry has nearly reached technological limits of productivity increases under existing conditions; and that, without revolutionary changes in technology, significant productivity increases cannot be realized (in the near future.) In examining these revelations, one might conclude that limited short term productivity improvements rest on the shoulders of the individual worker and the innovative vision of management and that really significant increases must await changes in technology. While obvious improvements can be achieved through better training, proper mine planning, and improved work attitudes, it need not stop there. Significant payoffs can be realized by more fundamental changes, such as improved scheduling, more effective utilization of equipment, improved face haulage, and the adoption of rigid maintenance plans.

To illustrate, a continuous miner, having the ideal cutting capacity of 500-600 tons/hour, usually can expect only between 300-600 tons/shift. Many inherent delays are built into the production process as shown in Figure I-4.



Distribution of Continuous Miner Activities

Figure I-4

Because of roof control restrictions, for instance, the miner often makes multiple cuts from different directions to facilitate roof bolting. This iterative cycle causes major delays in tramming that are unavoidable, yet costly to production.

Another problem facing the industry is that the "easy coal" in the East has nearly been mined out. With higher restrictions on pyritic sulfur and ash content, more "selective" coal is in demand, requiring more difficult mining practices, including pillar extraction of high overburden, low seam coal. These conditions are added variables with which mine management must contend in structuring its roof control plans for MESA approval.

It takes years before new technology has a significant impact on productivity. Figure I-3 shows that the continuous miner, a major breakthrough in technological advancement, appeared in 1950. It was 10 years before this method contributed 25 percent of total production. New mine systems will require a long lead time before major contributions to increased productivity are attained. This emphasizes the importance of concentrating on improving existing systems.

The USBM is also interested in determining the areas in which they should concentrate their efforts for research and development over the next ten years. To this end, areas have been identified where efforts should be made to develop new technology that will have a major impact on the industry.

As with any study which encounters a multitude of variables, judgment must be exercised in order to use limited observation time profitably. Accordingly, it was crucial to weed out spureous and inconsequential data and to concentrate on the most important questions:

- Where are people getting injured?

- How can roof control better contribute to safety, efficiency, and recovery?
- What operational methods and procedures can be applied?
- How can we optimize equipment utilization?
- How can we efficiently extract pillars?

The first step was to concentrate on analyzing pillar related accidents. These reports offered empirical insights into what occurred before, during, and immediately after each accident.

Throughout the analysis phase, efforts were focused towards expected high payoffs in safety and production efficiency. For example, determination of drill bit efficiency for a mobile face drill machine might provide the optimum drill cutting characteristics for a given job. The impact on productivity of the mine section, however, might be marginal compared to evaluating placement of temporary support systems.

Because roof control is so important to safe and efficient pillar removal, roof control plans were reviewed with regard to their adequacy and potential improvement. From preliminary field studies and accident analysis, a great variance between "perceived" knowledge of roof control plans and actual knowledge of these plans was discovered.

Another area of concentration was in evaluating equipment efficiency, capacity, and economics. Having the proper equipment matched to specific production requirements is crucial to any efficient operation, and the mining industry is no exception.

II. SUMMARY AND RECOMMENDATIONS

II. SUMMARY AND RECOMMENDATIONS

This section outlines the organization of the report and summarizes the findings and recommendations made as a result of the study.

1. OUTLINE OF REPORT

The following paragraphs condense the results of each of the major report chapters. The purpose of this summary is to help the reader to understand the content and organization of the report and to identify sections which may be relevant to his particular interests.

(1) Accident Analysis - (Chapter III)

This chapter documents the results of fatal accident analysis for retreat pillar mining. In the first section, the construction of the data base, the statistical techniques affecting the data, and normalization data are documented. In Section 2 specific recommendations are made as a result of the analysis. Section 3 includes discussion of field study hazard analysis results and includes some specific recommendations to reduce these hazards. Section 4 discusses the analysis of the non-fatal accident data base. Section 5 includes the details of all relevant analysis.

(2) Pillar Extraction Methods - (Chapter IV)

In the chapter on pillar extraction methods, Section 1 discusses concepts of retreat pillar mining, as well as some considerations in the utilization of such techniques. Section 2 discusses pillar recovery, using continuous mining techniques and summarizes the current state-of-the-art in pillar extraction. Although other techniques of pillar extraction may be used, this section discusses the most common techniques used in this country. Among the techniques discussed are pocket and wing, split and fender, split and fender with simultaneous fender extraction, multiple split and fender, radius pillaring, diagonal pillaring, and several novel and interesting pillaring plans. Section 3 discusses pillaring techniques commonly used in mines employing conventional mining equipment. Among the techniques discussed are open ending and split and fender.

Chapter IV was organized to focus on the various options for the extraction of pillars. It does not include any detailed discussion of pillar extraction practices. Such details are covered in subsequent report sections.

(3) Analysis of Pillar Extraction Practices - (Chapter V)

Chapter V highlights some of the benefits and penalties associated with various pillar extraction plans. The discussion includes plans for both continuous and conventional equipment but is restricted to the more common techniques. The comparisons of this section were used to produce a comparative analysis of alternative pillar extraction plans and a quantitative rationale for the selection of alternative pillar extraction plans for particular sets of conditions.

(4) Recommendations for Improvement in Pillar Extraction (Chapter VI)

Alternative considerations and variations associated with retreat pillar mining are identified and discussed in this chapter. Specific problems are highlighted as are several recommendations for improvements. Among the topics discussed are the following:

- sequence of cuts in pillar extraction,
- roof control requirements,
- effects of basic pillar dimensions,
- variations in section layouts,
- angle of the pillar line,
- sequence of extraction,
- alternative ventilation plans,
- logistical considerations,
- recovery rate considerations (i.e., percent of coal extracted), and
- equipment requirements.

(5) Rock Mechanics of Pillar Extraction - (Chapter VII)

This chapter summarizes the rock mechanics considerations for pillar extraction (discussed in detail in Appendix F of this report). Topics presented are room spans, intersection design, and production pillar design. Also included are recommendations for future studies of rock mechanics in pillar extraction.

(6) Appendices

Six appendices are included to provide back-up data for this report. Appendix A provides a complete listing of the fatal accident analysis variables and the definitions used in the fatal accident analysis. Appendix B contains a matrix summarizing all cross tabulations and frequency distributions run for the fatal accident analysis. The characteristics of mines included in the underground studies are tabulated in Appendix C. Appendix D provides summaries of the studied mines. Appendix F is a detailed discussion of rock mechanics theory as it pertains to retreat pillar mining. The MESA respirable dust file data used in normalization of fatal accident findings is found in Appendix E.

(7) Analysis of Haulage System

A separate but parallel study is made of haulage systems pertaining to retreat mining. This is published in a separate report. Chapter III of Volume II contains an analysis of fatal accidents related to haulage. Chapters IV and V of the report discuss general haulage practices as they pertain to retreat mining. Among the topics discussed are two-car cable shuttle car systems, three-car cable shuttle car systems, diesel powered shuttle car systems, and loader hauler dump (LHD) haulage systems. Also discussed are several continuous haulage systems, including bridge conveyors, extensible belt conveyors, and suspended belt haulage systems. All of these systems are discussed within the context of retreat mining. Chapter VI discusses the development of standard data for both shuttle car systems and continuous haulage systems. Included in the appendix is an equipment digest listing various haulage equipment used in coal mines today, including cable shuttle cars, diesel equipment, loader hauler dumps, and continuous haulage equipment.

2. SUMMARY OF RESULTS AND RECOMMENDATIONS

The following pages contain the summary of results and recommendations of this study. Each result or recommendation has been referenced to that part of the report where detailed discussions and back-up data can be found. This section has been organized into several classifications of results to ease the reading. While some of our proposed recommendations are currently being funded by the Bureau of Mines, we nevertheless have listed programs that we consider would have an impact towards improving pillar retreat safety and efficiency.

(1) Equipment

The following recommendations and findings deal with equipment development and modification.

- Develop Mobile Roof Support System. USBM should sponsor a project to develop a Mobile Roof Support Unit. This unit could be used instead of breaker posts in retreat mining and as temporary support in two pass advance mining. A unit-powered, remote controlled and fully recoverable under bad conditions, could in conjunction with remote controlled continuous miners, totally eliminate the need for men in a partially removed pillar or in the immediate vicinity of the face during advance mining. (Chapter VI, 13)

- Develop Timber Machine. The USBM should initiate a project to redesign and build a timber machine for use in pillar extraction. Such a unit could eliminate the handling and rehandling of timbers, lost tools and other supplies, while providing a mobil station for temporary support setting. Such a unit should be self-powered, equipped with a saw, have ample space for carrying material and incorporate a means of recovering timbers that have been set. (Chapter VI, 14)

- Develop Remote Controlled Loading Machine. In the conventional mining cycle the loading cycle is by far the most hazardous. Development of remote controlled loading machines could substantially reduce the hazards associated with this cycle. (Table III-37)

- Develop High Capacity Section Belt Feeder. The U.S. Bureau of Mines should develop a belt feeder system that can accept 8-10 tons of coal in 10-15 seconds, that can rapidly be moved and set up and that can accept coal from three directions. (Chapter VI, 37-38)

- Develop Self-advancing Belt System. The U.S. Bureau of Mines should develop a self-advancing belt system for use with shuttle car or continuous haulage systems. Such a belt should be able to advance in 15 minutes and be capable of advancing up to 150 feet. (Chapter VI, 37)

- Develop Automatic Cable Handling System. An auto-cable hauling device should be designed for the continuous miner and loading machines. (Chapter III)

- Develop Warning Devices. Articulated shuttle cars, scoop and tractor-trailers which when loaded are often limited in operator visibility should have an audible automatic signal to indicate that they are moving in a reverse tram mode. (Chapter III, 40)

- Develop Safe Braking System. All personnel vehicles should have a failsafe braking system that automatically activates in the event of a power failure to the piece of equipment. (Chapter III, 39)

- Develop Section Status Display. A status display system should be developed to provide a more effective means for foremen to communicate the status of a Section to alternate shift foremen. (Chapter III, 41, 42)

- Develop Guide for Equipment Selection. The USBM should develop a handbook which would guide operators in the selection of equipment. Such a guide would demonstrate how to properly match the various operating components on the section. Particular attention should be placed to the continuous miner/shuttle car interface and the shuttle car/discharge interface. Also, attention should be given to the advantages and disadvantages of full-face and two pass equipment, particularly in retreat mining. (Chapter VI, 34)

(2) Roof Control

The following recommendations and findings deal with roof control problems.

- Establish Uniform Standards for Roof Control Plans. A more uniform approach to the approval of roof control plans by MESA is needed. Wide variations in pillaring plans, bolting requirements, room widths, and other factors are found, many of which can not be attributable to local conditions. (Chapter IV, VI, 10)

- Develop Guidebook for Roof Control Specialists. The USBM should develop a manual and a hand calculator to assist the MESA roof control specialist to make on-the-spot assessments of roof conditions in coal mines. (Chapter VII)

- Develop Methods to Test Roof. Current methods of testing the roof are inadequate. Research must continue to develop means by which mine workers can quickly and accurately assess the condition of the roof as part of the normal routine. (Chapters III, 24-34; Table III-9; III-12; III-16; III-19; III-21; and III-45).

- Develop Roof Noise Analyzer. A significant problem is caused by the lack of a means to track the warning sounds given by the roof of impending falls. Continued development of such monitoring devices should be done with ultimately a requirement to have such devices on each pillar extraction section. (Chapter III 2-21, 22; Table III-45).

- Establish Lapsed Time Requirements for Roof Supports. Deterioration of the roof is an irreversible action. Many fatal accidents had, as a mitigating cause, the failure to provide temporary support of the roof immediately following mining. Current legislation does not set a specific period for the setting of temporary support following mining. This should be changed. (III 20; III 24-34; Table III-22).

- Conduct Analysis of Safety of Last Line of Support. An analysis of fatal roof falls shows the danger area is the immediate area at or inby the last row of bolts, yet regulations only address the area inby the last row of bolts. Some operators have recognized the dangers associated at the last row of bolts under certain roof conditions and have modified their plans to keep workers outby the second to last row of bolts. Additional research is required to establish the validity of this assertion. (III 23; III 34-36; Table III-8; III-9; III-13; III-14; III-15).

- Develop Informal Roof Control Plans for Unusual Conditions. Informal roof control plans to cover unusual conditions such as the supporting of kettlebottoms or slips should be adopted by mining companies to standardize the treatment of such abnormalities. Such plans would insure that inexperienced foremen know the specifics of how to treat unusual conditions and adequately respond to them. (Table III-17; III-47).

(3) Pillaring Techniques

The following recommendations and findings relate to mining methods and pillaring techniques.

- Split and Fender Method is Safest. Analysis of alternative plans for pillar extraction shows the split and fender plan using a continuous miner as the optimun plan from both a safety and an efficiency standpoint (Chapter V)

- Develop Handbook for Pillar Extraction. Selection of the proper technique to extract a pillar fender and utilization of the proper continuous miner cutter head can mean a savings of up to 18% of coal lost in stumps and an increase in productivity (Chapter VI-1). The USBM should prepare a handbook to assist the operator to make the proper decisions in planning and in selecting equipment for pillar extraction.

- Pillar Split Widths Are Critical to Safety. A major contributor to fatal accidents in retreat pillar extraction was the driving of the pillar splits too wide and in violation of the roof control plan. (Table III-4, 5, 7, 30, 31). Firm action on the part of MESA and of management to reduce these violations could significantly reduce fatal accidents. Further, a significant number of fatal accidents investigations also cited improper pillaring techniques as a contributing case.

- Comparative Safety of Conventional Versus Continuous. Under all conditions in retreat mining conventional mining is shown to have twice the probability of a fatal accident occurring as continuous mining. (Table III-23).

- Mining Conventionally with Shale Roof. Conventionally equipped sections retreat mining under a laminated shale top had three times the expected number of fatalities. If as it appears conventional mining under these conditions is inherently less safe, roof control plans should be suitably modified or conventional equipment should not be used in retreat mining. (Table III-11).

- Develop a Means for Mining Diamond Shaped Pillars. The USBM should determine a means of efficiently recovering diamond shaped pillars which are inherent when utilizing continuous belt equipment (Chapter VI- 20).

- Establish Experimental Pillaring Section. The USBM should set up a test operating section to experiment on different techniques in pillar extraction (Chapter V).

(4) Miscellaneous Topics

The following miscellaneous topics are presented.

- Develop Model Logistical System. The USBM should initiate a project to develop a model logistical system for coal mines. Such a system should track the supply requirements for operating sections, monitor and operate an inventory system, integrating an automatic replenishment capability with an optional inventory level system and incorporate a cost monitoring system (Chapter VI-31).

- Establish Guidelines for Use of Scoops. The USBM should establish guidlines for the safe usage of scoops. The increased usage of this machine could lead to an increase in scoop related fatalities (Chapter VI-36).

- Establish Guidelines for Remote Controlled Mining. The USBM should establish guidelines for the proper use of continuous miners with remote control with particular emphasis to safety and hazards eliminated by and associated with remote control (Chapter VI-35).

- Improve Inspection Programs for Small Mines. Smaller and lower seam mines have a higher than expected number of fatal accidents in retreat mining. (Tables III-24, III-25, III-26, III-27, III-28, III-29).

- Improve and Standardize Foreman Training. There is a need for a more comprehensive and standardized program to qualify and retrain foremen. Such a program should include the adoption of uniform standards for foremen established and maintained by MESA and regulated by the CMH&S Act,(Chapter III-19).

- Make Workmen Liable for Safety. As in the case of foremen, workers too should be liable to criminal action in the event they fail to follow the law as stated in the CMH&S Act or in the event they are judged negligent.

- Eliminate Job Bidding. The high incidence of job turnover as allowed by job posting or bidding should be eliminated since inexperience is a major contributor to the high accident rate. Alternatively, standardized federally regulated examinations and experience requirements could be required for all hazardous occupations,(Chapter III-24-25; Tables III-32; III-33; III-35; and III-36).

- Develop Reversible Ventilation System. The USBM should develop an efficient means of reversing the air flow on a section. Such a system, utilized with continuous haulage could eliminate the exhausting of air from the face over the mobile bridge operators (Chapter VI-28-31).

- Sponsor I.E. Study of Optimum Haulage. The USBM should sponsor an industrial engineering study of haulage systems, both shuttle cars and continuous belt and, through computer simulation determine optimum haulage paths, change points, mining sequences and belt moves for a variety of equipment capacities and section configurations. (Chapter VI-17-21).

- Develop Breakaway Lamp Cord. The cap lamp cord poses a serious threat to the safety of a miner working in the proximity of mining equipment. The cord should be modified providing a quick disconnect feature, (Chapter III-11).

III. ACCIDENT ANALYSIS

III. ACCIDENT ANALYSIS

This section documents the types and probable causes of more than 200 fatal accidents which occurred in retreat mining during the period 1966 to 1973. In cases where sufficient data was available, analyses of various types were performed to investigate and identify relationships and interactions related to the accidents. The purpose was to learn where possible the factors associated with the accident which could help to reduce possible future underground coal mine accidents.

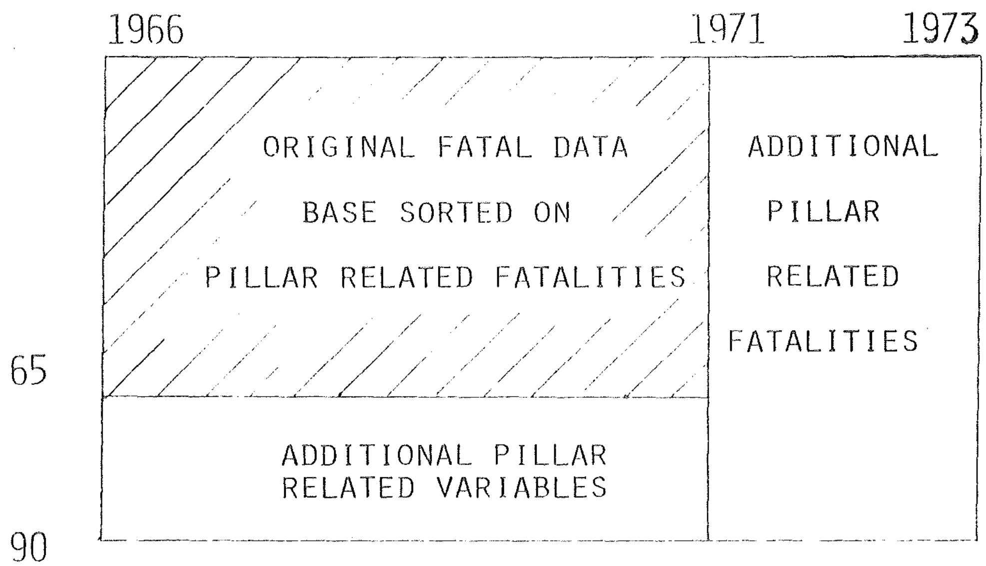
1. STATISTICAL ANALYSIS TECHNIQUES

A detailed statistical analysis was completed of fatal accidents which occurred in retreat mining during the period 1966-1973. The analysis included frequency distributions of each of 90 variables and crosstabulations of a total of 930 combinations. A detailed list of variables and their definitions is included in Appendix A. A list of the crosstabulations analyzed is included in Appendix B. The data base development and details of analysis are included in Part 1 of this section. The details of the statistical results are included in Part 2. The reader is cautioned to read Part 1-(4) entitled "Constraints of Analysis" prior to reading the details of the analysis.

(1) Data Base Construction

The first task was the preparation of data for analysis. The coding structure for fatal accidents was initially developed for the Bureau of Mines under Contract S0110601. Of the 250 variables provided, 65 variables were extracted which could be related specifically to pillar-extraction accidents occurring in the retreat pillaring area.

An additional 25 variables were developed for this study which were important in the analysis of pillar related facilities. Appendix A describes all 90 variables. Utilizing the coding structure developed in the Barry study, all pillar related fatality reports were reviewed for the period 1966-1973 and coded for the additional 25 variables. (The original 65 variables had previously been coded for the Bureau of Mines). Next, all 90 pillar related variables for pillar related fatal accidents in the period 1972-1973 were coded. Figure III-1 depicts the resulting fatal accidents entered into the computer system.



Fatal Accident Reports
Figure III-1

(2) Statistical Techniques Used

Two kinds of statistical techniques are used in this report. The first are the descriptive form of statistics, that is, the frequency distribution, crosstabulations, histograms and other presentation formats used to classify and describe the fatal accident data. In addition, percentages, sample means, ratios, and various numerical and descriptive comparisons are used to amplify and give meaning to the data. The second kind of statistic used are inferential statistics. Inferential statistics are used to decide with confidence if observed results are different than what was to be expected or different from the results observed in the other populations studied. A note of caution regarding statistics: statistical significance should not be considered to indicate casuality. Statistics can indicate that the observed variables are probably related in some fashion, but this cannot be taken to mean one factor "caused" the other. The cause of some observed effect is usually more complex than the relationship of two factors, and establishing the cause of some effect usually requires systematic research under carefully controlled conditions.

Two standard statistical techniques were utilized. These included:

- Chi-Square Evaluation and
- Spearman's Rank Correlation Coefficient

Each technique is briefly described below.

- Chi-Square Test - A Brief Explanation[1]

  In the ensuing analysis, the "chi-square (χ^2) test" is often used. This statistic measures the compatibility of a set of frequencies with "expected results". In a one-way table, the expected results must be generated by the user in some logical manner. For example, if the same number of miners worked each of 3 production shifts, one would expect 1/3 of the output on each shift. Thus, if 9000 tons of coal were produced daily, 3000 tons are expected from each shift. In a two-way table, the expected values are generated from the row column totals of each table itself.

  In either case, the comparisons reduce to looking at the actual and expected frequencies of each data cell. A χ^2 value of 0 means exact agreement; as the value of χ^2 increases, it is less probable that the events are compatible with expectation.

1 W. Dixon and Massey, Introduction to Statistical Analysis, 3rd Edition, McGraw Hill, (N. Y. 1969), Chapter 13.

Dependent upon the number of data cells (which in turn generate a value called "degrees of freedom (df)"), the statistician accepts or rejects a generated χ^2 value, and compatibility of the data. To make this judgment, the statistician must assign a level of probability and read a "critical value" of χ^2 from tables - if the sample χ^2 is less than the tabular value, compatibility is accepted; if not, compatibility is rejected.

- Spearman's Rank Correlation Coefficient - A Brief Explanation[2]

Spearman's Rank Correlation Coefficient compares the agreement of two sets of ranked data and is defined by the expression:

$$r_s = 1 - \frac{6 \sum_{i=1}^{n} D_i^2}{n(n^2 - 1)}$$

where n = number of paired observations (X_i, Y_i)

D_i = rank (X_i) - rank (Y_i) = $R_i - S_i$

If the X and Y random variables from which pairs of observations are derived are independent, the r_s has zero mean and a variance.

When $r_s = 1$, this indicates complete agreement in the order of the two compared ranks.

Conversely, when $r_s = -1$, this indicates complete agreement in the opposite order of the ranks.

(3) Method of Analysis

To analyze the data, one-dimensional data displays were prepared for each variable. These displays were reviewed for inconsistencies in the data, to identify abnormally distributed frequencies, and to tentatively select pairs of factors that might be inter-related for analysis using crosstabulation. Each frequency distribution for each variable was reviewed.

[2] J. D. Gibbons, Nonparametric Statistical Inference, McGraw Hill, 1971.

Next, all meaningful pairings of variables were identified for further analysis. Computer runs would be made to produce crosstabulations for each such pair. For instance, fatal accidents might be examined when classified by pillar extraction method and mining method. The crosstab would consist of four categories: open-end - conventional, open-end - continuous, split and fender - conventional, and split and fender - continuous.

Crosstab run planning sheets were prepared for each variable. These sheets displayed each selected crosstab to be run and served both as a worksheet for the computer runs and as an index of the completed runs. These were prepared by considering each of the more than four thousand possible crosstab combinations, and eliminating those which had little or no chance of resulting in useful conclusions.

Utilizing these planning sheets, computer runs were prepared for over nine hundred separate crosstabs. Crosstabs were run at the lowest level of detail possible using the coding structure of the fatal accident data base. Although the detail in many categories would be inappropriate for the sample size, it provided maximum detail and subsequent tests were not constrained.

Each resultant crosstab was examined to identify abnormal frequency distributions in any category. The need for re-grouping of codes for the crosstabs was determined and where required, crosstabs were rerun. Each case of statistical significance was documented for later review.

The results of all crosstabs were reviewed, and hypotheses were developed for examination during the underground field studies. Each result was carefully documented for subsequent analysis.

As part of the initial analysis of frequency distributions and crosstabulations, the data were reviewed to determine both which specific analysis techniques would be required, and whether the data sample would require regrouping prior to analysis. In several cases, data was regrouped or tables changed to better support the objectives of the analysis. Categories, in the initial results, were regrouped if sample sizes were insufficient for analysis or several categories were combined under a single category "other" to provide more meaningful analysis results.

Selection of the specific analysis techniques depended upon the objectives of the analysis, the type of data represented, sample size and whether or not normalization data was available to provide perspective for the results. The specific statistical techniques selected and a brief description of normalization data used is included with each analysis presented in this section. The following paragraphs discuss normalization and other factors affecting interpretation of the data base and results of the analyses.

(4) Factors Affecting the Data Base

This section describes various factors influencing the results of the data analysis. The data analysis portion of the study was designed to identify safety related problems and considerations as well as to give direction to future USBM studies and R&D efforts. These results, however, must be weighed by the reader after considering:

- the effects of normalization, and
- possible data biases.

Each of these areas is discussed below:

- Effects of Normalization

Normalization is necessary to provide a context for interpreting results. The analysis of accident data must include a careful consideration of the total underlying population from which this data is drawn. Frequencies alone cannot provide an adequate picture. It is normal to expect that when the underlying population consists of a high frequency in a particular classification, a specific sub-population would ordinarily be expected to contain a similar percentage. Thus, if sixty percent of pillaring is done by conventional mining methods, then one would expect sixty percent of the accidents to occur when using conventional mining methods. Consequently, if seventy percent of accidents occurred when using conventional means, then conventional mining is associated with a higher proportion of accidents than one would expect.

Throughout the analysis, close attention was given to requirements for normalization. Those areas where normalization could affect the results are noted. Some of the results which raised interesting and provocative questions are presented here, even though normalization data was not available to complete the analysis. This is because such results may be of sufficient importance to provide a basis for subsequent USBM R&D efforts even though they are not high confidence findings. We have carefully noted such cases and the reader should recognize such results are presented for information only. (See discussion of Normalization Data Used in the Analysis).

- Data Biases

In any data analysis one must be cautious of possible biases. There is little the analyst can do about the biases, but it is imperative that these biases be kept in mind to prevent erroneous conclusions. Among the more signficant are:

- shifting trends,
- fatal accident report preparation,

- coding biases, and
- low frequencies.

Each of these is discussed in detail below:

- <u>Shifting Trends</u>

  The data base utilized in the study covers a period of eight years during which a number of significant changes have occurred in underground mining. Among the more noteworthy are the shift from conventional mining to continuous mining; the shift from open-end pillar extraction to split and fender or pocket and wing; the influx of younger and more inexperienced workers; and the impact of the 1969 Coal Mine Safety Act. As an example, conventional mining methods were used in about 50% of the underground mines in 1955. By 1973, only about 35% of mining was by conventional methods. During the same period, the number of mines dropped by several hundred while the number of men employed increased by several thousand. Table III-1 was extracted from the Statistical Abstract of the United States 1974 - U.S. Department of Commerce Social & Economic Statistics Administration Bureau of the Census and displays the changes in the accident rates over recent years.

TABLE III-1

	1950	1960	1965	1970	1972
Number of Injuries					
Fatal	643	325	259	260	156
Non Fatal	37,264	11,902	11,138	11,552	12,330
Injury Rate Per Million Man Hours					
Fatal	0.90	1.15	1.04	1.00	0.53
Non Fatal	52.38	42.28	44.73	44.40	45.75

Accident Rate - Coal Mining

All of these items and many more could have a dramatic effect on the data that was analyzed. These must be carefully considered by the reader to prevent drawing erroneous conclusions.

- <u>Fatal Accident Reports</u>

  The data base analyzed in Phase I was drawn from the fatal accident reports prepared by MESA inspectors and provided to JJDA in both coded and written form at the onset of this study. These reports were prepared, obviously, by a number of different inspectors, each emphasizing those factors believed to be casual, but each prepared under different circumstances and with certain inherent and unknown biases.

- <u>Coding Biases</u>

  The actual data base was constructed by analysts examining fatal accident reports and then coding the report for each of the 275 variables. This coding process was completed by many different people over an extended period of time. The reports for the period 1966-1971 were coded for 250 variables in the year 1972. The reports for the period 1972-1973 were coded for all variables in the year 1974, along with the coding of an additional 25 variables for the original set of reports. Consequently, errors in interpretations of variables, ambiguous terms, individual knowledge, and fatigue factors all could lead to inadvertant coding biases.

- <u>Low Frequencies</u>

  Sometimes low frequencies were found either from lack of accident occurrences or from missing data. Thus, the analyst could either draw no conclusions or had to make conclusions as based upon extremely small sample sizes. Also the laws of chance can cause the analyst to draw erroneous conclusions. This is an important constraint whenever the analyst deals with small sample sizes. These results were reported for information only. In other cases, conclusions will be investigated further during subsequent USBM R&D programs.

(5) Normalized Data Used in the Analysis

As previously discussed, the use of normalized data is essential in relating the findings of accident studies to an appropriate non-accident environment and population. From a humanitarian standpoint accident rates, no matter how low, are of little interest to families of miners killed in roof fall accidents. However, if we are to study the circumstances of past fatal roof fall accidents and usefully apply what we learn to the prevention of future roof fall fatalities, we must determine the specific details of each accident and relate such details both to the details of other similar accidents and to all other situations where accidents have not occurred (i.e. the population of coal mines and miners). In effect, we must learn what is safe as well as what is not safe. To do this, the study group analyzed the records of more than 200 fatal accidents which occurred in pillar mining during the retreat phase of underground mining from 1966 to 1973. But before these accidents analyses can become more than "interesting statistics", a large amount of information concerning non-accidents must be obtained and related to the accident analysis results. This process is called normalization. The types of information needed for normalization include the number of mines doing retreat mining, the number of mines doing conventional versus continuous mining, the number of miners employed, the number of miners in each job classification, accident experience for the general industrial context and so on. In short, information is required which will place in perspective accident study findings and/or permit pinpointing problem areas. These problem areas in turn, will serve as guidelines for subsequent research and development aimed at reducing or preventing fatal accidents and injuries.

The study groups' efforts to obtain all the necessary normalization information in the appropriate format turned out to be an unrealistic task. Non-accident exposure times were not available. Coal industry summary information was not broken out by retreat mining. Some information was not sorted by conventional versus continuous mining. More than one method of mining was used by some mines, and some information was considered not releasable by the potential donors. Furthermore, when normalization data was obtained, there was no known method of assessing the extent of ambiguities in categorizations or other possible sources of bias in the data. Therefore, such problems and constraints should be considered when interpreting the normalized results. Specific constraints are identified with the various analysis results. By usual standards, the uses of normalization data in this report must be considered crude. However, it is believed that the normalization data available provides an adequate context for interpreting and applying the results of this study.

Several types of normalization were obtained for the analysis. The following briefly lists the various major sources:

- MESA Respirable Dust File, a computer data base containing job classifications of miners sampled industry wide. This data was obtained through the MESA Coal Mine Health and Safety Office. The period covered by these data was 1971-1974 (1971-1973 was used).

- Keystone Coal Industry Manuals 1970-1974, McGraw Hill Publishing Company. A sample of mines listed was compiled including numbers of men employed and tons produced by seam height class intervals. (The sampling procedure used every fourth mine listed).

- Fatality Analysis, Data Base Development, Theodore Barry and Associates, USBM S0110601, Mod. 2. (The original fatalities data base).

- Industrial Engineering Study of the Hazards Associated with Underground Coal Mine Production, Theodore Barry and Associates, PB-207-227, 1971. (Estimated hazardous exposure times for various job classifications).

- National Safety Council, Work Injuries in Pennsylvania, 1969, (Accidents by hours worked at the time of injury).

- Bituminous Coal Facts, 1972, National Coal Association. (Work output per man and accident rate historical data).

- The Miner, His Job and His Environment: A Review and Bibliography of Selected Recent Research on Human Performance, Charles Fried, et. al., PB-211-732, August, 1972.

- Accident Prediction Investigation Study (Pittsburgh Seam Study). Theodore Barry and Associates, 1966-1971.

- Injury Rates by Industry, 1970, Bureau of Labor Statistics; U.S. Dept. of Labor, BLS Report 406.

- Miscellaneous Issues of Coal Age, U.S. Dept. of Commerce, and coal industry publications.

Two items of normalization were used extensively in this report; the Keystone Annual "Tons Produced" and "Number of Men Employed" sample, and the MESA Respirable Dust File Data. Table III-2 presents the results of the randomized sample (every fourth page of state-by-state mine listing) of the 1974 Keystone. In order to compare normalization data based upon "Annual Tons" and "Number of Men Employed", a Chi Square Evaluation was run (see previous discussion for explanation of this test). The Chi Square test indicated that there was a statistically significant difference between the two distributions. This means that lower seam mines employed a greater percentage of the total miners to produce their proportion of the annual production (i.e., $\chi^2 = 19.29$, df = 6).

Respirable Dust File data was used primarily for job classification normalization. Appendix F contains the Dust File listings received from the MESA Health and Safety Analysis Center (HSAC). In addition, Table III-2 provides a limited time frame comparison of the total numbers of miners engaged in the two mining methods and phases of interest in this study. It can be seen that a gradual shift away from pillar mining using conventional methods has occurred even during the time frame of the Dust File data presented. We also know that during the beginning of the time frame of interest in this study (i.e., 1966), conventional mining methods were used in about 50% of underground bituminous mining. Also of interest in part 2 of this table is the declining percentage of men employed during the retreat mining phase. It is believed that this may be more related to increased numbers of mines doing advanced mining.

Table III-2

Mining Method			
Year	Total Miners	Continuous	Conventional
1971	49500	57.0%	43.0%
1972	52493	59.7%	40.3%
1973	48583	64.6%	35.4%
1974	50004	67.9%	32.1%
Mining Phase			
Year	Total Miners	Development	Retreat
1971	51563	73.5%	26.5%
1972	54638	78.1%	21.9%
1973	50462	77.8%	22.2%
1974	50807	79.5%	20.5%

Historical Production by Mining Method and Phase

These data were used as the population data base for normalizing various accident findings in this study. However, since these data were not available for the entire eight year period of interest in the study (1966-1973), the data for 1971 was taken to be the single year most representative of the entire period.

It should be noted that the HSAC data system which provided these data was unable to sort the data in such a way as to permit determination of how many miners doing retreat mining were using continuous versus conventional methods. Since this is an important piece of normalization data, an estimate was formulated. This estimate was based upon miner classifications which are uniquely used in conventional mining such as cutting machine operators, coal drill operators and shotfirers. By comparing the total numbers of miners in these three job classifications doing both development and retreat mining, it was estimated that between 14.7 and 18.5 percent of miners doing retreat mining in 1971 use conventional methods. (See Table III-23) By 1973 the percentage had dropped to between 7.8% and 12.6%. Admittedly there may be problems with such estimates, but in the absence of better estimates, they permit a reasonable approximation of the distribution of conventional retreat mining.

A final note should be made concerning the above normalization criterion for fatal accidents. Perhaps it is traditional in the coal industry to think in terms of some "acceptable" number of fatal accidents per thousand tons of coal produced. This should not be considered a heartless approach concerned only with production. Most coal industry people intuitively relate the large investment, the large amount of equipment and supplies and the large number of miners exposed to the risks of accidents during underground mining to the task of removing coal from the ground. Thus, stated or unstated, their concern is for minimizing the risks to the entire underground mining system. For these reasons, the study group has used both annual tons output and exposure rates of miners in underground mining and expects the reader to use his own judgment and rationale in selecting the "most representative" normalization criterion. (See Table III-3)

Table III-3

Seam Height Inches	Annual Employment		Annual Tons Produced	
	Number of Men	Percent	Tons	Percent
0-36	1474	6.4	4355	4.6
37-48	5369	23.4	18963	20.1
49-60	4555	19.9	20814	22.0
61-72	4999	21.8	24195	25.6
73-84	1164	5.1	5071	5.4
85-96	2663	11.6	10860	11.5
97+	2678	11.7	10297	10.9

Example Use of Normalization - On the basis of the number of men working in the 0-36" seam height class interval of mines (i.e., 6.4%), one would expect, everything else being equal, that 6.4% of fatal accidents would occur in such mines. Hence 6.4% of 219 fatal accidents is 14. However, if the actual number of fatal accidents in such mines was 20 (or 9%) then we can say that the observed number was higher than we would expect, normalized on the basis of exposure of men working in these mines. Normalization of this type is used extensively in the results of this study.

Number of Men Employed in the Mine
As An Approximation of Exposure

(6) Description of Analyses

Frequency distributions and cross tabulations were prepared on nearly every possible and logical combination of variables in the study. These frequency distributions and cross tabulations were run in order to identify possible trends and findings. Many provided useful results to pinpoint safety problems and are presented in subsequent sections. However, many frequency distributions and cross tabulations were eliminated as not satisfying the purposes of this study. Among the reasons for eliminating these data were the following:

- inadequate sample size precluded analysis,
- coding errors or ambiguities were present, and
- preliminary analysis or statistical tests indicated no difference and results were inconsequential.

In some cases even where no differences were found, analyses were presented in this report under the following circumstances:

- differences were initially thought to exist,
- findings were thought to be of general interest to the industry, or
- objective and comprehensive reporting was desired.

The next section summarizes the major findings and recommendations of the study.

2. SUMMARY OF STATISTICAL RESULTS

The following findings and recommendations are organized to assist readers with differing interests to use these results. For example, for use by mine operators, findings and recommendations related to mine planning, foreman control and self-enforcement of safety regulations were identified. For various USBM and MESA readers, findings related to the needs and potential payoffs of current and possible future research and development programs as well as those pertaining to health and safety regulations were highlighted. The sequence of presentation starts with findings related to mine parameters and methods and concludes with findings related to personnel and human factors.

On the whole, the fatal accident record of the segment of the coal industry considered in this investigation is reasonably good considering that the work is underground in a wide variety of geological environments. Being underground means that miners are subject to not only "normal" industrial risks (e.g., vehicle collision, electrical shock, and machinery accidents), but to roof and rib falls and other defects as well. Review of the data and field study results indicates that most operators and miners are concerned with safety and adhere to both health and safety regulations and good industry practices.

On the other hand, a significant number of fatal accidents occur as a result of careless or negligent actions on the part of coal operators, foremen, and/or miners. Other accidents occur because there is insufficient information available to foremen and roof bolters which will indicate the structural nature of the immediate roof as a guide to their decisions regarding specific roof control measures to be used. Additional research and development is needed for devices which can provide more than is currently detectable visually or acoustically. Specific discussion of the necessary technology is beyond the scope of this investigation, however, what is needed is a device providing continuous monitoring of the local roof situation in each section.

Another problem of insufficient information is clearly within the scope of this study. Indeed, it is one of the purposes of this study. This problem can be stated as follows. It is likely that operators, foremen, and miners engage in "unsafe practices" because they have insufficient convincing evidence of the risks and consequences of such practices on both their individual safety and productivity. The coal industry is more than 100 years old and has evolved largely on the basis of intuitive and empirical assessments of safety and productivity practices. In such circumstances, the occurrence of an accident would probably be viewed as a rare or random event by individual mine operators and miners rather than as specific evidence that some current practice is less safe than another alternative means of accomplishing the job. It is likely that miners view underground accidents in much the same way as many people view highway accidents, as simply random events unrelated to their own driving habits or skills. This is because, in most cases, individuals are not privy to analysis of causes and interactions associated with the accident situation. Thus, what is needed is information which will clearly point out both the increased risks and the consequences to the individual when he or his people engage in unsafe work practices. This information should address specifics of safety and productivity errors and practices being used by mine operators, foremen, and miners. Subsequent discussions highlight the findings of both the analysis of fatal accident data and field surveys. It should be noted that these findings combine both "raw" and analyzed results. Raw statistical results were sometimes used because adequate normalization data could not be acquired by the study group to place in perspective all study findings. For example, where statistical results indicated that a poor mining practice may be related to an increased accident rate, it was not possible to determine how many mines <u>industry wide</u> employ this practice using a field survey of only 25 mines. On the other hand, the field survey indicated that some poor practices were not rare events. For some analyses hypothetical normalization examples were formulated to indicate the <u>possible</u> impact of certain findings. As previously indicated, field surveys and statistical results indicated that most mine operators and miners follow both federal and state regulations and good safety practices. Thus, it is likely that only a few mines and miners engage in poor practices or commit safety-related errors. If this is true, the actual risk is greatly understated in the data. As an example, if half the fatal accidents in a category are the result of some poor practice engaged by only 10% of miners, then the actual risk of a fatal accident would be nine times greater for those using that practice.

The following are brief summaries of findings from review of about 225 accident reports organized by subject area. Single thread sketches of functionally related findings have been prepared on mine operators/supervisors, mine foremen, mine workers, and their training. These single thread sketches are a loose form of correlation for raw results because people are generally consistant in their behavior. If a man engages in unsafe or shoddy practices in one aspect, it is likely he will do the same in other aspects. No attempt is made to establish blame or responsibility, only to report the findings presented by categories believed to be useful influencing decision-making regarding safety practices.

(1) Mine Owners/Operators

Statistical data indicates that a non-trivial number of mine operators knowingly or inadvertently increase the risk of fatal roof fall accidents in several ways. They:

- failed to insure adequate training of foremen,
- did not provide and/or enforce policy regarding adequate qualifications for machine operators and substitute mining equipment operators, and
- failed to provide adequate self enforcement of mine safety regulations and good work practices.

As indicated in subsequent discussions and analyses, these types of policy and management failures are indirectly reflected in the probable causes of numerous fatal accidents.

(2) <u>Mine Foremen and Superintendents</u>

Statistical results have indicated that section foremen and/or superintendents for any of several possible, but unknown, reasons:

- did not adequately test the roof in 118 cases (V305) and were not precluded from proper roof testing by top coal (V306);*
- failed to provide temporary roof support when it was judged later that such supports could have helped in 129 cases (V319);
- permitted roof to remain unsupported from 1-19 hours (V312), permitted or failed to prevent unnecessary exposure of miners under unsupported roof for extended periods. Ninety-three percent of fatalities in this category occurred within six hours after the place was mined out (V312 vs V144);
- directed or permitted mining of pillars not in compliance with approved mining plans (<u>V323</u> vs <u>V144</u>);
- permitted abnormal roof pressures from improper support, overburden, and insufficient support from adjacent pillars and/or gob area in 105 cases (V325);
- directed or permitted overcutting of adjacent pillars (V320);
- permitted or failed to prevent use of improper work procedures in 219 cases (V120, also V66, V67, V68, and V69);
- permitted or failed to prevent improper operation of equipment in 53 cases (V135) and miners working exposed under inadequately supported roof in 158 cases (V144);
- failed to sense and/or recognize critical signs of bad roof conditions (V129);
- did not issue proper instructions in 162 cases (V68);

* V306 refers to variable number 306 in the analysis and crosstabulations.

- did not act to prevent accident in 219 cases (V35);
- failed to react appropriately to unusual situations (V39, V53, V144, V192, and V194).

From reading raw fatal accident reports and from review of accident analysis results, it is believed that a nontrivial number of foremen and supervisors do not handle well the unusual, unscheduled or unforeseen events in mining. When it is considered that such events occupy only a small fraction of their total time (i.e., responses normalized by percent of time spent in such events), the impact of this deficiency on risk of accident is even more pronounced than indicated by the data. Such events ordinarily require immediate and effective adjustment of actions and resources to minimize the impact on productivity and safety. In general, effective response to such events requires that the foreman must do the following:

- sense or perceive the true nature of the problem,
- assess the impact on current operations and safety,
- determine and assess alternative means and resources required for solving the problem considering both safety and productivity tradeoffs, and
- select and implement an optimum (or at least adequate) solution.

The requirements of such decision-making are beyond scope of this effort but should be the subject of further study.

(3) Mine Workers

Statistical data indicates that workers knowingly and unknowingly or inadvertently exposed themselves to increased accident dangers when they:

- exercised poor judgment in his course of action in 112 cases (V58);*

- ignored warning given by others in 200 cases (V60);

- ignored warning given by mine environment in 181 cases (V61). This may be a result of stimulus saturation (i.e., too many and too frequent indications of such events), noise masking, or perhaps bravado (i.e., reluctance to indicate fear to fellow workers);

- failed to take action to prevent accident in 219 cases (V35);

- ignored or delay response to roof "working" in 101 fatal accidents (V307);

- engaged in improper work procedures in 219 situations (V120), inattention to job, improper operation of equipment and operated from unsafe position.

- worked under unsupported roof unnecessarily in 158 cases (V144);

- attempted to save equipment in 6% of fatal accidents (V308).**

- had tacit approval of supervisors (V66 and V67) concerning unsafe actions; and

- did not understand roof control plans in 117 cases (V309);

Only 10% of roof fall fatalities did not involve some poor practice or procedure (e.g., unsupported roof, roof support not in compliance with approved plans, and roof left unsupported for more than two hours). Ninety percent of fatalities in the sample were associated with such errors on the part of both foremen and miners. Unsupported roof is 600 to 1000 times greater risk (see page III-23).

* Interestingly, raw results indicated that certain jobs such as recovering equipment, cleaning up fall, testing roof, etc. were safer than so called "routine jobs". However, if such jobs could be normalized by a fraction of the time in which they are conducted, this finding would probably be reversed (V53);

** This may have been in "poor" mines where livelihood depended on individual pieces of equipment, but this cannot be confirmed.

(4) Training

Training considerations appear to reflect a paradox in the fatal accident data.

- Training was indirectly considered insufficient by the following references in accident reports.
  - seventeen reports considered training insufficient. (V310)
  - improper work procedures were cited as the accident cause (V120), and
  - victim exercised poor judgment. (V58)
- However,
  - only two fatal accident reports (V196) cited lack of training as the primary personal factor contributing to accident, and
  - 124 reports (V310) considered training sufficient.
  - only 10 mines reported formal training, only 7 reported regular on-the-job training.
- Therefore,
  - training may not be considered necessary to specific mining tasks by the accident investigating committees. The exception is supervisor training which was directly and indirectly considered inadequate in numerous reports.

Few reports cited specific training programs given. The most common notation of training provided was a "combination of programs" (V17)

(5) MESA Regulations

Some of the findings of this study have a possible impact on current mine safety regulations.

- Fifty percent of fatal roof falls occurred at or inby the permanent line of support. This finding was a result of a crosstabulation between "Classification of Accident Type" (V029) and "Failure Nature" (V146). This crosstab indicated that roof failures at or inby the last line of support are most prevalent failures of permanent and temporary supported roof. However, when it is considered that less than 1% of a section's immediate area would not be supported by permanent roof supports at any given time (e.g., assuming 7 entries and 5 crosscuts back with pillars on 70' centers and 20' rooms) the results are even more important. In addition, less than 16% of the section's total man-minutes per shift (from previous and current studies), would be exposed under roof not supported by permanent supports. When these two normalizing factors are considered, the risk of fatal roof fall accident is at least 600 times greater at or inby the permanent line of roof support than elsewhere in the section's area. In addition, this greatly increased risk is understated if the total area of permanently supported roof in an entire mine is considered. In a large mine, the total area of supported roof would be many times greater than the roof area of one section's immediate work area.

- The longwall and shortwall mining systems appear to be inherently more tolerant of inexperience related or careless errors by foremen and miners. This is probably due to the continuous powered roof supports used by each. Conventional mining is probably the least tolerant. This mining method by design requires greater hazardous exposure by miners (e.g., loading under unsupported roof).

- If the best use is to be made of the above findings, the most important findings of this study should be compiled in a concise handbook readily usable by operators, foremen, and miners actually in a situation where such information could be used to influence day-to-day decision making in mines industry-wide.

(6) Mine and Job Attrition

Accident data and field studies indicate a rather high turnover among underground job classifications. Lack of experience in the underground mine environment appears to be related to increased risk of fatal accidents and injury. Similarily, inexperience on a specific job appears to substantially increase the risk of accidents. Job inexperience may result from several possible causes:

- normal job attrition,
- job posting and bidding,
- expansion of mines (i.e., greater manpower needs),
- less than adequate job training or cross-training,
- less than adequate immediate supervision,
- absenteeism (hence job substitution), and
- variability in the level of mine employment.

Accident data (non-normalized) indicates that job errors were cited as the cause of many accidents. (See Tables III-46, III-36, III-33). Job errors may result from:

- inexperience,
- poorly designed equipment and controls,
- poorly designed procedures and support equipment,
- equipment and procedure designs intolerant of minor operator errors,
- insufficient information and coordination, and
- boredom and inattention.

The current method of job posting and bidding appears to encourage job turnover.

Nationally, the average annual turnover of job classifications (based on Respirable Dust File data portrayed in Appendix F), is about 28% with a range of 13% to 46%. Not surprisingly, the turnover is highest for helper jobs (which probably implies a less than one year on-the-job training cycle). Remarkably, hand loaders had the lowest turnover (i.e., 13.4%) which may only imply that the few mines still using hand loading do not offer many advancement possibilities.

It should be noted that the "observed" turnover in job classifications in the Dust File data is partly accounted for by variability in the number of miners employed per year and by gradual shifts in the proportions of continuous and conventional mining. The effect of these considerations is to reduce the observed job turnover estimates by 10% to 12%, since these are factors over which individual miners have little control. As an example, continuous miner operator turnover in 1973 was 28.2% of the category. When the increased number of CM operators from 1972 to 1973 was considered, the effective turnover was 19.9%. This means that while job posting and job preference factors caused some job turnover and job inexperience problems, industry employment problems and technology-method changes also are major contributers to the job turnover.

Interestingly, all Longwall miner classifications (taken together) had a substantially higher apparent turnover than the national average (i.e., 37%). However, when the increased number of mines using Longwall method is considered, most of the observed turnover is accounted for by this single factor. Hence, actual job turnover is relatively low for Longwall mining.

An attempt was made to interpret the job classification versus mine experience fatal accident data (V042 vs V046). However, the only mine experience population data available for non-accident miners was from a limited context, the Pittsburgh Seam studies by Theodore Barry

Associates. On the basis of those data, a sharp decline in accident rate was observed during the 1 to 4.9 years-of-mine-experience class interval, but the 5 to 9.9 and 10 to 25 years-of-mine-experience intervals increased. These results are considered incomplete and may not be representative of industry-wide experience.

The continuous miner (CM) operator's job (V042) had the maximum fatal accident frequency and was 16.44% of the total. (See Table III-33). This finding was consistent throughout variables associated with mining method, accident method and work cycle (V911, V931, and V039). However, when the raw frequency of CM accidents is normalized by the number of continuous miner operators working in retreat mining, (based upon 1971 Respirable Dust File Data), the frequency of CM accidents is only slightly higher than would be expected, based upon their proportion of the population. Table III-34 (Analysis of V042) defines that relationship as the risk factor (i.e., the accident frequency divided by the expected accidents based on population). For CM operators the risk factor is 1.18, or 18% higher than the expected value. Thus, their job is somewhat more dangerous than their proportion of the retreat mining population would suggest.

On the other hand, loading machine operators, second ranked in frequency of accidents, had nearly two and one-half times the number of fatalities than would be expected. In the same category, hand loaders, although today their numbers are small, have the highest risk factor, 3.11. In other words, hand loaders suffer more than three times the expected number of accidents. However, roof bolters (in retreat mining), initially believed to be a dangerous job, are twelfth ranked and have a risk factor of 0.6 (or slightly more than half the expected accidents). Similarly, shuttle car operators experienced only about half the number of accidents their proportion of the population would suggest.

Entry level jobs such as laborer, timbermen and some helper jobs are more dangerous than their part of the miner population would suggest. Obviously such miners are less experienced in avoiding accident situations, and their inexperience probably contributes to this accident rate. However, most helper jobs are safer than jobs of the operators for whom they work. Perhaps when canopies are in wider use, this statistical difference may be reversed.

In an effort to explain some of the effects noted above Table III-35 was prepared. In this analysis, the frequency of fatal accidents by job classification was related to estimates of hazardous exposure. The source of these measurements is indicated on the table. However, it should be noted that hazardous exposure estimates should be considered only a rough estimate of the extent to which operators and helpers exposed themselves to unsupported roof. In this table, estimates of hazardous exposure for various job classifications were ranked from most to least exposed to unsupported roof. Following this ranking procedure, a rank correlation coefficient was prepared which indicated a significant relationship between the amount of time spent under unsupported roof and high accident frequencies, hence high risk jobs. It is believed that this problem area represents one of several possible payoffs for improvements in training and self-enforcement of safety regulations and safe working practices. It also represents a probable payoff for installation of canopies on major face mining equipment.

In a further effort to analyze the possible causes of high accident rates, the accident victim's job experience was analyzed in Table III-36. This table compares the victim's job classification with his experience at the mine where the accident occurred. The table suggests no apparent pattern of accidents with mine experience, except as previously noted. In other words, the higher accident frequencies occur with the continuous miner operator and loader operator and so on. Unfortunately, the class intervals of mining experience selected make it difficult to assess the effect of years of experience on accident frequency. However, if it is assumed that both experience and accident frequency are linearily distributed among the years in each class interval, the average number of accidents per year declines from 1.9 per year interval to 0.09 per year. Obviously, the assumption that accidents and experience are linearily distributed by year is not true, especially in the less than five year experience intervals. However, it can suggest that some declining rate of accidents per year of experience is being observed. In other words, an accident avoidance effect learned with years of experience appears to reduce the accident frequency in the miner population.

When compared to other industrial conditions, the mining environment is a comparatively hostile work place. There are many ways in which an inexperienced man can get into trouble. Some bad roof conditions are visually obvious, but some situations require complex information and subtle clues for detecting and avoiding serious trouble. Much of this accident avoidance information can be learned with continued experience. However, learning to avoid accidents can be a serious problem when increasing proportions of the work force are made up of inexperienced miners. This can occur with expansion of existing mines and as experienced miners retire. It can also occur as new mines are opened and compete with older mines for the existing work force. The effect is a serious problem in training inexperienced miners in the accident avoidance procedures underground.

There is also a problem in accident avoidance unrelated to years of experience. From the analysis of the data in this report, it is clear that there are accident avoidance information problems which are beyond the physiological capabilities of men. For example, correctly assessing a bad roof condition requires information which can be seen and heard as well as experienced. In addition, it requires information not detectable by the human senses. To provide the information needed for comprehensively assessing and correcting bad roof conditions, new sensors or other detection methods will be required. One alternative means of providing this kind of information is a USBM program currently underway using microseismic sensors, and a display of rock "noise" activity versus time. There may be other sensor alternatives or procedural alternatives which can provide the same kind of information. However, the essential element is the information necessary to adequately assess the structural nature of the immediate roof, and to identify any bad roof conditions. Obviously, if new roof detection systems become available to operating mine sections, miners and section foremen must be trained to relate their experience and knowledge of roof conditions to the information provided by the detection system. Then the combination of experienced miners and new information provided by systems should markedly reduce serious roof control and safety problems.

In related investigation, shift times when the accident occurred were compared. Table III-40 presents the distribution of accident frequency versus hours-on-shift-at-the-time-of-the-accident. The most prominent feature of this table is the high frequency of accidents which occurred both in the first hour and in the first and second hours following the break for lunch or dinner. These peak accident frequencies are especially interesting if one considers that during the first hour and the last hour of the shift some significant portion of the time is consumed by tramming into or out of the work face area. This means that in terms of actual exposure in the face area the hours actually worked (hence, exposed) per hour on shift may be low. For example, due to time spent tramming into (or out of) the face area, the time exposed at the face during the first (or last) hour of the shift may be as low as 0.5 hours exposed per shift hour. This means that if the accident frequencies are normalized with time actually exposed to the increased dangers in the face area, the rate of accidents in the first hour on shift may be as much as double the raw frequency indicated. (See Tables III-11 and III-12).

This study primarily considered face area accidents in pillaring, therefore, haulage accidents or other accidents not related to the face were not a major factor. To add perspective to these findings, a National Safety Council study of more than 70,000 manufacturing and non-manufacturing, non-fatal injuries in Pennsylvania during 1969 were compared with the data in this study in Table III-41. A Chi square evaluation was conducted and it was determined that the distribution of hours when fatal accidents occurred in coal mines was significantly different from the National Safety Council results. The accident peak during the first hour was not radically different from NSC industrial experience. However, the high rate of fatal accidents following lunch break in this study appears to be the strikingly different feature of this comparison. The cause of this difference are not precisely known. However, the fatal accident analysis associated with time-the-roof-was-unsupported may provide a clue. In many cases, section crews simply stop work for lunch and the roof just mined remains unsupported during that period. Other possible explanations may be an initial disorganization of the crew after resuming work or perhaps a psychological attention lag during the transition from a non-working to a working state. Since casual factors cannot be determined at this time, an interim solution may be a MESA campaign to increase miner awareness of this danger and improve emphasis on safety when beginning or resuming work.

As previously noted, inexperienced workers appear to suffer a greater number of accidents. In Table III-43 the number of hours the victim worked prior to the accident was compared with the victim's experience at the mine. This Table indicated that the previous findings were consistent in that less experienced miners were more likely to become the fatal accident victims during the first two hours of the shift and immediately after the lunch/dinner break.

Table III-45 compares the accident victim's experience and his job classification with the shift during which the accident occurred. Only about seven percent of mines report using a third production shift. At these mines 15 to 17 percent of fatal accidents occurred during that shift. The reason is probably found in the higher percentage of inexperienced miners in the third shift. Another possible explanation would be difficulties in physiological adjustments caused by changes of the diurnal cycle. This phenomenon has been observed in other industrial studies for second and third shift operations.

An important apparent cause for many fatal accidents involves some form of human error. Table III-47 summarizes the primary unsafe acts by the victim that accident investigating committees believe contributed to the accident. In approximately 87 percent of accidents some unsafe act was determined to have been related to the accident. The higher accident frequencies were compiled when the victim did not test the roof properly, did not follow warnings or instructions, was unnecessarily under unsupported roof, improperly operated equipment, or placed himself in an unsafe position. Probably the only solution to such problems is a continued strong emphasis by MESA and mine operators on training and restraining for safe work practices along with conscientious and consistent self-enforcement by mine operators of safe and effective working procedures. In some cases mine operators must be made aware of the payoffs for safety and productivity when miners use safe working practices.

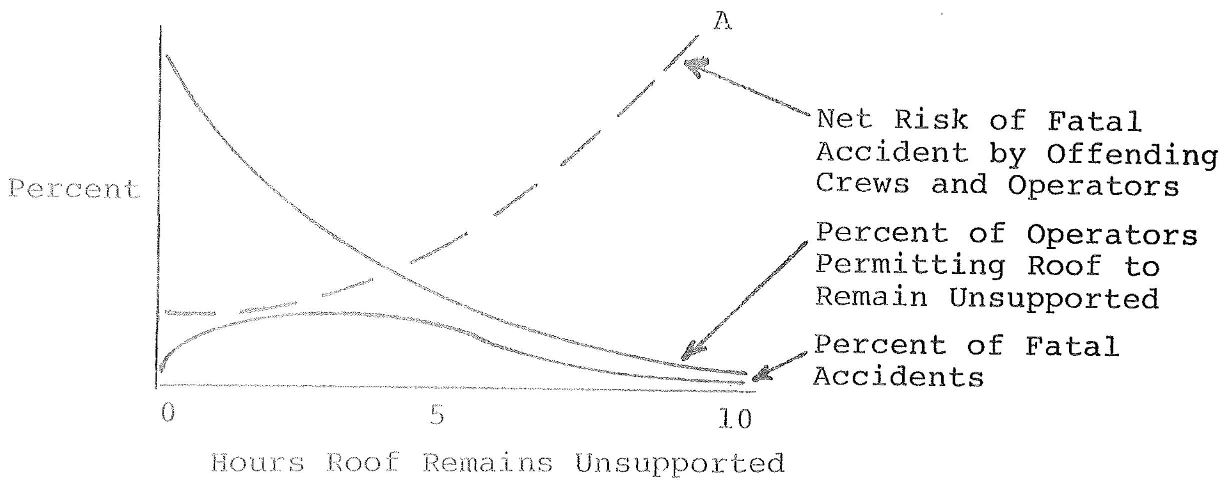
Miners must be initially trained to be aware of the safety problems. But that is not enough. They, like mine operators, must remain vigilant for the reoccurrence of these problems over many years. An underground coal mine is similar to other industrial circumstances in that with appropriate attention it can be a safe work place. However, a high degree of vigilance must be continuously maintained to avoid trouble which may occur. Many years ago an expression was coined to reflect such vigilance requirements. The expression is "hours of boredom interrupted by moments of stark terror". The routine and boredom leads to complacency regarding safety which in turn causes some miners to thoughtlessly engage in unsafe practices. This same complacency tends to resist periodic efforts by mine operators and MESA to focus on the dangers and refresh the awareness of individuals to specific safety hazards. Considerable human factors research remains to be done on effective ways to break through the complacency barrier many miners place between themselves and efforts to heighten awareness of safety problems. One possible mechanism for increasing the awareness of miners to such problems may be a "Zinger's Handbook". This handbook would be a brief compilation of the findings regarding fatal accidents from this and related studies. Such a handbook would be most effective if published in a small, easily readable form, using a presentation format designed to improve reader interest in the factual material presented (e.g., drawings, cartoons and caricatures). Such a handbook could be widely distributed among miners, perhaps at union meetings, with pay envelopes, or at routine waiting locations going on or off the work shift. Such a handbook might also include safety "Zingers" developed in other research and training programs, such as the training programs by MESA. Wide distribution of such a handbook, along with increased emphasis on both formal and informal training, is believed to be one solution for reducing fatalities associated with underground coal mining.

(7) Roof Control as Related to Safety

Nearly half of the 81 fatal roof falls which occurred in the face area were at the last line of permanent support (Cross tab V124 vs V146). The high frequency of such failures has caused some mining companies, on their own initiative, to rule that miners should not advance closer than one row of bolts behind the support line. Also significant is that more than 25% of failures occurred above the bolt anchor line (horizon). It is believed that these two findings provide ample evidence that miners have insufficient information to assess the true nature of the roof. Furthermore, few approved roof control plans provide systematic provisions for controlling bad roof conditions. Usually selecting the specific supplemental roof control provisions is left to the judgment and experience of individual foremen.

This means that variations in experience and judgment of section foremen and roof bolters can be expected to result in differing abilities to effectively deal with various roof problems encountered in each operating section. It is believed necessary that MESA review both regulations and enforcement objectives concerning advancing to the support line as well as test bolt procedures to assess quality of anchors. In the interim period, training of foremen and roof bolters concerning rock mechanics principles may be helpful in reducing the effects of this problem. It is also believed that USBM Research and Development should vigorously continue to develop roof sensors to provide essential roof information for face crews.

Some mine foremen and operators are permitting roofs to remain unsupported longer than either necessary or permitted by MESA regulations. (Cross tab V-124 vs V-312) Such unsupported roof is related to 65 fatal accidents. It was not possible to determine how many mines permit such poor practice. The field sample indicated no instance of this practice. However, even if few operators permit such poor practice in their mines, those few offending mine crews and mine operators are subjecting themselves to an inordinately high risk of accidents. The following hypothetical example illustrates the effect believed to be operable. (See Figure III-2).



Hypothetical Example on Unsupported Roof
Figure III-2

As illustrated in Figure III-2, as fewer operators allow the roof to remain unbolted for any length of time, the net risk of incurring a fatal accident in areas where the roof remains unbolted increases sharply as the length of time the roof is unbolted increases (see line A). It is believed that the inevitable deterioration which occurs in the strength of the immediate roof is irreversable, i.e., that eventual bolting does not achieve the level of strength obtainable with immediate bolting; it is not only dangerous but may be futile to attempt to obtain adequate roof support after extended periods of time. Good mining practice and safety regulations should be stressed in training and rigidly self-enforced by all mine operators.

3. FIELD STUDY RESULTS

There can be no question that coal mining is one of the most hazardous industrial environments in the United States today. Miners work with large, cumbersome equipment in restricted areas under conditions of poor visibility and high noise. These conditions alone represent a serious hazard, but added to them are hazards of roof and rib falls, making efforts to achieve a safe work place, a continuous "uphill fight". Furthermore, during retreat mining, coal is extracted in such a way as to systematically induce roof falls, adding to the dangers.

A comment from one mine foreman interviewed as he described his most hazardous mine is most interesting. This mine had an extremely bad roof, yet it had the safest record of any of his operations. He attributed this to the fact that his men never trusted the roof. "Because they always feared it, they never forgot it". This is often true in retreat mining. Many men recognize the hazards of the work and they exercise the proper caution in the completion of their assignments. Unfortunately, there are exceptions.

In the course of the underground visits, several unsafe acts were observed. This was not unexpected, nor is it meant to be critical, but such incidents are indicative of the hazards encountered day by day in mining. Some of the hazardous acts observed should not have occurred. Foremen, as leaders, should provide good examples of safe practices. On one occasion, the foreman, in showing the study team around the section, walked out into the gob area to point out where a piece of equipment had been caught in a roof fall a few days before the visit. He walked approximately 30 feet into the mined out area to the approximate location of the previous fall. Later, as the team returned to that location, a major roof fall occurred in the gob area where he had been standing. The foreman's rationalization for his unsafe act was that he knew and understood the roof and that it was not working at the time he had exposed himself. On a second occasion, the same foreman, while observing the section extract a pillar using the pocket and wing method, walked around to the back side of the pillar and entered the pocket through the gob side of the breakthrough, much to the amazement of his crew. Another foreman was observed walking some 40 feet out into a mined out area to recover posts which had been left in place. These three acts of carelessness could each have cost the man his life. Each was a display of bravado or at best thoughtlessness. Each was unnecessary. These essentially random events illustrate the need to continually emphasize to workers and foremen the hazards of their work place and the need for constant vigilance.

The setting of temporary supports is one of the more hazardous tasks in coal mining. Workers must go under unsupported roof in order to set the timbers or jacks. This task must be completed, but it can be completed in a safe manner. In a number of instances, workers were observed throwing timbers into or adjacent to an unsupported area, then needlessly working under the unsupported roof to measure, cut, and set timbers. Obviously, the major part of this task could be completed safely by working in the supported area.

One piece of equipment now in great demand is the twin-boom roof bolter. With the current union contract requiring that two men work in any hazardous face area, the use of a twin boom bolter appears to have many advantages in some applications. However, where the room width is 16 feet or less, only three roof bolts are required for the span. In such confined areas, use of the twin boom bolter is difficult, both in terms of work and sharing operations within limited clearance. A worker between the booms could be crushed between the two booms. An operator outside the booms must work in close proximity to the rib and is vulnerable to rib rash. The work space requirements for this type of equipment must be thoroughly analyzed. Consideration should be given to restricting its use to those areas where sufficient space for safe operation is available.

On another occasion during field observations, a continuous miner developed a malfunction which required corrective maintenance. Checkout and repair required that the cutting head be raised approximately two feet off the ground. The mechanic, upon arriving at the scene, set the head in the proper position and set a safety switch to prevent the head from being accidently dropped. A second man assisting the mechanic was equally concerned about the possibility of the head being dropped so he, too, set the safety switch, unaware that he had, in fact, returned the the switch to its original position. The result was that the head could have inadvertently been dropped on the mechanic. Such safety switches should be redesigned. A key locking device or highly visable and unambiguous flag which could not be inadvertently reset are possible solutions to this problem.

Electrically powered mantrip jeeps or buses occasionally are subject to loss of power when the trolley arm bounces off the trolley wire. This, in itself, creates a hazard because the trolley arm bounces against the roof causing debris to break off and fall. In most cases, the operator reaches up and pulls the trolley arm back on the piece of equipment while the vehicle is moving. In doing this, the operator risks injury from roof support beams and overhangs, and is working in an unstable position. Normally, when the trolley arm comes off, the operator slows the vehicle using a hand brake. Occasionally, due to malfunction, the brake system on the rail car becomes inoperative. On several occasions, field study team members observed mantrip rail cars being operated without proper braking systems. On one such occasion, a car with defective brakes was being driven out of the mine with the intention of using motor reverse in lieu of the brake system. While going down a fairly steep grade, the trolley arm bounced off the trolley wire. Without power for reversing the motor to stop the vehicle, the car was entirely without brakes or lights while going downgrade. Obviously, adequate maintenance would prevent this problem. However, it is believed that there is also a need for fail-safe brakes on mantrip vehicles which would automatically operate brakes to bring the car to a stop in the event of a power failure. An override could be provided on the system to allow towing of the vehicle to a maintenance area.

Maintenance of equipment in a mine is routinely performed under difficult conditions. A continuous miner may malfunction at the face under unsupported roof or shuttle cars may fail when heavily loaded with coal. Mechanics must repair these malfunctions quickly and safely. On one occasion, the field study team observed the replacement of a tram motor on a shuttle car. The activity was performed while the car was fully loaded. The car had been jacked up and cribbed for additional support while the motor was replaced. The wheel had been removed and the new tram motor was installed. As the motor was secured, the mechanic sat down with both legs under the shuttle car frame and between the cribbing blocks supporting the car. From that unsafe position, he tightened the bolts securing the tram motor risking a serious accident. The motor could have been installed in a manner which would not have exposed the mechanic to such hazards. Such an incident again emphasizes the possible hazards associated with each individual's work place and the need to be vigilant for safety problems at all times.

A piece of mine equipment increasingly being used for face haulage is the articulated shuttle car. In this unit the payload unit is permanently coupled using an articulated drawbar connected to the trailer. In operation, the shuttle car is backed to the loading point, turned and then backed to the discharge point. Because of this design configuration and the position of the operator, direct visibility behind the unit is virtually impossible. Consequently, the operator establishes and uses a guidance track for backing into position. He may use roof bolts or tire tracks, chalk marks on the roof, or other indirect guidance devices. In cases where the equipment is powered by diesels, the torque convertor power train noise does not vary sufficiently during movement (vis a vis idling) to provide an indirect audible signal to nearby miners that the equipment is moving. As emphasized before, the operator cannot see the path of the vehicle. Such vehicles constitute a serious safety hazard to miners who may cross or use the path of the vehicle. Such equipment should be equipped with backup warning devices clearly audible to other miners. The warning signal must be easily discernable from the noise background and must be given in sufficient time (e.g., 3-5 seconds) to permit miners to clear the path prior to any movement of the vehicle.

During the course of a work shift, section members cover most of the section area. Each miner observes hazardous situations in the performance of his job. If the hazards warrant it, the are pointed out to the foreman or others. However, under the best of conditions, communications are inadequate within section crews. Communications between sections and shifts are even worse. Thus, hazards observed by one miner or section may not be transmitted to others. A systematic method is needed to readily mark danger areas that he observes. The markers must be highly visable and perhaps be in two stages (e.g., a "caution" and a "Stop" marker). One possibility would be fluorescent coated tags attached to a roof bolt in the immediate vicinity. Alternatively, use of fluorescent paint spray on roof or walls in an area considered potentially hazardous might be feasible. Such a system would serve to warn all others of possible or known hazards to safety.

Cap lamps are universally utilized by miners underground. The lamp is attached to the hardhat and a cable runs to the battery pack carried on the belt. Many miners route the cable in such a way that it runs under the arm and over the shoulder keeping it in close contact with the body. Others permit the cable to hang loosely from the hardhat where it could be snagged by passing equipment or other obstructions. If this occurs, the cord has sufficient strength to pull a man into operating equipment or other hazards. This piece of equipment could be re-designed to provide a feature which would quickly disconnect the cord if caught, but would resist unwanted disconnections.

During the normal shift, the foreman must continuously maintain a knowledge of the status of each function and manage all major activities of his section's mining assignment. With poor verbal communications and visibility, determining the status of each function requires that he continuously circulate through the section checking on job activity problems and the status of supporting activities and supplies. It is believed necessary that section foremen be given some type of simple, effective and durable status display system to assist him in organizing and managing section activities, including contingency actions. The foreman's status display system should provide for real time or near real time updating of the status of section functions and major environmental conditions. Alternatives for implementing such a concept could range from simple mechanical check list style displays to electronic systems. Availability of such a system could greatly decrease the variability of safety and job performance among section foremen for routine, contingency, and emergency procedures and activities.

The foreman's job is the most difficult job on a section. Foremen work an eight hour shift with his crew underground, but following the shift, reports and other administrative demands require that he be at the mine for several hours per week beyond the regular shift. During the shift change, foremen have only a few minutes to transmit status information on problems or hazards observed during the previous shift to the foreman of the incoming shift. Availability of the foreman status display discussed above would also improve the exchange of status information between foremen working different shifts in the same section.

Bridge conveyor systems offer numerous productivity benefits when properly utilized. However, they have limited maneuverability in the limited space of the mining environment. Because of maneuvering problems, bridge equipment operators must work very close to the ribs. The operating position on the bridge carrier provides no protection to the operator. In fact, an improper maneuver could easily cause the operator to be pinned against the ribs. Some systems provide controls on each side of the carrier to permit the operator to select the control position least exposed. Unfortunately, during operation, many operators will not take the time to climb over the bridge to operate the unit from a safer position. Some situations would require the operator to move his control position several times during a place change. This equipment should be analyzed to determine ways of reducing the hazardous exposure of the operator. In addition, during advance mining, the bridge conveyor's position behind the continuous miner causes return air from the face to pass over the bridge carrier operator. In most cases, this return air is extremely dusty. A system or technique is needed to exhaust return air, utilizing tubing or other devices to eliminate this dust problem for bridge carrier operators.

Mining equipment, in general, is poorly designed from a human factors standpoint. Control and display arrangements are poorly positioned, lighting is poor and visibility may be very limited on some equipment. Operation of many types of equipment requires considerable training and some have little tolerance for operator error or misjudgment. Complete discussion of this subject is beyond the scope of this study. However, some general comments are believed necessary.

Mining equipment should be designed with more consideration for the environmental conditions in which operators must operate the equipment. (i.e., limited visibility, noise, dust, vibration, etc.) Near future and future underground equipment should be designed to:

- minimize operator training (for regular and substitute operators),
- increase visibility (and general illumination of work areas),
- increase maneuverability in narrow clearance areas (both to decrease the risk of damage to the machine and to reduce safety risks),
- increase the tolerance for expected operator errors, and
- increase operator comfort and safety (including such considerations as remote control and canopies).

If such human factors design criteria were included in new mining equipment, improvements could be expected in effectiveness, safety, and availability.

4. NONFATAL ACCIDENT ANALYSIS

Nonfatal accident data was reviewed to determine its applicability for use in the safety analysis of retreat mining. Criteria was established to determine its applicability. It was desirable for the data to:

- be representative of the same period (1966-1973) as the fatal accident data base,
- utilize the same coding structure as the fatal base,
- be sorted to extract nonfatals in retreat mining only, and
- have a means by which severity could be determined.

Meeting with HSAC representatives established that all of these criteria were not possible. The nonfatal accident data base was newly established, hence data was not available for the entire period of this study. Nonfatal accidents were not investigated thoroughly in this study, since the coding structure was not sufficiently detailed and did not always directly correspond to the fatal data base. The structure would allow the data base to be sorted on retreat mining only and did have a means of determining severity. This data base was not computerized for most convenient access. Thus, it was determined that the benefits of a parallel analysis did not outweigh the costs to analyze the large volume of data. It would be beneficial, however, for HSAC representatives to follow up recommendations of the fatal accident data study with analysis of similar nonfatal data over the period 1972-1975. This could serve to support the conclusions of this study and establish new areas for research.

5. STATISTICAL ANALYSIS RESULTS

The following tables present the results of frequency distributions, analyses, and crosstabulations of mine conditions, mining methods, and work force variables. It should be noted that these results may appear to be duplications of data. This is because the analyses systematically compared each variable with numerous other variables in an effort to detect meaningful relationships and interactions in the data base. Frequently, more than one cross tabulation of roughly similar data is presented to insure that different standpoints and audiences are objectively represented.

(1) Mine Conditions

Tables and results in the immediately following section relate to:

- Roof Control Plans
  - Approval Effects
  - Regulations and Compliance
- Assessment of Local Conditions,
- Victim's Exposure, and
- Work Cycle.

Table III-4

FREQUENCY DISTRIBUTION

of

V303 - Maximum Width of Pillar Split - (in feet)

f	%	DESCRIPTION
68	70.10	Greater than Plan
25	25.77	Equal to Plan
4	4.12	Less than Plan

STATISTIC: none

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: none possible

COMMENTS: The analysis of variable V303 has shown that over seventy percent of the fatalities occurred when the split of the pillar exceeded that specified in the plan. Maximum safe split widths are a function of the characteristics of the coal, the coal overburden pressures, and the seam heights. (See Rock Mechanics Section.) Although the Coal Mine Health and Safety (CMH&S) Law specifies this distance should not exceed 20 feet (75.200-11 c) and that the distance should be approved (75.201-1 a), the above statistics and those shown for the actual distance of pillar splits indicate a high number of violations of the pillar split width or permitted deviations coincidental with fatal accidents.

CONSTRAINTS: Field survey sample size was insufficient for either normalization or assessment of the extent to which regulations are enforced.

PROBABLE CAUSES:

RECOMMENDATIONS: MESA continue enforcement efforts regarding violation of approved pillar split widths of openings. There is a payoff in reduction of fatalities by enforcement of CMH&S Laws. Specific emphasis should be placed on (75.201-1 b) which require additional support when there are violations. Also, mine operators should increase self enforcement of regulations by emphasis with foremen.

CROSSTABULATION
of
V031 - Mining Method in Effect at Accident Site or Section
vs
V303 - Maximum Width of Pillar Split

Pillar Split Widths	Mining Method			
	Conventional		Continuous	
	f	%	f	%
Less than or Equal to Plan	10	29.41	15	34.09
Greater than Plan	24	70.59	29	65.91

STATISTICS: none   STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: none possible

COMMENTS: In the time frame of this study the percentage of conventional mining has decreased from about 50% of underground mining in 1966 to about 35% in 1973. The percentage of mines using conventional methods is approximately 44%. This is about what would be expected if there were no difference between accidents resulting from violations using conventional vs continuous mining methods.

CONSTRAINTS: This table is presented primarily for information only as no normalization was possible.

PROBABLE CAUSES: See above.

RECOMMENDATIONS: Increase enforcement and self-enforcement (as previously noted.)

Table III-6

CROSSTABULATION
of
V011 Mining Method in Effect at Accident Site or Section
vs
V144 - Victim's Exposure Nature

	Victim's Exposure Nature	Mining Method: Conventional f	Conventional %	Continuous f	Continuous %
Unsupported	Unsupported Unnecessary	17	30.91	22	29.73
	Unsupported Necessary	4	7.27	9	12.16
Temporary	Temporary in Compliance	2	3.64	3	4.05
	Temporary Not in Compliance	9	16.36	9	12.16
	Setting Temporary	1	1.82	0	0.00
Permanent	Permanent in Compliance	8	14.55	21	28.38
	Permanent Not in Compliance	9	16.36	7	9.46
	No Adjacent Support	5	9.09	3	4.05
	Total Unsupported	21	42.00	31	43.66
	Total Temporary	12	24.00	12	16.90
	Total Permanent	17	34.00	28	39.44

STATISTIC: none   STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: MESA Respirable Dust File

COMMENTS: The percentage of underground face area miners employed in conventional vs. continuous methods has changed during the 1966 to 1973 period. Starting with about a 50 - 50 split at the beginning of this period, conventional mining has decreased to about 35% of the number of miners employed. This can be roughly considered exposure of miners to accident risks. If the "average" percentage of conventional mining for the period is about 40 to 41%, this corresponds to 41 to 42% of the fatal accidents occurring under all roof exposure conditions attributable to conventional mining. Thus, a difference attributable to mining method cannot be shown.

<u>CONSTRAINTS</u>: Data available for MESA Respirable Dust File for years 1971 to 1973 could provide only a rough normalization (see text).

<u>PROBABLE CAUSES</u>:

<u>RECOMMENDATIONS</u>: none

Table III-7

CROSSTABULATION
of
V323 Was Pillar Block Mined From Multiple Directions
vs
V144 - Victim's Exposure Nature

Victim's Exposure Nature	Was Pillar Mined From Multiple Directions							
	Yes	Proper Method	Yes	Improper Method	No	Improper Method	No	Proper Method
	f	%	f	%	f	%	f	%
Unsupported Unnecessary	3	75.00	4	25.00	24	30.00	2	50.00
Unsupported Necessary	0	0.00	1	6.25	9	11.25	1	25.00
Temporary in Compliance	0	0.00	3	18.75	15	18.75	1	25.00
Temporary Not in Compliance	1	25.00	4	25.00	9	11.25	0	0.00
Setting Temporary	0	0.00	0	0.00	4	5.00	0	0.00
Permanent in Compliance	0	0.00	3	18.75	15	18.75	0	0.00
Permanent Not in Compliance	0	0.00	0	0.00	1	1.25	0	0.00
No Adjacent Support	0	0.00	1	6.25	3	3.75	0	0.00

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None

COMMENTS: Only 8% of fatal accidents occurred when the pillar block was mined properly while 92% were associated with improper mining. 77% of fatalities occurred when mining was not from multiple directions and proper mining methods were not used. This must be considered a foreman and/or supervisory error.

CONSTRAINTS: There is no method for checking the accuracy of assessments of proper method vs improper method.

PROBABLE CAUSES: Poor self-enforcement of approved practices exist.

RECOMMENDATIONS: USBM should investigate the effects of improper mining methods (see Rock Mechanics section of this report). MESA and mine operators should improve enforcement and self-enforcement of proper and safe mining methods.

CROSSTABULATION
of
V124 - Type of Fall
vs
V146 - Failure Nature if Permanent or Temporary Support

Failure Nature	Type Fall: Roof Fall at Face f	Roof Fall at Face %	Roof Fall at Intersection f	Roof Fall at Intersection %
Self supporting	0	0.00	1	5.00
Above the anchor	21	25.93	3	15.00
At the anchor	3	3.70	0	0.00
Below the anchor	4	4.94	1	5.00
Spalling	2	2.47	2	10.00
At or inby the support line	40	49.38	11	55.00
Over permanent support	3	3.70	2	10.00
Over temporary support	6	7.41	0	0.00
Between wood	2	2.47	0	0.00

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: The large percentage of roof falls at or inby the support line is an important finding. The high frequency of such failures have caused some mining companies, on their own initiative, to rule that miners should not advance closer than one row of bolts behind the support line. Also significant is the large number of failures above the bolt anchor line (horizon). It is believed that these two findings provide ample evidence that miners have insufficient information to assess the true nature of the roof.

CONSTRAINTS: Two types of falls are summarized in this cross tabulation.

PROBABLE CAUSES: Insufficient information is available to assess the strength and characteristics of the immediate roof. Few approved roof control plans provide systematic provisions for controlling bad roof conditions. Usually, selecting the specific supplemental roof control provisions is left to individual foremen.

RECOMMENDATIONS: MESA should review regulations and enforcement concerning advancing to the support line and testing bolts to assess quality of anchors. USBM Research and Development should vigorously continue to develop roof sensors or other techniques to provide essential roof strength and flaw information for face crews.

CROSSTABULATION
of
V079 - Nature of Accident
vs
V146 - Failure Nature if Permanent or Temporary Support

	Nature of Accid.	
	Victim Crushed By Roof	
Failure Nature	f	%
Self Supporting	1	.78
Above the Anchor Point	26	20.16
At the Anchor Point	3	2.33
Below the Anchor Point	7	5.43
Spalling	5	3.88
At or inby the Support Line	64	49.61
Over Permanent Support	9	6.98
Over Temporary Support	9	6.98
Between Wood	5	3.88

STATISTIC: None

STATISTICAL SIGNIFICANT: N/A

NORMALIZATION: None Possible

COMMENTS: The important findings in this table concern the large number of failures at or inby the support line and above the anchor point on"permanently" supported roof.

CONSTRAINTS: It is believed that these findings reflect the inability of foremen, bolters, etc. to adequately assess the true nature of roof conditions. In effect, they cannot obtain sufficient information with current technology to decide what roof support is necessary.

PROBABLE CAUSES: There was insufficient information to assess roof and incomplete application of current state-of-the-art in rock mechanics.

RECOMMENDATIONS: USBM should give current research programs to develop roof sensor and/or roof assessment systems and procedures additional emphasis and priority. As an example, if systematic core sampling was done as development progresses, a strata map could be prepared which may be useful as assessment and prediction of roof conditions. MESA should reconsider (at least temporarily)

Table III-9

the regulations permitting mining up to the last line of support. Perhaps the second row of bolts should be considered safe roof.

Table III-10

FREQUENCY DISTRIBUTION

of

V124 - Type of Fall

f	%	Type of Fall
106	48.40	Roof Face/Retreat Area
25	11.40	Roof Intersection Area
14	6.40	Roof - Other
74	33.80	All Others

STATISTIC: none

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: Calculated roof areas in typical section

COMMENTS: There were more than four times the number of roof fall fatalities which occurred in the face area than in the next most dangerous area--intersections. In a typical section, there is at least 5 times the roof area above intersections than above face areas. Thus, the risk is about 20 times greater at the face than in intersections. There is about 3 times more roof area in rooms and crosscuts than in intersections. Thus, there is more than 5 times the risk of roof fall in intersections than in other areas (not considering exposure time).

CONSTRAINTS: none

PROBABLE CAUSES: Insufficient information to assess true roof conditions and poor mine safety practices (in some cases) in allowing the roof to remain unsupported too long (see discussion elsewhere).

RECOMMENDATIONS: MESA should point out the differences in risks and the reasons for the differences (discussed elsewhere). In addition, the USBM should continue research into reasons for differences and continue development of roof sensors and detection systems perhaps for these specific areas.

CROSSTABULATION
of
V[illegible] Mining Method in Effect at Accident Site or Section
vs
V126 - Immediate Roof Composition

Immediate Roof	Mining Method: Conventional f	Conventional %	Continuous f	Continuous %
Laminated Shale	28	54.90	17	25.37
Fractured Shale	3	5.88	13	19.40
Draw Rock	8	15.69	5	7.46
Bone Coal	1	1.96	10	14.93
Sandstone	5	9.80	5	7.46
Coal (Head Coal)	1	1.96	15	22.39
Sandstone and Shale	5	9.80	2	2.99

STATISTIC: None STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None available

COMMENTS: The most important feature of these findings is the differences between roof fall fatalities in laminated shale conditions for conventional vs. continuous mining. Such differences probably must be attributed to basic differences in the mining method. For example, it is possible that shock to the immediate roof caused by blasting may more adversely affect laminated shale than other types of roof. Similarly the large percentage of fatalities using continuous mining where head coal is left at the roof may be a result of greater difficulties in judging how much head coal is left to prevent weathering of shale roofs.

CONSTRAINTS: Conventional mining was used in approximately 15% to 20% of retreat sections during the period of the study (i.e. 1966-1973), hence varies within the data samples.

PROBABLE CAUSES: Unknown

RECOMMENDATIONS: MESA should further investigate these problems to identify the nature and resolution of any differences in safety inherent in each mining method. Also MESA should investigate the high incidence of fatal accidents associated with conventional mining under a laminated shale roof.

Table III-12

CROSSTABULATION
of
V126 - Immediate Roof Composition
vs
V129 - If Applicable the Type of Bad Roof Condition Present
That Contributed to the Accident

Immediate Roof	Type Bad Roof Condition							
	Clay Veins f	Slips and Faults f	Horsebacks & Kettle bottoms f	Cracks f	Slickensides f	Roof Changes f	Previous Falls f	Drawrock f
Fireclay	0	0	0	1	0	0	0	0
Laminated Shale	1	8	1	6	1	1	3	0
Fractured Shale	0	4	1	1	1	0	0	0
Drawrock	0	0	0	0	0	0	0	0
Bonecoal	2	2	0	1	1	0	0	0
Sandstone	0	3	0	0	0	0	0	1
Sandstone/Shale	0	1	1	0	0	2	1	0
Coal (head coal)	2	2	0	1	1	0	1	0

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: This crosstabulation, while not statistically analyzed, may point up an information problem. As discussed elsewhere, it is believed that miners have insufficient information to assess the true nature of the roof. It appears from this chart that the nature of laminated shale in the immediate roof increases the problems of identifying slips, faults, and cracks in the roof. This would make the use of roof sensors even more important in such roof conditions. (also see page III-30).

CONSTRAINTS: none

PROBABLE CAUSES: Insufficient information.

RECOMMENDATIONS: USBM should continue development and application of roof sensor and/or roof assessment systems.

CROSSTABULATION
of
V126 - Immediate Roof Composition
vs
V146 - Failure Nature if Permanent or Temporary Support

	Failure Nature																	
	Above Anchor		At Anchor		Below Anchor		Spall		At or inby Support line		Over Perm. Support		Over Temp. Support		Between Wood		Totals	
Immediate Roof	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%
Laminated Shale	11	42.00	0	0.00	1	16.67	0	0.00	23	38.00	3	42.86	4	57.14	3	100.00	45	39.00
Fractured Shale	2	8.00	0	0.00	2	33.33	0	0.00	11	18.00	1	14.29	1	14.29	0	0.00	17	15.00
Drawrock	3	12.00	0	0.00	1	16.67	2	50.00	6	10.00	1	14.29	0	0.00	0	0.00	13	11.00
Bonecoal	2	8.00	0	0.00	0	0.00	1	25.00	6	10.00	1	14.29	1	14.29	0	0.00	11	9.00
Sandstone	2	8.00	0	0.00	1	16.67	0	0.00	5	8.00	0	0.00	1	14.29	0	0.00	9	8.00
Sandstone & Shale	1	4.00	1	50.00	0	0.00	1	25.00	3	5.00	1	14.29	0	0.00	0	0.00	7	6.00
Coal (Hard Coal)	5	19.00	1	50.00	1	16.67	0	0.00	6	10.00	0	0.00	0	0.00	0	0.00	13	11.00

Table III-13

Table III-14

CROSSTABULATION
of
V124 - Type of Fall
vs
V126 - Immediate Roof Composition

Immediate Roof	Type Fall: Roof Fall at Face f	Type Fall: Roof Fall at Face %	Type Fall: Roof Fall at Intersection f	Type Fall: Roof Fall at Intersection %
Fire clay	2	2.11	0	0.00
Laminated shale	37	38.95	8	36.36
Fractured shale	11	11.58	2	9.09
Drawrock	11	11.58	2	9.09
Bone coal	6	6.32	3	13.64
Sandstone	6	6.32	3	13.64
Shale & Sandstone	7	7.37	2	9.09
Coal (head coal)	15	15.79	2	9.09

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: A review of this table indicates, as expected, that problems with laminated shale are not limited to the face area. Not surprising, perhaps, is that sandstone roof, thought by many to be a safe roof, is substantially safer than several others in these results.

CONSTRAINTS: This table is presented for information since no normalization data was available.

PROBABLE CAUSES: Insufficient information to assess roof conditions (as previously discussed).

RECOMMENDATIONS: See previous recommendations.

Table III-14

CROSSTABULATION
of
V124 - Type of Fall
vs
V126 - Immediate Roof Composition

Immediate Roof	Type Fall: Roof Fall at Face f	Type Fall: Roof Fall at Face %	Type Fall: Roof Fall at Intersection f	Type Fall: Roof Fall at Intersection %
Fire clay	2	2.11	0	0.00
Laminated shale	37	38.95	8	36.36
Fractured shale	11	11.58	2	9.09
Drawrock	11	11.58	2	9.09
Bone coal	6	6.32	3	13.64
Sandstone	6	6.32	3	13.64
Shale & Sandstone	7	7.37	2	9.09
Coal (head coal)	15	15.79	2	9.09

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: A review of this table indicates, as expected, that problems with laminated shale are not limited to the face area. Not surprising, perhaps, is that sandstone roof, thought by many to be a safe roof, is substantially safer than several others in these results.

CONSTRAINTS: This table is presented for information since no normalization data was available.

PROBABLE CAUSES: Insufficient information to assess roof conditions (as previously discussed).

RECOMMENDATIONS: See previous recommendations.

CROSSTABULATION
of
V079 - Nature of Accident
vs
V146 - Failure Nature if Permanent or Temporary Support

	Nature of Accid.	
	Victim Crushed By Roof	
Failure Nature	f	%
Self Supporting	1	.78
Above the Anchor Point	26	20.16
At the Anchor Point	3	2.33
Below the Anchor Point	7	5.43
Spalling	5	3.88
At the Support Line	64	49.61
Over Permanent Support	9	6.98
Over Temporary Support	9	6.98
Between Wood	5	3.88

STATISTIC: None STATISTICAL SIGNIFICANT: N/A

NORMALIZATION: None Possible

COMMENTS: The important findings in this table concern the large number of failures at or inby the support line and above the anchor point on "permanently" supported roof.

CONSTRAINTS: It is believed that these findings reflect the inability of foremen, bolters, etc. to adequately assess the true nature of roof conditions. In effect, they cannot obtain sufficient information with current technology to decide what roof support is necessary.

PROBABLE CAUSES: There was insufficient information to assess roof and incomplete application of current state-of-the-art in rock mechanics.

RECOMMENDATIONS: USBM should give current research programs to develop roof sensor and/or roof assessment systems and procedures additional emphasis and priority. As an example, if systematic core sampling was done as development progresses, a strata map could be prepared which may be useful as assessment and prediction of roof conditions. MESA should reconsider (at least temporarily)

CROSSTABULATION
of
V124 - Type of Fall
vs
V129 - Type of Bad Roof Condition Present

Bad Roof Condition	Type Fall: Roof Fall at Face f	Roof Fall at Face %	Roof Fall at Intersection f	Roof Fall at Intersection %
Clay vein	4	10.26	0	0.00
Slips or Faults	14	35.90	1	11.11
Horsebacks	1	2.56	3	33.33
Cracks	8	20.51	0	0.00
Slickensides	3	7.69	1	11.11
Roof changes	2	5.13	1	11.11
Previous fall	3	7.69	0	0.00
Drawrock	4	10.26	3	33.33

STATISTIC: None STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: This is another aspect of the problems of insufficient information to assess the true condition of the roof. Some slips or faults can be difficult to detect under the environmental lighting conditions underground.

CONSTRAINTS: No normalization was possible on this, but refer to the discussion elsewhere in this report on Rock Mechanics and the strengths of roof with example defects.

PROBABLE CAUSES: Insufficient information to assess the roof, difficulties in detecting some types of conditions, and lack of systematic provisions in roof control plans for controlling various bad conditions encountered (see page III-30).

RECOMMENDATIONS: See previous discussions.

the regulations permitting mining up to the last line of support. Perhaps the second row of bolts should be considered safe roof. (See discussion of Rock Mechanics, Chapter VII). Another example of a current research program which should be given increased emphasis is the USBM project entitled Development of Technology for Design and Evaluation of Roof Control Plans being conducted at Spokane.

CROSSTABULATION
of
V129 - If Applicable the Type of Bad Roof Condition Present That Contributed to the Accident
vs
V214 - Type of Permanent Support Used

Type of Bad Roof Condition	Type of Permanent Support Used						
	Bolts Only f	Bolts & Crossbars f	Bolts & Posts f	Bolts, Posts & Crossbars f	Rib Posts f	Temporary Posts f	Breaker Posts f
Clay Vein	0	0	1	1	1	0	0
Slip & Fault	0	3	3	0	4	1	2
Other/Combination	1	0	5	2	0	0	0
Horsebacks & Kettlebottoms	1	0	0	0	0	1	0
Cracks	2	0	4	1	0	0	0
Slickensides	0	0	1	0	1	0	0
Roof Changes	1	0	0	1	0	0	0
Previous Fall	1	0	1	0	0	1	0
Draw Rock	0	0	0	0	2	0	0

STATISTIC: none

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: none possible

COMMENTS: As noted, no statistical testing was possible on this table. However, perhaps what is most interesting are the combinations of bad roof condition and type support used where no fatal accidents have occurred. It is possible that this indicates a quasi effectiveness in controlling bad conditions. Furthermore, bad roof features easily detectable and correctable with supplemental permanent supports do not appear to result in fatalities. On the other hand slips and faults and cracks do not appear to be reliably correctable. It is also possible that the true nature of the roof condition cannot reliably be determined as a guide to selection and installation of effective supplemental roof support.

CONSTRAINTS: Sample sizes were insufficient for statistical tests

PROBABLE CAUSES: There is no systematic method for correcting such conditions in approved roof control plans.

RECOMMENDATIONS: USBM expand this type of investigation to consider non-fatal accidents and unintentional roof falls in an effort to determine the most effective roof support for various bad roof conditions as well as improved systems to detect poor roof conditions.

FREQUENCY DISTRIBUTION

of

V144 - Victim's Exposure Nature

f	%	DESCRIPTION
49	31.01	Under unsupported unnecessarily (e.g., under unsupported roof to mine out a coal stump in retreat mining.)
31	19.62	Under permanent support in compliance with roof control plan.
22	13.92	Within temporary support not set in compliance to plan (too few posts).
18	11.39	Under permanent support which did not comply with roof control plan.
17	10.76	Under unsupported necessarily (set temporary support, test roof).
11	6.96	Under support in minimal compliance with plan but not adjusted for faulty conditions present.
6	3.80	Within temporary support set in compliance with plan.
4	2.53	In act of setting temporary support -- not fully protected yet.

STATISTIC: none   STATISTICALLY SIGNIFICANT: N/A

NORMALIZATION: none possible

COMMENTS: These findings reaffirm well known empirical knowledge regarding the dangers of venturing under unsupported roof. This problem points up a major advantage of remote control mining equipment. Only about 23% of roof fall fatalities occurred where there was not some type of procedural error on the part of the victim.

CONSTRAINTS: Normalization was not possible for this analysis. However, previous estimates of hazardous exposure are used in later analysis of this section.

PROBABLE CAUSES: Poor training and poor enforcement of safe working procedure.

RECOMMENDATIONS: MESA continue to inform miners of the extent of risks for such procedural errors. USBM continue development and support of remote control continuous miners, loaders, and roof bolters as well as methods of setting temporary supports and gas testing safely.

Table III-19

CROSSTABULATION
of
V124 - Type of Fall
vs
V144 - Victim's Exposure Nature

	Victim's Exposure Nature	Type of Fall: Roof fall at Face f	%
Unsupported	Unsupported Unnecessary	31	30.69
	Unsupported Necessary	11	10.89
Temporary	Temporary in Compliance	4	3.96
	Temporary Not in Compliance	17	16.83
	Setting Temporary	2	1.98
Permanent	Permanent in Compliance	20	19.80
	Permanent Not in Compliance	11	10.89
	No Adjacent Support	5	4.95
	Total Unsupported	42	43.75
	Total Temporary	23	23.96
	Total Permanent	31	32.29

STATISTIC: none

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: none possible

COMMENTS: The relatively large percentage of fatal accidents occurring under permanent supported roof installed in compliance with MESA approved plans again point up the need for roof sensors. Without sensors (e.g., microseismic) or some other means, roof bolters and foremen have incomplete information to assess actual roof conditions. It should also be noted that accidents occurring under unsupported roof unnecessarily and setting temporary supports not in compliance are two areas of considerable potential for reduction of fatalities by changes in regulations or procedure or by better enforcement of current regulations.

CONSTRAINTS: No normalization was possible for this analysis. In addition, other types of fall were eliminated from this analysis due to insufficient sample size.

PROBABLE CAUSES: Insufficient self enforcement of safe work practices by operators and insufficient information to assess the true nature of the roof.

RECOMMENDATIONS: MESA continue enforcement as previously noted. USBM continue development of remote control equipment and safety techniques for setting temporary supports, gas tests, etc.

CROSSTABULATION
of
V031 - Mining Method in Effect at Accident Site or Section
vs
V144 - Victim's Exposure Nature

	Victim's Exposure Nature	Mining Method: Conventional f	Conventional %	Continuous f	Continuous %
Unsupported	Unsupported Unnecessary	17	30.91	22	29.73
	Unsupported Necessary	4	7.27	9	12.16
Temporary	Temporary in Compliance	2	3.64	3	4.05
	Temporary Not in Compliance	9	16.36	9	12.16
	Setting Temporary	1	1.82	0	0
Permanent	Permanent in Compliance	8	14.55	21	28.38
	Permanent Not in Compliance	9	16.36	7	9.46
	No Adjacent Support	5	9.09	3	4.05
	Total Unsupported	21	42.00	31	43.66
	Total Temporary	12	24.00	12	16.90
	Total Permanent	17	34.00	28	39.44

STATISTIC: none

STATISTICALLY SIGNIFICANT: N/A

NORMALIZATION: MESA Respirable Dust File

COMMENTS: The percentage of underground face area miners employed in conventional versus continuous methods has changed during the 1966 to 1973 period. Starting with about a 50 - 50 split at the beginning of this period, conventional mining has decreased to about 35% of the number of miners employed. This can be roughly considered exposure of miners to accident risks. If the "average" percentage of conventional mining for the period is about 40 to 41%, this corresponds to 41 to 42% of the fatal accidents occurring under all roof exposure conditions attributable to conventional mining. Thus, a difference attributable to mining method cannot be shown.

CONSTRAINTS: Normalization data from MESA Respirable Dust File is available only for years 1971 to 1973 and the tabulations obtained could not be used to determine actual percentages of conventional versus continuous methods for retreat mining only. (See page III-14)

PROBABLE CAUSES:

RECOMMENDATIONS: none

CROSSTABULATION
of
V039 - During Which Work Cycle Did Accident Occur?
vs
V144- Victim's Exposure Nature

Work Cycle	Victim's Exposure Nature																					
	Unsupported						Temporary								Permanent							
	Unnecessary		Necessary		Total		In Compliance		Not in Compliance		Setting		Total		In Compliance		Not in Compliance		Total		No Adjacent Support	
	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%
Loading	9	25.7	4	36.4	13	28.3	3	60.0	8	40.0	0	0.0	11	37.9	4	14.8	5	33.3	9	21.4	3	37.5
Undercutting	4	11.4	0	0.0	4	8.7	0	0.0	2	10.0	1	25.0	3	10.3	0	0.0	1	6.7	1	2.4	1	12.5
Face Drilling	1	2.9	0	0.0	1	2.2	0	0.0	2	10.0	0	0.0	2	6.9	1	3.7	0	0.0	1	2.4	0	0.0
Face Shooting	2	5.7	1	9.1	3	6.5	0	0.0	1	5.0	0	0.0	1	3.5	1	3.7	4	26.7	5	11.9	1	12.5
Bolting and Support	13	37.1	2	18.2	15	32.6	1	20.0	2	10.0	2	50.0	5	17.2	5	18.5	1	6.7	6	14.3	0	0.0
Continuous Mining	6	17.1	4	36.4	10	21.7	1	20.0	5	25.0	1	25.0	7	24.1	16	59.3	4	26.7	20	47.6	3	37.5

Table III-21

STATISTIC: none

STATISTICALLY SIGNIFICANT: N/A

NORMALIZATION:

COMMENTS: The fatal accidents occurring under unsupported roof while loading and bolting probably can be attributed to improper work methods. (also see Table III-35) Of equal interest are the fatalities under permanent roof support installed in compliance with an approved roof control plan. It is possible that mechanical-seismic disturbances of the continuous mining machine are associated with this problem.

CONSTRAINTS: Accurate normalization was not possible, however, approximations of exposure times from the literature are discussed in Section V

PROBABLE CAUSES: Improper work methods, roof problems at the support line, and mechanical-seismic disturbances of continuous mining machine.

CROSSTABULATION
of
V124 - Type of Fall
vs
V312 - Length of Time Roof Unsupported (in hours & fractions)

Time Unsupported (Hours)	Type Fall: Roof Fall at Face f	Roof Fall at Face %	Roof Fall at Intersection f	Roof Fall at Intersection %
1	9	13.85	2	14.29
2	12	18.46	0	0.00
3	8	12.31	2	14.29
4	14	21.54	3	21.43
5	10	15.38	3	21.43
6	6	9.23	2	14.29
7	6	9.23	1	7.14
8	0	0.00	1	7.14

STATISTIC: none

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: none possible

COMMENTS: Roof falls at the face are relatively uniformly distributed over hours unsupported. (Roof support is defined as any type of permanent or temporary support.)

CONSTRAINTS: Normalization was not possible. However, logically, mine sections and operators would be decreasingly likely to allow roof to be unsupported for extended periods of time. Thus, the population at the 6-8 hour intervals would probably be substantially less making the risk factors higher than would be assumed by the raw frequency numbers.

PROBABLE CAUSES: Failure of mine operators to enforce strict adherence to current MESA regulations as well as good mining practice. If as suspected, few operators permit such poor practice, the risk of fatal accident to the offending operator and crews may be increasing rapidly during the time the roof is left unsupported.

RECOMMENDATIONS: MESA take a hard line on leaving roof unsupported because of the deterioration which occurs with time, deterioration which is irreversible.

(2) Mining Methods

Frequency distributions, analyses, and crosstabulations in the next section relate to:

- Conventional vs Continuous,
- Seam Height Factors, and
- Pillar Block Factors.

Table III-23

ANALYSIS
of
V011 - Mining Method(s) Used to Produce Coal Throughout Mine
Normalized by Annual Output

Mining Method	Fatals		Output 1966			Output 1970		
	Frequency	Percentage	Million tons	Percentage	Million tons/ Total Accidents	Million tons	Percentage	Million tons/ Total Accidents
Continuous	78	53	152.8	50	1.96	187.0	55	2.40
Conventional	70	47	152.7	50	2.18	155.4	45	2.22

STATISTIC: none

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: 1974 Keystone Coal Industry Manual (1973 data tons/year and men/mine) and MESA Respirable Dust File data: number of men employed by job classification for continuous vs conventional mining and for retreat vs advance mining.

COMMENTS: Based upon the shifting trends in mining method indicated by the normalization data, it would be difficult to say with confidence that there was a difference in fatal accidents due to mining method since the proportions of fatal accidents attributable to each appears to be closely related to both the output annual tons and the number of men employed in each mining method.

CONSTRAINTS: Annual tons output in retreat mining only was not available, hence output is for both development and retreat mining. However, using the MESA HSAC Respirable Dust File data and Job Classifications which are uniquely conventional (see summary below and page III-14) it was estimated that roughly 16% of retreat mining was by conventional methods in 1971 If that estimate is reasonably accurate, then conventional mining methods were associated with more than twice the number of fatal accidents than would be expected for 16% of the men employed (i.e., 24 expected versus 70 actual).

Number of Miners - (Used only in Conventional Mining)

Development and Retreat	Total 1971	Retreat only 1971	
Shotfire	1389	213	15.3%
Coal drill operator	1336	196	14.7%
Cutting machine operator	4453	824	18.5%

PROBABLE CAUSES: May be due to greater inherent hazardous exposure (See Table III-35).

RECOMMENDATIONS: none

ANALYSIS
of
V006 - Average Thickness of Coalbed (Inches)

Seam Height (Inches)	Fatals		Output			Exposure		
	Frequency	Percentage	Tons/ Man Year	Percentage	Frequency/ Output	Face Workers per Class Interval	Percentage	Frequency/ Exposure
0-36	20	9	2953	10	.0068	1474	6.4	.014
37-48	47	22	3532	13	.0133	5369	23.4	.009
49-60	49	22	4569	16	.0107	4555	19.9	.011
61-72	37	17	4840	17	.0076	4999	21.8	.007
73-84	26	12	4356	15	.0060	1164	5.1	.022
85-96	20	9	4078	14	.0049	2663	11.6	.008
97-	20	9	3845	14	.0052	2678	11.7	.007

Expected Fatalities
As a Function of
Average Thickness of Coalbed (Inches)
Normalized for Production Output & Exposure of Face Workers in Mine (# of Men)

Seam Height (Inches)	Observed Frequency	Expected Output	Expected Exposure
0-36	20	22	14
37-48	47	28	51
49-60	49	35	44
61-72	37	37	48
73-84	26	33	11
85-96	20	31	25
97-	20	31	26

STATISTIC: Chi Square Evaluation

Output, $\chi^2 = 27.97$
df = 6

Exposure, $\chi^2 = 28.81$
df = 6

STATISTICALLY SIGNIFICANT: Yes, however, caution should be used in interpreting this result since the exposure data in the 73-84" interval may be suspect.

NORMALIZATION: Output tons/man year source: 1974 Keystone Coal Industry Manual, Number men/mine seam height interval source: Sample from 1974 Keystone Coal Industry Manual. This is considered equal to exposure of men underground.

COMMENTS: Small seam thickness mines show a higher than expected frequency of fatal accidents in retreat mining in terms of output and exposure of miners in the underground environment.

CONSTRAINTS: Normalized data from 1974 only. Data base is from 1966-1973. During this period, more small mines existed. This tends to further support the conclusions.

Ratio of face to support personnel is different in small mines vs large mines, i.e., there are fewer support personnel in small mines. Thus, part time support functions would be assumed by face crews, reducing their exposure at the face. In addition, mines with less than 48 inches seam height often are not able to achieve full extraction of pillars in retreat due to mining problems of maneuvering equipment to obtain full extraction. This means that a greater proportion of the output of such mines was obtained in development rather than retreat mining. Therefore, our conclusions are conservative.

PROBABLE CAUSE: Shoestring operations, lower capital investment; poorer training in small mines; difficulties in movement; greater room spans; and poor equipment.

RECOMMENDATIONS: MESA forms increased enforcement efforts on smaller (low seam) mines and USBM pursue new technology for pillar extraction safely at smaller seam thickness.

CROSSTABULATION
of
V006 - Average Thickness of Coalbed (Inches)
vs
V011 - Mining Method (s) Used to Produce Coal Throughout Mine

	Mining Method			
	Conventional		Continuous	
Seam Height	f	%	f	%
0 - 36	4	5.71	1	0.95
37 - 48	20	28.57	15	14.29
49 - 60	24	34.29	19	18.10
61 - 72	8	11.43	25	23.81
73 - 84	4	5.71	20	19.05
85 - 96	1	1.43	16	15.24
97 - >	9	12.86	9	8.57

Frequency Histogram
of
Fatal Accidents by Seam Height and Mining Method

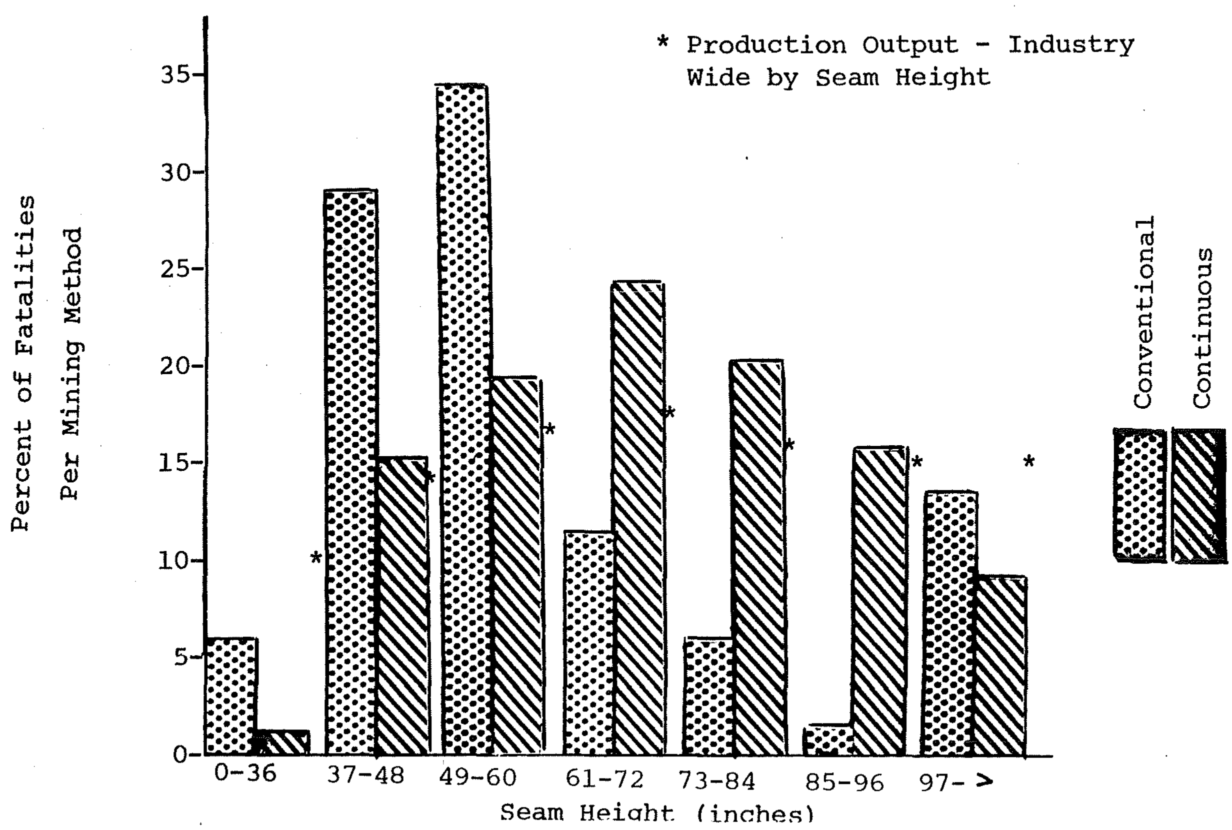


Table III-25

STATISTIC: none

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: n/a

COMMENTS: Chi square analysis of these results indicated significant differences in fatal accident frequencies between mining methods normalized by output and miner exposure, industry wide without respect to mining method. Further analysis using MESA Dust File data indicates, in terms of men employed, conventional mining over the time period of the study ranged from 50% to 35% of the mining (years 1966 through 1973 respectively). Conventional mining methods were being employed in 40% of the fatal accident occurrences. However, because appropriate normalization data could not be obtained, these results should be viewed with caution.

CONSTRAINTS: Normalization was not possible because industry data in the Keystone Coal Industry Manual was not desegregated by conventional versus continuous mining nor development versus retreat mining.

PROBABLE CAUSES: n/a

RECOMMENDATIONS: None

CROSSTABULATION
of
V006 - Average Thickness of Coalbed (inches)
vs
V029 - General Classification of Accident Type

Seam Height (Inches)	Type Accident: Roof Fall f	Roof Fall %	Rib/Face Fall f	Rib/Face Fall %	Machine f	Machine %	Haulage f	Haulage %
0-36	15	9.38	1	4.76	2	15.38	1	5.26
37-48	41	25.60	1	4.76	2	15.38	2	10.53
49-60	37	23.10	6	28.57	3	23.08	2	10.53
61-72	23	14.38	2	9.52	2	15.38	7	36.84
73-84	15	9.38	4	19.05	1	7.69	6	31.58
85-96	15	9.38	2	9.52	3	23.08	0	0.00
97 -	14	8.75	5	23.81	0	0.00	1	5.26

STATISTIC: Chi Square Evaluation   STATISTICALLY SIGNIFICANT: Yes

NORMALIZATION: 1974 Keystone Coal Industry Manual sample.

COMMENTS: Mines in low seam coal (i.e., less than 5 feet) have a significantly higher number of fatal roof fall accidents than could be accounted for by either their proportion of either annual tonnage or number of men employed in such mines.

CONSTRAINTS: Fatal rib face falls, machine and haulage accidents had too few observations for Chi Square analysis and were deleted.

Roof-fall Analysis

Mine Seam Height-Inches	Actual Fatalities: Roof Fall Fatalities	Expected Fatilities: Expected : Tonnage basis	Expected Fatilities: Expected: Exposure basis
0-36	15	7.4	10.2
37-48	41	32.2	37.4
49-60	37	35.2	31.8
61-72	23	40.9	34.9
73-84	15	8.6	8.1
85-96	15	18.4	18.6
97	14	17.4	18.7

CROSSTABULATION
of
Three Work Cycle Variables
vs
Seam Height, Normalized for Output and Exposure

Seam Height	Loading			Bolting & Support			Continuous Mining			Normalization	
		Expected Ratios			Expected Ratios			Expected Ratios			
	Actual	Output	Exposure	Actual	Output	Exposure	Actual	Output	Exposure	Output	Exposure
0-36	5	4.6	3.0	1	2.2	1.4	5	5.8	3.7	.10	.064
37-48	13	6.0	10.7	7	2.9	5.1	11	7.5	13.6	.13	.234
49-60	10	7.4	9.1	4	3.5	4.4	12	9.3	11.5	.16	.199
61-72	10	7.8	10.0	1	3.7	4.8	10	9.8	12.6	.17	.218
73-84	1	6.9	2.3	4	3.3	1.1	6	8.7	2.9	.15	.051
85-96	2	6.4	5.3	4	3.1	2.5	8	8.1	6.7	.14	.116
97 -	5	6.4	5.4	1	3.1	2.5	6	8.1	6.7	.14	.117

STATISTIC: Chi Square Evaluation

STATISTICALLY SIGNIFICANT: no

Table of Chi Square Results

	Loading		Continuous Mining	
Exposure	$\chi^2 = 3.90$	df = 4	$\chi^2 = 5.17$	df = 6
Output	$\chi^2 = 16.68$	df = 4	$\chi^2 = 3.94$	df = 6

NORMALIZATION: n/a

Table III-27

χ^2 Tonnage = 24.19 df = 6

Reject Hypothesis at significance level .99

χ^2 Exposure = 15.27 df = 6

Reject Hypothesis significance level .975

For the other categories Chi Square Analysis could not be performed, the expected (E)* number of fatalities normalized for exposure of miners for deleted types of accidents are compared below:

Other Fatal Accident Categories

Seam Height	Rib/Face Fall		Machine		Haulage	
	f	E*	f	E*	f	E*
0-36	1	1.3	2	0.8	1	1.2
37-48	1	4.9	2	3.0	2	4.4
49-60	6	4.2	3	2.6	2	3.8
61-72	2	4.6	2	2.8	7	4.1
73-84	4	1.1	1	0.7	6	1.0
85-96	2	2.4	3	1.5	0	2.2
97	5	2.5	0	1.5	1	2.2

Review of the normalized expected fatalities table above indicates the following:

1. Smaller mines in low seam coal appear to have fewer rib/face falls and haulage fatalities.

2. Higher seam mines appear to have a greater number of rib/face falls fatalities than would be expected by their exposure of miners in the mines. Conversely, higher seam mines appear to have fewer haulage accidents.

3. Mines in the 5-7 foot seam height appear to have somewhat higher numbers of accidents than would be expected.

<u>PROBABLE CAUSES</u>: Greater room spans in small mines may have an effect, as well as poor training in small mines.

<u>RECOMMENDATIONS</u>: MESA should investigate the differences in lower seam height mines and smaller mines which may account for such a finding and apply corrective action in the form of a different regulation and/or increased enforcement. MESA should review regulations to determine if differences are required to acknowledge characteristic differences in mining methods, roof control, etc., as a function of seam height.

<u>COMMENTS</u>: The individual work cycle has no apparent effect. All are equally dangerous at various seam heights. Only loading versus output was significantly different. However, as previously noted, this is probably due to proportionately less use of conventional mining methods in high seam coal.

<u>CONSTRAINTS</u>: The last three seam heights were combined due to small sample size problems.

<u>PROBABLE CAUSES</u>: n/a

<u>RECOMMENDATIONS</u>: none

CROSSTABULATION
of
V006 - Average Thickness of Coalbed (Inches)
vs
V039 - During Which Work Cycle Did Accident Occur?

Seam Height (Inches)	Work Cycle											
	Loading		Under Cutting		Face Drilling		Face Shooting		Bolting & Support		Continuous Mining	
	f	%	f	%	f	%	f	%	f	%	f	%
0-36	5	10.9	3	30.0	1	14.3	1	7.7	1	4.6	5	8.6
37-48	13	28.3	1	10.0	2	28.6	6	46.2	7	31.8	11	19.0
49-60	10	21.8	4	40.0	2	28.6	3	23.1	4	18.2	12	20.7
61-72	10	21.8	0	0.0	0	0.0	1	7.7	1	4.6	10	17.2
73-84	1	2.2	1	10.0	0	0.0	0	0.0	4	18.2	6	10.3
85-96	2	4.4	0	0.0	1	14.3	0	0.0	4	18.2	8	13.8
97	5	10.9	1	10.0	1	14.3	2	15.4	1	14.6	6	10.3

NOTE: THIS TABLE IS PRESENTED FOR INFORMATION ONLY

The categories for undercutting, face drilling, bolting, face shooting and support have too few observations for statistical analysis. The loading and continuous mining category distributions are probably different for various seam heights since more conventional equipment would be expected in low seam heights. However, when each fatal accident category is compared to the expected fatal accidents normalized by exposure for all types of mining, loading in lower seam heights appears somewhat higher than expected. Similarly, continuous mining in higher seams is somewhat higher. Complicating this assessment is the fact that the proportions of conventional and continuous mining has changed over the time period of the study. Starting in the 1966 period the proportions were roughly 50%: 50% with a gradual change to about 35% conventional to 65% continuous underground mining. Many small low seam mines have lagged behind this trend toward continuous mining methods.

CROSSTABULATION
of
V006 - Average Thickness of Coalbed (Inches)
vs
V144 - Victim's Exposure Nature

Seam Height (Inches)	Victim's Exposure Nature																				
	Unsupported						Temporary								Permanent						
	Unnecessary		Necessary		Total		Compliance		Non-Compliance		Setting		Total		Compliance		Non-Compliance		Total		Totals
	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%	f	%	
0-36	5	10	2	12	7	11	1	17	1	4	2	50	4	12	2	6	1	6	3	6	14
37-48	11	22	6	35	17	26	1	17	10	43	1	25	12	36	4	13	5	28	9	18	38
49-60	10	20	1	6	11	17	3	50	7	30	1	25	11	33	8	26	5	28	13	27	35
61-72	8	16	3	18	11	17	0	0	3	13	0	0	3	9	5	16	3	17	8	16	22
73-84	6	12	3	18	9	14	0	0	1	4	0	0	1	3	4	13	0	0	4	8	14
85-96	6	12	1	6	7	11	0	0	0	0	0	0	0	0	5	16	1	6	6	12	13
97-	3	6	1	6	4	7	1	17	1	4	0	0	2	6	3	9	3	17	6	12	12

Table III-29

Table III-29

STATISTICS: Chi Square Evaluation   STATISTICALLY SIGNIFICANT: See Comments

NORMALIZATION: Keystone 1974; sample productive output and number of men.

COMMENTS: A chi square test was run on each category of victim's exposure. Expected fatalities were computed based on both output and exposure normalization data. Only fatalities associated with temporary supports produced a significant difference in the chi square test. Seam heights less than 60" were associated with a greater number of fatal accidents during activities related to temporary supports.

CONSTRAINTS: Sample size problems required combining some seam heights to perform statistical tests.

PROBABLE CAUSES: There was less maneuverability of crew and equipment during setting of temporary supports. There were possible seam height to width problems in small mines. Poor procedures and enforcement of temporary support activities in lower seam height mines existed.

RECOMMENDATIONS: MESA should investigate the differences in setting temporary supports in lower seam mines; MESA should consider methods to improve enforcement of temporary support regulations.

Table III-30

CROSSTABULATION
of
V124 - Type of Fall
vs
V313 - Pillar Lift Widths

Pillar Lift Widths	Type Fall: Roof Fall at Face f	Type Fall: Roof Fall at Face %	Type Fall: Roof Fall at Intersection f	Type Fall: Roof Fall at Intersection %	
5-8 ft.	4	6.15	0	0.00	Probably Within MESA Regulations
9-12	2	3.08	2	16.67	
13-16	13	20.0	2	16.67	
17-20	30	46.15	3	25.00	
21-24	8	12.31	2	16.67	Probably Exceeded Regulations
25-28	2	3.08	1	8.33	
29-32	4	6.15	1	8.33	
33-36	2	3.08	1	8.33	

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: Nearly 25% of fatal accidents at the face and more than 40% of fatalities at intersections occurred when pillar split widths exceeded MESA regulations.

CONSTRAINTS: No normalization or violations data was available.

PROBABLE CAUSES: Unknown

RECOMMENDATIONS: MESA should continue enforcement regarding pillar split widths as well as work to gain cooperation of mine operators in improved self-enforcement of regulations.

Table III- 31

CROSSTABULATION
of
V031 - Mining Method in Effect at Accident Site or Section
vs
V313 - Pillar Lift Widths

Pillar Lift Widths	Mining Method				
	Conventional		Continuous		
	f	%	f	%	
5-8 ft.	2	5.13	2	4.35	Probably Within MESA Regulations
9-12	0	0.00	5	10.87	
13-16	4	10.26	11	23.91	
17-20	16	41.03	22	47.83	
21-24	7	17.95	3	6.52	
25-28	3	7.69	1	2.17	
29-32	3	7.69	1	2.17	
33-36	2	5.13	1	2.17	
37-40	2	5.13	0	0.00	

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: The observation which may be important in this crosstab is that 43% of the fatalities occurring at the split widths exceeding regulations were in conventional mining while only 13% were in the same category for continuous mining.

CONSTRAINTS: No normalization data was available.

PROBABLE CAUSES: Smaller mines still using conventional mining methods are permitted to drive pillar split widths greater than 20 feet. Such practice may result in reduced safety. Another possibility may be more numerous small mines are not receiving enforcement attention from MESA that larger mines receive. (Refer to Appendix E for a discussion of rock mechanics including the effects of seam heights).

RECOMMENDATIONS: MESA continue enforcement of mining plan provisions as well as reconsider the advisability of permitting some low seam mines to drive pillar splits wider than the current regulation.

(3) Work Force Variables

This final section of frequency distributions, analyses, and crosstabulations relate to:

- Safety vs Job Classification,
- Hours Worked Prior to Accident,
- Experience Factors, and
- Foremen and Crew Errors.

FREQUENCY DISTRIBUTION

of

V042 - Job Classification of Victim

Fatalities	Percent Fatalities	Population*	Percent Population*	DESCRIPTION
36	16.44	2012	14.7	Continuous Miner Operator
23	10.50	617	4.5	Loading Machine Operator
21	9.59	2571	18.8	Shuttle Car Operator
18	8.22	1247	9.1	Working Supervisor - Immediate Supervisor for Crew (Assistant Section Foreman, Section Foreman)
16	7.31	1749	12.7	Roof Bolter
16	7.31	533	3.9	Timberman (Support Man, Prop Man)
14	6.39	304	2.2	Hand Loader
9	4.11	110	0.8	General Supervisor (Mine Foreman, Superintendent, Owner)
9	4.11	538	3.9	Continuous Miner Helper, Jacksetter
8	3.65	200	1.4	Undercutting Machine Operator
8	3.65	247	1.8	General Laborer
7	3.20	213	1.6	Shot Firer

Fatalities	Percent Fatalities	Population*	Percent Population	DESCRIPTION
4	1.83	805	5.8	Mechanic, Repairman (Excludes Electricians)
4	1.83	139	1.0	Loading Machine Helper
3	1.37	193	1.4	Supplyman, Utility Man
2	.91	200	1.5	Undercutting Machine Helper
2	.91	195	1.4	Ventilation Man (Brattice Man)
1	.46	36	0.2	Rock Duster (Rockman)
1	.46	222	1.6	Electrician
1	.46	42	0.3	Fireboss
1	.46	176	1.3	Conveyor Beltman, Boomman (any job directly associated with conveyors at the face
1	.46	196	1.4	Face Driller
12	5.48	851	6.2	Other including Fire boss, Mainline locomotive, brakemen, Motormen and mantrip operators, Track Layers, etc.

* Population using 1971 Respirable Dust File Data for Retreat Mining. This table is presented for information only. (See the next Table III-34)

ANALYSIS
of
V042 - Job Classification of Victim
Normalized for Population and Risk Factor

Job Classification	Retreat Mining Only								
	Frequency	Percentage	Rank of Accident Frequency	Population: 1971 Number	Population: Percent of Sub Sample	Population: Rank	Expected Fatalities Based on Population	Risk Factor	Expected Rank
Continuous Miner Operator	36	19	1	2012	16.2	2	30.4	1.18	7
Loading Machine Operator	23	12	2	617	5.0	7	9.4	2.44	2
Shuttle Car Operator	21	11	3	2571	20.7	1	38.9	0.54	13
Foreman	18	10	4	1247	10.0	4	18.8	0.96	9
Timberman	16	9	5	533	4.3	9	8.1	1.97	5
Roof Bolter	16	9	6	1749	14.1	3	26.5	0.60	12
Hand Loader	14	7	7	304	2.4	10	4.5	3.11	1
Continuous Miner Helper	9	5	8	538	4.3	8	8.1	1.11	8
General Laborer	8	4	9	247	1.9	11	3.6	2.22	3
Cutting Machine Operator	8	4	10	824	6.6	6	12.4	0.65	11
Shot Firer	7	4	11	213	1.7	12	3.2	2.19	4
Repair Man (Mech & Elect)	5	3	12	1027	8.3	5	15.6	0.32	15
Loading Machine Helper	4	2	13	139	1.1	15	2.1	1.90	6
Cutting Machine Helper	2	1	14	200	1.6	13	3.0	0.67	10
Face Driller	1	1	15	196	1.6	14	3.0	0.33	14

Risk Factor is the ratio of Fatal Accidents to Expected Accidents for each job classification based upon the propor-[illegible]ns of the total underground mines population in each job classification. This factor indicates how much more (or [illegible] dangerous each job is than would be expected on the basis of population alone.

STATISTIC: Spearman Rank Correlation Correlation   STATISTICALLY SIGNIFICANT: Yes

NORMALIZATION: MESA Respirable Dust File Data 1971

COMMENTS: This table arrays the frequency of fatalities by job classification and indicates the population in one representative year, 1971 (for which normalization data was available). In addition, a "Risk Factor" was computed by dividing observed accidents in each job classification by the expected accidents based on the percentage each job classification was of the total underground miner population in retreat mining. Rank ordering was used to relate the standings of various job classifications. A Spearman Rank correlation coefficient indicated good general agreement of Accident Frequency and Population of Job Classification rankings (i.e., $r_s = .75$, $Z = 2.81$, reject HO @ .99).

Probably the most illuminating aspect of this table is the Risk Factor calculation for each Job Classification. This factor indicates how much more (or less) risk of a fatal accident is associated with each job. A value of 1.0 indicates that the number of fatal accidents is about what would be expected strictly on the basis of population of each job classification. For example, hand loaders appear to suffer more than three times the number of fatalities. Similarly, loading machine operators suffer nearly two and a half times the number of accidents that their proportion of the population would suggest. Conversely, roof bolters whose job was thought to be one of the more dangerous, actually have one of the safer jobs. Similarly, face drillers, cutting machine operators, and their helpers, although face occupations are among the safest job classifications. Interestingly, helper jobs are slightly safer statistically than the machine operators for whom they work. However, when large numbers of machines with canopies to protect the operators are in use, we would expect to see improvements in the safety of operator jobs, but not in the unprotected helper jobs. In addition, entry level jobs in underground mining, such as timberman, laborer, and most helper jobs, are more dangerous than their proportion of the population would suggest. Obviously such miners are less experienced in avoiding risks of underground mining. Hence, their inexperience may be contributing to the increased frequency of accidents in such jobs.

CONSTRAINTS: Dust File data was only available for years 1971 to 1974. Since years 1966 to 1970 were not available, 1971 was taken to be representative of the entire period.

PROBABLE CAUSES: Poor work safety practices (i.e., going under unsupported roof unnecessarily) caused accidents. Poor self enforcement of safe working regulations, poor job design and equipment design which forces miners to work under unsupported roof necessarily and poor training on safe operation of equipment and safe work practices all contributed to higher accident frequencies.

RECOMMENDATIONS: USBM should continue to develop safety features on mining machines such as canopies and remote controls. USBM should continue to improve safety in job design such as methods of setting temporary supports with greatly reduced hazardous exposure to miners. USBM and operators should continue to upgrade training of crews.

ANALYSIS
of
V042 - Job Classification of Victims Normalized for Population and Hazardous Exposure

Job Classification	Rank: Accident Frequency	Rank: Population for Retreat Mining 1971 - Only	Rank: Risk Factor Accidents/ Expected (based on Population)	Estimates: Hazardous Exposure %	Estimates: Rank
Shuttle Car Operator	3	1	13	est 0%	14
Roof Bolter	6	3	12	21-69%	3
Repairman (Mech. & Elec.)	12	5	15	est 0%	15
Loading Machine Operator	2	7	2	5-50%	5
Foreman	4	4	9	est 0%	11
Continuous Miner Operator	1	2	7	0-21%	9
Continuous Miner Helper	8	8	8	0-21%	10
Hand Loader	7	10	1	est 100%	1
Timberman	5	9	5	100%	2
Cutting Machine Operator	10	6	11	0%	12
Shot Firer	11	12	4	24-41%	7
Loading Machine Helper	13	15	6	5-50%	6
General Laborer	9	11	3	est 0%	8
Cutting Machine Helper	14	13	10	0%	13
Face Driller	15	14	14	47-54%	4

Table III-34

STATISTIC: Spearman Rank Correlation Coefficient. Ranks: Risk Factor vs. Exposure: $r_s = .54$ - $z = 2.03$

STATISTICALLY SIGNIFICANT: yes

NORMALIZATION: 1971 Respirable Dust File for Population and Ted Berry, I.E. Studies and Field Data for Hazardous Exposure Estimates

COMMENTS: This table was an attempt to determine if a relationship existed between hig risk factors (as defined in the previous table) and extensive hazardous exposure (under unsupported roof). This analysis indicated a significant relationship between the amount of time spent under unsupported roof and high risk jobs.

CONSTRAINTS: Estimates of hazardous exposure are only approximations and can be expected to vary widely in the industry.

PROBABLE CAUSES: Poor work safety practices, poor training and poor self enforcement of safety regulations cause the above accidents.

RECOMMENDATIONS: USBM and MESA should continue to disseminate information on the risks of such poor practices as operators improving self enforcement of safety regulations, and USBM and manufacturers should continue to make improvements in machine safety (i.e., canopies, remote control, etc.).

CROSSTABULATION
of
V042 - Victim's Job Classification
vs
V046 - Victim's Experience at That Mine in Years/Months

Job Classification	Mine Experience					
	0-9	1-4.9	5-9.9	10-25	Total	Pop. %
C/M Operator	3	8	4	5	20	16.2
C/M Helper	2	3	0	1	6	4.3
Cutter Operator	0	3	0	0	3	6.6
Cutter Helper	1	1	0	0	1	1.6
Face Driller	0	1	0	0	1	1.6
General Laborer	2	2	1	1	6	1.9
Hand Loader	3	6	-	0	9	2.4
Loader Operator	2	8	2	5	17	5.0
Loader Helper	2	0	0	0	2	1.1
Roof Bolter	2	4	1	3	10	14.1
Shot Firer	0	3	0	2	5	1.7
Shuttle Operator	4	9	1	3	17	20.7
Timberman	3	6	0	1	10	4.3
Shift Supt.	3	3	6	1	13	10.0
General Supt.	1	3	2	1	7	-

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: No adequate experience data was available on the miner population.

COMMENTS: Because the experience class intervals represent different numbers of years (i.e., 1, 5, 10, 15 years) an attempt was made to estimate the accident frequency per year. If it could be assumed that both experience and accidents were linearly distributed among years in each class interval, the average number of accidents per year would be 1.9/year, 1.0/year, 0.25/year and 0.09/year respectively for the experience class intervals above. And although we know that neither accidents nor experience is uniformly distributed by year in the above intervals, especially in the less than 5 year experience intervals, these initial estimates suggest the rate of decline of accidents per year of experience. This study and several previous studies have shown an accident avoidance learning effect with experience.

CONSTRAINTS: An attempt was made to use experience data on a sample of 363 miners in five job classifications taken during the T. Berry Pittsburgh Seam studies. However, extrapolation of these data to all job classifications does not produce an acceptable normalization data base.

PROBABLE CAUSES: A mine environment is a comparatively hostile work place. There are many things that can get an inexperienced man in trouble. Some of these situations involve complex and subtle clues to detect and avoid serious trouble. Some of these clues are learned with continued experience. However, other problems of information may be beyond the physiological capabilities of men, hence unrelated to experience. For example, correctly assessing a bad roof condition requires both experience and information not detectable by human senses. For such situations, sensors or other sources of the necessary information are required to provide a sound basis for assessing the true situation. Obviously, if such systems become available, miners and foremen must be trained to relate their experience and sensory detection capabilities to the new information to make the best use of all available information to avoid serious trouble underground.

RECOMMENDATIONS: USBM continues development of systems to provide supplemental information necessary to avoid serious accidents. MESA and operators continue training enforcement and self enforcement of safe working practices.

Table III-36

CROSSTABULATION
of
V031 - Mining Method in Effect at Accident Site or Section
vs
V042 - Victim's Job Classification

	Fatal Accidents		Estimated Typical Crew		Expected Fatalities
Victim's Job	Rank	Frequency	Crew	%/Crew	%/Crew
Loading Machine Operator	1	(20)	1	7.3	4.9
Cutter Operator	2	(8)	1	7.3	4.9
Shot Firer	3	(7)	1	7.3	4.9
Shuttle Car Operator	4	(5)	2	14.6	9.8
Timberman	4	(5)	1	7.3	4.9
Worker Supervisor	4	(5)	1	7.3	4.9
Loading Machine Helper	7	(3)	1	7.3	4.9
Roof Bolter	7	(3)	2	14.6	9.8
Supplyman	7	(3)	1	7.3	4.9
General Supervisor	7	(3)	.2	1.4	.9
Cutter Helper	11	(2)	.5	3.7	2.5
General Laborer	11	(2)	1	7.3	4.9
Face Driller	13	(1)	1	7.3	4.9

See Next Page

COMBINED CREW CATEGORIES FOR ANALYSIS

	f	Estimated Typical Makeup of Crew Crew	% Crew	Expected Fatalities	Risk Factor = Accidents/ Expected Accidents
Loading Op. & Helper	23	2	14.6	9.8	2.35
Cutter Op. & Helper	10	1.5	11.0	7.0	1.42
Shotfirer & Face Drill	8	2	14.6	9.8	.81
Shuttle Car Operator	5	2	14.6	9.8	.51
Timber, Supply, Labor	10	3	21.9	14.7	.68
Roof Bolter	3	2	14.6	9.8	.30
Supervisor	8	1.2	8.7	5.8	1.37

STATISTIC: Chi Square Evaluation

STATISTICALLY SIGNIFICANT: Yes

NORMALIZATION: Estimates of typical conventional mining section face and support crew in retreat mining.

COMMENTS: Review of these tables indicates that the loading machine operator's risk factor is more than twice what would be expected on the basis of population. The cutting machine operator also has somewhat higher risk of accident. Conversely, the roof bolter and shuttle car operator have significantly lower risk factors. These results are consistent with similar analyses for both conventional and continuous mining.

CONSTRAINTS: Small sample sizes required combining several job classifications from Table III-34 for analysis in Table III-37. It should be noted that the shot fire classification was found to have a risk factor more than twice what would be expected on the basis of population (See Table III-34). However, when the shot firer accident data was combined with the lower risk face driller (to permit analysis) an apparent lower risk factor was indicated.

PROBABLE CAUSES: Poor work safety practices, lack of protective canopies on equipment during periods of the study, inadequate training of machine operators and self-enforcement of safety regulations by foremen and mine operators contributed to the causes.

RECOMMENDATIONS: MESA and mine operator should continue or increase emphasis on training of section crews for safe operation of equipment and USBM continue development and testing of safer equipment (e.g., canopies, remote control, sensors, cable reels, etc.) and mining procedures.

CROSSTABULATION
of
V031 - Mining Method in Effect at Accident Site or Section
vs
V039 - During Which Work Cycle Did Accident Occur?

	Mining Method at Site				
Work Cycle	Fatalities Conventional Mining f	%	Number of Men Employed #	Expected Exposure %	Expected Fatalities Re. Exposure
Loading	30	54.6	1501	43.6	24.0
Undercutting	6	10.9	824	23.9	13.1
Face Drilling	7	12.7	196	5.7	3.1
Face Shooting	8	14.6	213	6.2	3.4
Bolting/Support	4	7.3	708	20.6	11.3

STATISTIC: Chi Square Evaluation
$X^2 = 20.68$ df = 4

STATISTICALLY SIGNIFICANT: Yes

NORMALIZATION: 1971, Respirable Dust File Data, Conventional Retreat Mining.

COMMENTS: The Chi Square Evaluation results for the observed frequency against the expected frequency based on exposure of men in elements of the work cycle gives a significant result. As previously discussed, loading is the most hazardous function in the conventional mining cycle. Bolting and support is one of the safer work cycle elements. Some crews may have higher exposure rates for both loading and support. This will further support the result.

CONSTRAINTS: n/a

PROBABLE CAUSES: Unnecessary exposure under unsupported roof, poor work safety practices and inexperience of operators may be major causes.

RECOMMENDATIONS: MESA should continue emphasis on safety in the loading cycle. USBM should continue development of remote controlled loading machines and/or protective canopies.

FREQUENCY DISTRIBUTION

of

V023 - On Which Shift Did Accident Occur?

f	%	Description
117	54.17	First Production Shift
60	27.78	Second Production Shift
29	13.43	Third Production Shift
6	2.78	Maintenance shift (no Production)
4	1.85	Idle day, not regular production, maintenance shift, etc.

The above data was reorganized to eliminate the maintenance shift and idle day work.

Class	f	%	Industry Norm '74 %	Based on Expected Industry Norm	Ratio
1st	117	57	56	115	1.12
2nd	60	29	37	76	.79
3rd	29	14	7	14	2.07

STATISTIC: Chi Square Evaluation STATISTICALLY SIGNIFICANT: Yes

NORMALIZATION: 1974 Keystone Industry Manual sample of 347 mines

COMMENTS: By comparing the ratio of the distribution of fatal accidents to the mines operating by shift, we see that an abnormal proportion of accidents occurred during the third shift. Conversely, fewer accidents occurred during second shift work.

CONSTRAINTS: None

PROBABLE CAUSES: A greater percentage of third shift crews are less experienced workers. In mines rotating shifts, difficulties exist in adjusting to differences in the diurnal cycle (e.g., daily habits).

RECOMMENDATIONS: Increased emphasis should be placed on safety by MESA for third shift workers.

FREQUENCY DISTRIBUTION

of

V025 - Number of Hours Victim Worked Prior to Accident

	1st	2nd	3rd	4th	5th	6th	7th	8th	9th/ Later
Number of Fatalities	33	23	18	25	28	35	27	17	4
%	15.7	11.0	8.6	11.9	13.3	16.7	12.9	8.1	1.9

STATISTIC: Chi Square Evaluation   STATISTICALLY SIGNIFICANT: Yes

NORMALIZATION: USBM Information circular, 1973; National Safety Council, Accident Facts, 1973 (See text); Estimated Hours Worked per Hour on Shift As An Estimate of Work Exposure (See text).

COMMENTS: In an attempt to interpret these data, normalization data reflecting industry experience relative to accidents vs time on shift was obtained. USBM data on 1969 injury experience[1] indicates that peak accident rates occurred at three to four hours after beginning work and about three hours after resuming work following lunch break (regardless of shift worked). Interestingly, peaks occur in the data from this study during the first hour on shift and again on the sixth hour (i.e., 1 1/2 to 2 hours after resuming work following lunch break). The high accident rates immediately after starting or resuming work are especially interesting. These raw rates become even more important when it is remembered that productive work time (hence, work exposure) is low during the first hour on shift due to tramming to the work face and starting up activities, yet a high frequency of work place accidents has been observed. (See subsequent discussion) A similar high first hour frequency of accidents was found in a National Safety Council study of 31,987 manufacturing and 43,295 non-manufacturing non-fatal injuries in Pennsylvania during 1969.[2]

Table 40 reproduces that summary data. However, a major difference is that the lowest frequency of the eight hour shift occurs in the fifth hour. In our study the highest rate occurs in the fifth and sixth hour intervals.

[1] USBM Information Circular 1973, #IC8599

[2] Accident Facts 1973 Edition, National Safety Council

Chi Square Evaluation of Hour of Work Day

In Which Accident Occurred

	1st	2nd	3rd	4th	5th	6th	7th	8th	9th/later
Frequency (Total 210)	33	23	18	25	28	35	27	17	4
Expected *	29	27	33	29	21	27	27	10	4
Frequency**	33	27	29	25	17	21	25	21	10

* = All Coal Mining ** = Manufacturing

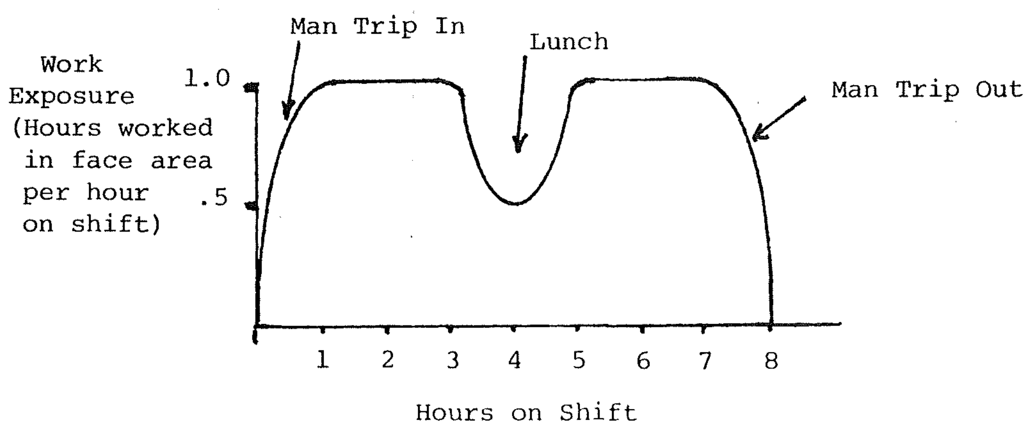
$X^2 = 18.12$
df = 8 All Coal Mining

$X^2 = 25.74$
df = 8 Manufacturing

Thus, the distribution of fatal accidents in underground retreat mining is significantly different than both manufacturing and all coal mining (general) industrial experience. However, this analysis indicates that the high observed accident rate in the first hour is not radically different from industrial experience. But, the after-lunch peaking of fatal accidents remains a striking feature of this Table in both coal mining in general, and retreat mining.

In addition, if accident frequencies from each shift hour were normalized for "work exposure" (i.e., hours worked in the face area per shift hour) approximated below, the higher frequencies within the first hour after starting the shift or returning from lunch break are even more startling. For example, if in that hour after beginning or resuming work (after lunch) only half an hour is actually worked (i.e., exposure = 0.5 hr/hr), the average risk of accident per hour worked is 2.5 times higher for those two hours as for any other shift hour worked. Range is 1.6 to 3.8 times more risk for these two hour periods.

APPROXIMATIONS OF WORK EXPOSURE



Thus, at the beginning of a shift and resumption of work after lunch, we see an abnormally high rate of fatalities. This phenomenon is probably different from industrial experience. Hence, if fatal accident data is normalized for both industry experience and work exposure, the effective accident rate for both the first hour after starting and resuming work are higher than for other industries.

CONSTRAINTS:

PROBABLE CAUSES: There are inherently more hazardous conditions than in any other industrial environment. There is poor communication of possible hazards among crews working different shifts. There is initial disorganization starting or resuming work. There is psychological (attention) lag when transitioning from a non-working to working state. There is not enough emphasis on safety at the beginning of each work period. There are short crews and/or job substitutions at the beginning of the shift or during staggered lunch breaks. The data and supporting literature available in this study are insufficient to establish actual causes of this phenomenon.

RECOMMENDATIONS: MESA should distribute these facts for the purpose of informing miners of the increased risk of accidents (due to various possible causes in the period immediately after starting or resuming work).

CROSSTABULATION
of
V025 - Number of Hours Victim Worked Prior to Accident
vs
V029 - General Classification of Accident Type

Hours Worked	General Classification of Accident Type								
	Roof Fall		Rib/face Fall		Machinery		Haulage		
	f	%	f	%	f	%	f	%	
1	25	16.34	5	26.32	1	7.69	2	10.53	
1-2	15	9.9	4	21.05	2	15.38	1	5.26	
2-3	12	7.84	2	10.53	1	7.69	0	0	
3-4	18	11.76	0	0	4	30.77	3	15.79	Lunch
4-5	24	15.69	1	5.26	1	7.69	2	10.53	Break
5-6	27	17.65	2	10.53	1	7.69	4	21.05	
6-7	18	11.76	4	21.05	1	7.69	4	21.05	

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: n/a

COMMENTS: When work exposure hours are considered (i.e., hours worked/shift hour), 49 (i.e., 32%) of fatal roof fall accidents occurred within two 1/2 hr. periods after beginning the shift and resuming work after lunch break. An average of 16.8 accidents were recorded for any other shift hour. The risk is nearly three times greater for these two 1/2 hour work periods than for any other shift hour. Although sample sizes are small, there is no indication that the same increased risk effect is present for any other general class of accident. See discussion of exposure hours on page 54.

CONSTRAINTS: Small sample sizes precluded statistical analyses. This table is presented for information only.

PROBABLE CAUSES: n/a

RECOMMENDATIONS: n/a

CROSSTABULATION
of
V025 - Number of Hours Victim Worked Prior to Accident
vs
V046 - Victim's Experience at That Mine in Years/Months

	Victim's Experience at Mine							
	0-.9 Years		1-4.9 Years		5-9.9 Years		10-25 Years	
Hours Worked	f	%	f	%	f	%	f	%
1	4	11.11	16	25.40	1	7.14	4	14.81
1-2	6	16.67	8	12.70	0	0.00	2	7.41
2-3	4	11.11	5	7.94	0	0.00	3	11.11
3-4	2	5.56	6	9.52	5	35.71	5	18.52
4-5	3	8.33	10	15.87	3	21.43	2	7.41
5-6	5	13.89	9	14.29	2	14.29	6	22.22
6-7	6	16.67	7	11.11	2	14.29	4	14.81
7-8	4	11.11	2	3.17	1	7.14	1	3.70
8-9	2	5.56	0	0.00	0	0.00	0	0.00

	Grouped Crosstabulation			
	0-4.9 Years		5-25 Years	
Hours Worked	f	%	f	%
0-2	34	35.0	7	17.1
2-4	17	17.5	13	31.7
4-6	27	27.8	13	31.7
6-8	19	19.6	8	19.5
% Accidents Per Experience Group		70.3		29.7

STATISTIC: None STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: Inspection of the data suggested there may be two different distributions for two major experience groupings. The table above (Group Crosstabulations) confirms this finding and suggests that it is the lessor experienced mine workers who are more likely to become a fatal accident victim during the first two hours of the shift.

CONSTRAINTS: Due to the small sample sizes no statistical tests were performed on these data.

PROBABLE CAUSES: See previous discussion

RECOMMENDATIONS: None

CROSSTABULATION
of
V047 - Victim's Experience at His Job Classification in Months
vs
V023 - On Which Shift Did Accident Occur?

Mine Experience Years	Accident Shift							Approximate Accidents/Year of Experience
	First		Second		Third			
	f	%	f	%	f	%	Total	
0-9	21	28.00	15	31.25	7	28.00	43	43
1-4.9	23	30.67	19	39.58	8	32.00	50	12
5-9.9	16	21.33	9	18.75	6	24.00	31	6
10-25	15	20.00	5	10.42	4	16.00	24	2

CROSSTABULATION
of
V046 - Victim's Experience at That Mine in Years/Months
vs
V023 - On Which Shift Did Accident Occur?

Mine Experience Years	Accident Shift							Approximate Accidents/Year of Experience
	First		Second		Third			
	f	%	f	%	f	%	Total	
0-9	18	27.50	11	27.50	5	23.81	34	34
1.0-4.9	31	41.89	19	47.50	9	22.86	59	15
5-9.9	12	16.22	5	12.50	0	0	17	3
10-25	13	17.57	5	12.50	7	53.33	25	2

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: No adequate experience data was available on the miner population.

COMMENTS: To interpret these tables, total accidents by experience class intervals and shift can be compared. Without adequate normalization, high confidence conclusions cannot be made. However, intuitively the frequency of accidents at any shift or experience interval is not unexpected. Perhaps a bit more illuminating are the approximations of accidents per year of experience on the right of each table. Another interesting feature of this comparison is the similarity of the distributions for mine experience and job experience. Again, without adequate normalization it is only an impression. A fact previously discussed is also evident in this comparison - that is a higher percentage of the total accidents on third shift. Where only about 7% of mines report using a third working shift, 15-17% of fatal accidents occur on that shift. The reason is probably a higher percentage of inexperienced miners on third shift.

CONSTRAINTS: Approximations of accidents per year of experience were based upon the assumption of a constant accident rate by year within any experience interval. This assumption is almost certainly not true for the 1-5 year interval, however, if viewed with caution, these approximations could be considered a quasi-average number of accidents per year within any experience class interval.

PROBABLE CAUSES: n/a

RECOMMENDATIONS: n/a

FREQUENCY DISTRIBUTION

of

V307 - Did Roof "Work" Before Fall?

f	%	DESCRIPTION
101	80.16	yes
25	19.84	no

STATISTIC: None STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: Over eighty percent of the fatal accidents occurred after the roof had previously worked. This highlights a major problem in retreat mining. Roof warnings are one of the major indicators to miners of impending falls. Unfortunately, workers cannot keep track of the number of warnings provided by the roof. For example, a worker in an area hears the roof work. He then moves to a new area and a second worker enters the area. He does not have the knowledge of the prior worker's observation and is caught in a roof fall. This is even more apparent in retreat mining where roof noise is very frequent.

CONSTRAINTS: n/a

PROBABLE CAUSES: There is no systematic method for tabulating roof working warnings. There is no systematic communication of such warnings between sections or shifts. There is no information individual miners have to help them associate specific roof warnings with incipient roof failure. Individual roof working events tend to become disregarded because there are many such events without a means of relating such events to useful roof fall warnings.

RECOMMENDATIONS: USBM should continue development of noise monitoring equipment to pinpoint location, intensity, and time relationship of noise in retreat mining. A graph, shown in Mining Congress Journal's October, 1974, issue shows the results of Microseismic Investigations, and displays the relationship of noise over time. Further development of this concept could provide a warning system by which after a certain roof noise rate or intensity is reached an area is abandoned, mined after installation of additional support, or mined remotely.

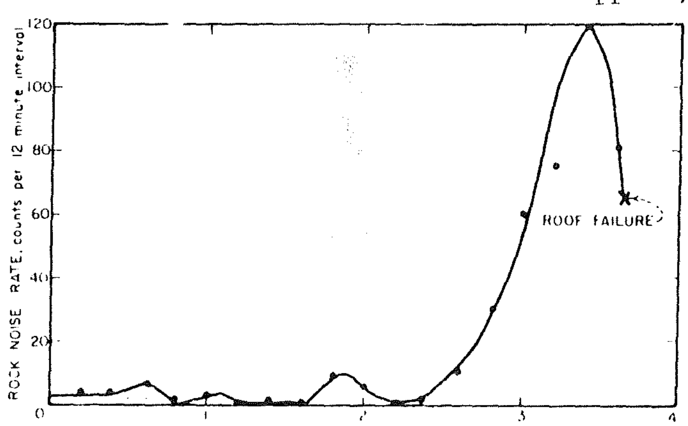


Fig. 12. Roof noise rate versus time before roof failure

FREQUENCY DISTRIBUTION

of

V194 - Primary Unsafe Act by Victim that Contributed to Accident

f	%	DESCRIPTION
24	22.22	None
17	15.74	Tested improperly or did not act
14	12.96	Did not follow warning/instructions
11	10.19	Proceeded under unsupported roof
11	10.19	Improper equipment operation
10	9.26	Placed self in unsafe position
6	5.56	Did not put up support
5	4.63	Removed support purposely
4	3.70	Deviated from standard operating procedure
3	2.78	Did not adjust for conditions
2	1.85	Did not scale loose material
1	.93	Removed support accidentally

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

COMMENTS: Noteworthy is the fact that in twenty two percent of the cases, nothing unsafe was determined to have been done. Conversely, other categories indicate the consequences for unsafe and improper procedures for a variety of reasons.

Table III-46

CONSTRAINTS: This table is provided for information only

PROBABLE CAUSES: Improper training and poor work safety practices contributed to these accidents.

RECOMMENDATIONS: MESA should continue emphasis on training for safe and effective work procedures. Mine operators should improve industry awareness of the safety and productivity payoffs for better qualified miners using safer work practices.

FREQUENCY DISTRIBUTION

of

V192 - Primary Unsafe Physical Conditions That Contributed to Accident

f	%	DESCRIPTION
35	25.36	No support
35	25.36	Improper pillar extraction (out of sequence cuts)
24	17.39	Unusual conditions present (bad roof, faults)
8	5.80	Not enough temporary support
8	5.80	Other
6	4.35	Not enough permanent support
5	3.62	Loose material; not scaled/supported or improperly scaled
4	2.90	None
4	2.90	Support removed - accidentally/ purposely
3	2.17	Excessive delay in setting support
2	1.45	Excessive entry width
1	.72	No plan to cover event
1	.72	Excessive cut depth
1	.72	Irregular blasting
1	.72	No proper tools/material available or used

STATISTIC: None

STATISTICALLY SIGNIFICANT: n/a

NORMALIZATION: None possible

COMMENTS: These categories of unsafe conditions stand out from this distribution: (1) No support - Over twenty five percent of the fatalities occurred where the primary condition was "no support" - consequently, workers were either necessarily or unnecessarily under unsupported roof. Again, this further supports the need for USBM projects to provide safer methods of setting temporary supports. (2) Improper pillar extraction - Over twenty five percent of fatal accidents were in this category. Apparently significant numbers of foremen are deviating from prescribed pillaring plans. Both MESA and mine operators should increase emphasis on strict adherence to the approved plan. (3) Unusual conditions present - Over seventeen percent of the fatalities were in this category. Seldom do roof control plans include specific steps to counter unusual or bad conditions. Thus, section crews are left to use their experienced judgment to both assess and solve such problems.

CONSTRAINTS: n/a

PROBABLE CAUSES: Poor process control by foremen in adhering to an approved pillaring plan and failure of roof control plans to provide contingency plans for countering bad roof conditions contributed to these accidents.

RECOMMENDATIONS: MESA should institute regulations requiring approved roof control plans to include systematic provisions for countering bad roof conditions. USBM and MESA should provide the necessary rock mechanics information as a basis for selecting and implementing standardized procedures for controlling bad roof conditions.

IV. PILLAR EXTRACTION METHODS

IV. PILLAR EXTRACTION METHODS

The following pages discuss the basic considerations facing mine operators in the selection of a pillaring plan. Each of the major types of pillar plans is discussed for both conventional and continuous mining. Only the major plans are presented. An analysis of variations in plans is presented in subsequent sections.

1. ROOM AND PILLAR MINING

Room and pillar mining consists of driving openings to divide the coal into blocks. The blocks, or pillars, support the overlying strata. The openings are called rooms, hence the term "room and pillar." Mining is normally divided into two phases: advancing, where the rooms are driven into the coal; and retreating, where the resultant pillars are extracted or removed. Where full pillar recovery is practiced subsidence or breaks often occur on the surface. Because of this, full retreat mining is not always practiced.

There are a vast number of general factors which can affect the decision of the mine operator in selecting the proper mining method and techniques to be used for room and pillar mining. These include the roof pressure, the character of roof and floor, the characteristics of the coal, the inclination and thickness of the seam, the end product desired, among others. Each is discussed below.

- **Roof Pressure**

  Heavy roof pressure requires larger pillars and smaller rooms. Squeezes or creeps are often the result of the failure to recognize excessive roof pressure. These are usually observed in retreat mining.

- Character of Roof and Floor

The character of the roof and the floor have a tremendous effect on retreat mining. Even a "good" roof which can sustain large room spans can be a problem. Such a roof might not break off at the pillar line, transferring excess pressure to adjacent pillars. This additional pressure makes them more difficult to mine. Conversely, a frail roof could break too soon while workers are in the process of extracting a pillar. Support of such a roof becomes critical. The addition of roof bolts in a previously unsupported area may cause the roof to become too strong. The addition of too many timbers inside a pillar may give too much support and prevent pillar stumps from crushing out.

Impurities present overlying the coal seam may affect the strength of the roof. Draw slate is often found in the roof. It readily separates and must be taken down along with the coal. In other areas, shales are of such poor quality that air causes rapid deterioration. Consequently, a thin layer of coal must be left in the roof to prevent air from deteriorating the roof.

The character of the floor is also a problem. A soft bottom with poorly designed pillars could result in the pillars' being driven into the floor and the floor's heaving, partially filling the room.

- Character of Coal

The character of the coal itself can be a problem. A hard coal produces a strong, firm rib. There is little chance of spalling but a chance of a rib roll and resultant injury of workers. A soft coal does not hold a good rib, posing a lesser danger of injury but requiring a tremendous amount of clean-up effort. Driving splits under these conditions is extremely dangerous and must be done with caution.

- Seam Pitch

Pitched seams cause pillars to be designed to keep rooms at an angle to the pitch and crosscuts parallel to the contour lines. The results are diamond-shaped pillars. Pillar extraction must be planned so that gravity works with the haulage system.

- **Seam Thickness**

  The thickness of the seam must be carefully considered. Thinner seams tend to support wider rooms and thick seams, narrower rooms. Pillars much over 8 feet in height pose grave hazards in the pillar extraction process. Difficulties in supporting the roof and hazards from rib rolls are some of the problems.

- **Methane Gas**

  The presence of methane gas is also a problem. Normally, however, advancement is done far ahead of retreat mining and any gas present in the pillar tends to bleed off. This helps but does not eliminate the problem.

- **End Product**

  The end product desired must be considered. Prior to the high degree of automation found in mines today, the mine operator had no problem blending the various qualities of coals found in his mine. Impurities such as draw slate and rock could be easily separated out by hand and were normally left in the mine. Today, with the high degree of automation, blending or selective mining cannot be performed by section, that is, within a section, but can only be performed by blending material from the various sections within the mine.

- **Size of Mine**

  The size of the mine can affect the methods employed. Larger mines have new and better equipment. Smaller mines often operate with poor equipment or without the knowledge and sophistication or training found in the larger mines.

All of these factors and others must be considered in order to select the "best" method. This is the method which will produce the maximum recovery, at the lowest cost, with the least danger to the miner.

Room and pillar mining is a flexible system. Basic system parameters can be readily changed to suit the particular conditions of each section within the mine. Unusual conditions can be coped with or even avoided when encountered. There are, however, some drawbacks to the system. While initial developmental costs are less than other methods such as longwalling, the overall cost for the system is considerably higher. The section is large, causing numerous delays while moving production equipment from place to place. This large work area also makes supervision and logistics extremely difficult.

Planning for room and pillar mining is of critical importance in order to enhance production and maintain safety. This is a difficult task, since the life of a mine is often in excess of 20 to 30 years. During this period of time, equipment undergoes many evolutions, changing basic requirements and mining techniques. This makes long-range planning as difficult as it is important.

Perhaps the first considerations are room widths and center-to-center dimensions for driving the openings. Consideration must be given to the overburden pressure and to the consistency of the immediate roof. Characteristics of the coal must also be considered. Opening widths must be sufficient for the size of the equipment that is to be utilized in the section. Center dimensions on rooms and crosscuts must be designed to support the overburden and to allow for a systematic recovery plan. Although occasions occur where less than full recovery is practiced, these discussions will be limited to full recovery of pillars. All of the pillars must be properly dimensioned to allow full extraction of each pillar in an efficient and speedy manner.

Pillars vary in size from those in excess of 100 feet square to those as small as 20' x 30'. These variations in dimensions can be better visualized by the following table:

Table IV-1

Room Centers (ft.)	Crosscut Centers (ft.)	Room Witdths (ft.)	Crosscut Widths (ft.)	Extraction During Advance		Extraction During Retreat	
				Area	%	Area	%
100	100	20	20	3600	36	6400	64
90	90	20	20	3200	40	4900	60
80	80	20	20	2800	44	3600	56
70	70	20	20	2400	49	2500	51
60	60	20	20	2000	56	1600	44
50	50	20	20	1600	64	900	36

Percent Recovery for Square Pillars of Varying Dimensions

Economic considerations might dictate the use of centers which are close together. The closer together they are, the higher the percentage of extraction during advance mining. Consideration should also be given to the production rate during advance and retreat mining, and the cost of extraction of coal during both of these phases.

In many of the eastern mines, numerous seams of coal are present. The extraction of coal in one seam must consider the workings in the underlying seams. Care must be taken to properly align the mining in the underlying seams, above and below. Deterioration in one seam could cause problems in the underlying seam.

Consideration should also be given to cleavage plans which exist in the coal. Bituminous coals generally contain two cleavage planes. They run at approximately right angles to one another. The longer, more pronounced irregular cleat is called the face cleat, the other, the butt cleat. Proper design of the mine can use the cleats to insure good breakage of the coal, stronger ribs, ability to withstand roof pressures, and proper bleed-off of gas.

During the retreat phase of mining the production pillars are recovered progressively from the bleeder blocks towards the entries in the barrier pillars. There are so many techniques of pillar recovery that only the major techniques can be discussed here. These differ from one another on the basis of the sequence and pattern of successive cuts (or lifts) taken from the pillar. These techniques can also be distinguished by the mining equipment utilized to extract the pillars. Most common in these discussions will be continuous mining equipment and conventional mining equipment.

The selection of the sequence of cuts in the preparation of the mining plan is one of the most important steps in insuring good production. Numerous sequences exist, and each has its own individual characteristics, advantages, and disadvantages. No one plan is universally applicable, and many plans can be used in a given set of conditions. The roof control specialist must select the best plan for the given conditions. The content of the plan can have a substantial impact on safety and productivity. The following section discusses some of the more common plans. Each will be analyzed to optimize the basic dimensions and support requirements. Utilization of the results will allow the reader to select a particular pillar plan and tailor it to his conditions and equipment in such a way as to maximize production and safety.

Throughout this report emphasis is placed on systems compatible for continuous miners. Conventional mining sections, are also discussed but emphasis is definitely placed on the former. This is representative of what is characteristic of the industry. In fact, during the period 1970 to 1974, the number of men employed in retreat mining varied from 27% to 21% and the percent using retreat mining conventional equipment varied from 10% to 20%. Only in scattered areas was conventional equipment being used in pillaring operations. Most often, it was due to geological conditions or to smallness of the mine. In one particular case, an 18-inch seam of rock ran through the center of the coal bed. Where operators were given a choice, they tended to use conventional equipment on advanced work and continuous equipment on retreat work. This was due, in part, to the ability of conventional equipment to cut efficiently "harder" solid coal, and also due to the need for speed of extraction in retreat mining which the use of continuous equipment provides.

The discussions of pillaring plans are presented first for continuous mining, then for conventional mining. Following these sections, each method is discussed in greater detail analyzing among others the effects of dimensions, equipment, haulage, and support. Finally, a comparison of each pillaring method is presented in terms of productivity, safety, and utilization of resources. The methods presented are in use in mines today. They vary widely in terms of their inherent safety. Some are obviously poor methods of pillar extraction and are not recommended. Caution should be used before selecting a plan to ensure that it is compatable with safety requirements stated by MESA.

2. CONTINUOUS MINING PILLARING TECHNIQUES

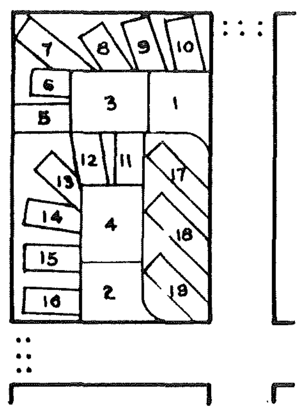
During the underground studies, numerous techniques were observed in extracting pillars using continuous mining equipment. In some cases the variations in methods were minor; in others, they entailed totally different approaches to the extraction of the pillar block. It was impossible to observe all the different methods of extracting pillars using continuous equipment, however, each major method is discussed. The techniques discussed include:

- pocket and wing,
- split and fender,

- splits with simultaneous fender extraction,
- multiple splits,
- radius pillaring,
- diagonal pillaring, and
- block mining.

(1) Pocket and Wing

This technique is commonly found in the Pittsburgh seams and is the best known of any of the multi-directional pillaring techniques. The reader should refer to Figure IV-1 during the following discussion.



Pocket and Wing
Figure IV-1

In the pocket and wing technique, lifts are taken of successive sides of the pillar block in a manner which will leave an eight to fifteen foot pillar of coal between the pocket and the gob. This pillar is commonly called the wing.

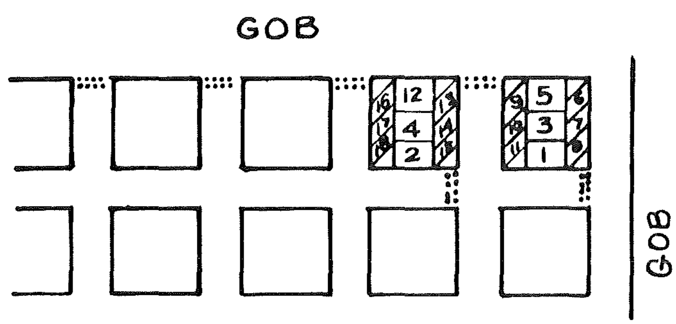
Following completion of the first series of lifts through to the gob end, the remaining coal in the wing is extracted. The cutting of the coal in the pocket is sequenced in a way which provides two working places within the pillar block. That is, a cut is taken off the first pocket, then the continuous miner is trammed to the second pocket to take cut number two. At the same time the first cut is bolted. The continuous miner then continues to exchange places with the bolter until the pocket is driven through to the gob. The remaining coal in the wing is then extracted prior to driving the second pocket through to the now completed first pocket. Normally, depending on the basic dimensions of the pillar block, the second pocket is then one cut away from completion. The lift is taken and the wing is then extracted. The pillar now has been reduced to a substantially smaller block. Again, depending on the basic dimensions, this remaining block may be extracted using a split and fender technique or may be reduced in size in a manner similar to that previously described.

The sequence of cuts in pocket and wing is very important to establish a tempo between the miner and the bolter and to prevent the miner from being forced to wait upon the bolter's completing an operation. Because of this, an additional lift often must be started in an adjacent pillar block to keep the continuous miner busy.

This technique normally results in leaving square blocks of coal after each stage of mining. It allows operations to be concentrated in one block at a time and provides an easy means to coordinate the mining and bolting operations. However, place changes can be confusing, and bolters must work internal to the block at the same time as the mining is being completed.

(2) Split and Fender

The split and fender technique is common throughout the country and is the best known single directional technique of pillaring. It requires that several blocks be mined at the same time in order to coordinate the mining and bolting. The reader should refer to Figure IV-2 during the following discussion.

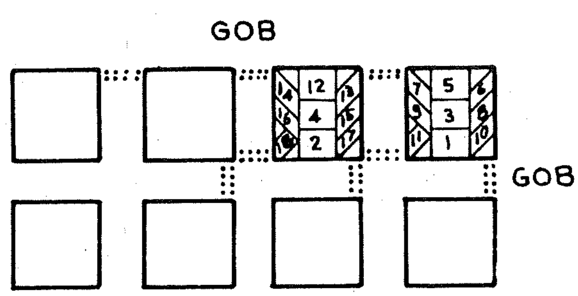


Split and Fender
Figure IV-2

Through a sequence of lifts, a split is driven through the center of the pillar block. Normally, in the case of rectangular blocks, this split is driven the length or the long distance through the block. Two and occasionally three pillars are extracted at the same time. The continuous miner moves from one block to the other to provide a new place for the roof bolter. Following completion of the driving of the split, the miner then turns and extracts the fender on the gob side. Depending on the basic dimensions of the pillar block, the extraction of the fender may require the miner to cut at a 45° angle or as great as a 90° angle. Following completion of the extraction of the gob side fender, the miner then positions himself on the room side of the pillar to extract the second fender. This technique requires that two pillar blocks be simultaneously active in the extraction phase. Sequencing of cuts is much easier than pocket and wing.

(3) <u>Splits with Simultaneous Fender Extraction</u>

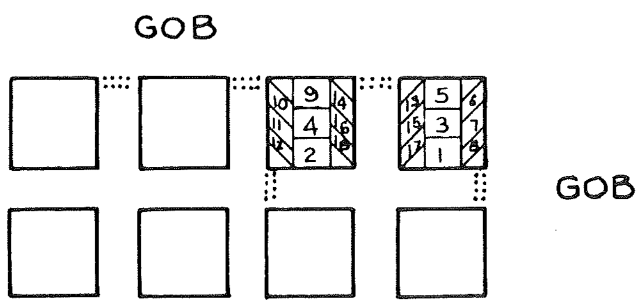
"Christmas Treeing" is one name for the technique of pillar splitting in which both fenders are extracted at the same time. In this method a pillar block is initially cut in a manner similar to split and fender, but then cuts are taken off both the right and left fenders as the miner retreats out of the pillar split. Roof bolting is completed in the same manner as in Split and Fender. See Figure IV-3.



Split with Simultaneous Fender Extraction
Figure IV-3

In this manner the entire pillar can be extracted from the pillar split. This technique is often used in areas where access to the pillar from adjacent rooms is prevented because of roof falls, floor heaves, or other such obstructions. It is also the standard practice of pillar extraction in some mines. It appears to offer little advantage over the conventional split and fender method, and increases the hazards since on the process of fender extraction there is a high amount of exposure of workers.

Another technique is a combination of split and fender and simultaneous fender extraction, shown here in Figure IV-4.

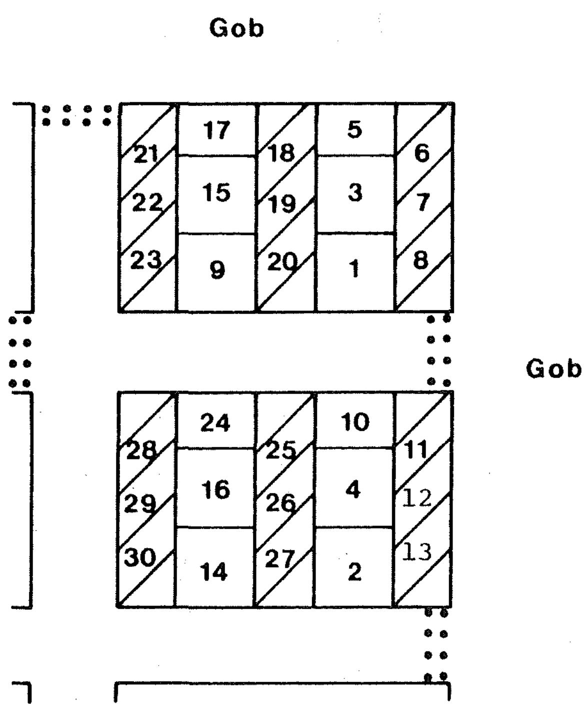


Split and Simultaneous Fender
Extraction from the Room
Figure IV-4

Splits are driven in adjacent pillars in a manner which facilitates roof bolting. The pillar block on the gob side is holed through first, then the gob side fender is extracted. The miner then returns to the split of the second pillar, extracts this, and completes the out-by fender. The two remaining fenders border the room and are extracted by moving the continuous miner into the room entry and extracting both fenders by taking subsequent lifts off the left and right sides.

(4) Multiple-Splits

Where pillar dimensions are sufficiently large, multiple splits are required to achieve complete extraction. In Figure IV-5, two splits are driven through two different pillar and sequenced in such a way to provide for the bolting functions.

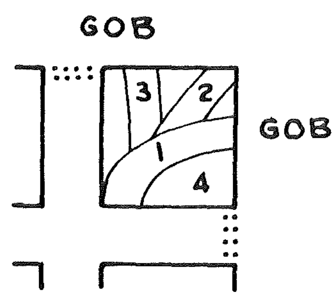


Multi-Split and Fender
Figure IV-5

Following the completion of the driving of the first split, the fender is extracted. The split in the second pillar is then completed and its fender extracted. The remaining pillar is again split in much the manner described earlier.

(5) Radius Pillaring

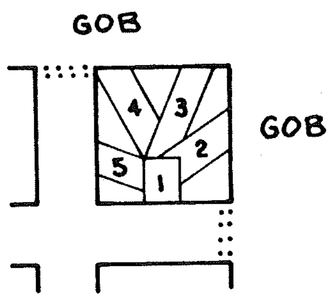
Another pillaring technique is shown in Figure IV-6.



Radius Pillaring
Figure IV-6

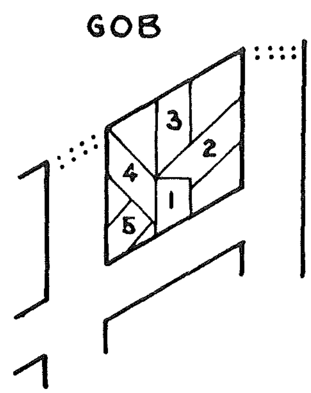
Here the pillar block is entered from the corner. As the first cut is taken, it is turned sharply to the gob side. The miner is then withdrawn and the pillared area is timbered off. A second cut is started by merely enlarging the first cut towards the center of the block. The completed second cut is timbered off. In a similar manner, the third cut is taken, subsequently enlarging upon the first cuts. The remaining stumps adjacent to the first cut are extracted during the final phases of removing the pillar. No bolting is done during the extraction process. This sequence of cuts appears to be extremely hazardous and is not recommended under any conditions.

A similar pattern is shown where the pillar block is entered not from the end point but from the mid-point. (Figure IV-7). The first cut is turned sharply to the gob side and is holed through.



Unnamed Pillaring Plan
Figure IV-7

Subsequent cuts are made following timbering. Each subsequent cut enlarges the first and is cut through the pillar block. The stump adjacent to the first cut is removed during the final phases of extraction. A similar technique is used in the extraction of pillars driven with angles crosscuts. (Figure IV-8).

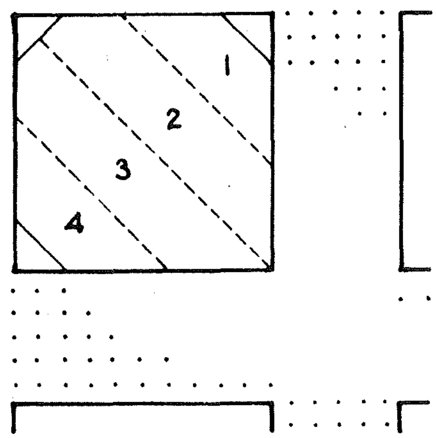


Pillaring of Diamond Block (Continuous Haulage)
Figure IV-8

Both of these pillaring plans are utilized using a conventional timbering roof control plan.

(6) Diagonal Pillaring

Diagonal pillaring was observed in only one mine. See Figure IV-9.



Diagonal Pillaring
Figure IV-9

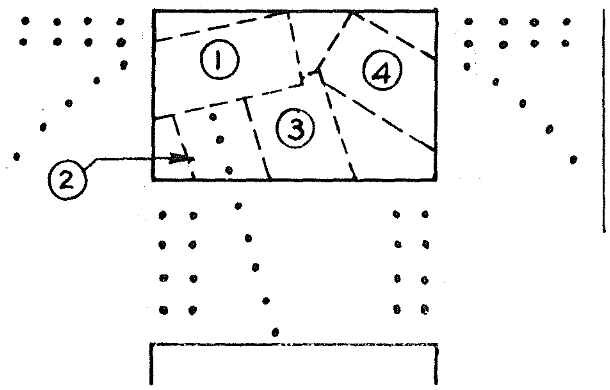
The first cut is taken approximately 10 feet in from the corner on the gob side and is driven entirely through to the gob. The second cut follows the setting of timbers in the first cut. The cut is not always driven through. If the roof is holding well and little spalling is observed on the stump adjacent to the first cut, then the second cut can be driven the full length through the pillar block. Subsequent cuts are taken until the entire block is extracted, leaving a small stump in the far corner. This method of pillar extraction is utilized with a conventional timbering plan, and where the roof conditions are extremely good.

(7) Other Pillaring Plans

The following text and figures describe some of the other unusual pillar recovery techniques observed or described during the course of the field visits.

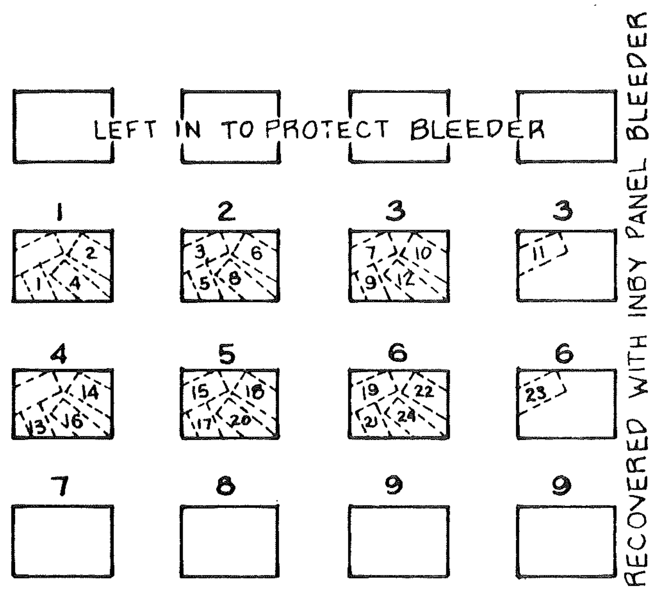
One plan which was observed has been utilized in a mine which had an extremely bad top. This mine had numerous slips and clay veins. Resin bolts were required along with a considerable amount of timbering with crossbars. The pillaring plan was designed to avoid the necessity of roof support internal to the pillar block. The plan left a considerable amount of coal. This was unavoidable, however, if the men were to be protected from the hazardous conditions inside the pillar block.

Blocks 34 ft. x 20 ft. were developed in advance. During the retreat phase, a single pass was taken on the narrow side of the block. It was driven as deep into the block as possible on a slight angle to break through the back side of the pillar. The miner then moved to a position on the long side of the pillar block. Subsequent cuts were then driven through the center of the block to the first cut. The miner was then moved to the far room side where a fourth cut was driven through, and as much coal extracted as possible. Figure IV-10 shows the extraction plan.



Pillar Extraction without Entering Block
Figure IV-10

A variation to this plan is shown where the second cut is taken from the adjacent room side and a third cut is taken into the adjacent pillar in a similar manner to the first cut shown in Figure IV-9. The miner is then brought back and the fourth cut is taken adjacent to the second cut. Figure IV-11 shows the entire sequence of cuts.



Complete Plan not Requiring Entry into Block
Figure IV-11

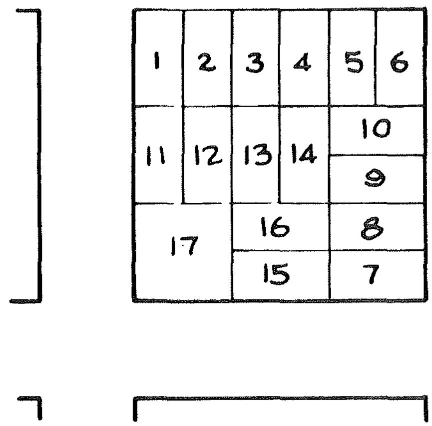
3. CONVENTIONAL MINING PILLARING TECHNIQUES

During the literature search portion of the study numerous methods of extracting pillars were reviewed using conventional equipment. During the course of the underground visits, however, conventional equipment was observed doing pillaring work on only one occasion. This is not unexpected however, since data shows that conventional mining comprises less than 20% of retreat mining and retreat mining comprises less than 20% of total mining.

The mine in question was using an open end method. However, team members spoke with operators and foremen who have used other techniques. As mentioned before, conventional equipment is seen less and less in pillar work. This is a natural result of the increased use of the continuous miner within the coal industry. However, there will always be some mines that will operate with conventional equipment. Continuous miners designed today cannot be used effectively in mines with certain geological conditions. Also, some operations will never be large enough to justify the expense of a continuous miner. Because of this, discussions on conventional mining of pillars have been included.

(1) Open Ending

Figure IV-12 shows a typical sequence of cuts for open ending. Successive cuts are taken on the gob side of the pillar, without leaving a fender between the working place and the gob. Subsequently, support is set adjacent to the cut to protect workers from the gob when making the next cut and the place is fully bolted. The sequence of cuts continues, until the end of the block is reached. The operations then shift to the opposite side, cutting in a similar manner, until the break-through. This pattern continues, leaving a square block, which is removed in a manner similar to the first few sequences of cuts. The final block is removed in the push phase and completes the entire sequence of extraction. Because of the make-up of the conventional mining crew, i.e., five basic operations, a minimum of five and often six pillars are active at any one time, and the pillar line is normally kept on a diagonal. The progress of extraction of each pillar is normally staggered to prevent undue pressures along the entire line. Some variations do exist in the open ending method, which are elaborated on in a subsequent chapter.

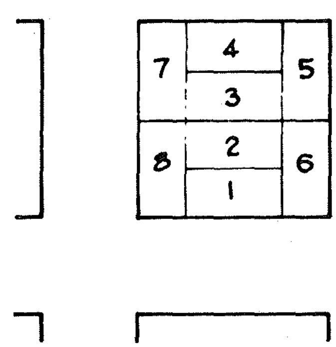


Open End
Figure IV-12

(2) Split and Fender

Numerous variations in the split and fender method exist for conventional mining. Again, these were not actually observed during the underground phase of the study; however, discussions were held with operators and men to gain an understanding of the inherent advantages and disadvantages of each system. Several of the variations are presented below, each of which offers a different recovery rate. Each one can be adjusted to the particular dimensions of the pillar. Also, while one variation might offer a higher percentage of pillar recovery, it will most likely require additional support. Depending on the particular conditions of the roof, the mine operator might opt for a lesser amount of recovery in return for reduced support requirements and correspondingly faster and safer recovery. Another deciding factor is whether the remaining stumps can crush out, or whether additional blasting will be required to remove them.

The first plan (see Figure IV-13) is a conventional splitting plan that offers near full recovery potential and has been utilized in areas with extremely good roof conditions.

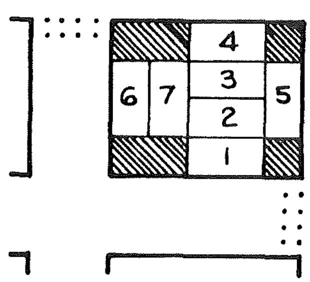


Split and Fender
Figure IV-13

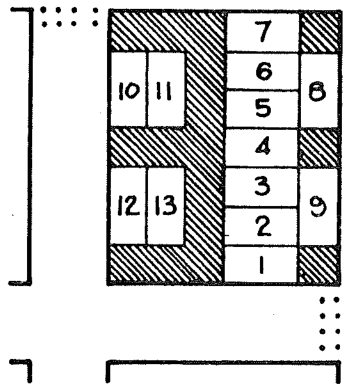
Prior to beginning recovery efforts the entire pillar is encircled with timbers on four-foot centers. Then, the first series of cuts are taken through the pillar to the gob. Following each cut timbers are set against the rib and the area is fully bolted. Following the breakthrough to the gob, the number 5 cut is taken. The number 6 cut is taken from the room side, number 7 cut from the split, and finally, the number 8 cut is taken from the room side. Of course, areas are timbered off as they are opened up to the gob.

Interestingly, this plan was used because the open ending technique used previously was not getting the desired recovery percentage. The coal seam observed had an 18-inch line of rock through the center which required a considerable amount of explosives to break. The result when open ending was that the rock would fall neatly in front of the work area, but much of the coal would be shot out into the gob, where recovery was impossible. Utilization of the splitting method kept much of the coal confined between the adjacent fenders and resulted in a higher recovery percentage.

The second and third methods shown (Figures IV-14 and IV-15) are basically the same except for the dimensions of the pillar block.



Variation of Split and Fender
Figure IV-14



Variation of Split and Fender
Figure IV-15

A series of cuts are taken through the pillar to the gob; each one in turn is fully bolted. Following the breakthrough, the split is timbered off. A cut or several cuts are then taken off the in-by fender, leaving a series of stumps. These may or may not be blasted, depending on the crushability of the stump. Finally, a series of cuts is taken off the remaining block from the room side. Final recovery for these methods ranges between 50% and 70%.

Undoubtedly other methods utilizing splits can be designed or are actually in use. Each of them however, requires the extraction of coal off the fender from the split. This presents a risk in terms of exposure which is not required in open ending.

V. ANALYSIS OF PILLAR EXTRACTION PRACTICES

V. ANALYSIS OF PILLAR EXTRACTION PRACTICES

In the section which follows criteria are established by which alternative pillar extraction plans can be compared. Each of the basic pillar extraction methods --split and fender, and pocket and wing, for continuous mining; and open ending, and split and fender for conventional mining--are compared for alternative dimensions and sequences. Each method is compared on the basis of the amount of free coal, the distance the continuous miner or loader must tram, the support requirements, the rate of recovery, the safety considerations, and the productivity rate.

The analysis shows that the dimensions of the pillars, the method of extraction, and the sequence of extractions can affect the safety and productivity of the pillaring section. Although no one technique is recommended as optimum, a basis by which mine operators can compare alternative plans is provided.

1. CRITERIA FOR EVALUATION

It is difficult, if not impossible, to select one optimum technique from the many used to extract pillars because each mine operates under unique conditions and selects and tailors a method to fit its conditions. In fact, several methods may be equally effective. It is important, however, for operators to be cognizant of other techniques and how and why they are used. Few operators can say without reservations that they have analyzed all alternatives and selected their current method because it optimizes safety and productivity.

In the sections which follow each of the major types of pillar extraction is analyzed. In order to provide a basis for comparison, several criteria have been established which are felt to be of importance. They include:

- Free coal - This is the percentage of total coal extracted where roof bolting is not required. This calculation includes the bolting of rooms and cross-cuts during advancement.

- Tram moves - This is the distance that the continuous miner or loading machine must move outside the pillar block in removing a pillar as dictated by the sequence of cuts.

- Support - This is the amount of support that must be set in the normal extraction of a pillar.

- Safety - This is an evaluation of the relative safety considerations inherent in the system.

- Productivity - This is an estimate of the overall rate in which a pillar can be extracted using the method given certain assumptions for standard times.

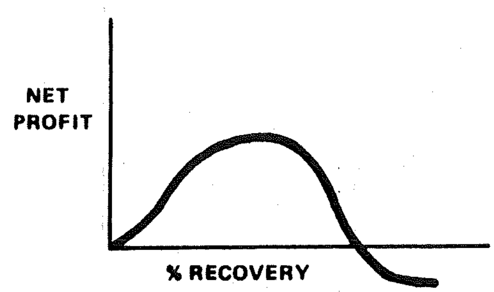
For comparative purposes, each measure is reduced to a common base. Thus, when a measure of the support requirements is established, it is adjusted to allow for relative differences in block dimensions for different methods.

The concept of free coal is important. There are mines operating under conditions where bolting is not required and where mining from under unsupported roof is routine and considered safe. In fact, many operators and workers view bolting inside of pillars as extremely hazardous and would rather mine the coal quickly, not being held up for the roof bolting. Most mines, however, must bolt inside the block. This bolting is time-consuming and hazardous. The extraction activity could be made more productive if the bolting process could be eliminated without sacrificing safety.

The tram moves required by the continuous miner or loading machine are critical, for this is pure non-productive time. Each equipment move requires time from the crew that is totally lost to production. A system that reduces or eliminates these moves is going to be more productive.

Reduction of support requirements for pillar extraction could mean cost savings, increased productivity, and reduced hazards. Cost savings are involved because fewer posts and bolts mean fewer materials, not to mention the labor involved. This means increased productivity, because a reduction in support requirements eliminates activities which interfere with the normal productive cycle. Hazards are reduced, because anything that eliminates or reduces the involvement of men in pillar extraction is going to reduce hazards.

Recovery rate is becoming more and more important today, both in terms of the current value of the coal resources today, and in terms of the loss of part of a resource which can never be recovered again. Also, consideration should be given to the effect the remaining coal might have on impending good roof falls. The mine operator is faced with a decision to increase the recovery rate at an additional cost or, if not, to determine where the break-even point is. The curve displaying the economics of the decision may look like Figure V-1.

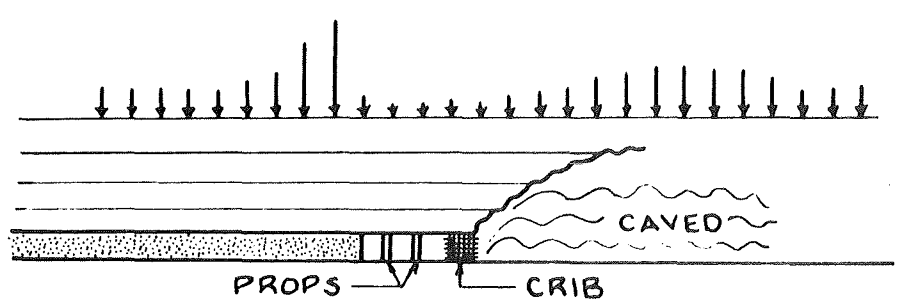


Symbolic Cost/Benefits of Pillar Recovery
Figure V-1

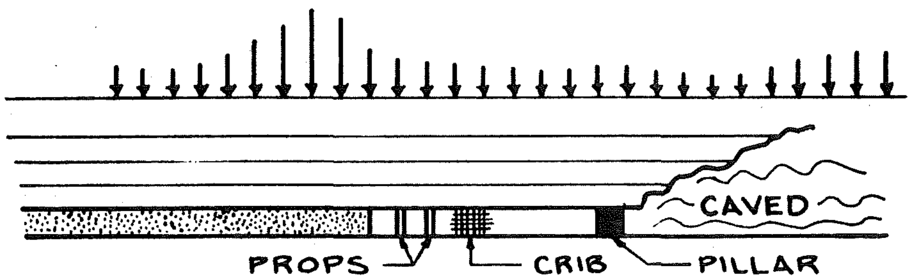
The actual break-even point is unknown. It is a function, of course, of individual conditions. But determination of the point might lead to decisions to eliminate coal recovery from "pushout stumps", substituting blasting, or perhaps increasing the percent of total extraction on advance to a point at which lifts taken around the perimeter of pillars might be a sufficient recovery rate. (See IV-1 (7)).

On the other hand the short-term cost of increasing the percent of pillar recovery cannot be the only consideration. Coal lost during pillar recovery can never again be mined. The coal extracted in the final stages of the recovery process is certainly less profitable than the earlier stages, but it has value and, if it is not mined now, it is lost forever. Operators might even be faced with a decision not to mine certain sections until the technology exists to do it efficiently.

Also very important in selecting coal recovery rates is the knowledge that stumps left in a pillar could impede a good roof fall. This in turn affects the mining of adjacent pillars. Bad falls prevent the relief of pressure on adjacent pillars. The cantilever effects of such problems can be seen by comparing the vertical pressures resulting from good mining practices, in Figure V-2, to the vertical pressures resulting from bad mining practices, in Figure V-3.



Vertical Pressure - Good Mining Practice
Figure V-2



Vertical Pressure - Bad Mining Practice
Figure V-3

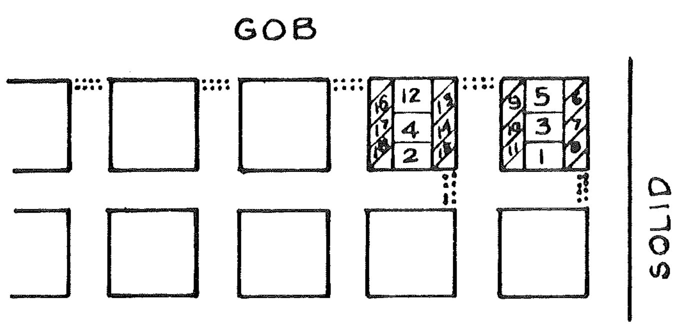
Safety considerations are of paramount importance. Unfortunately, it is often difficult to quantify the relative safety factors of alternative methods. Some of the considerations include the amount of solid coal adjacent to the work area, the extent in which men are required to expose themselves in the pillar, and the safety designed into the pillar during various stages of extraction.

The productive rate for alternative systems is difficult to assess. Each system observed differed due to local conditions, and the section was staffed accordingly. In these discussions productivity rate will be evaluated by considering delays inherent to the system caused by requirements for roof bolting, the availability of shuttle cars, and the necessity for the continuous miner to tram to different locations. These summarize the major delay factors which are common to the application of the system in any mine.

2. SPLIT AND FENDER PILLAR EXTRACTION - CONTINUOUS MINING

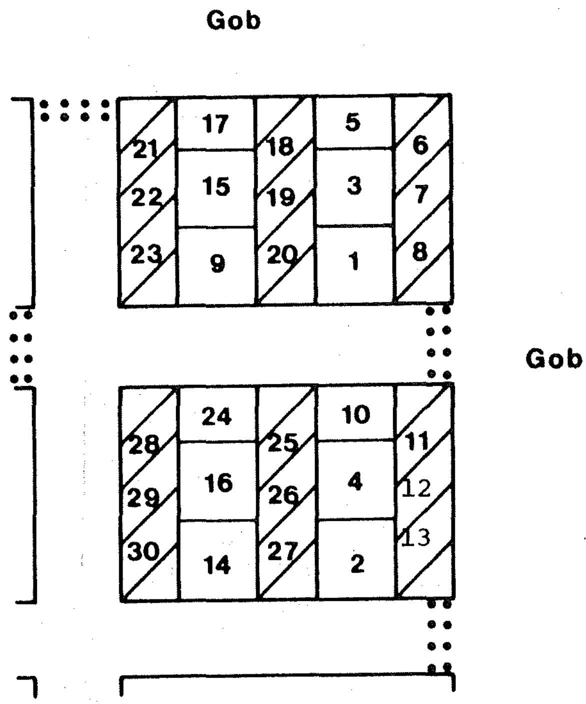
The technique of split and fender with continuous mining equipment, with its numerous variations, is the most common technique of pillar extraction found in retreat mining today. Both single split and multiple split systems are used. Often the multiple splits are both driven from the room entries.

The basic technique of split and fender entails the driving of a split through a pillar block to the gob in a manner which leaves a fender of coal narrow enough to be extracted by the continuous miner as it retreats out of the split. Under normal conditions, the pillar split is bolted to within 15 feet of the gob. This often necessitates working two pillars at the same time to allow production to continue while the roof is bolted. Fenders are extracted without any roof bolting. Figure V-4 shows the sequence of cuts.



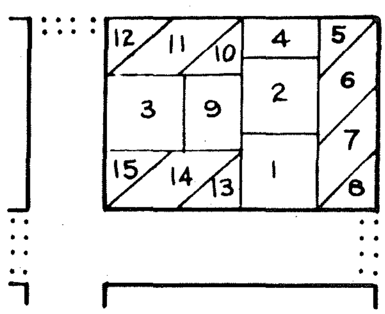
Split and Fender Method 1
Figure V-4

A second common technique of split and fender is driving two splits through the same block. This is used where blocks are too large to be mined with only a single split. This is shown in Figure V-5 below.



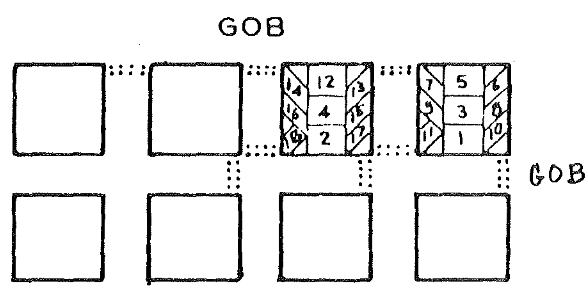
Split and Fender Method 2
Figure V-5

A third method seen is where splits are driven from two directions. This method, shown in Figure V-6 varies little from pocket and wing other than the dimensions of the fender versus the dimensions of the wing.



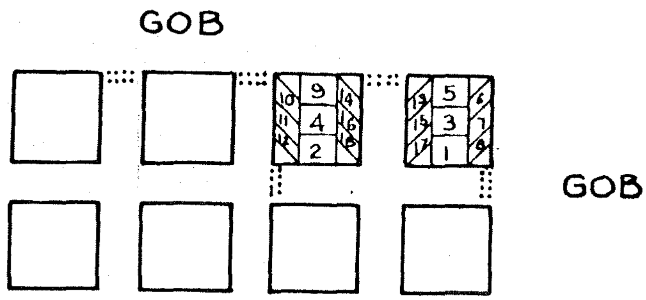
Multiple Split and Fender Method 3
Figure V-6

The next variation of split and fender differs in that the fenders are extracted to both the left and right side as the continuous miner retreats out of the split. This is common where access to the side of a block may be blocked by fallen roof debris or bottom heaving. It is, however, the standard technique in some mines. This technique is shown in Figure V-7.



Split and Fender - Method 4
Figure V-7

The last major variation of split and fender to be discussed in this section utilizes the technique of simultaneous fender extraction, but only from the entry. The plan is shown in Figure V-8.



Split and Fender - Method 5
Figure V-8

Each of the above techniques has its advantages and disadvantages. These are discussed in detail in the paragraphs below, considering the criteria established in the introduction to this section. The following assumptions are made in the evaluation:

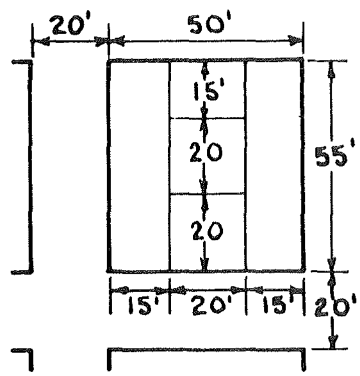
- splits will be driven cleanly through to the gob at a width of 20 feet;
- fenders and lifts will be taken 10 feet wide on a 60 degree angle;
- breaker posts are required any time any area is opened to the gob except when extracting fenders or wings;
- turn posts are required for each fender or wing lift and for each split or pocket driven;
- all splits and pockets are fully roof bolted to within 15 feet of the gob;
- a cut can be taken a depth of 20 feet prior to support;
- haulage systems cannot tram parallel to the gob, closer than the second crosscut outby; and

- rooms and crosscuts are driven 20 feet wide.

The percentage of free coal (unbolted area) varies only between single and multiple splits. The total area bolted for a single split technique is the same regardless of the method used to extract the fenders based on the assumptions stated above.

Before comparing the free coal available for different methods, it is interesting to note the effect on the percent of free coal when fender widths and pillar lengths are varied.

Consider first a 50' x 55' pillar, split 20 feet wide. The dimensions are shown in Figure V-9 below.



Typical Dimensions - Split and Fender - Method 1, 4, 5
Figure V-9

In the extraction of the 70' x 75' block, the percentage of the seam bolted is calculated as follows:

- the 20' x 55' room is bolted - area 1100 sq. ft.,
- the 20' x 50' crosscut is bolted - area 1000 sq. ft.,
- the 20' x 20' intersection is bolted - area 400 sq. ft., and
- the 40' x 20' split is bolted - area 800 sq. ft.

The total bolted area is 3300 square feet. Note that the last 15 feet of the split and the fenders were not bolted. This may not be the case for some mines, but for the analysis in this section all pillars will be treated in that manner. The total area extracted is 70' x 75' or 5250 square feet. Thus, the percent of free coal for this pillar plan is 37 percent.

The table below summarizes the effects of changes in the length of the pillar and the width of the fender and splits. The example just described is shown first.

Table V-1

Method	Case	Pillar Length	Fender Width	Split Width	Pillar Width	Bolted Area	Unbolted Area	Total Area	Percent Unbolted
1	1	55	15	20	50	3300	1950	5250	37.1
1	2	75	15	20	50	4100	2550	6650	38.3
1	3	55	10	20	40	3850	1400	5250	26.7
1	4	55	20	20	60	3500	2500	6000	41.7
1	5	55	15	15	45	3000	1875	4875	38.5
1	6	75	20	15	55	3825	3300	7125	46.3

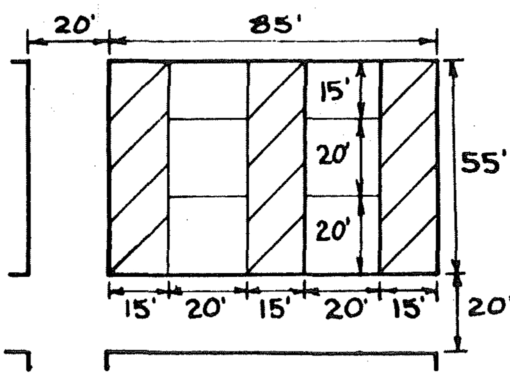
Free Coal - Split and Fender

Using Case 1 as a base, examine the effect of changes of pillar length, fender width, and split widths. Case 2 shows an increase in pillar length. This results in an increase in the amount of free coal. In Case 3 the fender width is decreased with a substantial decrease in the amount of free coal. Case 4 shows the corresponding effect by increasing the width of the fender. Case 5 shows that reducing the width of the split results in an increase in the amount of free coal. In summary, then:

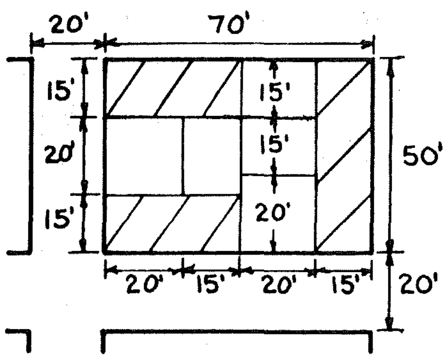
- an increase in the length of the pillar increases slightly the amount of free coal;
- an increase in the width of the fender increases significantly the amount of free coal; and
- a decrease in the split width increases slightly the amount of free coal.

Suppose it is possible to achieve all these gains, that is, increase the length of the pillar, widen the fenders by extracting them in a 90 degree angle to the split and decrease the split width. The net effect is shown in Case 6; 46.3 percent of the area extracted is unbolted, an increase of almost 9 percent just by changing basic dimensions. While this action does not have a direct effect on production, it could increase productivity and would certainly decrease the labor and material requirements for roof bolting.

As discussed before, the only other effect on free coal is the number of splits driven. For Methods 2 and 3, the amount of free coal can be calculated and compared to the other methods. Figures V-10 and V-11 show the basic dimensions for the calculations.



Typical Dimensions - Method 2 Split and Fender
Figure V-10



Typical Dimensions - Method 3 Split and Fender
Figure V-11

As before, the room, crosscut, intersections, and splits are fully bolted, except for the last 15 feet of the split. The results are shown in Table V-2 below. Case 7 is the multiple split from one direction (Method 4). Case 8 the multiple split from two directions (Method 5).

Table V-2

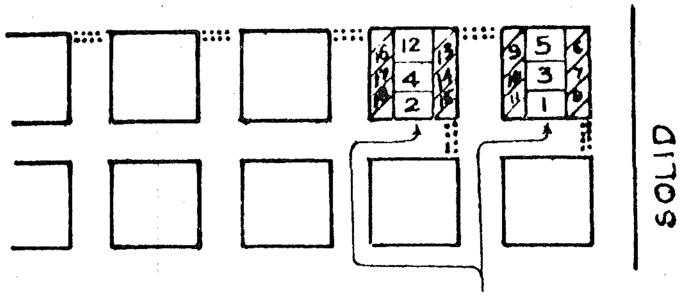
Method	Case	Pillar Length (ft.)	Fender Width (ft.)	Split Width (ft.)	Pillar Width (ft.)	Bolted Area (sq.ft.)	Unbolted Area (sq.ft.)	Total Area (sq.ft.)	Percent Unbolted (%)
2	7	55	15	20	85	4800	3075	7875	39.0
3	8	50	15	20	70	3900	2400	6300	38.1

Free Coal Split and Fender

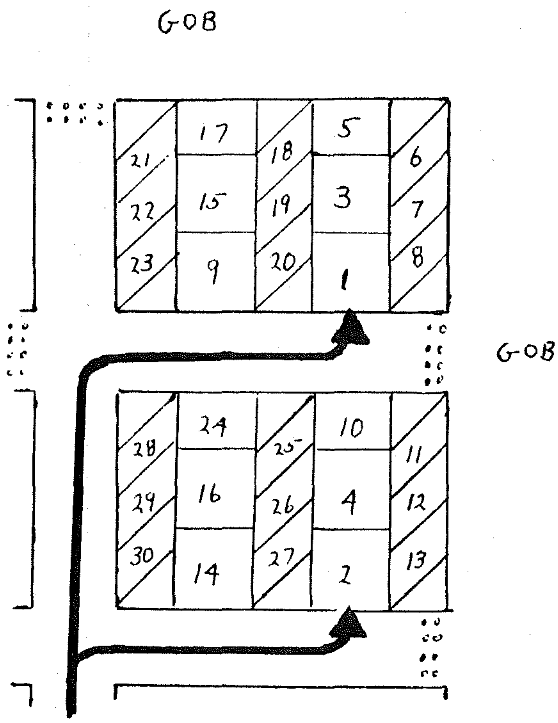
In both cases a slight increase in the amount of unbolted area results but hardly enough to justify a change in method from single split to multiple splits.

In summary, then, the split and fender variations offer little difference from the standpoint of free coal. The major gains occur not from changing the methods, but from varying the basic dimensions of the pillar and split.

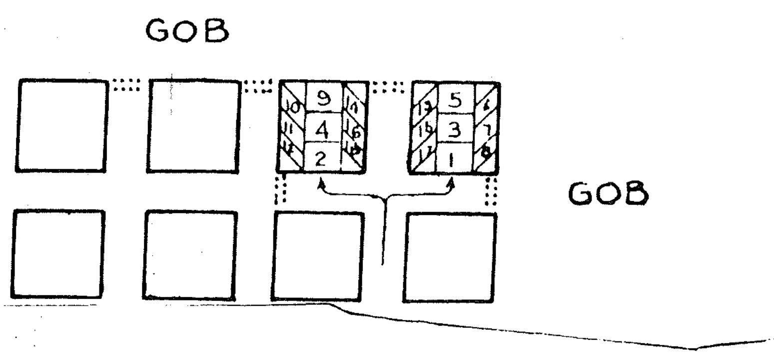
Next consider the tram moves required for the different methods. The evaluation must consider the distance moved by the continuous miner for each cut. Not included is the distance moved within the pillar block. Figures V-12 and V-13 and V-14 show examples of the paths used. Breaker posts are set during the course of extraction which greatly increases the tram distances for several methods. The tram paths for methods 3 and 4 are not shown.



Continuous Miner Tram Path - Method 1
Figure V-12



Continuous Miner Tram Path - Method 2
Figure V-13



Continuous Miner Tram Path - Metho
Figure V-14

After completion of cut number 8 for method 1 (see Figure V-12) the breaker posts are set in the crosscut. The continuous miner must then tram the entire distance around the outby pillar to get in position for cut number 12. For method 2 (see Figure V-13) the continuous miner must tram the entire length of the pillar between each cut. For method 3 (not shown) the continuous miner must tram between adjacent positions on the block with one cut taken out of an adjacent pillar. The tram requirements for method 4 (not shown) are nearly the same as method one without the tram requirements for fender extraction. Method 5 (see Figure V-14) eliminates the requirement to tram around the adjacent pillar. Table V-3 summarizes the results.

Table V-3

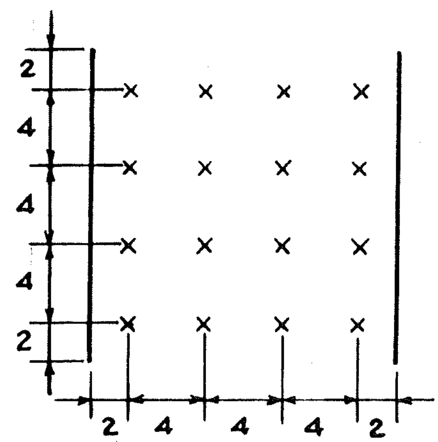
Method	Room Center (ft)	Crosscut Centers (ft)	Continuous Miner Tram Requirements: Driving Splits		Extracting Fenders		Total Area (sq. ft.)	Total Distance (ft)	Tram Ratio (Dist/ Area)
			No. of Moves	Distance (ft)	No. of Moves	Distance (ft)			
1	70	75	5	570	2	70	10500	640	.061
2	105	75	10	1795	2	70	15750	1865	.118
3	90	70	5	1210	1	-	6300	1300	.206
4	70	75	5	570	0	-	10500	1750	.054
5	70	75	5	350	1	35	10500	385	.037

Continuous Miner Tram Distance - Split and Fender

Since some of the pillar designs above contain different amounts of coal, it is convenient to summarize the results by comparing the area of the coal extracted to the distance trammed. This ratio is shown in the last column in Table V-3, and is found by dividing the total tram distance by the area of coal removed, (the product of room center times cross-cut center times the number of pillars extracted).

The results show that method 5 requires substantially less time for tramming the miner from place to place. Also the results show that both multi split methods require a considerable increase in the total tram requirements.

Support requirements can be separated into roof bolt requirements and timber requirements. The discussions on free coal are the best basis for analyzing roof bolt requirements. Assuming a bolt pattern on 4' centers, we can calculate a bolt density of .04 bolts per square foot. Figure V-15 shows the bolts required for a 20' x 20' segment of bolted area.

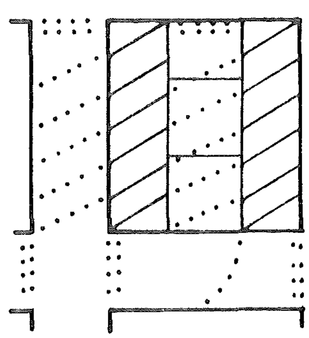


Bolt Pattern for 20' x 20' Segment
Figure V-15

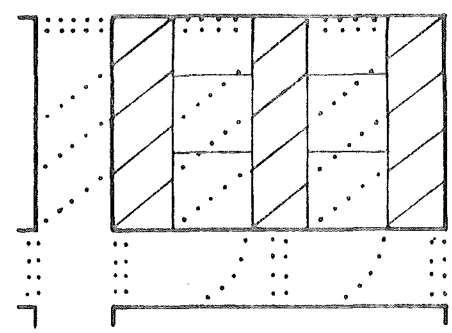
Sixteen bolts are placed in 400 square feet. Multiplying the bolted area times the bolts per unit area gives the total bolts required for each pillar methods. Dividing this total by the total area mined gives a ratio which makes possible the comparison of alternative pillar methods.

Breaker posts are normally set when a split breaks through to the gob. They are also set adjacent to any opening of rooms and crosscuts to the gob. Turn posts are set adjacent to each split. Roadway posts and rib posts are used under certain hazardous conditions. Roadway posts are used to reduce the roadways to sixteen feet or less thus reducing the chance of roof falls. Rib posts are used to reduce the hazards of rib rolls.

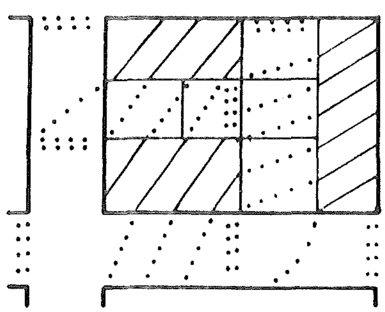
While the use of posts may vary somewhat for the pillar methods under discussion, these variances are not included in the analysis. Figures V-16 to V-20 below show the post requirements for each method. Throughout these figures dots are used to show post positions.



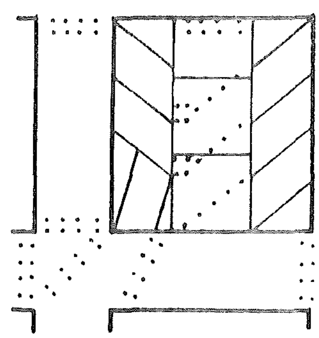
Timber Pattern - Method 1
Figure V-16



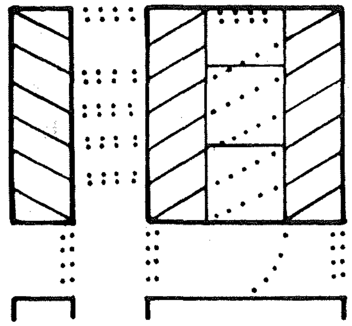
Timber Pattern - Method 2
Figure V-17



Timber Pattern - Method 3
Figure V-18



Timber Pattern - Method 4
Figure V-19



Timber Pattern - Method 5
Figure V-20

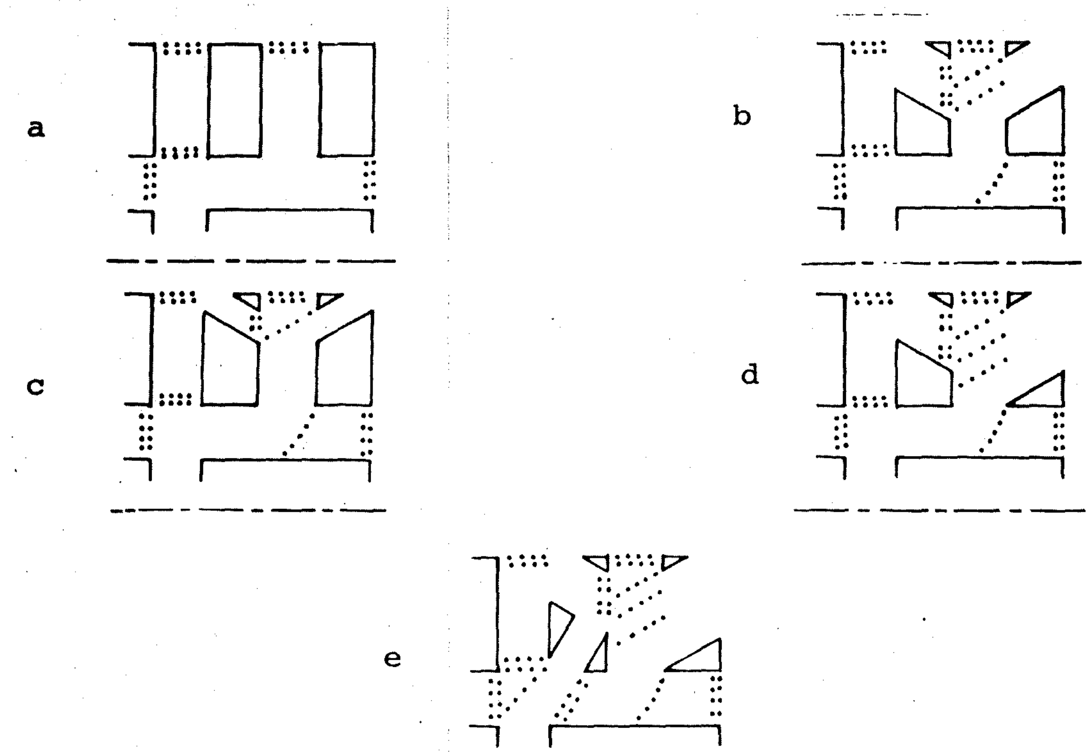
Table V-4 below summarizes the support requirements for alternative methods. The post ratio shown in the last column is the ratio of posts to the area of the coal extracted.

Table V-4

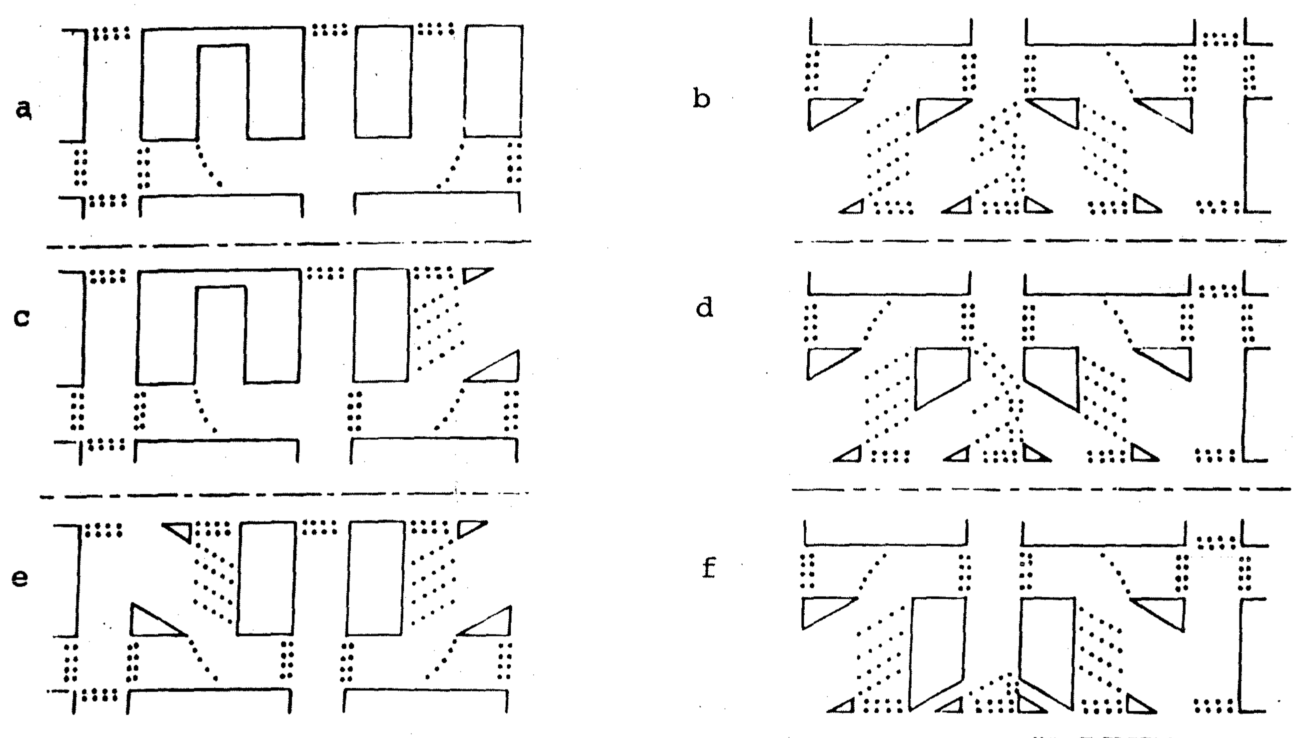
Method	Total Area (sq.ft.)	Bolted Area (sq.ft.)	Total Bolts	Bolt Ratio	Turn Posts	Breaker Posts	Total Posts	Post Ratio
1	5250	3300	132	.025	55	32	87	.017
2	7875	4800	192	.024	85	48	133	.017
3	6300	3100	156	.025	60	48	108	.017
4	5250	3300	132	.025	30	32	62	.012
5	5250	3300	132	.025	30	48	78	.015

Support Requirements - Split and Fender

For all of the split and fender methods the bolt ratios are essentially the same. The post requirements are different however. Method 4 uses the fewest post of those considered.



Sequence of Timbering - Method 4
Figure V-21



Sequence of Timbering - Method 5
Figure V-22

Both methods 4 and 5 are controversial. This is even more apparent if one visualizes the structure of the remaining coal pillar at varying stages in the extraction process. The exposure of men and equipment to hazards is apparent from Figures V-21 and V-22. Also method 3 requires two work places in the same pillar. Mining in one part of a pillar while roof bolting in another could increase the chances of a roof fall.

The last criteria for evaluation is productivity. In order to assess productivity one must be able to accurately determine the manpower requirements, the time requirements for each job assignment and the inherent delays or interference between operations. This is impossible since each pillar extracted is unique in its own way. However some broad generalizations can be made on the basis of the data already presented.

The two major operations in pillar extraction are the mining of the coal and the support of roof. Production can be tied directly to the mining of coal provided the support of roof takes less time, as is normally the case under fair to good roof conditions. In addition less than 40% of a pillar is normally bolted. Thus a productivity assessment can be tied directly to the mining cycle. The next consideration is the actual mining operation. It is reasonable to assume that there is no difference in the time and effort to make a cut using one pillar method versus another.

Using these assumption, production and productivity can be directly related to delays to the mining cycle. There are two types of delays that are attributable to the method, that is excluding maintenance or face haulage or other delays common to all methods. These are: delays for tramming and set-up for a new cut and delays to set support. A review of the previous discussions shows significant variations in the tram ratios.

Methods 1, 4 and 5 have significantly less delays due to repositioning of the continuous miner. Of these, method 5 should have the least amount of delay time attributable to tramming. Support delays are minimized in methods 4 and 5. An analysis of the variations in support requirements shows that the additional posts required for method 5 are breaker posts which can normally be set without causing a delay to the mining cycle. Thus support delays should be about the same for both.

On the basis of this analysis then method 5 appears to have the least amount of lost time and assuming equal staffing should be the most productive.

In summary, the advantages and disadvantages of each system should be obvious. Each system must be evaluated against the conditions of the individual mine. If alternative systems are acceptable to the mine and to MESA, then these criteria will allow the operator to select the optimum split and fender technique. Table V-5 summarizes the results.

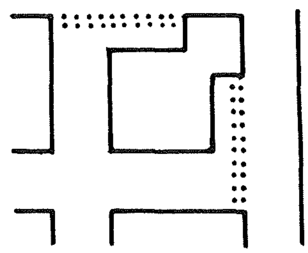
Table V-5

Method	Description	Percent Unbolted	Tram Ratio	Post Ratio	Safety Considerations
1	one split	37.1	.061	.017	No Adverse Considerations
2	one direction-multiple split	39.0	.118	.017	No Adverse Considerations
3	two direction single split	38.1	.206	.017	Two work places in the same pillar
4	simultaneous fender extraction	37.1	.054	.012	High exposure in split
5	simultaneous fender extraction from the room	37.1	.037	.015	High exposure in room

Summary of Analysis - Split and Fender - Continuous Mining

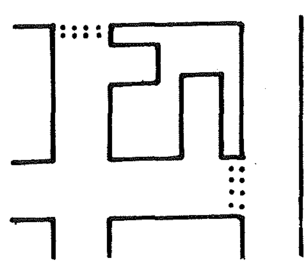
3. POCKET AND WING - CONTINUOUS MINING

The technique of pocket and wing is one that evolved naturally with the development of the continuous miner. Open end mining was the counterpart of pocket and wing when conventional equipment was utilized. In open end mining, successive lifts were taken off the gob side of the pillar, normally at a depth of about ten feet. Following the loading out and bolting of a place, a double row of breaker posts was set at the gob line. The pillar looked much like Figure V-23.



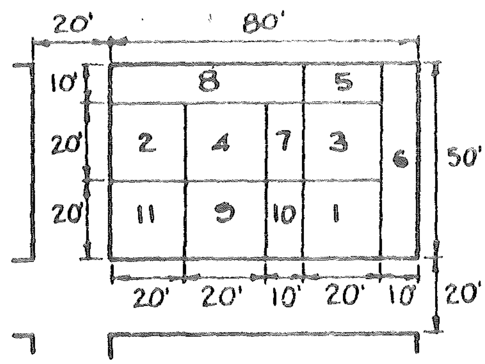
Open End Pillar Extraction - Two Active Places
Figure V-23

When the continuous miner was utilized with its capability to "reach" into unbolted areas, it was natural to substitute a wing of coal for the breaker posts. This wing provided the necessary support of the roof and protection from the gob and could be easily extracted as the miner retreated out of the pocket. This technique is shown in Figure V-24 below.



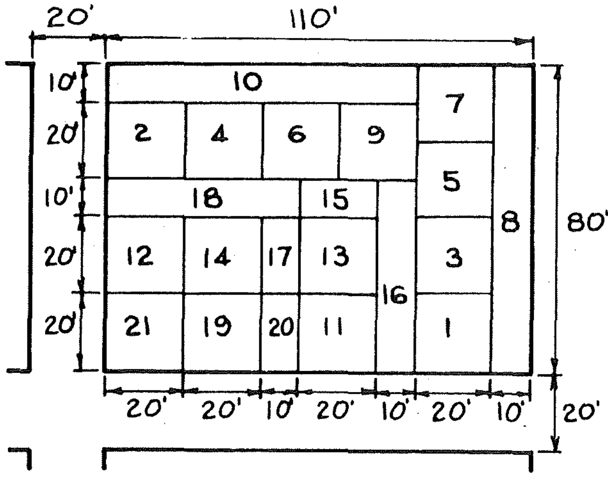
Pocket and Wing Pillar Extraction
Figure V-24

The basic differences in pocket and wing methods are either in the size of the block, or in the sequence of cuts required for working two blocks simultaneously. In the examples below, 20 foot wide pocket spans and 10 foot thick fenders are utilized to facilitate calculations. Method 1 in the analysis of pocket and wing shown in Figure V-25, is the basic block configuration for pocket and wing. This pattern entails a sequence of cuts entirely within one block. Only one block is worked at a time. Note that in the following diagrams the dimensions shown are consistant. They may be, however, too large for specific roof conditions. Although twenty foot wide pockets are not uncommon, smaller widths are often the case. In order to facilitate comparisons, all dimensions have been standardized.



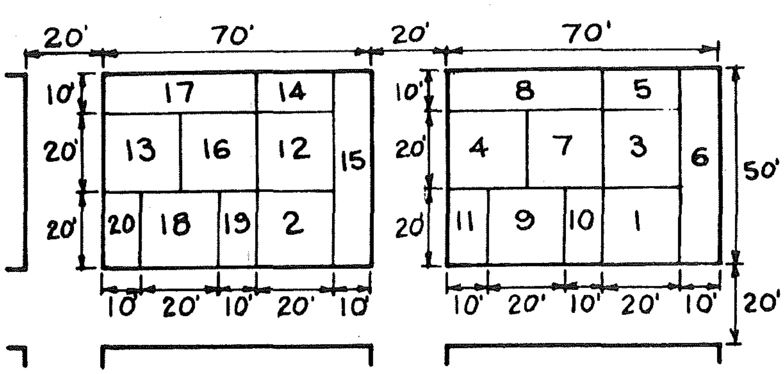
Pocket and Wing--Method 1
Figure V-25

Method 2 in the analysis shown in Figure V-26, is the same configuration as Method 1 but the basic block dimensions have been enlarged. In designing a pocket and wing system where one block is to be mined in its entirety, the dimensions should be such that the number of vertical cuts required to break through to the gob is equal to the number of horizontal cuts required to break through to the completed first pocket.



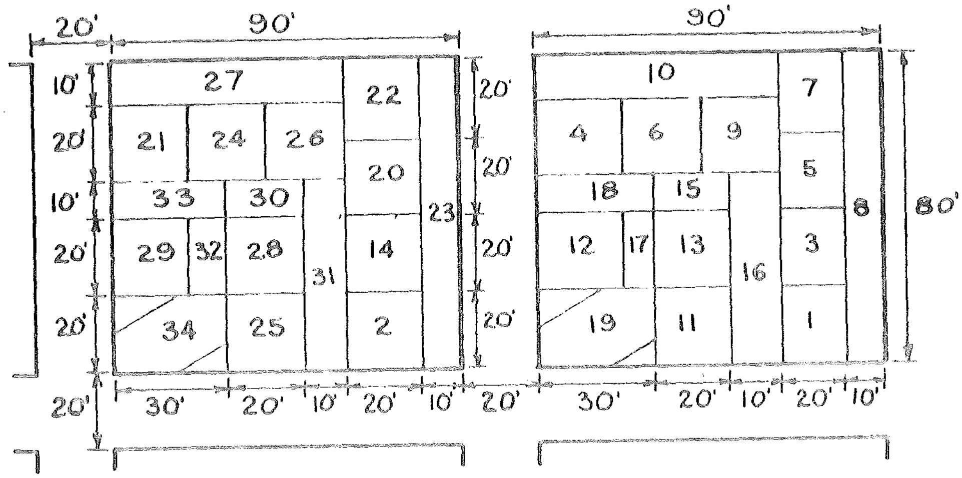
Pocket and Wing--Method 2
Figure V-26

Method 3 in the analysis, shown in Figure V-27, shows a sequence of cuts for completing a pair of blocks. It is impossible to complete mining in only one block with these basic dimensions without encountering a delay for bolting.



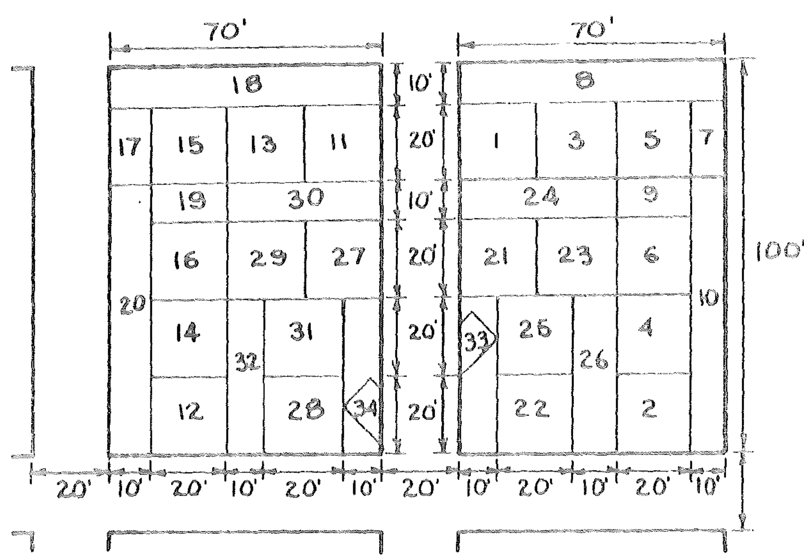
Pocket and Wing--Method 3
Figure V-27

variation of Method 3. The block size has been enlarged, and the pushout stump is extracted in a different manner. Again, two blocks are required to sequence miner and bolter operations.



Pocket and Wing--Method 4
Figure V-28

Method 5 in the analysis shown in Figure V-29, is a variation of Method 1, where the adjacent block sequence is reversed to allow simultaneous extraction of both blocks from a single entry. It should be noted that the plan presented in Method 5 was or is to be utilized only where adverse roof conditions or falls prevented using the normal mining sequence.



Pocket and Wing--Method 5
Figure V-29

The analysis of the five methods is based upon the same criteria established earlier for split and fender. Each method will be analyzed in terms of free coal, tram moves, support requirements, recovery rate, safety, and productivity rate.

The amount of free coal available for each method is summarized in Table V-6 below. The differences among the methods is strictly a function of the configuration of the pushout stump which remains rather than to the method itself. Recall that the calculation of the total area extracted includes the area of the rooms and crosscuts. No bolting is required within 15 feet of the point in which the pocket breaks through to the gob. All room and crosscut entries are 20 feet.

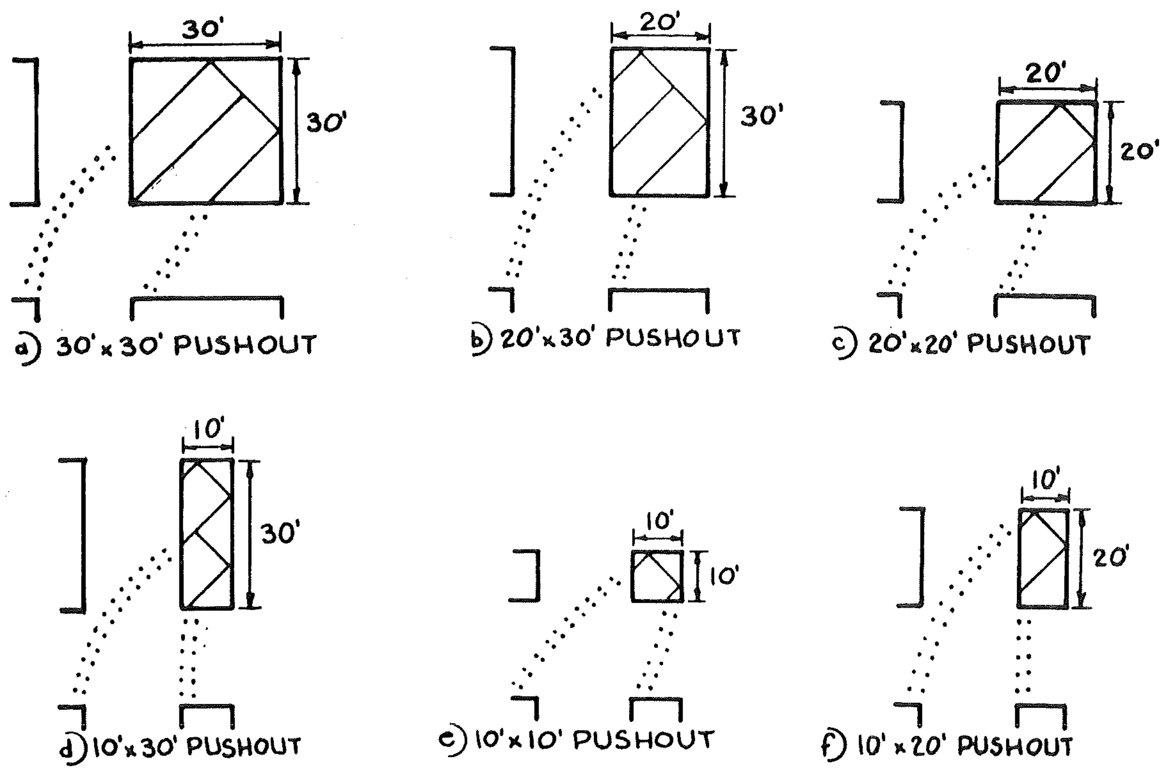
Table V-6

Method	Pillar Width (ft)	Pillar Length (ft)	Pocket Width (ft)	Wing Width (ft)	Bolted Area (sq.ft)	Unbolted Area (sq.ft.)	Total Area (sq.ft)	Percent Unbolted (%)
1	80	50	20	10	4400	2600	7000	37.1
2	110	80	20	10	8200	4800	13000	36.9
3	70	50	20	10	8000	4600	12600	36.5
4	90	80	20	10	14000	8000	22000	36.4
5	70	100	20	10	14000	7600	21600	35.2

Free Coal - Pocket and Wing

It can be seen that no one method shows any significant variation in the amount of free coal available. The difference that does exist is caused by the pushout stump configuration.

There are several different pushout stump configurations worthy of note. Figure V-30 displays each alternative. Notice that if a pushout dimension is larger than 40 feet, then it can be split to a smaller pushout.



Pushout Stumps Pocket and Wing
Figure V-30

The pushouts shown in a) and b), above, are difficult to mine because of their size. They are almost too small to split, yet too large to extract as a pushout. In any case, a pushout 20 feet square or smaller is preferable in terms of ease of recovery and recovery rate.

The tram distance for the continuous miner is the second criterion for evaluation. As in split and fender, the need to avoid tramming between two pillars should be apparent. The number of times a move is necessary is a measure of the disruptions to the productive cycle. The move ratio shown in Table

V-7 is the ratio of the number of moves to the area of coal removed.

Table V-7

Method	Room Center (ft)	Crosscut Center (ft)	No. of Moves	Total Distance (ft)	Total Area (sq.ft)	Tram Ratio (Dist/Arm)	Move Ratio
1	100	70	7	1050	7000	.015	1.00
2	130	100	16	3000	13000	.231	1.23
3	90	70	14	2190	12600	.174	1.11
4	110	100	26	4720	22000	.215	1.18
5	90	110	26	4340	21600	.201	1.30

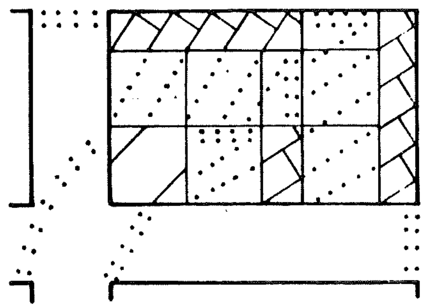
Continuous Miner Tram Distance -- Pocket and Wing

Analysis of Table V-7 shows that the methods using larger blocks require more tramming and a higher number of disruptive moves per square foot of coal extracted. Also, the mining of two blocks simultaneously requires additional tramming and additional moves compared to the mining of one block.

Support is the third criterion for evaluation. To allow comparisons with other techniques, the support requirements have also been separated into roof bolting and timbering requirements. The bolt ratio was again calculated using a bolt density of .04 bolts per square foot. The calculations for the number of turn posts and breaker posts were based on the following assumptions:

- one cut in the wing is taken for every 10 feet of advance in the pocket,
- five turn posts are set for every wing cut, and
- eight breaker posts are placed where needed, i.e., where the pocket breaks through the gob.

This is best illustrated by referring to Figure V-31, which shows the support requirements for Method 1. Support requirements for Methods 2 to 5 can be readily calculated by applying the above-mentioned rules to the respective mining plan, (Figures V-26 to V-29).



Timber Plan -- Pocket and Wing -- Method 1
Figure V-31

Table V-8 summarizes the results for the five methods discussed in this section:

Table V-8

Method	Total Area (sq.ft)	Bolted Area (sq.ft)	Bolt Ratio	Turn Posts	Breaker Posts	Total Posts	Post Ratio
1	7000	4400	.025	79	40	119	.017
2	13000	8200	.025	164	56	220	.017
3	12600	8000	.026	162	80	242	.019
4	22000	14000	.025	268	96	364	.017
5	21600	14000	.026	272	96	368	.017

Support Requirements - Pocket and Wing

It can be seen that the different methods do not require a significant difference in the number of bolts per unit area. The same is true for the posts, except for Method 3. In Method 3, the sequence of cuts includes two blocks. This leads to a slightly higher ratio of posts per unit area.

From the prospective of safety it should be noted that each method requires two working places in the same pillar during some portion of the extraction process, and each requires the extraction of a pushout stump. The technique of twinning required for Method 5 is obviously more hazardous. This is apparent from an examination of the exposure of men to the gob while driving the first pocket in the second pillar.

Again the assessment of productivity shows little variation between the methods when considering roof support requirements. The single significant factor appears to be tram delays. Method 1 stands out as the pocket and fender method which has the fewest disruptive moves and the least total distance required for tramming when considered on the basis of the area of coal to be removed.

Table V-9 summarizes the results. Method 1 with the smallest block offers the greatest advantages under all conditions considered.

Table V-9

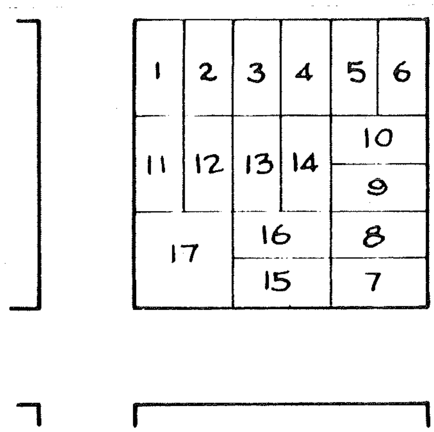
Method	Description	Percent Unbolted	Tram Ratio	Move Ratio	Post Ratio	Safety Considerations
1	50'x80'Block	37.1	.150	1.00	.017	Two working places
2	80'x130' Block	36.9	.231	1.23	.017	Two working places
3	(2) 50'x70' Blocks	36.5	.174	1.11	.019	Two working places
4	(2) 90'x80' Blocks	36.4	.215	1.18	.017	Two working places
5	(2) 70'x100' Blocks (Twinned)	35.2	.201	1.20	.017	High exposure

Summary of Analysis - Pocket and Wing

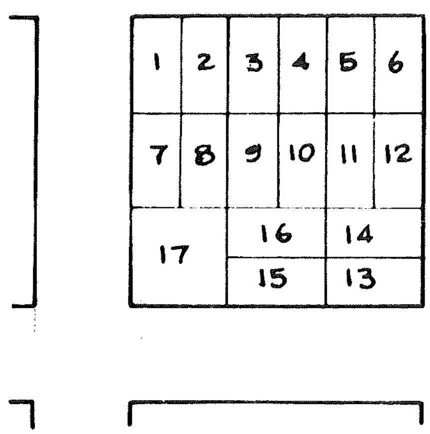
4. OPEN END - CONVENTIONAL MINING

Two variations for open end mining of pillars are common. The first variation calls for the extraction of the pillar from one direction, the second from two directions. Unlike continuous mining, the normal practice is to have only one active work place in each pillar at a given instant. On occasions when a new section is being retreated or an old one completed, more than one place may be mined in a pillar at the same time to provide adequate work places for the section crew. This activity is rarely pursued for more than a few of the pillar cuts.

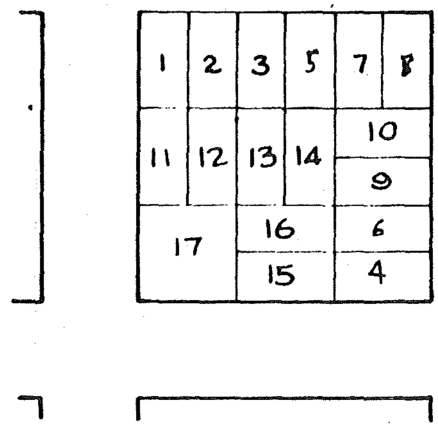
The two variations used in this discussion differ only in the geometry of the remaining pillar. Figures V-32 and V-33 show two methods to be evaluated. A third method similar to method 1 differs only in the sequence of cuts and is shown in Figure V-34.



Open End - Method 1
Figure V-32



Open End - Method 2
Figure V-33

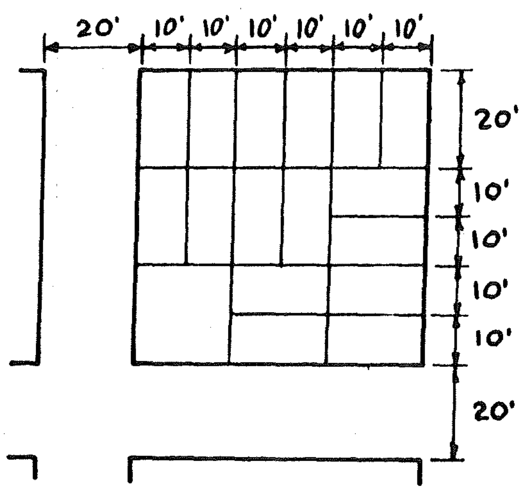


Open End - Method 3
Figure V-34

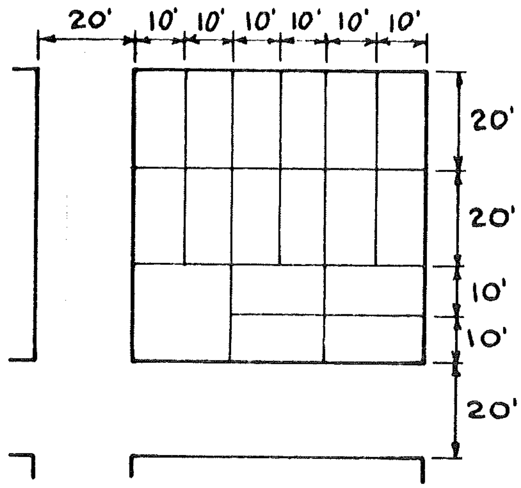
The criteria for evaluation of conventionally mined pillars are the same as those used in evaluation of continuous mined pillars, except:

- bolting must be completed to within 10 feet of the gob, and
- a cut can be taken at a depth of 10 feet prior to support.

To compare these methods, first consider the free coal. Figures V-35 and V-36 show the basic dimensions of the pillars.



Open End Dimensions - Methods 1 and 3
Figure V-35



Open End Dimensions - Method 2
Figure V-36

For method 1, cuts 6, 10, 14, 16, and 17 do not require roof support. For method 2, cuts 6, 12, 14, 16, and 17, do not require roof support. For method 3, cuts 8, 10, 14, 16, and 17 do not require roof support. There is no difference is free coal for these methods. Table V-10 shows the calculations.

Table V-10

Method	Pillar Length (ft)	Pillar Width (ft)	Room Width (ft)	Crosscut Width (ft)	Total Area (sq.ft)	Total Unbolted Area (sq.ft)	Percent Unbolted
1	60	60	20	20	6400	1200	18.8
2	60	60	20	20	6400	1200	18.8
3	60	60	20	20	6400	1200	18.8

Free Coal - Open End

The amount of free coal is substantially less than that for continuous mining pillar techniques.

The analysis of tram distances for conventional mining must differ slightly from continuous mining, since five separate operations are involved instead of two. It is not necessary to determine the total tram distance of all equipment combined, since the constraining factor is the resultant time delay.

A typical section has five work places in conventional pillaring. Table V-11, below, shows the results.

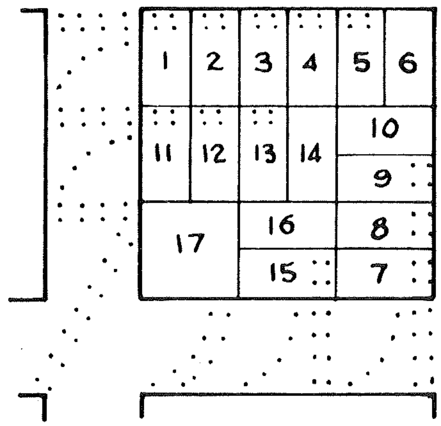
Table V-11

Method	Pillars	Width (ft)	Length (ft)	Total Area Per Pillar (sq.ft)	Total Area (sq.ft)	Tram Distance Total (ft)	Ratio
1	5	60	60	3600	18000	29480	1.64
2	5	60	60	3600	18000	28640	1.59
3	5	60	60	3600	18000	26960	1.50

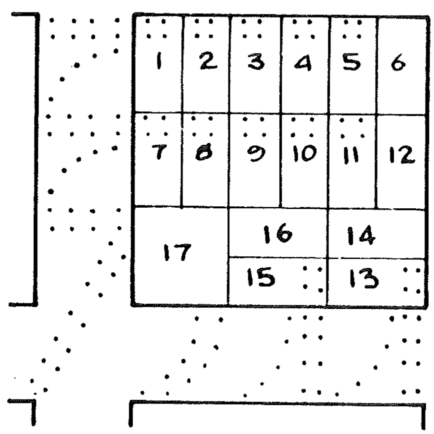
Tram Distances - Open End

The slight difference between methods 1 and 2 can be accounted for by the tram distance inside the pillar block. As expected, the results are quite similar. Method 3 shows a fairly substantial decrease in the amount of tramming required. This is because on several occasions two operations are performed on a block in succession.

Support requirements are again separated into bolting and timbering requirements. A bolt pattern on 4' centers gives a ratio of .04 bolts per square foot. Timbering plans for each method are shown in Figures V-37 and V-38. The support requirements are displayed in Table V-12.



Timber Plan - Method 1 and 3
Figure V-37



Timber Plan - Method 2
Figure V-38

Table V-12

Method	Total Area (sq.ft)	Bolted Area (sq.ft)	Bolts	Bolt Ratio	Turn Posts	Breaker Posts	Total Posts	Post Ratio
1	6400	5200	208	.033	20	112	132	.021
2	6400	5200	208	.033	20	112	132	.021
3	6400	5200	208	.033	20	112	132	.021

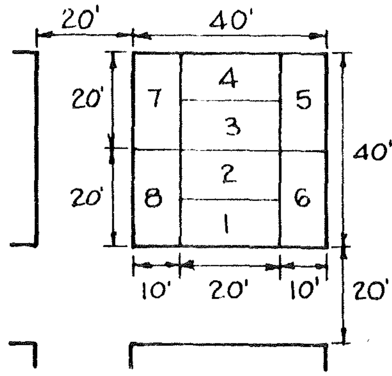
Support Requirements - Open End

Little can be said concerning the safety aspects of each method. Open ending is a hazardous system due to the high amount of exposure. Certainly of the three methods the first has the greatest inherent safety since the block is mined in only one direction at a time. In terms of production and productivity the advantage appears to be with method 3 since a slight decrease in tram time is inherent in the method.

In summary, the analysis has shown few differences among the three methods. It apperas that the selection of one over the other is merely a matter of local preference.

5. SPLIT AND FENDER - CONVENTIONAL MINING

Several variations exist for split and fender pillar extraction for conventional mining. However, little basis for comparison exists with the exception of the one method analyzed which leaves large stumps of unmined coal. All the methods have been described in detail in Section IV. The analysis of the split and fender method is shown so that it might be compared to open end methods. The sequence of cuts and dimensions for a typical split and fender are shown in Figure V-39.



Split and Fender Conventional
Figure V-39

The percent of free coal for this technique can be easily calculated since cuts 5, 6, 7 and 8 require no bolting. The results are displayed in Table V-13.

Table V-13

Method	Pillar Length (ft)	Pillar Width (ft)	Room Width (ft)	Crosscut Width (ft)	Total Area (sq.ft)	Total Unbolted (sq.ft)	Percent Unbolted
1	40	40	20	20	3600	800	.222

Free Coal - Split and Fender (Conventional)

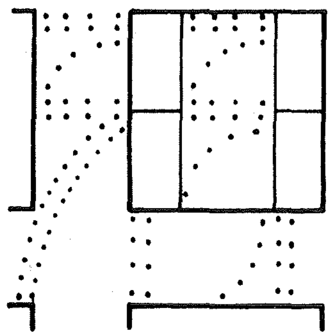
The tram distances are calculated based upon the total distance trammed by the loading machine in the process of extracting five pillars. The results are displayed in Table V-14.

Table V-14

Method	Pillar	Width (ft)	Length (ft)	Total Area Per Pillar (sq. ft.)	Total Area (sq.ft)	Tram Distance (ft)	Ratio
1	5	40	40	1600	8000	9550	1.19

Summary of Analysis - Split and Fender (Conventional)

Support requirements are separated into bolting and timbering, and depicted in Table V-15. A bolt pattern on 4 ft. centers gives a ratio of .04 bolts per square foot. The timber plan is shown in Figure V-40 below.



Timber Plan - Split and Fender - Conventional
Figure V-40

Table V-15

Method	Total Area (sq.ft)	Bolted Area (sq.ft)	Bolts	Bolt Ratio	Turn Posts	Breaker Posts	Total Posts	Post Ratio
1	3600	2800	112	.031	20	70	90	.025

Summary of Analysis - Split and Fender (Conventional)

The favorable effect on recovery rate for this system was a prime consideration for its adoption. In the coal bed where it was utilized the seam had a band of shale 10" thick through the center requiring a large shot charge to break it up. When the face was shot, the coal would not scatter into the gob but would be contained between the adjacent fenders, thus improving the recovery rate.

The major safety consideration is the hazards associated with the extraction of fenders. The crew is required to work from within the split with a thin fender of coal for protection.

6. COMPARATIVE ANALYSIS

It is convenient at this point to summarize the results of this section to provide an easy comparison of the different techniques. Obviously the factors analyzed are not all that must be considered in the selection of the "optimum" pillaring method. Trade-offs exist between productivity and safety that can only be resolved on an individual basis. However, this analysis does provide a first step in the decision process for the selection of a pillaring technique for any mine. Table V-16 summarizes the results.

Table V-16

Pillar Plan	Method	Description	Percent Unbolted	Tram Ratio	Post Ratio	Safety Considerations
S&F-Cont.	1	Single Split	37.1	.061	.017	None apparent
S&F-Cont.	2	(Multi Split Single Direction)	39.0	.118	.017	Two work places
S&F-Cont.	3	(Multi Split Two Directions)	38.1	.206	.017	None apparent
S&F-Cont.	4	(Single Split Simultaneous Fender)	37.1	.054	.012	Exposure
S&F-Cont.	5	(Single Split Simultaneous Entry)	37.1	.037	.015	Exposure
P&W-Cont.	1	50'x80' Block	37.1	.150	.017	Two work places
P&W-Cont.	2	80'x130' Block	36.9	.231	.017	Two work places
P&W-Cont.	3	(2) 50'x70' Blocks	36.5	.174	.019	Two work places
P&W-Cont.	4	(2) 90'x80' Blocks	36.4	.214	.017	Two work places
P&W-Cont.	5	(2) 70'x80' Blocks Twinning	35.2	.201	.017	Two work places High Exposure
O.E. Conv.	1	Multi Directional	18.8	1.64	.021	High Exposure
O.E. Conv.	2	Single Direction	18.8	1.59	.021	High Exposure
O.E. Conv.	3	Multi-Directional	18.8	1.50	.021	High Exposure
S&F-Cont.	1	Single Split	22.2	1.19	.025	High Exposure

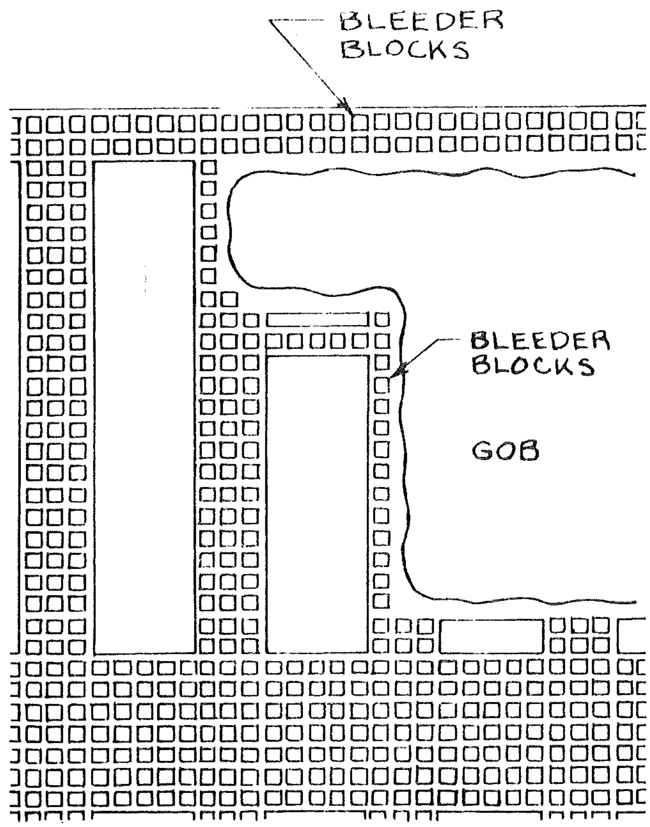
Summary of Analysis

The reader should recall that a high percent of pillar extraction without bolting is a favorable condition, e.g., high percent free coal. Similarly, a low tram ratio (feet trammed per square foot of pillar extracted) and a low post ratio (posts set per square foot of pillar extracted) are favorable conditions.

In summary the technique of split and fender with the pillar block dimensioned so that only one split is required is the best overall technique, although specific local conditions might dictate the selection of another technique.

7. MODIFIED OLD BEN SYSTEM

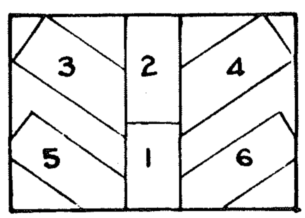
Another pillaring method not previously addressed merits discussion at this point: The Modified Old Ben System is widely practiced in central Illinois. This system calls for the development of long panel headings off a set of development headings. The inby end of the panel headings are opened into the bleeder blocks and the bleeder headings are advanced to the next panel. The blocks remaining are typically 300 feet wide and about a thousand feet long. After development the coal is systematically extracted from one end by driving a pair of rooms and crosscuts from one panel entry to the other, then extracting the chain and barrier pillars. Figure V-41 shows the typical layout.



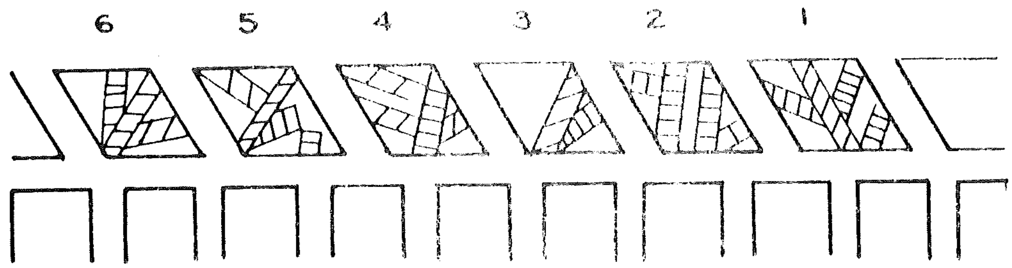
Mine Plan - Block System
Figure V-41

This system, as observed, does not lend itself to the type of analytical analysis present earlier in this section. First, local conditions permit the roof to remain unbolted on retreat if at least 8 inches of top coal are left in place. Second, no timbers are set inside the pillars under normal conditions. This system does not necessarily eliminate the need for bolting or timbering. If bolting is required, the apparent advantages of the system are lost, since only one work place is active at a time and men would have to wait on bolting with the resulting delay time severely affecting production.

Basically the three types of pillars extracted in this system are barrier pillars, chain pillars and bleeder blocks. The bleeder blocks are about 45' x 70' and are normally square or diamond shaped. These blocks are adjacent to the gob, have often stood for several months prior to mining, and have deteriorated greatly. Often adjacent entries are cluttered with debris from roof falls, rib spalling, and bottom heaving. Consequently, only one entry may be available for access to the block. Extraction of the block follows a standard split and fender technique without posting or bolting. Recovery rate is poor, mostly due to the hazardous conditions. Figure V-43 shows the actual cuts for a number of blocks. The variations in the approach are apparent.

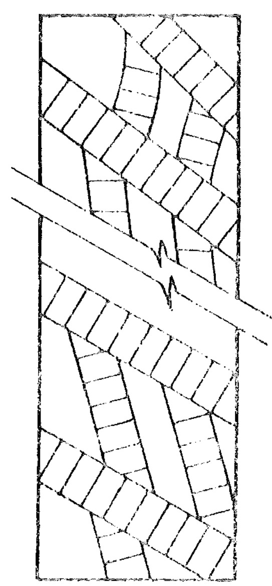


Sequence of Cuts - Bleeder Block
Figure V-42



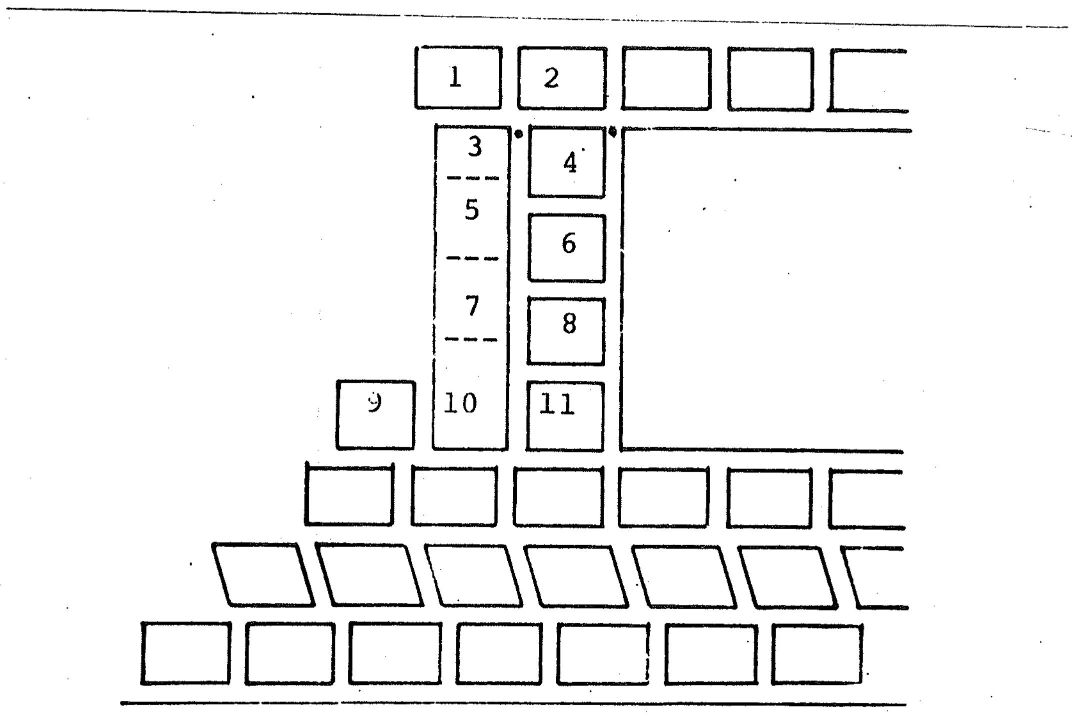
Typical Cuts in a Number of Bleeder Blocks
Figure V-43

The barrier pillars range from 12 to 34 feet wide and are about 300 feet long. They are extracted by driving splits off the end and extracting the resultant fender. The splits are driven at an angle of 45 to 90 degrees. Obviously, the angle is significant. For example, penetration of the split at a 45° angle means that the distance to drive through the gob increases by 1.4 times the width of the pillar. The time of exposure inside the pillar increases as well. Figure V-44 shows a typical extraction pattern.



Typical Cuts - Barrier Block
Figure V-44

The chain pillars are extracted at the same time as the barrier pillar in order to keep the pillar line reasonably flat. Figure V-45 shows the basic configuration of the section after completing the rooms. The sequence of pillar extractions is also shown.



Room Layout
Figure V-45

Chain pillars are extracted by the split and fender method. The process does not allow complete extraction of the pillar, since, without bolting stumps are vital for support. Figure V-46 shows several patterns.

a b

Typical Cuts - Chain Pillars
Figure V-46

The sequence for the extraction of pillars varies somewhat. The most significant difference is the extraction of pillar number 11. (See Figure V-45). As shown in the haulage system analysis section, Volume II, this pillar is difficult to extract because of its proximity to the tail piece. An alternative effectively utilized is to extract pillars 1, 2, 3, 4, 5, 6, 7, and 8. Next pillar 9 and 10, leaving 11 until the next set of pillars are extracted. This optimizes production by alleviating the haulage bottleneck.

Too often, a systematic technique is not used in the extraction process. The ability to cut under unsupported roof allows too much latitude to some sections and the selection of improper starting positions or improper cut sequences are often the result. Losses in coal and excessive time expenditures are frequent occurences. This can only be corrected by establishing a concrete plan for cuts and by the foreman's rigidly adhering to the plan. Each split should be carefully positioned so as to provide uniform fenders that can be easily extracted.

VI. RECOMMENDATIONS FOR IMPROVEMENT IN PILLAR EXTRACTION

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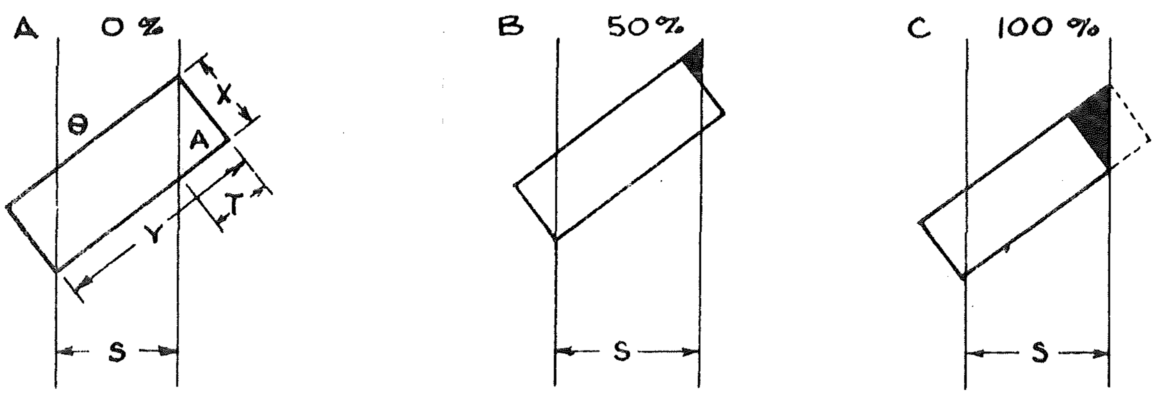
> Mine management must consider many alternatives and make numerous decisions in the establishment of a plan for development and extraction of pillars. This section deals with many of these decision areas encountered by mine management. For each area considered the critical factors are discussed, and favorable alternatives are pointed out. Among the areas discussed are the selection of the sequence of cuts, roof control requirements, basic dimensions, section layouts, pillar lines, sequence of pillar extraction, ventilation, logistics, dust control, percent of recovery, and equipment requirements.

In the planning process and actual production for retreat pillar extraction the mine manager or foreman must consider numerous factors, each of which plays a vital role in the overall productivity and safety of the operating section. Many of these areas are discussed in the following paragraphs. The discussions are general, encompassing both conventional and continuous mining and all methods of pillar extraction. It is the purpose to present these factors, emphasize their importance and discuss some of the alternatives. There is such a tremendous number of factors which affect the process that some are, no doubt, omitted in these discussions.

1. BASIC DIMENSIONS

The decision on the dimensions of the pillars is one that must be made far in advance of the pillar extraction. The mine engineers, when they make their initial projections for the development of a section, must decide upon the center-to-center distances of the rooms and crosscuts and the widths of the rooms and crosscuts. Their number one consideration, of course, is the stability of the roof and floor. Failure to consider these factors can lead to roof failure, pillar failure, or bottom heaving. Once an acceptable range is determined, however, consideration ought to be given to means of increasing production efficiency. The pillar must be dimensioned properly to maximize the recovery of coal while eliminating the need to go underneath unsupported roof.

The basic pillar block dimensions should be such that the entire block can be extracted in a systematic manner. For example, a 40 foot square block split in two passes at a width of 20 feet would result in two 10 foot wide, 40 foot long fenders. These fenders could be then extracted completely, provided the type of continuous miner had the reach and attacked the fender at the proper angle. Figure VI-1 shows three alternatives to the extraction of fenders. In Method A, penetration is 100%. In Method B, penetration through the block is 50%, and in Method C, penetration through the block is virtually non-existent. Each alternative has advantages.

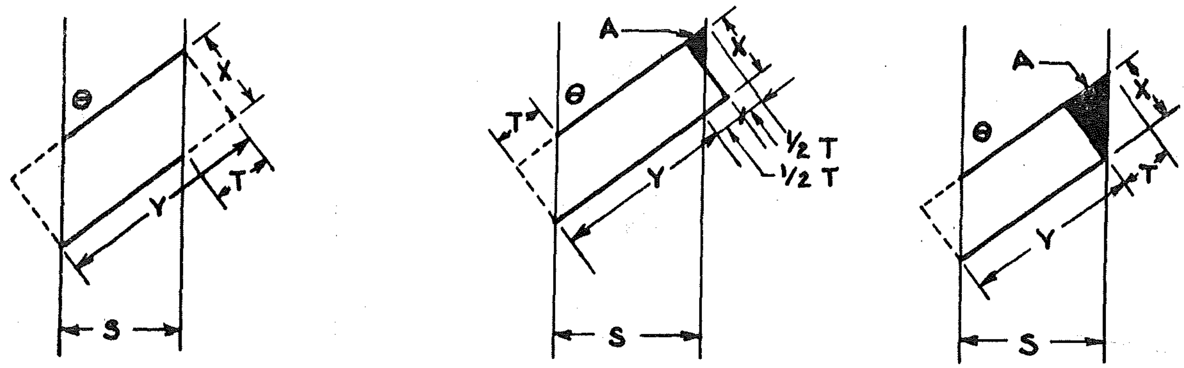


Fender Extraction Techniques
Figure VI-1

Figure VI-2 shows the path of the continuous miner in extracting a cut of coal from a fender for each method. The basic dimensions and distances are designated on the drawings. The following discussions shows the effects on extraction ratios and necessary fender dimensions for a variety of machine head widths and angles of attack.

In the diagrams which follow the symbols and relationships shown below are utilized.

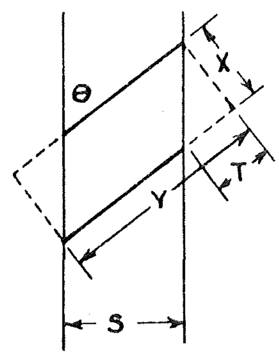
- Angle of Attack = θ
- Width of Cut = x
- Reach of Miner = y
- Width of Fender = s



Path of Head of Continuous Miner in Fender Extraction
Figure VI-2

The dimensions can be chosen to leave stumps of varying sizes, as in the examples b & c, where stumps are categorized as 50% and 100%. The desirability of stumps varies according to geological conditions, however, they also can affect productivity and, hence, profitability. Once the miner has broken through the back side of the fender, less coal is available to be cut, hence, productivity is reduced. Also, since attempting to extract this remaining coal necessitates a narrower fender, the overall dimensions of the pillar block are affected, and a smaller amount of the block is considered "free" coal again affecting productivity.

To optimize this operation, one must increase the angle of attack to approach 90^{o} while considering the trade-offs of turning the miner and maneuvering within the pillar split. For example consider the diagram (Figure VI-3) and the related table (Table VI-1) shown below.



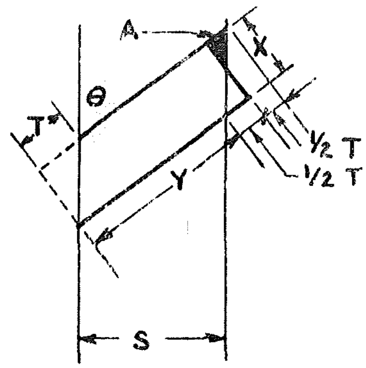
Fender Extraction
Figure VI-3

Table VI-1

Given x = 10, y = 20				
θ	tan ($\frac{\pi}{2}$ - θ)	t=x tan (π/2-θ) (ft)	y - t (ft)	s= (y-t) sinθ (ft)
30	1.73	17.24	2.76	1.38
45	1.00	10.00	10.00	7.11
60	.58	5.78	14.22	12.31
75	.27	2.68	17.32	16.73
90	0	0	20.00	20.00

Critical Dimensions - Method A

Referring to [illegible] θ = 30°, [illegible] the [illegible] with [illegible] foot [illegible] head and 20 foot [illegible], the [illegible] only be [illegible] feet wide. Turning a 90° cut can be done with a [illegible] fender, which is acceptable. Increasing the width of the cutter head reduces the size of the fender, requiring that the angle of attack be increased.



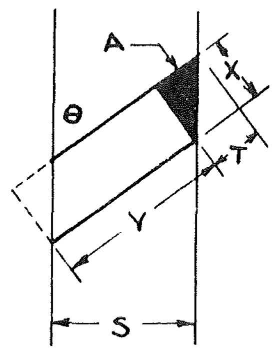
Fender Extraction
Figure VI-4

Table VI-2

Given x = 10, y = 20							
θ	tan (π/2−θ)	t=x tan (π/2−θ) (ft)	1/2 t (ft)	y−1/2 t (ft)	s=(y−1/2 sinθ (ft)	A=1/8 xt (sq.ft)	Percent Lost (%)
30	1.73	17.24	8.62	11.38	5.69	21.55	15.92
45	1.00	10.00	5.00	15.00	10.61	12.50	7.69
60	.58	5.78	2.89	17.11	14.82	7.23	3.91
75	.27	2.68	1.34	18.66	18.02	3.35	1.76
90	0	0	0	20.00	20.00	0	0

Critical Dimensions - Method B

Referring to Figure VI-4 and Table VI-2 above, if $\theta = 45^{o}$, the fender width is 10.61 and 12.50 ft^2 of seam area are left in the stump. Turning 15^{o} further allows a reduction of the seam area of over 5 ft^2 or, for a 6 foot seam thickness, a savings of 1.22 tons per stump.



Fender Extraction
Figure VI-5

Table VI-3

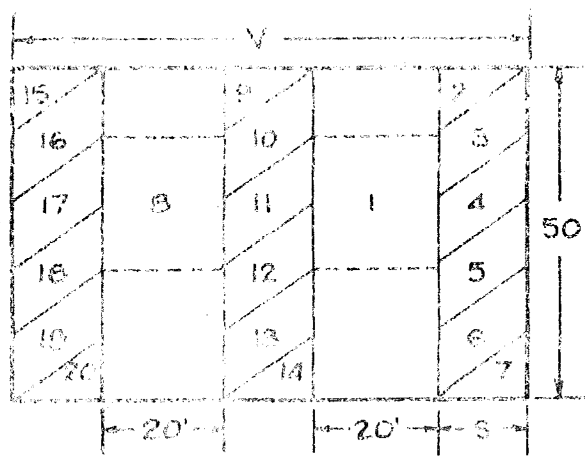
Given X = 10, 6 = 20					
θ	tan (π/θ)	t = x tax (π/2 - θ) (ft)	s=(y-t) Sin θ	A=1/2tx (sq.ft.)	Percent Lost %
45	1.00	10.00	7.07	50.00	50.00
60	.58	5.78	12.31	28.90	20.32
75	.27	2.68	16.73	13.40	7.74
90	0	0	20.00	0	0

Critical Dimensions - Method C

[illegible]

The improper design of the cutting of fenders, then, can [illegible] as well. Consider the effect of narrow fenders.

[illegible] of free coal which affect [illegible] the fenders in a pillar, the more [illegible] However, turning the required [illegible] to extract those fenders is more difficult. Look [illegible] at the impact of designing a pillar plan using two [illegible] splits and extracting fenders using different angles [illegible] This is pictured in VI-6. The results are summarized [illegible]



Multi Split and Fender
Figure VI-6

Table VI-[illegible]

[illegible]	Attack [illegible]	Width of Pillar V (ft)	Total [illegible] of Pillar (sq ft)	Tonnage of Pillar [illegible] (tons)	Coal Recovered [illegible] Bolting (tons)	Percent of Free Coal (%)	[illegible] Coal [illegible]
[illegible]	[illegible]	[illegible]	[illegible]	935	546	58	
[illegible]	[illegible]	[illegible]	4509.5	1096	707	65	
[illegible]	[illegible]	100.00	5000.0	1215	826	68	
[illegible]	[illegible]	71.83	3591.5	873	438	50	[illegible]
[illegible]	60	84.46	4223.0	1026	611	60	[illegible]
[illegible]	75	97.96	4703.0	1143	754	66	[illegible]
C	45	82.42	4121.0	1001	430	43	[illegible]
	60	91.96	4598.0	1117	623	56	[illegible]
	75	97.96	4898.0	1190	752	63	[illegible]

Dimensions for Multi Split and Fender

Consideration should also be given to the fact that widths [illegible] the ability of the continuous miner to turn a corner. Splits described in Table IV-5 include 12, 14, 15 1/2, and 20 feet, representing one-pass and two-pass miners. For this discussion, Method A will be used for 100% recovery with 60 degree [illegible]

Table VI-5

Split Width (ft)	Fender Width S (ft)	Width of Pillar V (ft)	Total Area (sq ft)	Total Tons (tons)	Coal Extracted Without Bolts (tons)	(Percent)
[illegible]	12.31	60.93	3046.5	740	507	68
[illegible]	12.31	64.93	3246.5	789	517	66
[illegible]	12.31	67.93	3396.5	825	524	63
[illegible]	12.31	76.93	3846.5	935	546	58

Impact of Variations in Split Width

Thus, changing split widths from 12 feet to 20 feet decreases the percentage of coal recovered without bolting by 10 percent. To install bolts on 4 foot centers in a 12 foot split requires 3 bolts across. To install bolts on 4 foot centers in a 14-or 15 1/2 foot split requires 4 bolts across. To install bolts on a 20 foot split requires 5 bolts. Thus, the changes in widths do not always necessitate additional bolting efforts.

As indicated, the proper dimensions for a pillar are a function of many items including the width of the split, the angle of attack on the fender, the head on the continuous miner, and the degree of penetration or the distance of penetration for the continuous miner. These decisions obviously have an impact on the type of equipment to be selected for the section. For example, if the split to be driven through a pillar were to be only 10 feet, a very maneuverable miner would be required. In addition, if the continuous miner head width can be selected, an effort to evaluate properly the advantages and disadvantages of going to a wider head must be made.

These discussions bring out the complexity and decisions for the proper design of pillars for retreat mining. The topics discussed are only a few of many of interest to the mining engineer. Even within the limited framework of this section numerous variations were omitted. A handbook should be prepared elaborating on this topic and others which can serve to guide the mining engineer in preparing pillaring plans. Such a handbook should evaluate the geometry of pillar design, the sequence of cuts, the widths of cuts, the angle of attack for fender extraction, section layout alternatives, belt retreating, shuttle car path alternatives and other related factors. This handbook could be an invaluable guide to aid mine operators in planning and in day to day operation of retreat mining sections.

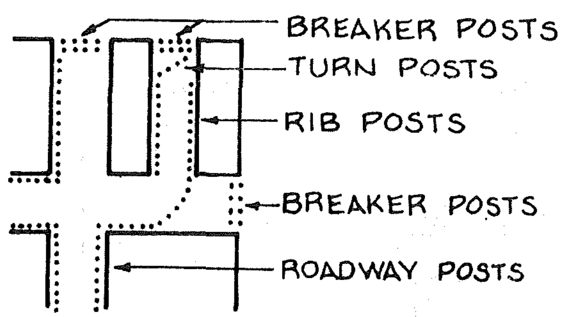
2. ROOF CONTROL REQUIREMENTS

Roof control in coal mining is of obvious importance. In advance mining, the sole objective is to support the roof adequately to prevent any roof fall. Little concern is given to over-supporting the roof except from a cost standpoint. In retreat mining, however, proper roof control demands that support be adequate for safety, yet at a level low enough to insure good falls. Safety is in no way to be compromised, but over-support could lead to poor falls. These could cause squeezes and abnormal pressures which jeopardize safety. In addition, a trade-off exists between the exposure of the man placing roof support in a partially mined pillar versus the inherent instability of the structure without that support. One additonal factor is the delay that roof bolting inside of a pillar causes, increasing the overall time of extracting a pillar. It is generally believed that the lapsed time from the initial cut into the pillar until the final stump is removed must be as short as possible in order to insure safety of the miners.

Many types of support are found in retreat mining. Each plays an essential role in the overall roof and ground control. Bolting, of course, is the usual support method used while driving the splits or pockets into the pillar. Some mines timber using cross bars instead of roof bolting. This allows the mining activities to continue as the roof support is set. In some cases only posts are set. In this case the series of lifts are completed and the area is timbered off. Where a mined out split is timbered, and roof bolting is not required, the continuous miner operator must be out under unsupported roof during some phase of mining. Obviously, the setting of the timbers requires workers to be out under unsupported roof. The study team observed, during the course of this study, mines within a few miles of one another mining in the same seam under similar conditions. Mine A uses a full bolting plan without going beyond supported roof, and mine B timbers off the cuts after they are completed mining routinely under unsupported roof. In both cases, the management and workers are totally satisfied with the method they are utilizing. And in both cases they can not understand how it could be done another way, both listing safety as the reason. Actually both ways may or may not be adequate, but, nevertheless, the difference in approach is striking. This type of contradiction exists throughout the industry: pillaring is considered an art, not a science.

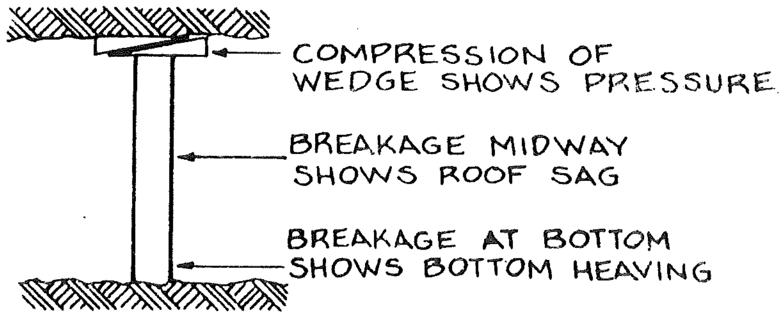
Breaker posts play a vital role in retreat mining. They are placed between adjacent pillars at the gob line, normally in double rows on four foot centers. Under severe conditions, cribs may be substituted. Breaker posts are also used when splits or pockets are driven through to the gob. The purpose of a breaker post is to provide a firm line of support at the pillar line, substituting for the lack of continuity between the two adjacent pillars by supporting the mined out opening.

Turn posts are used adjacent to the continuous miner or loading machine prior to mining out a fender or wing. They are generally left in place upon completion of mining. In some operations, rib posts are used on either side of the splits or pockets to prevent accidents from rib rolls. Roadway posts are commonly used on entries to the pillars. These are generally placed on four foot centers in such a way which reduces the span of the roadways to 16 feet or less. (See Figure VI-7)



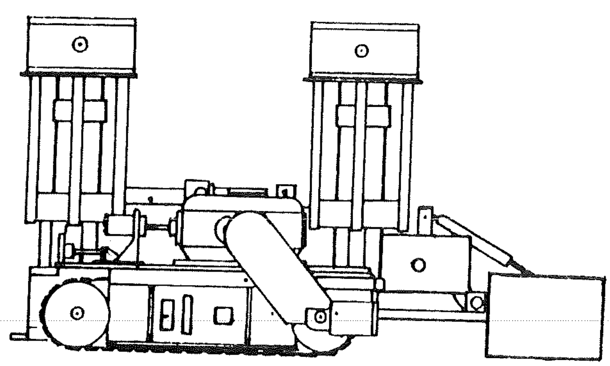
Timber Plan
Figure VI-7

In addition to roof support, posts provide a vital warning to the miners. Pressures due to sagging or bottom heaving can be readily observed and diagnosed by watching the posts. (See Figure VI-8).



Timber Pressure Signs
Figure VI-8

In virtually all systems of extracting pillars, breaker posts are set between adjacent pillars at the pillar line. These breaker posts are often recovered and reset as the pillar extraction process progresses. One major coal company has developed a mobile roof support unit on a cat chassis utilizing hydraulic jacks. (See Figure VI-9).



Mobile Roof Support Unit
Figure VI-9

In the original prototype version, the power for the equipment was slaved off the continuous miner. In operation, it was trammed into position, and the jacks raised against the roof. It was left in place while the split was driven and the fender extracted. Following completion of this operation, the jacks were lowered and the machine trammed to a position near the next cut. As a prototype the machine was successful. It showed that such a device could be used effectively. It did, however, suffer from several shortcomings. They include:

- the slaving of power off of the continuous miner meant additional cables and interference for equipment moves;
- the operation of lowering the jacks and moving the piece of equipment required a man to go into an unstable area; and
- in bad roof, the lowering of the jacks caused the roof to lower, which required the setting of temporary support.

The USBM should initiate a program to investigate the feasibility of a second generation mobile roof support unit. Such a unit should include the design of the jacks following the concept of longwall mining, where two jacks can be lowered and moved back while the other two remain in place in such a manner which enables the miner to "walk" the unit out of a bad area. Second, the machine should be equipped with remote control to eliminate the need for the operator to go into the hazardous area. Consideration should be given to battery power, since the number of moves in the course of a shift for the equipment would be minimal. These features could make the mobile roof support unit a valuable tool for mines with the proper conditions.

The typical pillar extraction, depending on the method, may use anywhere from 40 to 100 separate posts for roof support during the course of extraction of a single pillar. These timbers are generally brought forward to the pillar area by shuttle car having been loaded somewhere in the rear of the section. The timbers are then unloaded in the immediate vicinity of the pillar, where they remain until they are needed. In addition, other supplies, such as wedges and header blocks are also brought forward. Timbers typically are slightly longer than required to allow for variations of coal thickness. Each one is individually fitted at the location where it will be set. When timbering is required, generally all production activities cease. Miner, miner helper, and other team members all assist in the activity of timbering a split. The men select a position for a timber, measure the height, select a timber and measure and cut it by hand. The timber is then carried to its position and set upright, and wedges are driven to anchor it securely. As the extraction process continues, additional timbers are set as turn posts, rib posts, or roadway posts. When the fender is extracted completely, the extra timbers must be picked up and moved. The breaker posts and turn posts that have been set are generally left in place. The activity of setting timbers is a major contributor to delays in pillar extraction.

In the past a conventionally equipped pillar extraction section used a piece of equipment known as a timber machine. This machine is basically a wheel-driven, electrically-operated machine capable of carrying a load of timbers, wedges, and other supplies. It has, as an integral part, a saw for sizing the timbers prior to installation. This machine can operate independently and carry supplies and equipment needed for timbering. Although seldom seen today on the retreating section, this machine has the ability to increase production on the section. It can eliminate the handling and rehandling of timbers, and provide a workplace for cutting and consolidating all the equipment for timbering operations. The USBM should re-evaluate this equipment in light of today's requirements on a continuous mining section. The upgrading of this piece of equipment with the addition of self-contained power and a device to recover timbers could make it a valuable asset to the continuous mining section. Other possibilities would be the integration of the system with a shield to protect workers while they are setting timbers.

Perhaps one of the newer developments affecting roof control in retreat mining is remote control in continuous miner equipment. Remote control, both through the use of an umbilical cord and through radio control, is now available on new equipment and as a retrofit to old equipment. The study team has observed remote control being used in pillar extraction and believe that it can play a vital role in improving safety and productivity when it is used properly.

Remote control has many advantages. It offers additional reach and allows the operator to remain in a safe position. However, it reduces visibility, and the operator does not have the "feel" of the machine. He can, however, use his eyes and ears to make up for some of these shortcomings. For example, when a miner strikes the top or digs into a soft bottom, the change in noise is an easily audible sign to the operator. In addition, he can use his eyes to observe the flow of material off the boom and see the presence of rock or clay. The angle of the miner indicates whether the penetration is level, angled-up into the top, or angled down into the bottom. In driving the split the miner moves from a protected operating place and sometimes stands along the rib. Here he becomes susceptible to being crushed between the miner and shuttle car or to becoming a victim of rib rolls. When driving splits the machine operator should stay in a place where he is not apt to be struck by a piece of equipment. In the extraction of the split, the last cut prior to breakthrough to the gob is often less than 20 feet. Without remote control, the miner operator must stop mining and move to a new position while support is set prior to this cut. He can then remove the last cut. With remote control, he can often continue to cut the remaining distance until the split is finished or until the shuttle car operator is exposed beyond unsupported roof. This can be a decided advantage in productivity, by eliminating a place change.

However, there does exist the possibility that the machine would be advanced beyond supported roof, to a point where the unsupported roof would fall. Thus, a disadvantage does exist that the miner may, through overzealous advancement, be covered up in a roof fall.

In reviewing fatal accident reports occasionally a continuous miner operator had advanced beyond the last row of bolts even though he knew it was an unsafe act. This was necessary to extract a stump of coal that would not crush if left intact. Although it was unsafe to advance beyond the last row of bolts, it was also unsafe to leave a stump that would not crush out and would, in turn, cause overriding pressures on the adjacent pillars. Remote control would eliminate this type of accident.

Another major advantage to remote control is in the extraction of fenders. Here the operator can position the equipment to start a cut on the fender, and he can then take a position outside of the pillar split and mine out the entire fender from a safe psotion. This minimizes his hazardous exposure during the entire pillar extraction cycle leaving only the time when he is actually driving the split. In observing remote control mining of fenders the study showed that an operator qualified and experienced with remote control could extract a fender almost as fast with the remote control as he could with the conventional control.

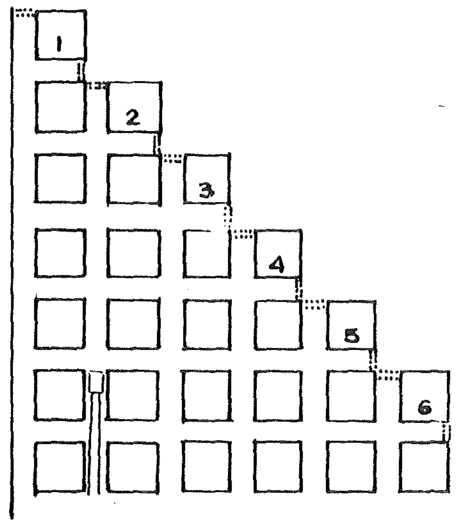
In all of the methods of pillar extraction with continuous equipment, a requirement normally exists to set posts inside of the pillar block. The setting of posts is one of the most hazardous operations in the pillar extraction process. The turn posts are designed to protect the operator and shuttle car driver. The use of remote control for continuous miners necessitates a reevaluation of the need for timbers during portions of the pillar extraction.

Also, prior to the driving of a split through a pillar, the mine foreman knows approximately where the split is going to break through the pillar block. Consideration should be given to the possibility of setting the breaker posts across the back side of the pillar, approximately where the split is projected to penetrate through. Although breaker posts set in this manner would not be exactly in the same position as if set from inside the split, they could be set prior to the completion of the previous pillar from a fully bolted, fully supported, and stable area. Thus eliminating the need for workers to enter the pillar block to install them. Alternatively the mobile roof support unit, described earlier could be used in the split in place of breaker posts. It could then retreat along with the continuous miner as the fender is extracted acting as both breaker posts and turn posts. Given the proper dimensions and remote control this could eliminate the need for men to enter the pillar block through the entire mining cycle greatly reducing the hazards.

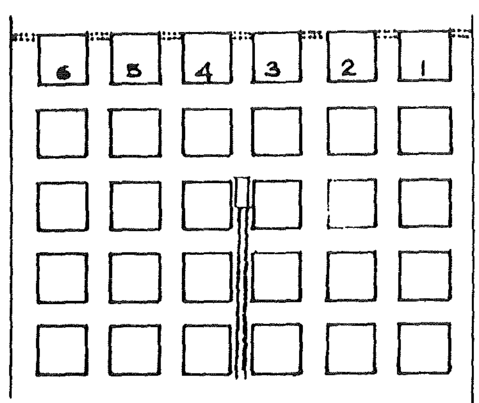
3. PILLAR LINES AND THE SEQUENCE OF PILLAR EXTRACTION

The shape of the pillar line and the sequence of pillar extraction are two areas of interest to the mining engineer. Pillar lines range from steeply angled, to flat, to the W-shaped line unique to continuous haulage. Each creates unique problems for the engineer as he selects the proper sequence of pillar extraction that will provide good roof falls with no pressure problems.

The selection of the angle of the pillar line can affect the production of the section. The proper position of the belt and selection of haulage paths are affected by this angle. The optimum positions required to reduce tram times and belt moves favor a flat pillar line rather than an angled pillar line. Figure VI-10 and VI-11 show the optimal positions for both a 45 degree pillar line and a flat pillar line. An analysis of the tram distances for the shuttle cars for the two types of lines shows that the travel



Belt Position - 45 degree Pillar Line
Figure VI-10



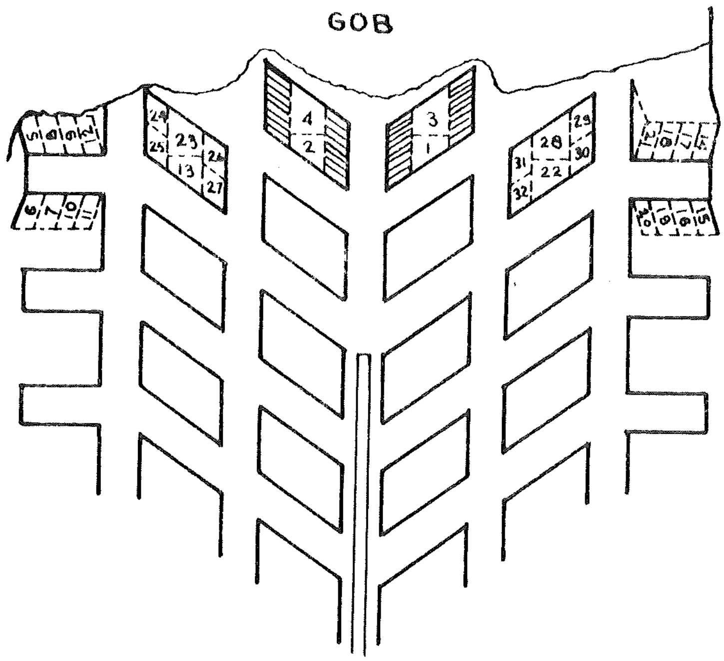
Belt Position - Flat Pillar Line
Figure VI-11

required for the angled line is greater than that for the flat line.

In a system that is haulage constrained, which is typical of retreat mining, increased delays could result from the use of an angled pillar line. The pillar line itself is constructed to secure a fall inby the pillar line. The jagged pillar line of an angled line results in portions of the pillar jutting out into the gob area. These areas will be under heavier pressure and will be more hazardous to mine than those of associated with a flat pillar line. Conventional mining requires multiple work places, and an angled pillar line provides protection during the extraction process. Each pillar is in a different state of recovery. Thus, all active pillars are not on stumps simultaneously. No clear choice appears to resolve these questions.

Numerous sequences for pillar extraction exist. Some mines work left to right, others right to left. Some mines work two pillars deep from the pillar line, while other work the pillars one deep. Some mines work from the left and the right towards the center, extracting the two center pillars together. Plans allowing extraction of two pillars from a single roadway require correspondingly less tram time for shuttle cars and continuous miner. Where one pillar is extracted at a time using two work places on the pillars, the total distance required for tramming the continuous miner and for movement of coal from the face by shuttle cars is often greater. These areas have been discussed in detail in Chapter 5.

Continuous haulage systems create a different type of pillar line. This is shown in Figure VI-12. Here it is difficult to devise a sequence of extraction which eliminates pressure points while maximizing coal recovery. Some suggestions of alternative sequences and further elaboration of these problems are found in Chapter 5 of "Face Haulage Efficiency in Underground Retreat Coal Mining" published separately as Volume II of this report.



Sequence of Pillar Recovery Using Continuous Haulage System
Figure VI-12

The use of continuous haulage systems, such as bridge conveyors, poses unique and interesting problems in the selection of the sequence and design of the pillar line. In this system, diamond-shaped pillars are formed due to the lack of mobility of the bridge conveyor system which makes the cutting of 90 degree intersections impossible. The left hand side of the section is a mirror of the right, with the result that the pillar line is neither flat nor angled, but W-shaped. Although a more detailed discussion of the problem is contained in Volume II it is appropriate to summarize the conclusions.

The current methods used to extract pillars of this design result in a loss of coal because of the inability to mine a W-shaped pillar line safely. No way currently exists to design the cuts and the pillar in a manner which eliminates pressure points in the pillars. The USBM should initiate a study of new techniques of extracting coal in the W-shaped pillar line situation. The trend in mining today is toward increased usage of continuous haulage systems. Most of the systems designed today will require the driving of angled crosscuts, resulting in section layouts similar to the ones discussed above. Until such a study has been performed, and methods have been developed for pillar extraction in W-shaped pillar lines, the new systems and concepts designed today will never obtain their full potential.

SECTION LAYOUTS

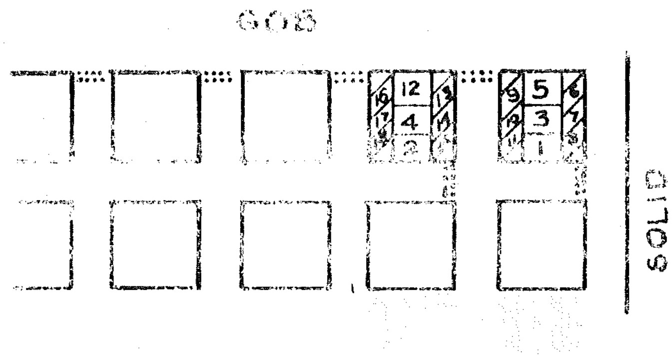
The layout of the section in retreat mining is important to the proper selection of haulage paths, the positioning of supplies, the supervision of work places, the rate of retreat and others. Numerous alternatives exist for the number of entries and, correspondingly, the number of pillars in a section. Three, four, five, seven, and nine entries are all viable alternatives, each having its advantages and disadvantages. Having fixed the number of entries, the position of the discharge point and the selection of shuttle car paths can then be considered.

A large section six or seven pillars wide requires that the belt be kept close to the pillar line because of the limited range of shuttle cars on cables. The total distance from the tail piece to the working place cannot be in excess of the length of the cable on the shuttle car. Although extra long cables can be fitted to shuttle cars, this does result in some power loss and additional cost.

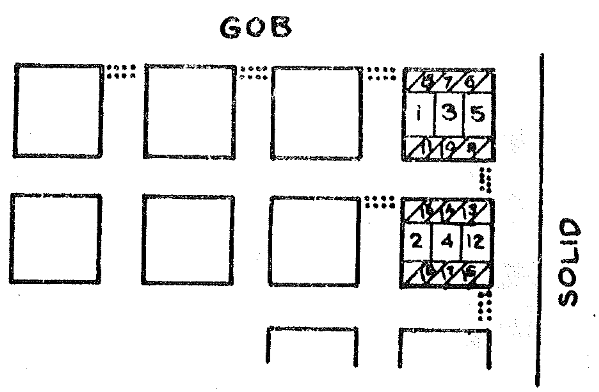
The foreman's management problems in retreat mining are very difficult ones, far more difficult than in advance mining. These problems are complicated by the necessity for him to cover a large work area. In a large section the rate of retreat of the pillar line is much slower. This may or may not create problems. Sections with fewer entries have a faster rate of retreat. As the rate of retreat increases, the rate of belt moves increase. If the belt is not kept close to the face, shuttle cars are required to tram longer distances with correspondingly increased delay times at the face. Logistic problems become more critical, since supplies cannot be stored in a central point as conveniently. The foreman's management problems, on the other hand, decrease as the range which a foreman must cover becomes smaller. However, because of the faster tempo of the rate of retreat in smaller sections, he must maintain more awareness of what is going on.

A three entry bridge conveyor system can be operated with two bridges and one mobile carrier. The particular rate of advance or retreat requires a tail piece movement at a certain frequency. This frequency may or may not correspond with normal breaks on the operating section. That is to say, it may occur every two shifts so that the move can be made during third shift with no loss in production. However, if the move is required after 1 1/2 shifts, a half of a shift of production may be lost every other move. This same system given an additional bridge and mobile bridge carrier, would allow the cutting of five entries. This is an added cost, but it may increase production by changing the timing for tail piece movement. The addition of another bridge and carrier to allow such a system to mine seven entries would again change the rate of advance or retreat and the timing for movement of belt.

The sequence of pillar extraction varies. Pillars extracted using single splits were worked in pairs and, occasionally in triplets to provide adequate work places for both the mining activity and the bolting activity. Two approaches were observed for working of pillars in pairs. The first, shown in Figure VI-13 had the two active pillars on the gob line. The second, shown in Figure VI-14, had one active pillar on the gob line, and the second active pillar outby the first.

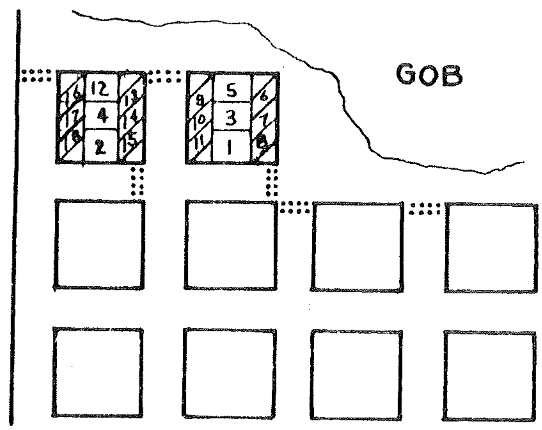


Split and Fender Pillar Sequence Pattern 1
Figure VI-13

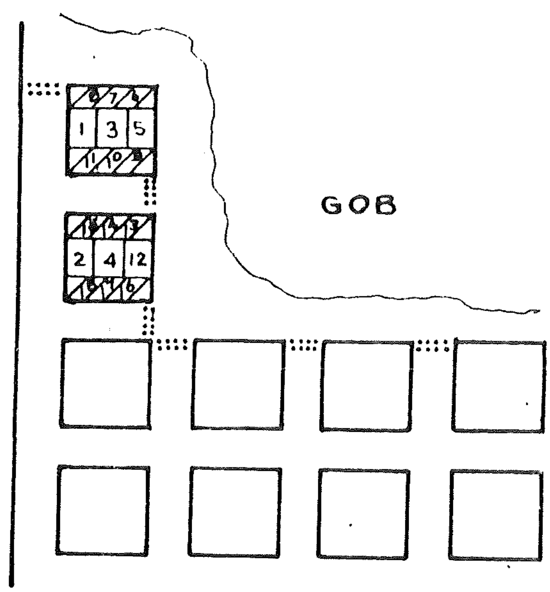


Split and Fender Pillar Sequence Pattern 2
Figure VI-14

Examination of the two approaches shows no specific advantages to either except in the completion of a row of pillars. In the second approach, the haulage access is restricted, which would affect productivity. This is shown in Figures VI-15 and VI-16.

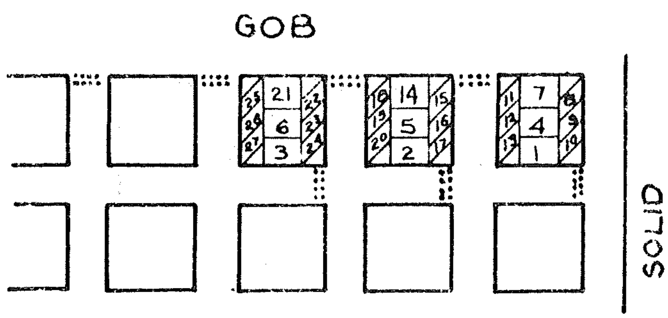


Split and Fender Pillar Sequence -- Pattern 1
Figure VI-15



Split and Fender Pillar Sequence -- Pattern 2
Figure VI-16

Three blocks can also be mined at the same time. (See Figure VI-17). This could be effective in a situation where mining is twice as fast as bolting, and two bolters were utilized. However, as a standard practice, the additional tram distance required by the continuous miner makes the system ineffective. Table VI-6 shows the average tram distance and tram ratio for two active pillars, (Method 1) and three active pillars, (Method 2).



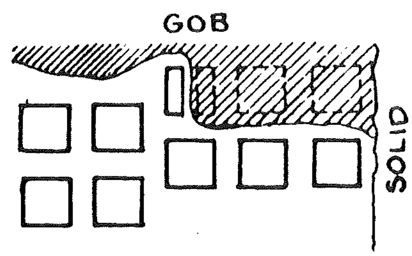
Split and Fender Three Active Pillars
Figure VI-17

Table VI-6

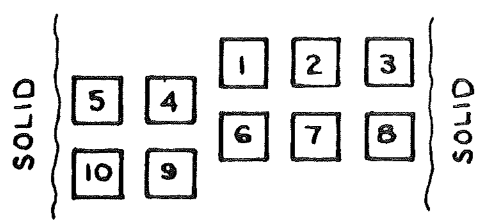
Method	Room Center (ft.)	Crosscut Center (ft.)	Pillars	Total Area (sq. ft.)	Continuous Miner Tram Requirements						Total Distance (ft.)	Ratio
					Driving Splits	# Moves	Distance (ft.)	Extracting Fenders	# Moves	Distance (ft.)		
1	70	75	2	10,500		5	290		2	90	1630	.15
2	70	75	3	15,750		5 3	290 360		3	90	2800	.17

Continuous Miner Tram Distance - Split and Fender - Two Versus Three Pillars

In some mines, offset intersections are used. These create a rather unique problem in haulage and in the pillar extraction process. With offset intersections, part of the pillar projects into the gob area. These pillars can be extracted safely, but only through careful planning and sequencing of cuts. Figure VI-18 shows the type of situation to be avoided. Figure VI-19 shows a sequence of pillars which allows the safe extraction of the pillar.



Offset Intersections
Figure VI-18



Sequence of Pillar Extraction - Offset Intersections
Figure VI-19

The USBM should initiate a study to evaluate alternative section layouts for different numbers of entries and different sequences of pillar extraction with respect to management requirements, logistic requirements, type of haulage, and rate of retreat in order to determine which system is optimal for a particular set of conditions.

5. SEQUENCE OF CUTS

The sequence of cuts in the extraction process of a pillar is of particular importance in terms of productivity and safety. Productivity is affected because a proper sequence of cuts will minimize the tram time required for the continuous miner to make a place change. Safety is affected because the proper sequence of cuts is critical to prevent a load imbalance within the pillar by extracting too much coal from one area at one time thereby destroying the load carrying strength of the pillar and causing a premature roof fall. The choice of sequence of cuts is also critical for ventilation. Splits should be driven through the pillars as quickly as possible to allow the flow of air through the pillar. Having two active work places on the same pillar requires splitting of the air at the intersection. Working two or three places in two or three separate pillars requires channeling the air in a manner which will ventilate each of the work places adequately. A problem in both advance mining and retreat mining is sequencing the cuts in a manner which prevents dust caused by the continuous miner from blowing over the place where roof bolting or other support activities are being conducted. The variations in the sequences of cuts has been discussed in detail in Chapter V. The USBM should sponsor a study on sequence of cuts, for both pocket and wing and split and fender, to determine which technique results in the least downtime and hence, the greatest productivity. Careful consideration must be given in the load carrying characteristics of the pillar. Such a study should also include a thorough investigation of the dimensions of the pillar block splits and overall layout of the section. In addition, the USBM should make recommendations about the type of equipment to be utilized, including analysis of the trade-offs between sizes of cutting heads on continuous miners for retreat mining. The Bureau should attempt to optimize pillar dimensions, the sequence of cuts, and selection of equipment for a variety of center-to-center distances. With this information any operator, after designing his pillars, could select the optimum sequence of cuts and equipment to fit his particular conditions.

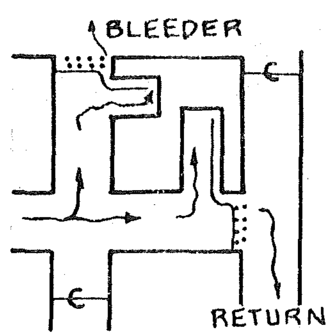
6. VENTILATION

There is no question that ventilation is the most critical element in underground mining. Because of the dynamic situation of the extraction of pillars ventilation becomes difficult. It is essential for air to flow over the gob area to prevent the entrapment of gas and subsequent explosion. Air is essential to prevent gas from accumulating in work areas. Air is essential for visibility to allow the miner to perform his work safely and productively. And, of course, air is essential to reduce the respirable dust.

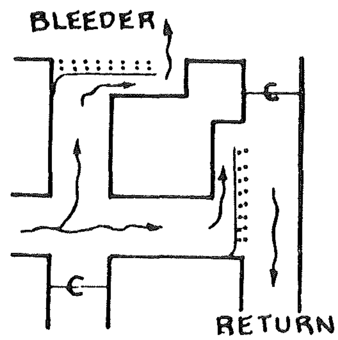
It is a requirement of the Coal Mine Health and Safety Act that where pillars are extracted either partially or in their entirety, that a bleeder system be established. This results in leaving a row or two or unmined or partially mined pillars around the perimeter of the area which is being mined. These rows of pillars are necessary to prevent falls in the perimeter which would block the passage of return air which must flow over the gob area. The parimeter bleeder system must be properly maintained. It is the duty of the fire boss to routinely walk the bleeder system to assure that they remain open.

Basically, in mining, mine management has the alternative of using a single split or a double split ventilation system. In a single split, air must be channeled to pass over and outby the pillar line. Curtains must be set at the pillar line to direct the flow of air. Since a single split requires air to flow across the face, care must be exercised to prevent short-circuiting the system anywhere along the way. Once the air has passed the working face, it can be allowed to flow across the gob or down the return. This flow of air can be controlled by regulators on the bleeder and the return entry. In a double split air can be directed to flow over two work places. In this manner a roof bolt crew can work along with a continuous miner crew and not be forced to work down wind and in the dust generated from the miner.

Ventilation of the pillar itself varies according to the method being utilized. Open end mining, pocket and wing, and splits all are ventilated differently. In both open end and pocket and wing, two work places exist in the pillar. Assuming both are being worked at the same time, air must be split in a way which provides ventilation to both the position being mined and to the roof position being bolted. Air is generally run across the crosscut split at the corner of the pillar and then channeled to each of the work areas. Figure VI-20 and VI-21 show the flow of air.

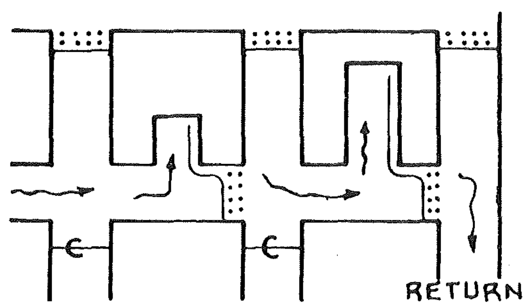


Split Air Ventilation - Pocket and Wing
Figure VI-20



Split Air Ventilation - Open End
Figure VI-21

In split and fender both faces must be ventilated off the same split of air. (See Figure VI-22). Alternatively air could be channeled up the room entry and split between the pillars to allow separate air for each entry.



Single Split Entry Ventilation
Figure VI-22

Numerous alternative ventilation plans exist depending on the section layout and the particular pillaring technique being utilized. In the overall process of selecting the proper mining system and evaluating the numerous alternatives, ventilation, either by a single split or double split, must be carefully analyzed in light of the entire mining plan.

The problems of ventilation become even more difficult in continuous haulage systems. The irregular intersection and diamond shaped blocks make air flow difficult. The tramming of equipment often interferes with brattice curtain. The use of tubing, both intake and exhaust is hindered by lack of clearance. Often mobile bridge operators must work downwind of the continuous miner. Many of these problems could be eliminated by the development of means of rapidly reversing the direction of air flow in a section. Thus air could flow clockwise when mining on the right side of the section or counter-clockwise when mining on the left side of the section.

7. LOGISTICS

The logistical problems are important for both advance and retreat mining. However, in retreat mining they are even more critical. As has been mentioned before, it is essential to extract pillars quickly and efficiently. This means that the proper supplies such as timbers, wedges, header blocks, rock dust, roof bolts and others be at the proper location at the proper time. A common occurrence is for the foreman to order an improper amount of supplies. Too few timbers, for example, means either a costly delay or continuing the mining activity with inadequate support. Too many timbers means that they must be reloaded onto a shuttle car or supply car and moved. And, because the immediate work area is soon to be mined out, these extra supplies must be moved immediately. At least some of these problems could be eliminated through the use of a timber supply car discussed in section 2 of this chapter.

Logistical problems are common to many mines. Supply systems operate by word of mouth without inventory control systems. Reordering is often completed only when supplies are exhausted, resulting in delays or "doing without". The USBM should initiate a project to develop a model logistics system for coal mines. Such a system should determine the supply requirements for the typical advancing and retreating sections, develop an automatic requisition system, set up an inventory control system, set reorder levels based on standard techniques, and establish an integrated inventory level and cost monitoring system. This system should be implemented with a participating mine and documented to demonstrate its feasibility and its economics to the industry.

8. RECOVERY RATE

The importance of pillar recovery cannot be overemphasized. If the coal is not recovered during retreat mining, it is lost forever. On the other hand, one must consider the extra time it takes to get a high recovery rate against the value on the additional coal. For example, if the cost of recovering a suicide stump is in excess of the value of the suicide stump, then perhaps it should be left in place and shot. In the course of extracting the push out stump, double rows of timbers must be placed along with cribs and other supports. Production virtually ceases during this phase. Finally, the continuous miner moves in to extract a few tons of coal. The value of the coal which is actually extracted is offset by the cost of men and equipment utilized in this phase. This analysis does not even consider the hazards involved in setting the temporary supports. Perhaps it would be better to shoot the stump in place and not waste the effort to obtain the coal.

In the extraction of coal, consideration must be given to the concept of "free coal." Free coal is defined as that portion of the pillar which can be extracted from supported areas without any additional support. For example, the miner drives a split through the pillar and bolts it. Next, the miner extracts the fender lift under supported roof. The coal in the fender is called free coal. The evaluation of various sequencing of cuts for extraction of coal shows little difference between split and fender and pocket and wing in terms of the amount of free coal available. Analysis includes giving due consideration to the percent of coal extracted during advance and retreat and the actual dimensions of and the actual dimensions of both. The percent of free coal is affected by the dimensions of the splits and fender lifts and by the angle of extraction.

On a conventional section, open end mining requires almost total bolting. Split and fender, however, does allow for a higher percent of coal to be extracted as free coal. However, the recovery rate for split and fender in conventional mining is generally not as great as in open ending.

In the desire to extract as much coal as possible, a decision to either run the continuous miner under unsupported roof to increase the extraction ratio or not to run under unsupported roof leaving a stump and losing the valuable resource must be made. This decision is a difficult one and one which occurs frequently in pillar extraction. How do you measure the tradeoffs between a poor roof fall which may cause overriding pressure against the hazards of sending a miner and an operator under unsupported roof to capture the stump? Tradeoffs that must be weighed include the ease with which the stump will crush out if left vs the cost of additional temporary support if the decision is to recover the stump. A possible long-term solution to this problem would be the utilization of remote controlled miners. The flexibility of the remote controlled miner makes it a powerful weapon in combating mistakes made in planning the extraction of the pillars. With its flexibility combined with its other inherent advantages, remote control will be a valuable tool in safety increasing the recovery rate.

The extraction ratio for fender reserves can vary. Turning the continuous miner to a full 90 degrees allows the entire fender to be extracted with no waste of coal. Conversely, taking less than a 90 degree turn often results in the leaving of stumps. The stumps can be a valuable tool in providing additional support and are occasionally required, depending upon the geological conditions. A trade-off exists between the extra time it takes a continuous miner to make a full 90 degree turn and the additional coal left when the miner is allowed to turn a lesser angle.

The USBM should further investigate these areas to increase productivity and safety, including the geometries for pillar extraction to determine the trade-offs between the percent of recovery and productivity, the geometries of pillars to determine trade-offs in different patterns in terms of free coal, and comparisons of different patterns to determine which plan results in an overall higher recovery rate.

9. EQUIPMENT REQUIREMENTS

The selection of the proper equipment for retreat mining cannot be overemphasized. Too often, however, a section is equipped without due consideration of the interface between pieces of equipment. A shuttle car selected for its high rate of discharge may be utilized on a section without a feeder with adequate capacity to take advantage of it. Consequently, the shuttle car operator has to continually turn the discharge conveyor on and off to avoid spillage or overloading the belt system. Either a slower rate of discharge should have been selected, or the section should have been outfitted with high capacity feeder equipment.

In retreat mining the equipment requirements are quite different from those of advance mining. Flexibility is the key to selection of continuous miners. The miner must be highly maneuverable, small, and dependable. Trying to jockey a large miner around to pull out a small coal stump is a needless waste of money, effort, and energy. Virtually all pillaring schemes for a continuous miner require driving of a split or pocket and then, turning the continuous miner at an angle and taking a series of cuts to extract the wing or fender. The angle of attack here is critical. The miner must be maneuverable in order to turn an angle without serious loss of time. A wide head for cutting could cause productivity to suffer. As the angle of attack turns from 90 degrees toward 45 degrees, the penetration of the miner through the fender in order to achieve full extraction increases significantly. Thus, the fender width must be made smaller and smaller until a point where it reaches its safety limit. Alternatively, an extraction ratio of less than total must be accepted. The angle of attack is not the only critical factor. The width of the miner or the width of the cutting head determines the amount of coal that is actually left in the stump. All these factors must be analyzed prior to equipment selection.

Remote control is an answer to many of these problems. It gives the added flexibility needed to increase penetration and to make up for mistakes in the positioning of cuts. Remote control is being used effectively in a number of mines in the country today. It will be of great value in terms of safety and productivity in retreat mining. There is no doubt that there is going to be a loss due to miners' being covered up, but, as the operators gain experience, this problem will tend to be minimized. Also, the operators will be able to remove a piece of equipment under treacherous roof conditions, where in the past he may have been forced to leave a piece of equipment in order to insure his own safety.

Another important piece of equipment on the section is the scoop. More and more often today scoops are proving to be a valuable tool on the section. Demand for them is high, with well over a two-year backlog existing. Unfortunately scoops are very dangerous when not used properly. The USBM should conduct an intensive hazard analysis of scoops and their operation.

Although roof bolting is a constraining activity in advance mining, this is seldom the case in the retreating section. This is because roof bolting requirements are not nearly so severe in retreat mining as in advance. The roof bolter selected for the retreating section should be capable of bolting a place within the same amount of time as the miner can mine out a place. More and more today the use of twin boom roof bolters is being seen. Twin boom roof bolters offer the advantage of allowing two men to work separately to bolt a place. They do, however, suffer delays from the interference of those two men. In retreat mining, and particularly in the driving of the first cut for the split, it is difficult to maneuver a roof bolter in position in a manner which allows both workers to operate. Consequently, a man must wait until the first man inserts enough bolts to allow the roof bolter to move forward where both can operate. Further, in the normal progression of bolting, interference is found between the two workers. Many of the splits are driven less than 20 feet wide and some with only 3 bolts across. A narrow split makes it hazardous for two men to be working. One man must be close to the rib, where he is susceptible to rib rolls. Three bolts across makes it impossible to portion out the work to the two men without having high amounts of down time. Thus, the utilization of a twin boom roof bolter in retreat mining does not double the rate of production. In fact, a good working team on a single boom roof bolter can probably nearly match the production of a twin and can do it more safely under some conditions.

Some of the major delays in retreat mining are due to waiting for shuttle cars. These delays are due to a number of causes, including improper haulage path layouts, low capacity feeder systems, and failure to move the belt. The delays are also due to the improper selection of the shuttle cars. As we mentioned before, shuttle cars must be selected and equipped to interface with other equipment. The discharge rate of a shuttle car must be mated with the acceptance rate of the feeder system. The side boards must be as high as possible in order to insure maximum load-carrying capacity.

One of the areas possessing major shortcomings on the typical section is the feeder or feeder-crusher. These systems have been overlooked too long. There is much that can be done to improve them, and the improvements most certainly can increase productivity. Shuttle cars built today can discharge 5 to 7 tons of coal in 20 to 30 seconds. Unfortunately, they are usually utilized with feeders that can barely accept 5 to 7 tons of coal in 40 seconds. Operators complain that they need feeder equipment which can be moved and set up in place quickly, which will be dependable, and which has a high capacity. At least one RFP put out by the USBM calls for development of a self-advancing tail piece that can set up in as little as 15 minutes. Both the feeder system and the belt advance can be combined. It is suggested that the above study incorporate a feeder system which can accept a complete load from a shuttle car in 10 to 20 seconds. Additionally, since the capacity of the normal 36" belt running at 500 feet per minute is approximately 600 tons per hour, the feeder should have an input/output ratio of at least 4 to 1. It has been noted that at least one equipment operator offers a feeder utilizing the telescoping hopper principle similar to that seen of many of the "ram type" shuttle trucks. It is suggested, therefore, that the optimum feeder system should be designed with a minimum of 6 tons surge capacity, that it incorporates a self advancing feature, that it is relatively flat bottomed to optimize installation, that it may be fed from any one of three sides and that it may be easily disassembled for major moves.

VII. ROCK MECHANICS OF PILLAR EXTRACTIONS

VII. ROCK MECHANICS OF PILLAR EXTRACTION

Much work has been done to develop and publish material concerning the structural mechanical aspects of ground stability, but little has been done to assemble and edit the relevant theory as it applies to room and pillar mining of coal. Appendix F assembles most of the major theories in structural mechanics as they apply to coal. This section is structured in a manner designed to provide results that are useful to MESA inspectors and company mine safety representatives. It is necessary that these individuals be able to collect and analyze the basic data in order to make decisions relating to the adequacy of a pillaring plan and to make on-the-spot assessments of impending roof or pillar collapse. With these tools, an individual will be able to:

- make basic measurements such as roof sag, room spans, pillar splits, angle of slips, etc:
- determine standard values from a table for tensile and compressive strength of coal, modules of rupture, and densities, for the particular mine;
- select the appropriate formula or formulae;
- calculate the related safety factor; and
- compare the calculated value against empirically proven safe values to determine the safety of a particular structure.

This procedure can provide a basis for decisions to take action to correct hazardous structures. For example, where rib rashing has increased the span of an intersection to a point where the structural integrity is possibly jeopardized, it is necessary to determine if additional support should be added. What effect has the rib rashing had on the ability of the intersection to withstand the overburden pressures? The procedure for calculating the safety factor is cumbersome, but manageable, and is shown in Appendix F.

There is no single document which thoroughly covers the state of the art, and it is beyond the scope of this project to attempt to produce one here. There are various publications available in the field; but, taken as a group, they contain many conflicts and inconsistencies. Empirical results to back up the theory are often lacking. Admittedly much is left to be done before an inspector has a single authoritative source, which can be consulted in every instance. The science has advanced, however, to a point where he should have all the necessary tools to offer a body of scientific fact to back up his intuitive judgements.

What is needed is: an indepth study of the state-of the-art theory of advance mining; the development of related theory of retreat mining; the testing of the theory in a variety of controlled conditions; the assembly of the theory and empirical results in a single document complete with tables of all needed constraints, or calculated data not readily available at the mine site; and the development of calculator devices to allow the inspectors to rapidly perform the needed calculations on site.

The theory discussed in this section is basic and, while much has been accepted universally, caution should be used prior to application of the formulas to in-situ problems. A more complete discussion is included in Appendix F. The reader should refer back to the original publication referenced to understand clearly the conditions and constraints of application.

Although the scope of this study is restricted to retreat mining the rock mechanics presentation starts with the designing of structures during advance mining. The time to properly design the mine for the retreat phase is during the design of the advance phase. Too often this is not done and the result is the pillars are not dimensioned properly for the equipment used to extract them. This can be costly. Either valuable coal is left behind or unsafe mining practices are used to extract it, and in both cases productivity suffers.

1. ROOM SPANS

One of the critical dimensions in room and pillar mining is the span of the room. From a structural safety standpoint, a room too wide could collapse. From a personal safety standpoint, a room too narrow places the worker in a hazardous position. The maximum safe room span is, of course, a function of many variables. Perhaps the most important are those describing the overburden, the depth of the seam, the composition of the immediate roof, and the density of the coal. Other factors include the seam thickness and the relationship of the room span to the pillar width. Room span analysis must also consider the presence of hydraulic pressure. The proximity of such bodies can radically affect the pressures. The calculations developed do not provide for the effects of additional support such as roof bolting or timbering.

Extracting the appropriate formulas from Appendix F Part I-16, the safety factors can be determined for failure of roof due to tensile-compress in strength or shear stress. These are shown in Table VII-I. With the basic parameters designated, mine safety officials or inspectors can determine the relative safety of a particular span and can determine appropriate action.

2. INTERSECTIONS

Perhaps one of the greatest constraints encountered in the determination of the basic dimensions of the advancing or retreating sections is the intersection. Many of the fatal accidents in retreat mining that have involved roof falls have occurred at a location which, could be defined as an intersection. This includes the location where the initial pillar split or pocket is driven, the position where lifts are extracted, and the position of the pushout stump. Fiqure VII-1 illustrates these cases. The shading indicates the intersection.

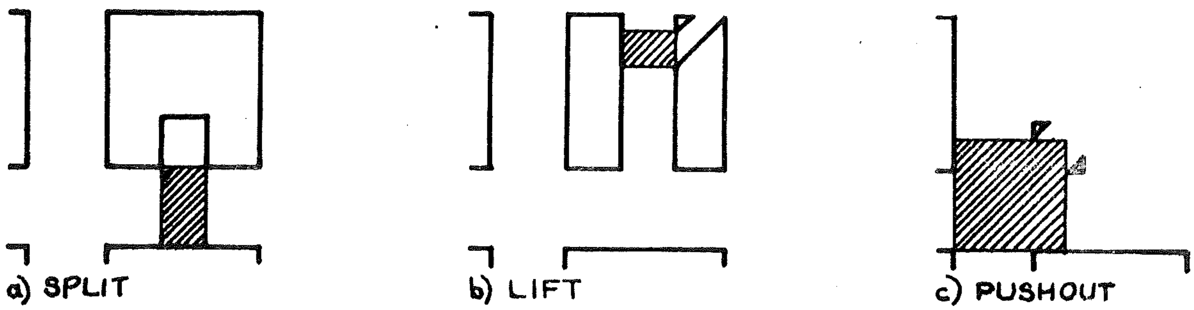


Figure VII-1
Typical Intersections

The safety factors for each case can be readily calculated, and the relevant safety quickly determined. Table VII-2 illustrates the required steps.

In many mines today staggered intersections are used exclusively. Their use greatly increases the strength of the intersection. The figures above show that many intersection in retreat mining can be defined as staggered. But in some cases such as shown in Figure VII-1c the intersection is a normal one. On the basis of rock mechanics staggered intersections can be shown to be less safe than alternative measures.

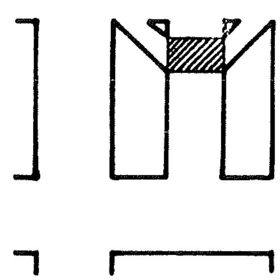


Figure VII-2
Simultaneous Fender Extraction

Table VII-1

SAFETY FACTORS FOR ROOM SPANS

Description	Formula	Failure Due to	Symbols	Reference (Appendix E)
Span of Room (Normal)	$K_4 = \frac{2d\ Ro}{L^2 \gamma}$	Tensile Strength	d = layer thickness Ro = modulus of rupture γ = density of rock K_4 = safety factor L = span of room	I - 16
	$K_5 = \frac{2d\ Ro}{3\ L}$	Shear Stress	J = Shear strength of rock K_5 = Safety factor	I - 16
Span of Room (Hydraulic)	$K_4 = \frac{2d\ Ro}{L^2 (\gamma + P/d)}$	Tensile Strength	P = Hydraulic or pneumatic pressure per unit width and length	I - 24
	$K_5 = \frac{4\ J}{3\ L}$	Shear Stress	Others as above	
Span of Room (Fragmented)	Due to the complex nature of the formulas the reader is referred to the discussion and examples in Appendix E			I - 6

Table VII-2

SAFETY FACTORS FOR INTERSECTIONS

Intersection Span	Steps	Formulas	Tables
	(1) Max. Neg. Moment at the Abutments	$M_{Max} = -\frac{qL^2}{12}$	
	(2) Max. Pos. Moment at the Center	$M_{(x=L/2)} = \frac{1}{24} qL^2$	
	(3) Pillar Factor	$C = 3/4\ (1-n^2) \cdot \frac{Ep}{ER} \cdot \frac{A^4}{Hd^3}$	
	(4) Multiplication Factors Based on M_{Max} and $M_{(x = L/2)}$	$I = f_1\ (c)$ Tabled	Curve R - Regular Pillar Plan Curve S - Staggared Pillar Plan
	(5) Multiplication Factor based on ratio of Room Span to Center to Center Distances	$II = f_2\ (L/A)$ Tabled	Figure 12
	(6) Multiplication Factor based on Ratio of Crosscut Width to Main Room Span	$III = f_3\ (B/L)$ Tabled	Figure 13
	(7) Multiply Results of (1) and 4, 5, 6	$M_{Max} = M_{Max} \cdot I \cdot II \cdot III$	Temporary Results
	(8) Multiply Results of (2) and 4, 5, 6	$M_{(x = L/2)} = M_{(x = L/2)} \cdot I \cdot II \cdot III$	Temporary Results
	(9) Maximum Tensile and Compressive Stresses	$\underset{\text{(tons comp)}}{\sigma} = \pm \frac{6Mx}{d^2}$	$M_x = Max\ (M_{Max}, M_{(x = L/2)})$

Table VII-2 (Continued)

SAFETY FACTORS FOR INTERSECTIONS

Intersection Span	Steps	Formulas	Tables
	(10) Ratio of Tensile Strength of the Roof Rock to the Compressive Stress	$K_8 = \frac{S_{Comp}}{f_{Comp}}$	
	(11) Ratio of Tensile Strength of the Roof Rock to the Tensile Stress	$K_4 = \frac{S_{Lens}}{f_{fens}}$	
	(12) Minimum of the two Ratios is the Safety Factor	$K_{10} = \text{Min}\ (K_8, K_4)$	

Fender extraction similar to that shown in Figure VII-1b should be used rather than that illustrated in VII-2 where possible. Push-out stumps should be narrowed (as illustrated in Figure VII-3) rather than met head on, (as shown in Figure VII-1c).

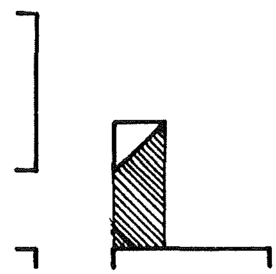
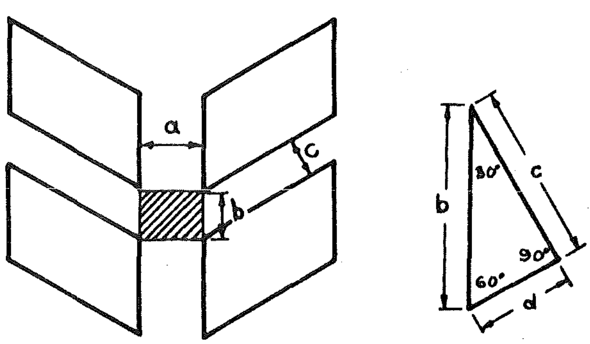


Figure VII-3
Alternative pushout configuration

One problem, discussed more thoroughly in the section on haulage, relates to intersections common to continuous haulage systems. There, equipment requirements force the development of diamond shaped pillars. The belt haulage system is in the center entry, causing a unique and interesting problem. The intersection configuration is shown below.



a) INTERSECTION b) GEOMETRY OF INTERSECTION

Figure VII-4
Continuous Haulage Intersection

The problem arises for several reasons. First, it is desirable to make the dimension b the same as dimension a, since both are required by MESA to not exceed 20 feet. This, in turn, forces c, the crosscut dimension, to be less than 20 feet. In the illustration (Figure VII-4b) the c dimension would be b sin 60° = 20 feet (.87) = 17.32 feet. This causes difficulties, since the continuous haulage equipment is large and at best could use additional room to maneuver.

Second, the main entry has a panel belt along the rib line. The severe crowded conditions make it desirable to expand the room width of the intersection to 22 feet. The added room width greatly reduces the hazards to the men of being crushed against the rib by equipment.

Third, the configuration of the diamond pillars leads to the crushing out of the leading edge of the diamond shaped pillars.

In some areas mine operators have received approval to open up the main room to 22 feet and to maintain crosscut widths of 20 feet. This, coupled with the crushing of the edge of the pillar, leads to the intersection dimensions illustrated in Figure VII-5.

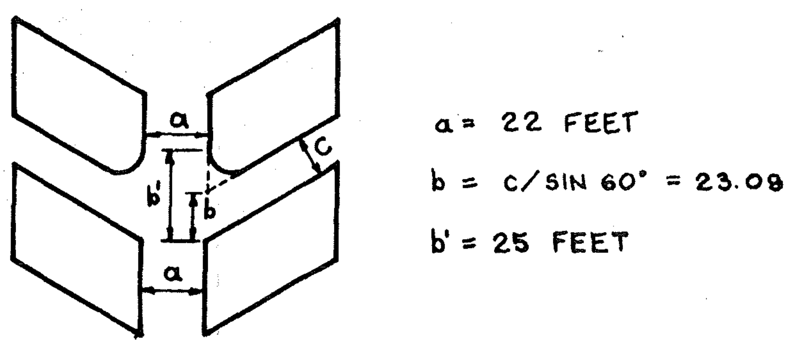


Figure VII-5
Intersection Dimensions

Thus, the area of the intersection has increased from 346 square feet to 550 square feet - a 59% increase. The safety factor can be calculated to decrease from 1.55 to 1.36 or 12%. Whether this is acceptable or not must be determined by MESA on an individual basis.

3. PRODUCTION PILLARS

Once room spans and intersections are defined, one can proceed to the dimensions and shapes of pillars. Numerous formulas exist to determine the dimensions. These are expressed in terms of the room span, the characteristics of the overburden and the coal, the percent of extraction on advance, the geometry, and other factors.

These formulas are presented in Table VII-3. The reader can note the striking similarity of the formulas. On the basis of such formulas, inspectors can determine the relative strength in terms of a safety factor.

Other formulas which express pillar strength as a function of flaws found in unit test cubes and the pillar shape are exhibited in Appendix F. Some significant relationships include the following:

- the strength of the pillar is inversely proportional to its height, and
- the strength of a pillar varies directly with the square root of the minimum width.

These formulas can also be used to determine the strength of fenders and the span of splits and pockets in pillars. This is discussed further in Appendix F,

Table VII-3

PILLAR STRENGTH

Description	Author	Formula	Symbol	Referenced (Appendix F
Pillar Strength	Holland-Gaddy	$K_2 = \frac{S\ T}{C_1\sqrt{W}}$	T = bed thickness W = Smallest lateral dimension S = Pillar Strength (psi) C_1 = Tabled compressive strength	I - 13
Pillar Strength	Morrison, Corlett & Rice	$K_2 = \frac{S^2}{C_2}\sqrt{\frac{T}{W}}$	C_2 = Crushing strength (psi)	I - 15
Pillar Strength	Obert (modified by Holland)	$K_3 = \frac{S}{C_3}(1.29+4.50\frac{T}{W})$	C_3 = Average Strength (psi) T = H (height of pillar)	I - 16
Room Span to Pillar Strength		$K_6 = \frac{W\ S_o}{1.1D(L+W)}$	S_o = Compressive strength of coal L = Span of room D = Overburden	I - 25
Pillar Strength		$K_{11} = \frac{S_{comp}}{D} \quad \frac{2\frac{W}{T}\frac{A^{11}}{A}+1}{2\frac{W}{T}+1}$	S_{comp} = Compressive Strength of coal γ = Density of rock D = Overburden T = H (Height of Pillar) W = Minimum width of pillar A^{11}= Area of pillar A = Area of pillar, room, cross-cut and intersection	I - 50
Pillar Strength	- South Africa	$K_{11} = \frac{S_{comp}}{D} \quad .91\frac{A^{11}}{A}$		I - 55

4. COMMENTS AND RECOMMENDATIONS

Appendix F was designed to summarize the relevant theory as it applies to coal mining and to utilize it to solve a variety of problems observed during the underground studies. The theory is rigorous by design and will, therefore, be difficult to read and absorb. In order to enhance the reading, however, each theory has been illustrated with examples. It is recommended that the reader at a minimum leaf through the examples, for many interesting problems are defined and solved there.

Within the constraints of this study the rock mechanics survey had to be limited to a literature survey. It is recommended that:

- research be continued on all aspects of rock mechanics related to coal mines with particular emphasis on the South African in-situ studies by Grobbelaar,
- an expanded study be conducted on pillar strength as it applies to U.S. mines regionally,
- an operating retreat mining section be established to test the validity of theories empirically, and
- reference material and calculating devices be developed with which inspectors, both company and MESA, can easily calculate relative safety factors for a variety of situations, routinely encountered.

APPENDIX A

Variable Number	Computer Level	Description

017 COURSES — What training or safety classes are offered?

0) none		
1) firstaid	- First aid	
2) acc prev	- Accident prevention	
3) fa/ap	- First aid and accident prevention	
4) rescue	- mine rescue	
5) safety	- safety	
6) roofcont	- Roof/ground control	
7) fa/safty	- First aid and safety	
8) haulage	- Haulage safety	
9) oth/comb	- Other courses or other combinations of above	

022 HOUR-ACC — Time of Day When Accident Occurred

0) unknown	5) 4-5 am	10) 9-10 am	15) 2-3 pm	20) 7-8 pm
1) 12-1 am	6) 5-6 am	11) 10-11 am	16) 3-4 pm	21) 8-9 pm
2) 1-2 am	7) 6-7 am	12) 11-12 noon	17) 4-5 pm	22) 9-10 pm
3) 2-3 am	8) 7-8 am	13) 12-1 pm	18) 5-6 pm	23) 10-11 pm
4) 3-4 am	9) 8-9 am	14) 1-2 pm	19) 6-7 pm	24) 11-12 mid-night

023 ACCSHIFT — On Which Shift Did Accident Occur?

0) maint	- Maintenance shift (no production)
1) first	- First production shift
2) second	- second production shift
3) third	- Third production shift
4) other	- Idle day, not regular production, maintenance shift, etc.
5) preshift	- Preshift examination
6) n.s.	- Not stated

025 HRSWORK — Number of Hours Victim Worked Prior to Accident

0) unknown	2) 1-2 hr	4) 3-4 hr	6) 5-6 hr	8) 7-8 hr	10) 9-10 hr
1) 1 hr	3) 2-3 hr	5) 4-5 hr	7) 6-7 hr	9) 8-9 hr	11) > 10 hrs

026 FATALS — Number of Fatalities Resulting from Accident

1) 1	3) 3	5) 5	7) 7-10	9) 21-30
2) 2	4) 4	6) 6	8) 11-20	10) 31-99

027 INJURED — Number of Persons Injured

0) 0	2) 2	4) 4	6) 6	8) 8
1) 1	3) 3	5) 5	7) 7	9) 9

028 NONFATAL — Number of "lucky" nonfatalities; i.e., persons injured or nearly killed. If a worker had been near the edge of a massive roof fall but escaped injury, he was counted as a "lucky" nonfatality.

0-9 same as #27

CODING STRUCTURE

FOR

1966-1973 FATAL ACCIDENT DATA BASE
SORTED FOR RETREAT MINING

Variable Number	Computer Level	Description
006	SEAM HGT	Average Thickness of Coalbed (Inches)

0) n.s.	4) 43-48	8) 73-84
1) 1-30 inch	5) 49-54	9) 85-96
2) 31-36	6) 55-60	10) 97-108
3) 37-42	7) 61-72	11) > 108 inch

Variable Number	Computer Level	Description
011	METHOD	Mining Method(s) Used to Produce coal Throughout Mine
	0) handload	- Report only stated that coal was loaded by hand
	1) handmine	- No machinery; pick/shovel only
	2) s.b./h.l.	- Blasted off the solid (no undercutting), hand loading
	3) s.b./m.l.	- Blasted off the solid, machine loaded
	4) conv/h/l.	- Conventional mining (undercut, blast), hand loaded
	5) conv/m.l.	- Conventional mining (undercut, blast), machine loaded
	6) conv/h&m	- Conventional mining, hand and machine loaded
	7) cont-r/b	- Continuous mining (ripper or borer)
	8) conv/cnt	- Conventional and continuous mining (ripper, borer)
	9) longwall	- Longwall system
	10) conv/lon	- Conventional and longwall units
	11) cont/lon	- Continuous and longwall units
	12) mechload	- Report only stated that coal was loaded mechanically; could be conventional or continuous
	13) auger ct	- Auger-type continuous mining
	14) auger/cv	- Auger and conventional mining
	15) cont-ns	- Continuous mining; specific type machine not stated
	16) cont-/cv	- Continuous and conventional mining; type of continuous mining machine not stated
012	STATUS	Stage of production in life of mine
	0) n.s.	- Not stated
	1) new open	- Opening new mine
	2) re-open	- Re-opening abandoned/closed mine
	3) advance	- Development stage (advance), not extracting pillar
	4) adv/ret	- Some sections advancing, some on pillar recovery
	5) part pil	- In retreat; however, pillars only partially recovered
	6) ful pil	- In retreat, pillars fully recovered
	7) lgw adv	- Longwall advance
	8) lgw ret	- Longwall retreat
	9) p&f pil	- Partial and full pillar retreat (varies with section)

Variable Number	Computer Level	Description
035	0) usbm	- U.S. Bureau of Mines
	1) mgt	- Management
	2) supvisr	- Victim's immediate supervisor
	3) vic-mgt	- Victim (supervisory or management level)
	4) vic-wkr	- Victim (worker level)
	5) othr wkr	- Other worker
	6) othr per	- Other person
	7) no one	- No one
	8) unknown	- Information was not sufficient to judge
	9) combo	- Combination of above
039	WKRCYCLE	During Which Work Cycle did Accident Occur?
	0) n.s.	- Not stated
	1) solblast	- Blasting off the solid
	2) loading	- Hand or machine loading
	3) undercut	- Undercutting
	4) facedril	- Face Drilling
	5) faceshot	- Shooting the face
	6) contmine	- Continuous Mining
	7) bolt/sup	- Roof drilling/bolting; setting support
	8) shuttle	- Hauling coal by shuttle car
	9) other	- Other, such as removing a roof fall, repair
041	INJ FREQ	Injury Frequency Rate per Million Manhours
	0) 0-10.0	4) 40.1-50. 8) 80.1-90.
	1) 10.1-20.	5) 50.1-60. 9) 90.1-100.0
	2) 20.1-30.	6) 60.1-70. 10) > 100.0
	3) 30.1-40.	7) 70.1-80.
042	VIC JOB	Job Classification of Victim
	0) brakeman	- Mainline locomotive brakeman
	1) c/m op	- Continuous miner operator
	2) c/m help	- Continuous miner helper/jacksetter
	3) beltman	- Conveyor beltman, boomman (any job directly associated with conveyors)
	4) cutter 0	- Undercutting machine operator
	5) cutter h	- Undercutting machine helper
	6) fac dril	- Face driller
	7) electrcn	- Electrician
	8) engr-tec	- Engineer, technical (I.E., etc.)
	9) genl lbr	- General laborer
	10) handload	- Hand loader
	11) loadm o	- Loading machine operator
	12) loadm h	- Loading machine helper
	13) longwall	- Longwall crew man (shearer operator, beltman, chockman, etc.)
	14) Mechanic	- Mechanic, repairman (excludes electricians)

Variable Number	Computer Level	Description
029	<u>TYPE ACC</u>	<u>General Classification of Accident Type</u>
	0) rf fall	- Roof fall
	1) rb/fc fl	- Rib/face/overhang fall
	2) machinry	- Equipment/machinery accident other than haulage (i.e., continuous miner)
	3) haulage	- Accident involving haulage of coal or men (main-line, shuttle , mantrip, etc.)
	4) intm tvl	- Intramine travel (carrying supplies, mine inspection, nonhaulage travel via tractor, jitney, shuttle car, etc.
	5) electric	- Electrical accident - includes those occurring on haulage equipment
	6) explosiv	- Explosives/ shot firing accident
	7) exploson	- Explosions caused by natural combustion (methane)
	8) bumps	- Bumps or bursts of face or ribs
	9) other	- All other accidents not falling into above categories (fall of person, flying objects, fires, inundations)
030	<u>LOCATION</u>	<u>Part of Mine in Which Accident Occurred</u>
	0) Main ent	- Main entry or air passageway (portal area, etc) where area is larger and better supported due to heavy travel.
	1) face adv	- Face area in advance stage of mining
	2) xcut adv	- Crosscut in advance area
	3) int adv	- Intersection in advance area
	4) face pil	- Face area in retreat or pillar recovery work
	5) int pil	- Intersection in retreat area
	6) xcut pil	- Crosscut in retreat area
	7) mainhaul	- Mainline haulageway
	8) sec haul	- Secondary haulageway - area designated for regular or heavy shuttle haulage
	9) abandond	- Abandoned area of mine - old workings, gob, etc.
031	<u>ACCMETH</u>	<u>Mining Method in Effect at Accident Site or Section</u>
	0) n.s.	- Not stated
	1) hand	- Hand mining (pick and shovel)
	2) s.b./h.l.	- Blast off the solid, hand loaded
	3) s.b./ m.l	- Blast off the solid, machine loaded
	4) conv/h.l.	- Conventional mining hand loaded
	5) conv/m.l.	- Conventional mining machine loaded
	6) cont	- Continuous mining
	7) longwall	- Longwall mining
	8) auger ct	- Auger continuous mining
	9) cont-ns	- Continuous mining; specific type machine not stated
035	<u>WHO PREV</u>	<u>Who Was in the Best Position to Have Prevented the Accident (not necessarily to blame for it)?</u>

Variable Number	Computer Level	Description
051	VIC SUB	Was Victim Substituting in Any Way?
	0) no	
	1) crew	- Substituting on crew but at regular job (roof bolter brought in from another shift/section)
	2) task	- Substituting at different job on regular crew
	3) equip	- Substituting on equipment such as at lunch (foreman operating miner)
	4) n.s.	- Not stated
	5) temptask	- Temporarily performing task outside normal duties
	6) diff eqp	- Operating a different piece of equipment (different shuttle car as opposed to regularly assigned car)
	7) combo	- Combination
053	VIC DO	What Was the Victim Doing at the Time of the Accident?
	0) brush fl	- Brush floor
	1) brush rf	- Brush roof
	2) chang bt	- Change bit (c/m or bolter)
	3) chang dr	- Change drill
	4) clean bt	- Clean bit
	5) Clean rb	- Clean rib
	6) clean up	- Clean up (shovel up loose coal)
	7) cont min	- Continuous mine (operate)
	8) dmp shut	- Dump shuttle (unload)
	9) drl xtnd	- Extend drill (bolter)
	10) drl face	- Drill face
	11) drl roof	- Drill roof
	12) mty dbox	- Empty dustbox
	13) xtnd cnv	- Extend conveyor
	14) elecmain	- Electrical maintenance
	15) get redy	- Get ready (for work element)
	16) handload	- Hand load
	17) hangtube	- Hang tubing
	18) hookwire	- Hookup wires (shot firing)
	19) nsrtbolt	- Insert bolt
	20) nsrt chg	- Insert charge (shot firing)
	21) nspt eqp	- Inspect equipment
	22) insp min	- Inspect mine
	23) op load	- Operate loader
	24) jack set	- Set hydraulic jack (temp)
	25) jack rel	- Relocate hydraulic jack
	26) jack rmv	- Remove hydraulic jack
	27) mv cable	- Move cables
	28) mrk hole	- Mark hole (for shot firing)
	29) machmain	- Machine maintenance (repair)
	30) machsrvc	- Service machine (oil, etc.)

variable Number	Computer Level	Description
042	15) minrhand	- Hand Miner (pick and shovel man as opposed to hand loader)
	16) motorman	- Mainline motorman, locomotive operator, mantrip operator, etc.
	17) r duster	- Rock duster
	18) rf bolter	- Roof bolter
	19) shotfire	- Shot firer
	20) shuttle	- Shuttle car operator
	21) timbrman	- Timberman (support man, prop man)
	22) trackman	- Trackman (track layer, track maintenance, etc.)
	23) vent man	- Ventilation man (brattice man)
	24) car load	- Car loader (at shuttle unloading point - mainline haulage)
	25) tractr	- Tractor operator (battery-powered)
	26) supplymn	- Supplyman, utility man
	27) wrk supv	- Working supervisor - immediate supervisor for crew (assistant section foreman, section foreman)
	28) gen supv	- General supervisor (mine foreman, superintendent, owner)
	29) fireboss	- Fireboss
	30) examiner	- Mine examiner
	31) nonmine	- Nonmine personnel
	32) other	- Other

046 MINE EXP — Victim's Experience at that Mine in Years/Months

0) < 1 mo	4) 6 -12 mo	8) 5 -15 yrs
1) 1-2 mo	5) 1 -2 yrs	9) 15 -25 yrs
2) 2 -4 mo	6) 2 -3 yrs	10) >25 yrs
3) 4 -6 mo	7) 3 -5 yrs	11) n.s.

047 JOB EXP — Victim's Experience at his Job Classification in years/months

0) < 1 mo	4) 6 -12 mo	8) 3 -5 yrs
1) 1 -2 mo	5) 12 -18 mo	9) 5 -10 yrs
2) 2 -3 mo	6) 18 -24 mo	10) > 10 yrs
3) 3 -6 mo	7) 2 -3 yrs	11) n.s.

050 TRAINING — Type of Training Victim had for Job

0) n.s.	- Not stated
1) cert/for	- Certification/formal training
2) ojt reg	- Regular on the job training - well supervised and watched
3) ojt sub	- On the job training by substituting off and on
4) lunch hr	- Learned through idle time or lunch-time operation

Variable Number	Computer Level	Description
053	81) escaping	- Escaping hazard
	82) unknown	- Unknown
	83) rep pole	- Replace trolley pole
	84) crosbelt	- Crossover (conveyor)
	85) walkwork	- Walking to or from work area
057	UNSAFPOS	Was the Victim Knowingly in an Unsafe Position (Under unsupported roof)?
	0) no 1) yes	3) unknown
058	JUDGMENT	Did the Victim Exercise Poor Judgment in his Course
	0) no 1) yes	3) unknown
059	ALONE	Was the Victim Working Alone?
	0) no 1) yes	3) unknown
060	MEN WARN	Was the Victim Warned of a Dangerous Situation by Other Men?
	0) no	
	1) yes	- 1) Victim had a warning but ignored it
	2) unknown	
	3) yes, no time	- 3) Victim had a warning but not enough time to escape
061	MINEWARN	Did the Mine Environment Give any Warning of Impending Danger? (Applicable mainly to roof falls)
	0) no	- Mine gave no warning
	1) yes-ign	- Mine gave warning but victim ignored it
	2) unknown	- Unknown
	3) yes-no time	- Mine gave warning but victim had no time to escape
	4) drummy	- Roof had a drummy sound when tested
	5) dribble	- Dribbling roof material
	6) cracking	- Roof made cracking sounds
	7) sagging	- Roof was sagging; posts took on weight
	8) working roof	- Roof was working
	9) combo	- Combination of above
062	WKRECORD	What was the Victim's General Work Record Like?
	0) safe	- Considered a safe, cautious worker
	1) conscien	- Considered to be conscientious and conservative about work and material - takes risks to save material/equipment
	2) careless	- Considered careless/unsafe in his work habits
	3) n obey	- Record of disobeying orders, warnings or established procedures

Variable Number	Computer Level	Description
053	31) mrk roof	- Mark roof (for bolting)
	32) meth tst	- Methane test
	33) pos eqpt	- Position equipment
	34) pry f/r	- Pry face/rib (scale)
	35) rockdust	- Rock dust
	36) prop set	- Set temporary support
	37) prop rel	- relocate temporary support
	38) prop rmv	- Remove temporary support
	39) rev wire	- Recover wire (shot firing)
	40) set brat	- Set Brattice
	41) shootcol	- Shoot coal
	42) shootflr	- Shoot floor
	43) cleanfal	- Clean-up fall
	44) sweepflr	- sweep floor
	45) survey l	- Survey layout
	46) scale rf	- Scale roof
	47) sweepunc	- Sweep undercut
	48) sump	- Sump
	49) testhole	- Drill test hole
	50) tram in	- Tram in
	51) tram out	- Tram out
	52) tgt bolt	- Tighten bolt
	53) testroof	- Test Roof
	54) transupl	- Transport supplies
	55) undercut	- Undercut (operate)
	56) cut vert	- Cut vertically
	57) rjck set	- Set rope (winch) jack
	58) rjck rel	- Release rope (winch) jack
	59) rjck tgh	- Tighten rope (winch) jack
	60) handpick	- Use hand pick
	61) shutl op	- Operate shuttle car
	62) jitny op	- Operate jitney
	63) trctr op	- Operate tractor
	64) mainln o	- Operate mainline haulage equipment
	65) mantrp o	- Operate mantrip car
	66) tracklay	- Lay/maintain track
	67) suprvise	- Supervise
	68) observe	- Observe operations
	69) idle	- Idle (eat lunch, sit down, etc.)
	70) ride	- Ride equipment
	71) coup.unc	- Couple/uncouple mine car
	72) switchin	- Switch tracks
	73) sprag/bl	- Sprag/block/chock mine car
	74) crlod/ul	- Load/unload mine car
	75) rerailng	- Rerail equipment
	76) other	- Other
	77) -	- -
	78) signalng	- Mainline signaling
	79) rcv mt/e	- Recover/retrieve material/ equipment
	80) belt op	- Operate conveyor belt

Variable Number	Computer Level	Description
066	AWR VIC	Was the Supervisor Aware of What the Victim was Doing? (answered only if victim was doing something unsafe or contrary to usual practice)
	0) no 1) yes 2) unknown	
067	APPROVE	If the supervisor was Aware of the Victim's Activity, did he approve, either verbally or through lack of admonition (e.g., did not reprimand bolter who was bolting out of sequence)?
	0) no	
	1) yes	
	2) unknown	
068	INST FLWD	Were the Supervisor's Instructions, Pertinent to the Incident, followed?
	0) no	
	1) yes	
	2) not give	- 2) Instructions not given pertinent to the accident
	3) unknown	- 3) Unknown if followed or given.
069	AWR COND	Was the Supervisor Aware of the Mine or Situation? condition (e.g., the support was bad, the roof was weak, the equipment was faulty)?
	0) no	
	1) yes	
	2) unknown	
074	ACC CLAS	Classification of Equipment Accident
	0) mainhaul	- Mainline haulage of coal
	1) mantrip	- Mantrip accident
	2) sec haul	- Secondary haulage - shuttle, tractor/trailer, etc.
	3) intm tvl	- Intramine travel - supplies, repair, inspection on tractor, jitney, or shuttle
	4) mach mov	- Moving face machinery - tramming loader, miner in action, etc.
	5) mach sta	- Stationary equipment - being repaired, conveyor belt, etc.
	6) elecmain	- Mainline haulage electrical accident - usually trolley wires
	7) elec oth	- Electrical repair or other electrical accident
	8) other	- Any equipment accident not falling in above categories
	9) rf fall	- Roof fall as a result of equipment operation or handling
079	ACCNATUR	Nature of Accident
	0) rf/crush	- Hit or crushed by roof or roof projection
	1) ovrhg ht	- Hit or crushed by roof-rib overhang
	2) ribcrush	- Hit or crushed by post or side obstruction
	3) post hit	- Hit or crushed by post or side obstruction
	4) mach-mac	- Machine-machine collision - crushed between or caused by
	5) pin undr	- Pinned under falling equipment (i.e., blocked up for repairs)

Variable Number	Computer Level	Description
079	6) run over	- Run over by equipment - either thrown out or standing near
	7) mvprt ht	- Hit by moving part (auger, drill head, boom, etc.)
	8) elec con	- Electrical contact
	9) other	- Other
092	MAINPART	If Maintenance Was Involved, What Part of the Equipment Needed Maintenance?
	0) lights	-
	1) brakes	-
	2) controls	- Steering, acceleration, etc.
	3) seating	-
	4) commun	- Communication equipment
	5) electric	- Electrical system
	6) hydraulic	- Hydraulic system
	7) mechanic	- Mechanical breakdown
	8) other	- Other
	9) combo	- Combination of above
095	PROTMISS	What Type of Protective Device Failed or was Missing?
	0) derail f	- Derail failed
	1) derail m	- Derail not there or used
	2) guard	- Guards needed for side/frontal protection
	3) signals	- Lights, warning devices needed
	4) self off	- Self return to neutral power needed
	5) refl mat	- Reflective material to indicate position of projection (bolts behind brattice, for instance)
	6) crossove	- Safe crossover facility, especially at conveyor
	7) emrgbrak	- Emergency brakes failed or not there
	8) elecprot	- Electrical protection missing, such as ground cover, etc.
	9) weelstop	- Mine car wheel stopper (block) failed
100	OPERATON	If Improper or bad Operation was Involved, What was Wrong?
	0) lostcont	- Operator lost control
	1) xces spd	- Excessive speed for conditions
	2) n fac tv	- Did not face direction of travel
	3) tooclose	- Followed other equipment too closely
	4) n clear	- Did not check if path was clear of people, equipment, etc.
	5) inattn	- Inattention (not watching roof conditions/ clearance closely
	6) n stop-id	- Did not turn off equipment when idling or stopped
	7) use impr	- Used equipment improperly (backing into trailer with tractor to prevent rolling away, etc.)
	8) nwarn ap	- Did not warn of approach when could have
	9) fault eq	- Knowingly operated faulty equipment

Variable Number	Computer Level	Description
101	OP QUALF	Was the Operator Qualified on that Equipment?
	0) n-ntrain	- No, had no training
	1) n-subst	- No, was substituting on it
	2) y-regop	- Yes, was the regular operator or operated regularly as part of job
	3) y-train	- Yes, had training on equipment
	4) y-sub	- Yes, was substitute operator
	5) y-mgt	- Was considered qualified by management per report
	6) unk-sub	- Was substituting, qualifications not stated
	7) d-smtran	- Debatable - had some training
	8) -	-
	9) n.s.	- Not stated
105	VISIBILITY	If visibility was a Factor, What Were Conditions?
	0) n.s.	-
	1) good	-
	2) brattice	- Brattice obstructed vision
	3) curve	- Curve in passageway prevented seeing ahead
	4) lighting	- Lack of or insufficient lighting/lighting too strong (blinding headlight)
	5) position	- Operator's position obstructed vision (low coal, leaning around side to see ahead)
	6) nface tv	- Did not face direction of travel; therefore, visibility poor
	7) dust	- Excessive amounts of dust reduced visibility
107	MOV TYPE	For Haulage or Moving Machinery Accidents, What Was the Nature of the Accident Cause?
	0) derail	- Equipment derailed
	1) ml coll	- Collision of mainline equipment (2 locomotives)
	2) mve coll	- Collision with moving equipment (shuttle and moving loader)
	3) side col	- Collision with rib or side obstruction (post, etc.)
	4) pkdequip	- Collision with parked equipment
	5) runaway	- Runaway haulage equipment (mine cars, etc.)
	6) fall/jmp	- Victim fell or jumped from moving equipment
	7) knockout	- Victim was knocked out of equipment (post caught leg and victim was pulled out)
	8) run over	- Victim was run over (had been outside machine prior)
	9) top coll	- Collision with roof or roof projection
120	MAJ CAUS	What Was the Major Cause of the Accident?
	0) imp proc	- Improper work procedures
	1) n qualf	- Worker not qualified
	2) nflw reg	- Worker not follow regulations

Variable Number	Computer Level	Description
120	3) nflw war	- Worker not follow warning
	4) n supvsn	- No or improper supervision
	5) no plan	- No mine policy or plan established/no instruction
	6) imp oper	- Improper equipment operation
	7) maint	- Lack of proper maintenance
	8) eqptfail	- Sudden or unforewarned equipment failure (power went out)
	9) canopy	- Lack of protective canopy
	10) guard eq	- Lack of equipment guards
	11) guard el	- Lack of electrical guards
	12) lighting	- Poor or lack of lighting
	13) visiblty	- Poor visibility
	14) seat pos	- Seating position
	15) contrl p	- Controls position
	16) contrl r	- Control response
	17) signal/w	- Lack of signals or warning device
	18) op-loc	- Location of operator
	19) oth desg	- Other design deficiency
	20) no seat	- Lack of seating facilities
	21) unsf pos	- Victim placed himself in an unsafe position
	22) inattn	- Inattention of victim or operator
	23) n enf sp	- Not enough roof support/improper roof support in accident area
	24) imp brak	- Improper or lack of braking device
	25) wkr comm	- Communication
	26) clearanc	- Faulty clearance
	27) nenf reg	- Regulations or policy there, but insufficient
	28) ndeenrgz	- Not de-energize or shut-off equipment
	29) imp tool	- Improper work tool
	30) bd floor	- Bad floor conditions
	31) imp block	- Improper blocking up of equipment for repair
	32) n use sf	- Did not use safety feature present on equipment
	33) -	-
	34) -	-
124	TYP FALL	Type of Fall
	0) rf-fac a	- Roof - face/advance area
	1) rf-fac-r	- Roof - face/retreat area
	2) rf-fac o	- Roof face/other/unknown
	3) rf-inter	- Roof - intersection area
	4) rf-other	- Roof - other area
	5) rbf-face	- Rib fall - face area
	6) rbf-intr	- Rib fall - intersection area
	7) rbf-othr	- Rib fall - other area
	8) ovhg-fac	- Overhang fall - face area
	9) ovhg-oth	- Overhang fall - other area

Variable Number	Computer Level	Description
126	IMM ROOF	Immediate Roof Composition
	0) n.s.	- Not stated
	1) fireclay	- Fire clay
	2) shal lam	- Laminated shale
	3) shalfrac	- Fractured shale
	4) drawrock	- Draw rock
	5) bonecoal	- Bone Coal
	6) sandston	- Sandstone
	7) snstshal	- Sandstone and shale mixed
	8) coal	- Coal
	9) oth/comb	- Other or combination
129	BAD COND	If Applicable, the type of bad roof condition Present that Contributed to the Accident?
	0) n.s.	- Not Stated
	1) clayvein	- Clay veins/spars
	2) slip/flt	- Slips/faults
	3) oth/comb	- Combination/other
	4) hrsbk/kb	- Horseback/kettlebottom
	5) cracks	- Cracks/joints/fractures
	6) slknside	- Slickenside
	7) roofchng	- Change in roof composition
	8) prev fal	- Previous roof fall
	9) drawrock	- Draw rock - very poorly consolidated rock
130	STEP TKN	If Aware of Bad Condition in Particular, What Steps Were Taken?
	0) n.s.	- Not stated
	1) none gen	- Knew it was a bad area in general, but no special care taken
	2) none spc	- Knew particular bad condition existed, but no special steps taken
	3) xtrabolt	- Extra bolts used
	4) xtrapost	- Extra posts used
	5) xbarbolt	- Used crossbars/ bolts
	6) xbarpost	- Used crossbars/posts
	7) xtracrib	- Used extra cribs
	8) stopmine	- Dangered off area, ceased mining, etc.
	9) atmptfix	- Tested, scaled, etc. until thought safe
133	HOW TEST	What Method Was Used to Test the Roof?
	0) n.s.	- Not stated
	1) visual	- Visual test only
	2) sv/+tool	- Sound and vibration - using proper roof test tool (hand/bar)
	3) sv/-tool	- Sound and vibration - improper tool
	4) sv/?tool	- Sound and vibration - unknown tool
	5) sound	- Sound method only

Variable Number	Computer Level	Description
135	IMP EQOP	If Equipment Operation Was Involved, How Was it Improper?
	0) hit supt	- Accidentally knocked out support with equipment
	1) rem supt	- Purposely knocked out support with equipment
	2) unsf pos	- Purposely knocked out support with equipment
	3) xces cut	- Cut too deep for safe loading (conventional mining)
	4) xces c/m	- Continuous miner run too deep for operator to remain under supported roof
	5) thin rib	- Cut the fender too thin - decreased natural support
	6) imp bolt	- Improper bolting sequence such that bolter exposed to unsupported roof or support not effectively provided.
143	VIC-FALL	Victim's Position in Relation to the Fall
	0) n.s.	- Not stated
	1) inby edg	- Inby edge
	2) outbyedg	- Outby edge
	3) rib edg	- Rib edge
	4) center	- Center
	5) off cent	- Off center
144	EXPOSURE	Victim's Exposure Nature
	0) n.s.	- Not Stated
	1) unsp unc	- Under unsupported roof unnecessarily (repairing machine under unsupported roof)
	2) unsp nec	- Under unsupported roof necessarily (set temporary support, test roof)
	3) perm com	- Under permanent support in compliance with roof control plan
	4) prm ncom	- Under permanent support which did not comply with roof control plan
	5) temp com	- Within temporary support set in compliance with plan
	6) tem ncom	- Within temporary support not set in compliance to plan (too few posts)
	7) set temp	- In act of setting temporary support - not fully protected yet
	8) no adjst	- Under support in minimal compliance with plan but not adjusted for faulty conditions present
	9) selfsupt	- Under roof considered self-supporting
146	FAIL NAT	Failure Nature if Permanent or Temporary Support
	0) n.s.	- Not stated

Variable Number	Computer Level	Description
146	1) selfsupt	- Failed when no support was required - self supporting roof
	2) bv anchr	- Failed above anchorline of bolts
	3) at anchr	- Failed at anchorline of bolts
	4) lw anchr	- Failed below anchorline of bolts
	5) spall	- Spall between bolts
	6) at splin	- Fell inby or at the permanent support line
	7) overperm	- Fell over permanent wood support
	8) overtemp	- Fell over temporary support
	9) btw wood	- Fell between wood supports
166	<u>TYP PILR</u>	<u>Type of pillar cut/extraction</u>
	0) n.s.	
	1) wing	
	2) pocket	
	3) lift	
	4) stump	
167	<u>% PIL RMV</u>	<u>Percent of pillar block removed</u>
	0) n.s.	
	1) none	
	2) 1-10%	
	3) 11-20%	
	4) 21-30%	
	5) 31-40%	
	6) 41-50%	
	7) 51-60%	
	8) 61-70%	
	9) 71-80%	
	10) 81-90%	
	11) 91-100%	
173	<u>TYP AREA</u>	<u>Type of Area</u>
	0) n.s.	- Not stated
	1) mainhaul	- Mainline haulageway
	2) crosscut	- Crosscut in working section
	3) gob/pil	- Gob/pillar area (old)
	4) mainenty	- Main entry (belt, air, etc.)
181	<u>TYP SUPT</u>	<u>Type of Roof Support Directly in Front of Rib Fall</u>
	0) none	-
	1) ribposts	- Rib posts
	2) tempposts	- Temporary posts

Variable Number	Computer Level	Description
186	WHT CAUS	If Applicable, What Led to Overhang or Irregular Condition?
	0) n.s.	- Not stated
	1) irr ucut	- Irregular undercutting
	2) irr cm	- Irregular continuous mining
	3) irr blst	- Irregular blasting
	4) longwall	- Long wall mining method
	5) prev fal	- Previous roof fall
	6) eq match	- Equipment mismatched (cutter cut deeper than blast holes, leaving overhang)
191	SUPVTEST	If Applicable, When Did a Supervisor Last Properly Test the Rib? (If prior to last test)
	0) n.s.	
	1) never	
	2) 0-15 min	
	3) 16-30 min	
	4) 31-60 min	
	5) 1-1.5 hr	
	6) 1.6-2 hr	
	7) 2.1-3 hr	
	8) 3.1-4 hr	
	9) 4.1-6 hr	
	10) 6.1-8 hr	
	11) > 8 hrs	
192	UNSFCONI	Primary Unsafe Physical Conditions that Contributed to Accident
	0) n.s.	- Not stated
	1) none	- None
	2) inadq ps	- Not enough permanent support
	3) inadq ts	- Not enough temporary support
	4) no supt	- No support
	5) xces wid	- Excessive entry width
	6) xces dep	- Excessive cut depth
	7) unuscond	- Unusual conditions present (bad roof, faults)
	8) imp pilr	- Improper pillar extraction (out of sequence cuts)
	9) no scale	- Loose material; not scaled/supported or improperly scaled
	10) no plan	- No plan to cover event
	11) lacktool	- No proper tools/material available or used
	12) spt dlay	- Excessive delay in setting support
	13) rmv supt	- Support removed - accidentally/purposely
	14) irrblast	- Irregular blasting
	15) n rpl sp	- Did not replace deteriorated support
	16) other	- Other

Variable Number	Computer Level	Description
194	UNSF ACI	Primary Unsafe Act by Victim That Contributed to Accident
	0) n.s.	- Not Stated
	1) none	- None
	2) no test	- Tested improperly or did not act
	3) condajus	- Did not adjust for conditions
	4) no scale	- Did not scale loose material
	5) hit supt	- Removed support accidentally
	6) rmv supt	- Removed support purposely
	7) imp eqop	- Improper equipment operation
	8) dvia sop	- Deviated from standard operating procedure
	9) not obey	- Did not follow warning/instructions
	10) sftydvic	- Did not use safety devices
	11) unsp rf	- Proceeded under unsupported roof
	12) unsf pos	- Placed self in unsafe position
	13) n up spt	- Did not put up support
	14) n mv ep	- Did not move equipment to safe place
196	PERSNALI	Primary Personal Factors in Regard to Victim that Contributed to Accident
	0) n.s.	- Not stated
	1) none	- none
	2) lcktrain	- Lack of training
	3) lck warn	- Lack of warning/instruction
	4) lck supv	- Lack of supervision
	5) imp attd	- Improper attitude
	6) ignr rul	- Willful disregard of rules
	7) n folo r	- Unconscious disregard of rules
SUPPORT		
204	PLANSTAT	Support Plan Status Relative to Approval by USBM
	0) n.s.	- Not stated
	1) none	- None
	2) verbal	- Verbal plan only - not submitted for approval
	3) sub-napr	- Submitted, not USBM approved yet
	4) approved	- USBM approved
	5) pstd-?ap	- Posted at mine, not stated if USBM approved
	6) not aprv	- Not approved (submitted and rejected by USBM)
214	TYP PERM	Type of Permanent Support Used
	0) n.s.	- Not stated
	1) not yet	- None set yet
	2) noneself	- None - self supporting
	3) boltonly	- Bolts only

Variable Number	Computer Level	Description
214	4) blt/xbr	- Bolts/crossbars (straps)
	5) blt/post	- Bolts/rib posts
	6) bl/xb/po	- Bolts/crossbars/posts
	7) blt/crib	- Bolts/cribs
	8) bl/xb/cr	- Bolts/crossbars/cribs
	9) arches	- Arches (H bars, etc.)
	10) rib post	- Rib posts only
	11) pst/xbar	- Rib posts/crossbars
	12) pst/crib	- Rib posts/cribs
	13) po/xb/cr	- Rib posts/crossbars/cribs
	14) tempjack	- Temporary hydraulic jacks only
	15) temppost	- Temporary wood posts only
	16) sprags	- Sprags
	17) brkrpost	- Break posts/permanent support
	18) crb/xbar	- Cribs/crossbars
	19) other	- Other
245	TS PLAN	Was Temporary Support Spaced According to Temporary Plan?
	0) n.s.	- Not stated
	1) no plan	- No plan
	2) No comp	- Non-Compliance to plan
	3) complian	- Compliance to plan
300		Year - Pillaring only
	1) 1972	- 1972
	2) 1973	- 1973
	3) 1974	- 1974
301	METHOD	Method
	1) oe	- Open end
	2) sf	- Split and fender
302	SPLIT	Pillar Splitting At Time of Accident
	1) yes	- Yes
	2) no	- No
303	WIDTH	Maximum Width of Pillar Split (in feet)
	1) le	- LE plan
	2) eq	- E to plan
	3) gt	- GT plan
304	ROOF FALL	Roof Fall - At What Phase of Pillar Split?
	1) drive	- While driving split
	2) at face	- At face of fender

Variable Number	Computer Level	Description
304	3) rem fend	- While removing fender
	4) set temp	- While setting temporary posts
	5) set xbar	- Setting crossbars
	6) set bolt	- Setting bolts
	7) cut bar	- While cutting barrier block
	8) Routine	- Routine testing/clean up
305	ROOF TEST	Was Roof Test Deemed Adequate?
	1) yes	- Yes
	2) no	- No
306	TOP COAL	Did Top Coal Preclude Proper Roof Testing?
	1) yes	- Yes
	2) no	- No
307	WORK	Did Roof "Work" Before Fall
	1) yes	- Yes
	2) no	- No
308	SAVE EQUIP	Did Victim Attempt to "Save" Equipment?
	1) yes	- Yes
	2) no	- No
309	UNDRSTND	Did Victim/Workman Fully Understand Roof Control Plan
	1) yes	- Yes
	2) no	- No
310	TRAIN	Training
	1) Suff	- Sufficient
	2) insuff	- Insufficient
	3) none	
311	FENDER	Fender Crush
	1) Partial	
	2) Total	
	3) n.s.	
312	TIME UNSP	Length of Time Roof Unsupported (in hours and Fractions i.e., 2.5 hours)
313	LFTWDTH	Pillar Lift Widths (in feet)
314	INSTRUCT	Roof Control Plans Taught to Workers/Victim
	1) yes	- Yes
	2) no	- No

Variable Number	Computer Level	Description
315	SF/OE	Could S & W Method Been Used in Lieu of OE?
	1) yes	
	2) no	
316	OE/SF	Could OE Method Been Used in Lieu Of S & W?
	1) y	- Yes
	2) n	- No
	3) unk	- Unknown
317	PREVENTION	Could Smaller Lifts/Splits/Pushouts Prevent Accident
	1) yes	
	2) no	
318	SEQUENCE	Was Suitable Sequence for Extracting Pillar Blocks Made?
	1) yes	
	2) no	
319	TEMPSUPP	Would More Temporary Supports Reduce the Likelihood of a Fall?
	1) yes	
	2) no	
320	ADJ PILLAR	Was Adjacent Pillar Cut - % Cut Raw Data - 0-100%
321	BARBLOCK	Were Barrier Blocks Used Adjacent to Gob Area
322	# WORKERS	No of Workers in Fall Area Immediately Prior to Fall Raw Data
323	MULTI DIR	Was Pillar Block Mined From Multiple Directions?
	1) yes	- Proper method
	2) yes	- Improper method
	3) no	- Incompliance w/ plan
	4) no	- compliance to plan
324	POSITION	Was Victim inby/outby Machine and Face or Rib?
	1) inby	
	2) outby	
325	YEAR	Year of Accident
	1966	1970
	1967	1971
	1968	1972
	1969	1973

APPENDIX B

MATRIX OF VARIABLES ANALYZED

The following pages show each of the fatal accident variables that were analyzed. These are provided to the reader in order that he might understand the scope of the fatal accident analysis. Details on why specific relationships were analyzed or excluded are available at JJDA.

The following sheets are organized according to the overlay which follows this page. Each sheet is numbered and the entire matrix can best be envisioned as being layed out on a flat surface in accordance with the overlay.

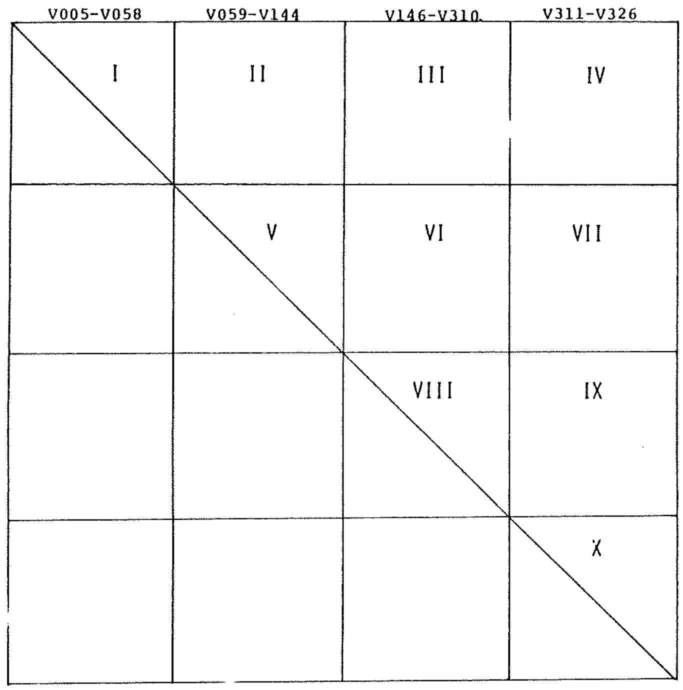
The matrix coding is as follows:

- Vxxx designates the variable analyzed. These are each described in detail in Appendix A;
- Frequency distributions were run for each variable designated by a "F";
- Crosstabulations were run for each pair of variables designated by a "C"; and
- All analysis was omitted for variables or pairs of variables designated by an "O".

Analysis was omitted for numerous reasons. These include:

- poorly defined variables,
- small sample sizes, and
- lack of a relationship between variables.

OVERLAY OF THE MATRIX VARIABLES ANALYZED



MATRIX OF VARIABLES ANALYZED - I

	V005	V006	V011	V013	V017	V022	V023	V025	V026	V027	V028	V029	V030	V031	V035	V039	V041	V042	V046	V047	V050	V051	V053	V057	V058
V005	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V006		F	C	C	0	0	0	0	C	C	0	C	C	C	0	C	0	C	0	0	0	0	0	0	0
V011			F	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V012				F	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V017					F	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V022						F	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V023							F	C	0	0	0	0	0	0	0	0	0	0	C	C	0	C	0	0	0
V025								F	0	0	0	C	0	C	0	C	0	0	C	C	0	0	0	0	C
V026									F	C	C	C	0	C	0	C	0	0	0	0	0	0	0	0	0
V027										0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V028											0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V029												F	C	C	0	C	0	C	C	C	0	0	0	0	0
V030													F	0	C	C	0	0	0	0	0	0	0	0	0
V031														F	0	C	0	C	C	C	C	0	0	0	0
V035															F	0	0	0	0	0	0	0	0	0	0
V039																F	0	0	C	C	0	C	C	C	0
V041																	0	0	0	0	0	0	0	0	0
V042																		F	C	C	C	0	C	0	0
V046																			F	C	0	C	0	C	C
V047																				F	0	0	0	C	C
V050																					F	C	0	0	0
V051																						0	0	0	0
V053																							F	0	0
V057																								F	C
V058																									F

MATRIX OF VARIABLES ANALYZED - II

	V059	V060	V061	V062	V066	V067	V068	V069	V074	V079	[illegible]	V095	[illegible]	V101	V105	V107	V120	V124	V126	V129	V130	V133	V135	V143	V144
V005	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V006	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	C
V011	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V012	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V017	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V022	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V023	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V025	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V026	0	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0
V027	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V028	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V029	0	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	C	0	0	0	C	0	0	0	0
V030	0	0	0	0	0	0	0	0	0	C	0	0	[illegible]	0	C	0	0	0	0	C	0	0	0	0	0
V031	0	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	C	C	0	0	0	0	0	C
V035	0	C	C	C	C	C	C	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0	0
V039	0	0	C	0	0	0	0	0	0	C	0	0	0	0	0	0	C	C	C	0	0	0	0	0	C
V041	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V042	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0	0	C	0	0	0	0	0	0	0	C
V046	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V047	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V050	0	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V051	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V053	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V057	0	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V058	0	C	0	C	C	C	C	C	0	C	0	0	C	0	0	0	C	0	0	0	0	0	C	0	C

MATRIX OF VARIABLES ANALYZED - III

	V146	V166	V167	V173	V181	V186	V190	V192	V194	V196	V204	V213	V214	V245	V300	V301	V302	V303	V304	V305	V306	V307	V308	V309	V310
V005	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V006	0	C	C	0	C	0	0	0	0	0	C	0	0	C	0	0	C	C	C	C	0	C	0	0	0
V011	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V012	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V017	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V022	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V023	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V025	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V026	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V027	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V028	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V029	C	C	0	0	0	0	0	0	0	0	C	0	0	C	0	0	C	C	0	0	0	0	0	0	0
V030	0	0	0	0	C	0	0	0	0	0	0	0	C	C	0	0	0	0	0	0	0	0	0	0	0
V031	C	C	C	0	C	0	0	0	0	0	0	0	C	0	0	C	C	C	C	0	0	0	0	0	0
V035	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V039	0	C	C	0	0	0	0	0	0	0	0	0	0	0	0	C	0	C	C	0	0	0	0	0	0
V041	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V042	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V046	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0
V047	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0
V050	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V051	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V053	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V057	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0
V058	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	0	C	0	0

MATRIX OF VARIABLES ANALYZED - IV

	V311	V312	[illegible]	V314	V315	V316	V317	V318	V319	V320	V321	V322	V323	V324	V325	V326
V005	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V006	0	C	0	0	0	0	C	C	C	C	0	0	C	0	0	0
V011	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V012	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V017	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V022	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V023	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V025	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V026	0	0	0	0	0	0	0	0	0	C	0	C	0	0	0	0
V027	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V028	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V029	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V030	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V031	0	0	C	0	0	0	0	0	0	C	0	0	0	0	0	0
V035	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V039	0	0	0	0	0	0	0	0	0	0	C	0	C	0	C	0
V041	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V042	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V046	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V047	0	0	[illegible]	0	0	0	0	0	0	0	0	0	0	0	0	0
V050	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V051	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V053	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V057	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V058	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

MATRIX OF VARIABLES ANALYZED - V

	V059	V060	V061	V062	V066	V067	V068	V069	V074	V079	V092	V095	V100	V101	V105	V107	V120	V124	V126	V129	V130	V133	V135	V143	V144
V059	F	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V060		F	C	C	C	C	C	C	0	C	0	0	0	0	0	0	C	0	0	C	0	0	0	0	0
V061			F	0	0	0	0	C	0	0	0	0	[illegible]	0	0	0	0	C	C	C	0	0	0	0	0
V062				F	0	0	C	0	0	C	0	0	0	0	0	0	C	0	0	0	0	0	C	C	C
V066					F	C	C	C	0	C	0	0	0	0	0	0	C	0	0	C	0	0	C	0	C
V067						F	C	C	0	C	0	0	0	0	0	0	C	0	0	C	0	0	C	0	C
V068							F	C	0	C	0	0	0	0	0	0	C	0	0	C	0	0	C	0	0
V069								F	0	C	0	0	0	0	C	0	C	0	C	C	0	0	0	0	0
V074									0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V079										F	0	0	0	0	0	0	C	0	0	0	C	0	0	0	0
V092											F	C	0	0	0	C	C	0	0	0	0	0	C	0	0
V095												F	C	C	0	C	C	0	0	0	0	0	0	0	0
V100													F	C	0	C	C	0	0	0	0	0	C	0	0
V101														F	0	C	C	0	0	0	0	0	C	0	0
V105															F	C	C	0	0	0	0	0	C	0	0
V107																F	C	0	0	0	0	0	C	0	0
V120																	F	C	C	0	0	0	C	C	C
V124																		F	C	C	0	0	0	C	C
V126																			F	C	0	0	0	0	0
V129																				F	0	0	0	0	0
V130																					0	0	0	0	0
V133																						F	0	0	0
V135																							F	0	0
V143																								F	0
V144																									F

MATRIX OF VARIABLES ANALYZED - VI

	V146	V166	V167	V173	V181	V186	V190	V192	V194	V196	V204	V213	V214	V245	V300	V301	V302	V303	V304	V305	V306	V307	V308	V309	V310
V059	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V060	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	0	C	0	0
V061	0	0	C	0	0	0	0	0	0	0	0	0	C	0	0	C	0	0	C	0	0	C	0	0	0
V062	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	C	C	0	0
V066	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	C	C	0	0
V067	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	C	C	0	0
V068	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	C	C	0	0
V069	C	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	C	C	C	0	0	0
V074	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V079	C	C	0	0	0	0	0	0	0	0	C	0	0	C	0	0	C	C	0	0	0	0	0	0	0
V092	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V095	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V100	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V101	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V105	0	0	0	C	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V107	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V120	0	0	0	C	0	0	0	C	C	C	C	0	0	C	0	C	0	C	C	C	C	C	0	C	C
V124	C	0	C	C	0	0	C	C	C	C	C	0	C	C	0	C	C	C	C	C	C	C	0	0	0
V126	C	0	C	0	0	0	0	0	0	0	0	0	C	C	0	C	C	0	C	C	C	C	0	0	0
V129	C	0	0	0	0	0	C	C	0	0	0	0	C	0	0	C	C	0	C	C	C	C	0	0	0
V130	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V133	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V135	0	0	0	0	0	0	0	C	C	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V143	0	0	0	C	0	0	0	C	C	C	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0
V144	0	0	0	C	0	0	0	C	C	C	0	0	0	0	0	C	C	0	C	0	0	0	0	0	0

MATRIX OF VARIABLES ANALYZED - VII

	V311	V312	V313	V314	V315	V316	V317	V318	V319	V320	V321	V322	V323	V324	V325	V326
V059	0	0	0	0	0	0	0		0	0	0	0	0	0	0	0
V060	0	0	0	0	0	0	0	C	0	0	0	0	0	0	C	0
V061	0	C	0	0	0	0	0	0	0	C	C	0	C	0	C	0
V062	0	0	0	0	0	0	0	0	0	0	0	0	0	C	C	0
V066	0	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0
V067	0	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0
V068	0	0	0	0		0	0	C	0	0	0	0	0	0	C	0
V069	0	0	0	0	0	0	0	0	C	0	0	0	0	0	0	0
V074	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V079	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0
V092	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V095	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V100	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V101	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V105	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V107	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V120	0	C	C	0	0	0	C	0	C	C	C	C	C	C	C	0
V124	C	C	C	0	0	0	C	C	C	C	C	0	C	0	C	0
V126	C	C	C	0	0	0	C	C	0	C	C	0	C	0	C	0
V129	C	C	C	0	0	0	0	C	0	C	C	0	C	0	C	0
V130	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V133	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V135	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V143	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V144	0	C	0	0	0	0	0	0	0	C	C	0	C	0	C	0

MATRIX OF VARIABLES ANALYZED - VIII

	V146	V166	V167	V173	V181	V186	V190	V192	V194	V196	V204	V213	V214	V245	V300	V301	V302	V303	V304	V305	V306	V307	V308	V309	V310
V146	F	0	C	C	0	0	0	C	0	0	0	0	C	C	0	C	C	C	C	C	0	C	0	0	0
V166		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V167			F	0	0	0	0	0	0	0	0	0	0	0	0	C	0	0	C	0	0	C	0	0	0
V173				0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V181					F	0	C	0	0	0	C	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V186						0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V190							0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V192								F	C	C	0	0	C	C	0	C	0	0	0	0	0	0	0	0	0
V194									F	C	0	0	C	0	0	C	0	0	0	0	0	0	0	0	0
V196										F	0	0	0	0	0	C	0	0	0	0	0	0	0	0	C
V204											0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V213												0	0	0	0	0	0	0	0	0	0	0	0	0	0
V214													F	C	0	C	0	C	0	0	0	0	0	0	0
V245														F	0	C	0	0	0	0	0	0	0	C	0
V300															0	0	0	0	0	0	0	0	0	0	0
V301																F	C	C	C	0	0	C	0	0	0
V302																	F	C	C	C	0	C	0	0	0
V303																		F	C	0	0	0	0	C	0
V304																			F	C	C	0	0	0	0
V305																				F	C	C	0	0	0
V306																					F	0	C	0	0
V307																						F	0	0	0
V308																							F	0	C
V309																								F	C
V310																									F

MATRIX OF VARIABLES ANALYZED - IX

	V311	V312	V313	V314	V315	V316	V317	V318	V319	V320	V321	V322	V323	V324	V325	V326
V146	C	C	0	0	0	0	0	C	C	C	C	0	C	0	C	0
V166	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V167	C	C	C	0	0	0	0	C	0	C	C	0	C	0	C	0
V173	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V181	C	C	0	0	0	0	0	C	0	C	C	0	C	0	C	0
V186	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V190	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V192	0	0	C	0	0	0	0	C	C	0	C	0	C	0	C	0
V194	0	0	0	0	0	0	0	C	0	0	0	0	C	0	0	0
V196	0	0	0	C	0	0	0	0	0	0	0	0	0	C	0	0
V204	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V213	0	0	0	0	0	0	0	0	0	0	0	0	0	0	[illegible]	0
V214	0	C	C	0	0	0	0	C	C	C	C	0	C	0	C	0
V245	0	C	C	0	0	0	0	0	C	C	C	0	C	0	C	0
V300	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
V301	C	C	C	0	C	C	C	C	C	C	C	0	C	0	C	0
V302	C	C	C	0	0	C	C	C	0	C	C	0	C	0	C	0
V303	C	C	C	C	0	0	C	C	0	C	C	0	C	0	C	0
V304	C	C	C	0	0	0	0	C	0	C	C	0	C	0	C	0
V305	C	C	C	0	0	0	0	C	0	C	C	0	C	0	C	0
V306	C	C	0	0	0	0	0	0	0	C	0	0	0	0	0	0
V307	0	C	C	0	0	0	0	0	0	C	C	0	C	0	C	0
V308	0	0	0	0	0	0	0	0	0	0	0	0	0	C	0	0
V309	0	0	0	C	0	0	0	0	C	0	C	0	C	0	0	0
V310	0	0	0	C	0	0	0	C	0	0	0	0	0	0	0	0

MATRIX OF VARIABLES ANALYZED - X

	V311	V312	V313	V314	V315	V316	V317	V318	V319	V320	V321	V322	V323	V324	V325	V326
V311	F	0	C	0	0	0	0	0	0	C	C	0	C	0	0	0
V312		F	C	0	0	0	0	C	0	C	0	0	0	0	C	0
V313			F	0	0	0	C	C	C	C	C	0	C	0	C	0
V314				F	0	0	0	0	0	0	0	0	0	0	0	0
V315					F	0	0	0	0	0	0	0	0	0	0	0
V316						F	0	0	0	0	0	0	0	0	0	0
V317							F	0	0	0	0	0	0	0	0	0
V318								F	0	0	0	0	C	0	0	0
V319									0	0	0	0	0	0	0	0
V320										F	C	0	C	0	C	0
V321											F	0	C	0	C	0
V322												F	0	0	0	0
V323													F	0	0	0
V324														F	0	0
V325															F	0
V326																0

APPENDIX C

MATRIX OF CHARACTERISTICS OF MINES INCLUDED IN STUDY

LOCATION	MINE SYSTEM	PILLAR SYSTEM	FACE HAULAGE	SECONDARY HAULAGE	SEAM HEIGHT	PRODUCTION (TPY) MIL.
West Virginia	Continuous	Split and Fender	Shuttle Car	Belt	60"	1.1
West Virginia	Conventional	Open End	Shuttle Car	Belt	46"	.7
Pennsylvania	Continuous	Pocket and Wing	Shuttle Car	Rail	68"	2.0
Pennsylvania	Continuous	Split and Fender	Shuttle Car	Rail	72"	.6
Pennsylvania	Continuous	Pocket and Wing	Shuttle Car	Belt	78"	1.6
Pennsylvania	Continuous	Pocket and Wing	Shuttle Car	Rail	40"	1.4
Pennsylvania	Continuous	Pocket and Wing	Shuttle Car	Belt	84"	.7
Pennsylvania	Continuous	Partial	Shuttle Car	Belt	54"	1.6
Pennsylvania	Continuous	Split and Fender	Belt	Belt	72"	.3
West Virginia	Continuous	Radius Cuts	Belt	Belt	50"	.5
West Virginia	Continuous	Radius Cuts	Shuttle Car	Belt	42"	.3
Virginia	Continuous	Split and Fender	Shuttle Car	Belt	42"	.8

MATRIX OF CHARACTERISTICS OF MINES INCLUDED IN STUDY

LOCATION	MINE SYSTEM	PILLAR SYSTEM	FACE HAULAGE	SECONDARY HAULAGE	SEAM HEIGHT	PRODUCTION (TPY) MIL.
Kentucky	Continuous	Split and Fender	Shuttle Car	Belt	96"	.8
Kentucky	Auger	Open End	Belt	Belt	32"	.1
Alabama	Continuous	Diagonal	Shuttle Car	Belt	48"	.3
Alabama	Conventional	Split and Fender	Shuttle Car	Rail	36"	.1
Alabama	Conventional	Open End	Shuttle Car	Rail	42"	.8
Alabama	Conventional	Open End	Shuttle Car	Rail	96"	.4
Illinois	Continuous	Split and Fender	Shuttle Car	Belt	84"	2.5
Illinois	Continuous	Split and Fender	Shuttle Car	Belt	106"	2.0
Illinois	Continuous	Split and Fender	Shuttle Car	Belt	84"	2.3
Illinois	Continuous	Partial	Belt	Belt	84"	4.1
Illinois	Continuous	Partial	Belt	Belt	84"	4.1
Colorado	Continuous	Split and Fender	Shuttle Car	Rail	72"	.6
Colorado	Continuous	Split and Fender	Shuttle Car	Belt	84"	.3

MATRIX OF CHARACTERISTICS OF MINES INCLUDED IN STUDY

LOCATION	MINE SYSTEM	PILLAR SYSTEM	FACE HAULAGE	SECONDARY HAULAGE	SEAM HEIGHT	PRODUCTION (TPY) MIL.
Utah	Continuous	none	Shuttle Car	Rail	76"	1.0
Utah	Conventional	none	HD	Belt	162"	.3

APPENDIX D

FIELD STUDY #1

This coal mine located in the Alabama area mined the American seam which ranges from 25 to 50 inches. The bottom consists of firm, sandy fire clay, making pretty good bottom conditions. The immediate roof is a hard shale approximately two to eight inches, followed by a sandy shale main roof. Roof conditions are very good. The American coal seam has numerous partings; however, in the particular area observed the partings did not create any problems. The roof control plan called for full pillar recovery following development. The rooms were driven on 75 foot centers, the room widths were 30 feet. Cross cuts were driven on 65 foot centers and widths were 20 feet. This left approximately 45 foot square pillars.

The section observed used a Joy 9CM Continuous Miner with a Wesco Shuttle Car Model 6632. The section was also equipped with a scoop. The seam height in the observed area was approximately 40 inches.

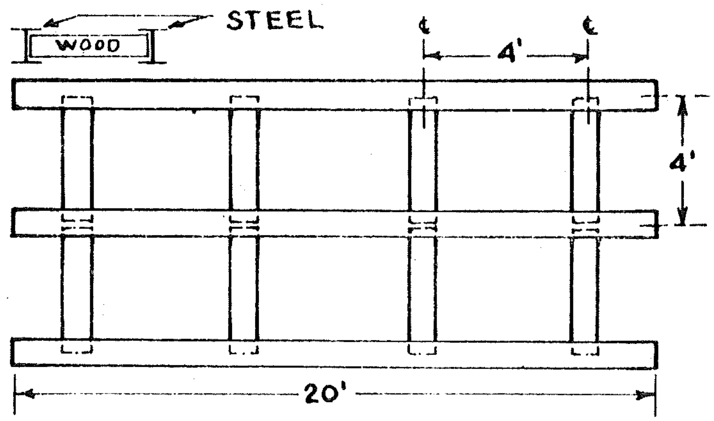
The crew consisted of a continuous miner operator and a helper, two shuttle car operators, a scoop operator, and two timber men, plus the foreman. This mine used a pillaring technique described in Section IV. Diagonal cuts were taken through the pillar moving from the right or the gob side to the left in such a manner to leave a triangle stump of coal in each of three corners. Following the completion of a lift the miner is backed out and repositioned for a second cut. At that time two men go into the pillar and set posts on four foot centers throughout the lift. During the course of the pillar extraction process the foreman stays on site watching for spalling that might take place on the triangle stumps. He also watches for any pressure signs that might be observed on the timbers.

The six wheel Wesco Shuttle Car is said to carry as much as eight to ten tons. Loading times for the continuous miners averaged one and one-half minutes. The delay time for timbering a lift took approximately fifteen minutes. However, the continuous miner could extract an entire pillar without having to tram to a new location to start a second cut. Production using this system averaged approximately 325 tons of raw coal per section shift. As pointed out before, roof conditions were ideal, which is probably the key to making the pillaring system work.

FIELD STUDY #2

The mine is located in Colorado and works the Allen seam which varies from 48 to 72 inches. The coal bed slowly undulates and progresses southwest at approximately three to seven percent incline. The entire seam plus about two feet of coal and black shale are mined. The roof is extremely hazardous and very weak. The section observed was equipped with a Lee Norse HH105 and two Joy 10SC22 shuttle cars. The section was advancing. No retreat mining was being done at the time of the visit. The section crew consisted of eighteen people, fifteen under the direct supervision of the foreman, miner operator and helper, three roof bolters, two steel beam men, two buggy drivers, two masons, one pipe man, one mechanic, two material handlers, two beltmen (who were responsible to the haulage boss working on the section), and one boss.

The roof is extremely weak and as a result the section was advancing four feet at a time and putting up a beam. The roof in this section was supported by eight inch by eight and one-half inch by three-quarter inch steel I beam, approximately eighteen feet side in length, with an eight inch by eight inch by four foot wood cross beam set on four foot centers. (See Figure D-1). The beams were supported on either end by an eight inch diameter timber angled to match the contour of the floor. The I beams were transported to the face by shuttle cars. The face of the beam was grooved and placed on the head of the continuous miner. It was



Roof Support System

Figure D-1

lifted into place by the continuous miner. Supports were set and the cross beams were put into place. Production at this section averaged approximately 22 shuttle cars per shift.

The plan for pillar recovery utilizes a continuous miner and shuttle cars. The pillar is split and mined to both the right and left. The lifts off the fender are approximately nine feet wide and are driven from a 45 degree angle from the pillar split. Roof bolts are installed in the pillar split for approximately 36 feet of the 56 foot pillar. Breaker posts are set at each opening to the pillared area. Temporary props are set adjacent to the mine operator if it becomes necessary for the operator to advance beyond the bolted or supported area.

At one time this mine used three cable shuttle car system in pillaring. They considered it very successful.

FIELD STUDY #3

This mine operates in the Alabama area. It has three operating sections. This mine does no retreat mining. Approximately 70 percent of the coal is mined during the advance phase. They do not attempt to recover the remaining 30 percent. This is a totally conventional equipment mine. In this area they are mining the

Blue Creek seam which averages approximately seven and one-half feet. Conventional equipment is used because of the high amount of rock in the seam of coal. It is their opinion that using a continuous miner under these conditions is impossible. They have encountered extremely hard rock partings in the center of the seam as thick as two feet. They experimented with continuous miners on two separate occasions. Both times it resulted in an early failure of the equipment.

In the past this mine did use an open end method of pillar extraction. They did not use the multidirectional approach; rather they took a series of lifts off the gob side and then started over and took a second series of lifts. An example of the plan is shown in Chapter V.

At one time they had experimented with pillar splitting using conventional equipment. They felt that pillar splitting required less timber and yielded higher recovery. With this seam of coal using conventional methods a large charge was required to split the rock parting. When using the open end technique, the coal would be blasted into the gob but the rock would fall right into place. Consequently, the coal desired was scattered all over the place and the rock that you didn't want was right at your feet. By utilizing the pillar split the solid fenders adjacent to the face helped to contain the coal; consequently, it improved recovery efforts. However, they continued to use the open end method in this mine because of roof conditions which they experienced using split and fender.

Occasionally they were mining in as great as a thirty-degree grade. In this case, they laid track along the hill so that the track bed was flat. Pillars would be mined above the track in a 45 degree pillar line. They would shape the pillar line so it intersected with the track at a 135 degree angle rather than a 45 degree angle. In other words, they would not allow the pillar line to intersect the track and create a very narrow point, as this point would receive a tremendous amount of pressure causing difficult mining.

FIELD STUDY #4

This company was located in West Virginia and operated numerous small mines in the area. The seam mined was #2 Gas which averaged approximately 66 inches in height. The seam contains a middle bank of approximately 14 to 20 inches in thickness. The section was equipped with a model 35 Lee Norse Continous Miner and Joy Shuttle Cars. The pillars in this mine were driven on 60 foot centers with 20 foot rooms; cross cuts were on 70 foot centers leaving 50 foot rooms. Pillars were extracted by driving a split down the long length of the pillar. This was followed by the extraction of the fender inby the split. Final extraction of the second fender was from the cross cut. Offset intersections were utilized. The offset intersection assisted in the driving of the splits by providing a straight on access for the initial cut. The section observed had a new foreman with approximately 4 years' total underground experience. This was his first opportunity to work in retreat mining and although he had been instructed on the proper techniques he had only pulled some 15 pillars prior to our study. Numerous delays occurred on the section, many of which could be attributed to supervision.

This operation was an excellent example for the need for good management in retreat mining. It is essential to properly plan each cut in order to insure the fenders are of the proper dimensions for extraction. Both mining and roof bolting have to be carefully coordinated to avoid delays. The timing of the movement of supplies to the face area is critical. Overall the production in this section averaged between 300 and 500 tons. The conditions were extremely good. Undoubtedly as this foreman gained experience, his ability in retreat mining will improve. Under these conditions he should have been easily able to double his production.

FIELD STUDY #5

This mine is located in central Pennsylvania in the lower Kittanning seam. Seam thickness is approximately 72 inches and the seam is located approximately 450 feet below the surface. This is a small mine, and had very poor conditions in sections. Normally they develop pillars approximately 40 feet by 60 feet which during pillar extraction are split through the center and extracted from the fenders in a relatively common technique. The roof conditions are fairly good. They have approximately six inches of slate which is removed at first mining. On top of this is a fairly firm shale which holds very well. Under normal conditions they use five foot bolts, utilizing resin grounded bolts in timbers when they encounter bad conditions. Roof conditions are generally pretty good. Very little sloughing is observed. The section that we observed was being developed for a shortwall. The shortwall pillar was approximately two hundred feet wide. The equipment included a Lee Norse 45HH, a Lee Norse tram car, and two Lee Norse extensible belts which fed onto a section belt. Also this section was equipped with a Lee Norse roof bolter and an S&S scoop. This set of entries was developed on 50 foot centers with crosscuts on 70 foot centers. The room widths were approximately 16 feet. Three entries were being driven at approximately distances of 4000 feet in preparation for the shortwall. The extensible belt was being used in the lift entry. Under operation the continuous miner was cutting a twenty foot deep pass approximately ten feet wide. This cut was completed in a little under thirteen minutes. With six and a half feet of coal in this section the cutting rate was slightly over 4 tons per minute. Physically the surge car had a grate over the top of it with openings on approximately six inch centers. The purpose of this grate is to draw out any heavy lumps of coal so they can be broken prior to going on the belt system. Unfortunately, a large number of the particles included pieces of shale which caught in the grate and caused delays. In operation the extensible belt sagged considerably. The stands that are to be set into place as the equipment moves forward were not properly positioned, causing the belt to sag slightly and to spill coal on the floor. The crew experienced some difficulty in coordinating their efforts. Consequently production was not as fast as it may have been. The continuous miner was able to cut far faster than the capacity of the surge car, consequently the miner had to be shut down occasionally to keep from overloading the surge car. Initially this system was used without chain drags on the surge car. However, this resulted in the overloading of the belt and it was not a satisfactory arrangement. Consequently they installed two chains to keep the belts from overloading, which slowed down the continuous mining process. The overall production, however, was an improvement.

The system required a miner, a miner helper, surge car operator, a man on each extensible belt, one general laborer to help set up stands, two roof bolters, plus the foreman. Under good conditions they normally expect to get between a hundred and sixty and two hundred feet of advance per day. They have not attempted to do any pillaring with this continuous haulage system.

FIELD STUDY #6

This mine was located in central West Virginia. The mine was very small, operating a single section over two shifts a day. The drift mine was operating under about two hundred feet of cover, mining the Number Two Gas seam. The seam locally averaged about 72 inches. On the section observed there was a fire clay bottom which was quite muddy. The roof consisted of ten to fifteen feet of firm shale and was generally very good. This mine did all developmental work using conventional equipment and retreat work using continuous equipment. In the section observed they used a 35Y Lee Norse continuous miner and Joy 16SC shuttle cars.

The continuous miner was in a bad state of repair with, among other things, a bad pump. The high speed tram was inoperative, consequently, cutting and loading times were not representative. The operator was not particularly experienced and encountered a tremendous amount of problems in pulling down top coal. The continuous miner was not able to fully reach to the top. Consequently mining had to be planned to leave approximately two feet of bottom coal during the first pass in order to reach all of the top coal. Following the advance of approximately ten feet the miner would back up and take the two feet of bottom coal. Often the miner operator did not do this. On one occasion he took all of the coal off the floor and attempted to reach the top coal using the stabilizing jacks on the side of the miner. These jacks were located too far back on the miner and are not designed for this. The front end of the miner tipped down to the bottom. In a second attempt posts were set under the miner in an attempt to give it the added reach. After three or four attempts the pillar split was widened to approximately thirty feet to put additional pressure on the top. A bar was used to pull down the top coal. Bolting was not required. The lift was timbered off following its completion. Approximately fifteen to twenty percent of the coal was left in stumps. The plan also required that the operator expose himself to unsupported roof, to distances as great as thirty feet from the last row of bolts. The internal span was also as great as thirty feet. The timbermen were required to go into the pillar to set posts without setting up temporary support. The miner operator, the foreman, and the miner helper were all underneath unsupported roof during the course of the extraction of the pillar.

The mine normally extracted one pillar per shift. Over the last twelve days they extracted eleven pillars. This is eighty cars per day average, or approximately four hundred tons per shift. The observed day saw forty-seven cars or two hundred thirty-five tons.

Overall the mine suffered from poor equipment maintenance, improper training and poor supervision.

FIELD STUDY #7

This mine, located in central Illinois, mines out of the Illinois Number Six coal seam. The maximum cover is approximately seven hundred fifty feet; it has a grey shale top, and a fire clay bottom. The coal seam is approximately one hundred eight inches in thickness. The coal is soft and approximately eight inches of top coal is left to prevent deterioration of the roof. The mining technique calls for the driving of 4 panel entries off the main to the end of the property, driving two rooms approximately four hundred feet in length to a bleeder block system, and extracting the chain pillar between the two rooms and the solid block adjacent to the two rooms. The room center is approximately fifty feet and the room width fourteen to fifteen feet, leaving pillars thirty-five by fifty feet. Roof bolting is done only on advance; extraction of pillars utilizes the split and fender method. The section averages approximately five hundred tons per shift and ranges anywhere from two hundred twenty tons to eight hundred eighty tons.

FIELD STUDY #8

This company is located in central West Virginia, mining the Number Two Gas coal seam. The depth of cover over this coal bed is approximately one hundred fifty to five hundred feet. The mine is opened through a drift. The immediate roof consists of ten to fifteen feet of firm shale, and the main roof is sandstone. The coal bed itself is approximately seventy-two inches high. The bottom is about ten feet of firm shale. This particular section observed is relatively small; the section is reached by walking. They use a thirty inch conveyor belt as their primary haulage system. This mine is part of a series of small coal mines located in this area. Each mine uses one to three sections. They operate two shifts per day. The equipment utilized here is old and not particularly well maintained. Interestingly, they utilize a conventional set of equipment to develop and a continuous miner to extract pillars. Typical daily production for this mine is approximately four hundred tons. The crew consists of six men plus a beltman and a foreman.

The continuous miner was a 35Y Lee Norse and was in a particularly bad state of repair. The operator was not experienced and continually encountered problems. The attitude of the section was not good. Numerous abuses were observed of the equipment. Section members were observed smoking. Mining practices were generally considered unsafe. An unusually wide split was opened in the pillar and the operation continued in areas which would generally be considered unsafe. This operation was not considered typical of what was observed during most of the studies.

FIELD STUDY #9

This mine, located in southwest Pennsylvania, mines in the Pittsburgh coal bed. The mine operates under approximately 400 feet of cover. The mine in the Pittsburgh coal bed has a thickness of approximately eighty-seven inches. The immediate roof is draw slate with a sandstone main roof. The bottom is a fire clay. Normally in the mining operation the draw slate is taken with the coal bed since it will normally fall if not otherwise taken. The mine is located along a river. It is the last operating mine in this area; consequently, it receives a tremendous amount of water that flows through other adjacent mines. A tremendous number of pumps are required to keep the water out of the mine. The mine is equipped with a Lee Norse CM 48-3K and a Fletcher single boom roof bolter. Joy 10SC shuttle cars are also utilized. The standard crew consists of six individuals, including the foreman, one miner operator, two shuttle car operators, one roof bolter, one ventilation man. Roof support consists of seventy-two inch bolts with four inch bearing plates along with a two by twelve header board. Under bad conditions additional roof bolts will be inserted. The mine has been developed on one hundred by eighty foot centers with eighteen foot wide rooms, leaving approximated an eighty-two by sixty-four foot block of coal. The coal is extracted using the pocket and wing technique. Production during the shift averages better than five hundred tons. The section crew operated very well as a team, supervision appeared good. The operation was one of the better observed during the course of the underground studies.

FIELD STUDY #10

This mine is located in northern Virginia. It is entered through a drift. The seam mined is approximately forty-eight inches thick. The cover is approximately four hundred feet. The immediate roof is a fragile to firm shale. The main roof is sandstone, and the floor is a soft shale. The roof has numerous faults and is considered quite dangerous. Primary haulage system for the mine is a forty-two inch belt with thirty-six inch section belts. Conventional mine equipment is utilized in most of the mine. The observed section had a Joy 12CM with remote control with an eleven foot head. The mining system for the extraction of pillars was split and fender. The section was developed with seven entries with five of the six pillars being pulled and the sixth one being left for bleeder. Room widths were twenty feet. Blocks were developed on seventy foot centers. FMC 5-L shuttle cars were used in the section, discharging into a Rosco feeder crusher. Discharge time was quite slow. Pillars were extracted by driving a sixteen to twenty foot wide split through the pillar block and then extracting the fender. Remote control was utilized in the extraction of the fender. However, conventional controls were used to drive the split. Three pillars were extracted at the same time. Total production for this shift ranged from one hundred thirty-five as much as one hundred ninety feet of advance per day, approximately four hundred fifty to five hundred tons. The crew was experienced with the use of remote control and it was an effective means for the extraction of pillars.

FIELD STUDY #11

This mine, located in southern Pennsylvania, was one of the top ten mines in the country. At one time they mined in excess of nine thousand tons per day. Currently, the operation mined about four thousand tons per day. They mine the Pittsburgh seam. Locally it was approximately sixty-eight inches of coal. The top was draw rock, the thickness of about twelve inches, followed by about six inches of boney coal. Generally the conditions were considered difficult, the draw rock and the roof coal would not stay up. In addition they had numerous vertical and horizontal clay seams which further complicated matters. The pillars are extracted by pocket and wing. Approximately eighty-five percent of the coal inside the pillar was recovered. In the section observed the roof was extremely bad; many clay veins caused numerous complications. In places the roof was coming down from as high as twelve feet. Additional support was set adjacent to the pillar consisting of cribs, crossheaders, posts, and additional roof bolts. Six foot and nine foot roof bolts were used. The team worked very well and was carefully supervised. During the observation days two hundred sixty four tons of coal were mined and three hundred ninety-nine tons of coal were mined.

FIELD STUDY #12

This mine, located in eastern Kentucky, was entered through a drift. The seam was approximately ninety-six inches in thickness with a fractured shale roof. Mining in the observed section was close to the outcrop. No pillaring plan had yet been established as the operation was comparatively new. The mine was developed with rooms on fifty feet centers and crosscuts every fifty feet. The crew consisted of one continuous miner, one helper, two roof bolters, one ventilation man, two shuttle car operators, and one foreman. The section was equipped with the Jeffrey 120H continuous miner and three cableless shuttle cars loading into a Stamler BF14B-5 feeder. Production in this mine averaged approximately 1400 tons per shift. The production was largely attributable to the attitude of the crew and the utilization of cableless haulage equipment.

FIELD STUDY #13

This mine was located in Alabama. It was a medium size and mined the Pratt seam which averaged five feet to ten feet in height with numerous shale partings. The immediate roof consisted of firm shale, firm sandy shale, or sandstone. The bottom consisted of six inches of shale and eighteen inches of fire clay. This mine used entirely conventional equipment. Their experience and the experience of adjacent mines have shown considerable difficulty in using continuous mining equipment due to the large and extremely hard shale partings which exist in the seam. On a minimum, this mine used forty inch roof bolts, 5/8ths in diameter, with a 5 x 5 bearing plate. Where conditions warranted, additional roof bolts were utilized. The mine itself had approximately six hundred feet of cover. The seam was relatively flat, dropping probably one foot for every sixty or seventy feet of distance. The open end method of mining utilized in the conventional sections gave a very high recovery rate. The crew consisted of fourteen men on the observed section. Three of the fourteen men were new employees and were being trained. The normal crew consisted of a loading machine operator, loading machine helper, a cutter operator, a driller operator, one roof bolter, one roof bolter helper, two shuttle car drivers, three timber men, and one foreman. In addition to the normal, conventional equipment the mine section was equipped with a timber machine which was used to move timbers to adjacent areas where timbering was required. A normal section in this mine runs approximately four hundred fifty tons per day, or under peak conditions as much as seven hundred tons. During the observation the section was just completing the retreat of this panel and were operating on three pillars with five active places. The result was extreme congestion with men, cables, and equipment operating virtually on top of one another. This in itself created a hazard as well as drastically curtailing production.

FIELD STUDY #14

This mine, located in southwest West Virginia, operated numerous small drift mines in the immediate location. The particular mine studied extracted coal out of the Cedar Gove coal bed which locally varied from forty-two inches to sixty inches of thickness. The depth of cover was approximately nine hundred feet. There were two adjacent coal beds above the seam. The immediate roof consisted of six feet of firm sandy shale, the bottom was approximately ten feet of firm shale. The roof control plan called for the insertion of bolts a minimum of thirty-six inches in length. Roof conditions underground were generally hazardous with numerous breaks and fractures observed. The section observed was utilizing continuous bridge conveyor system. They were in the advanced phase of mining. The low seam height made the continuous system compare favorably to shuttle car systems. In fact, the system consistently outproduced the comparable shuttle car system by as much as fifty percent. The section was developed on sixty-five foot centers. The section crew consisted of nine men, including a continuous miner operator, continuous miner helper, two bridge conveyor operators, one rock duster, two roof bolters, one electrician, and one ventilation man. The continuous miner was a Joy 11CM. The bridge conveyor system was built by Long Airdox. They were equipped for a five entry panel. Production during the day was approximately seven hundred tons. Overall the operation went quite smoothly. The crew appeared to be very experienced with the use of the equipment. The attitude was generally good.

FIELD STUDY #15

This mine located in central Illinois was quite large. Locally the coal bed averaged approximately seven feet. They have a fire clay bottom approximately fifteen feet thick and the shale and slate immediate roof of varying thickness. The roof control plan called for bolts of a minimum length of twenty-four inches, and larger where deemed appropriate. The maximum cover in this area was approximately three hundred fifty feet. Entry widths approximately twenty-two feet were driven on fifty foot center with seventy foot between crosscuts. Crosscuts were cut on a sixty degree angle to facilitate the use of a continuous haulage system. No retreat mining was practiced in this area. The section was equipped with a Marietta Miner which dumped into a surge car which fed a continuous belt haulage system suspended from the ceiling. This in turn dumped onto a panel belt. Other equipment on this section included a Long Airdox twin arm roof bolter and a Kersey scoop and tractor. Production on this section averages over a 1000 tons per shift with peaks approximately 1500 tons per shift. This equipment has not been used in retreat mining, however, it appears this use is highly feasible.

FIELD STUDY #16

This mine, located in southwestern Pennsylvania, operates in the Freeport seam, which locally is approximately seven foot nine inches in thickness. The maximum cover in the area is four hundred feet but averages approximately three hundred feet. Full retreat mining is planned and practiced. The roof consists of shale with approximately fourteen inches of cannel coal being left in the immediate roof. The bottom is level and consists of approximately three feet of fire clay. Approximately twelve inches of black slate is removed during mining. The room centers in the area to be mined were fifty feet by one hundred feet. The room widths were eighteen feet. This left pillar blocks approximately thirty-two feet by eighty-two feet in length. Four foot long roof bolts were used and inserted on four foot centers. A two split ventilation system was used with returns on both sides. The crew consisted of seven people including a foreman and miner operator and helper, two shuttle car operators, one roof bolter, one mechanic, and one foreman. The section was equipped with a Lee Norse 38H

Continuous Miner, two Torkar Shuttle Cars, one Fletcher Single Boom Roof Bolter, and two Bantam Rock Dusters. Coal was dumped into twelve ton mine cars on the main line. A split and wing system was used for pillar extraction. The block were not the proper size for this type of mining. This was because development of this section had taken place many years ago. The operator of the continuous miner was required to go beyond supported roof in order to punch through. The sequence of extraction was generally poor and inadequate. Too much coal was being left in each stump. This was due to poor planning. Production during the visit averaged approximately three hundred fifty tons. In general the attitude of the workers was very good and they worked well as a team; however, the problems that existed on the section were due partially to lack of good management and partially to improper dimensions of the section.

FIELD STUDY #17

This mine was located in central Pennsylvania. It was an extremely large mine, and mined both the upper and lower Freeport seams. The seam locally was approximately five feet in thickness. The immediate roof was sandy shale, approximately thirty-two inches in thickness, with a limestone shale main roof. The bottom was fire clay, approximately ten feet in thickness. Draw slate was removed as they mined the coal. The shale roof was extremely brittle and broke off in small chunks. Roof conditions varied dramatically throughout the mine, from areas which could be supported with a four foot bolt down to areas that had to be heavily timbered. Resin bolts were used predominately throughout the mine. They eliminated the need for straps. On retreat the pillars were developed with rooms on forty foot centers and crosscuts on fifty-four foot centers, leaving a 20 x 34 foot block of coal. This block of coal could be extracted by taking lifts into the blocks but without requiring men or equipment to go into the block in the extraction process.

FIELD STUDY #18

This mine, located in Utah, mined a coal bed that averaged twelve to fourteen feet in thickness. Approximately four to five feet of coal was left in the immediate roof, with sandstone above that. The bottom was sandstone. No retreat work was being done at this mine at this time. The mine was equipped with conventional equipment; places were drilled, cut and shot and scoops were used to load and haul the coal to the belt. A 15RU cutting machine was used; a Long Airdox Coal Drill and Stamler Feeder Breaker Equipment. The face crew consisted of two loader hauler dump drivers, one cutter operator, one driller, two shot and powder men, one ventilation man, and a foreman. Production averaged 640 tons during that ten hour shift.

FIELD STUDY #19

This mine located in Utah was mining a coal seam of approximately eight feet in thickness and on a seventeen degree pitch. The mining unit consisted of a 38H Lee Norse Continuous Miner, two Joy 10SC Shuttle Cars, and a twin boom Fletcher Roof Drill. The section was mining on a side pitch and doing very well. Stopper drillers were used to insert roof bolts. No retreat room and pillar mining was planned at this time.

<u>APPENDIX L</u>

EMPLOYEE CLASSIFICATION
CONTINUOUS VS CONVENTIONAL

CCC CODE	CONTINUOUS 1971 LAST	VARIED	1972 LAST	VARIED	1973 LAST	VARIED	1974 LAST	VARIED	CONVENTIONAL 1971 LAST	VARIED	1972 LAST	VARIED	1973 LAST	VARIED	1974 LAST	VARIED
FACE EMPLOYEES																
001 BELT/CONVEYOR MAN	320	79	250	133	242	90	323	124	389	118	186	77	156	51	184	68
002 ELECTRICIAN	257	47	436	174	525	156	522	168	430	91	531	161	447	110	366	93
003 ELECTRICIAN HELPER	3	1	10	5	5	3	20	13	9	1	15	7	10	3	11	4
004 MECHANIC	1938	388	2001	542	2039	495	2278	641	784	129	717	205	630	137	529	142
005 MECHANIC HELPER	26	20	52	33	62	27	66	35	15	10	32	13	25	6	23	10
006 ROCK DUSTER	39	12	42	22	23	5	35	15	70	17	35	14	19	6	23	2
007 SHOTFIRERS/SHOOTER/B	40	10	35	12	60	18	46	14	1324	294	1464	411	1125	283	984	272
008 STOPPING BUILDER/VEN	94	58	245	98	192	73	232	77	141	73	257	105	160	56	172	70
009 SUPPLY MAN	302	62	150	76	116	40	97	38	248	20	61	27	31	11	31	15
010 TIMBERMAN/PROPMAN/JA	1332	338	1521	418	1005	306	991	287	438	96	408	124	264	74	210	53
011 WIREMAN	10	5	18	11	20	5	9	2	10	7	7	4	4	1	6	2
013 CLEANUP MAN	1	1	52	44	43	14	76	16			112	53	67	14	54	10
015 FAN ATTENDANT	1	1	1		1		1		1	1	3	1	1		1	1
016 LABORER	532	246	825	355	930	264	1121	299	483	236	621	270	546	170	660	237
021 BEATER/SHOTFIRE HELP			1	1					6	3	6	3	11	5	8	2
022 [illegible] MAN	411	92	406	144	322	72	323	97	190	43	143	51	124	38	142	40
[illegible] COAL DRILL HELPER	40	7	6	3	4	1	1		263	23	69	29	50	24	41	14
[illegible] COAL DRILL OPERATOR	34	38	51	25	55	17	43	16	1167	635	1436	420	1132	303	1054	293
[illegible] CONTINUOUS MINER HEL	1722	695	1993	1011	1708	872	1877	886	24	12	26	20	20	10	15	6
[illegible] CONTINUOUS MINER OPE	6505	1814	5782	1689	6309	1780	6948	2090	79	40	62	37	53	32	35	17
037 CUTTING MACHINE HELP	41	20	27	18	38	17	41	25	808	287	658	273	456	181	365	140
038 CUTTING MACHINE OPER	93	44	75	31	74	24	47	28	4264	1105	3258	896	2783	717	2704	736
039 HAND LOADERS	4		9	3	24	2	17	3	84	58	85	17	38	6	66	15
040 HEADGATE OPERATOR	7	5	12	7	12	9	16	9	14	5	4	3	3	3	1	
041 JACK SETTER/LONGWALL	45	21	34	20	24	6	50	16	11	6	3	1	3	1	1	
042 LOADING MACHINE HELP	17	5	35	21	48	23	35	14	604	383	735	265	596	214	514	190
043 LOADING MACHINE OPER	588	173	562	167	519	144	509	171	2193	432	2055	563	1703	430	1538	438
044 LONGWALL SHEAR OPR/P	2	2	10	8	5	2	8	4	4	2	3	3	1			
[illegible]	6	3	6	5	11	5	16	11	35	27	46	30	52	25	50	24
[illegible] ROOF BOLTER	4435	945	4594	1305	4835	1241	5264	1499	2199	380	2269	621	1895	476	1727	450
[illegible] ROOF BOLTER HELPER	45	24	160	76	134	47	197	66	79	39	124	58	96	38	133	56
047 ROOF BOLTER MOUNTED	87	65	453	280	488	220	423	144	11	5	23	12	26	14	18	13
[illegible] SECTION FOREMAN	2803	403	3190	720	3378	676	3674	766	1433	194	1616	351	1294	224	1262	276
[illegible] SHUTTLE CAR OPERATOR	5779	1142	6250	1735	6182	1610	6420	1653	3039	535	3259	801	2617	622	2379	591
[illegible] DRIVER					2	2	1		2		1					
052 TAILGATE OPERATOR	13	6	33	20	39	10	33	14	13	7	26	6	18	3	16	[illegible]
053 UTILITY MAN	626	287	996	417	902	299	998	292	385	184	605	219	509	162	414	[illegible]
054 SCOOP CAR OPERATOR			242	138	356	185	551	258			183	80	194	[illegible]	305	109
[illegible] JACKSETTER			447	320	579	264	649	215			5	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
TOTALS	16237	7359	31314	10171	31393	8974	33958	10005	21263	5598	21149	6294	[illegible]	[illegible]	[illegible]5045	[illegible]

[illegible] EMPLOYEES

[illegible]

MESA RESEARCH FILE DATA

Appendix E
Page 1 of [illegible]

EMPLOYEE CLASSIFICATION
CONTINUOUS VS CONVENTIONAL

OCC CODE	CONTINUOUS 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED	CONVENTIONAL 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
NON FACE EMPLOYEES																
103 ELECTRICIAN HELPER	3	1	21	12	12	3	16	8	24	17	37	10	30	11	37	18
104 MECHANIC	904	238	970	279	932	231	994	231	493	112	517	120	454	96	557	153
105 MECHANIC HELPER	22	13	63	36	50	25	52	30	9	3	24	8	12	3	31	9
106 ROCK DUSTER	104	25	160	55	108	29	69	12	86	26	129	35	118	22	119	22
108 STOPPING BUILDER/VEN	266	140	420	113	362	92	354	93	236	114	306	96	220	52	194	52
109 SUPPLY MAN	291	88	333	121	279	74	245	66	199	53	245	67	216	53	211	58
110 TIMBERMAN	242	80	319	107	187	52	154	41	192	65	271	77	200	49	198	54
111 WIREMAN	76	34	102	23	84	21	108	35	56	30	58	11	29	9	32	10
112 BELT VULCANIZER	1	1	6	5	8	4	9	1	6	3	18	6	14	4	14	2
113 CLEANUP MAN	41	16	31	11	42	22	39	18	72	27	55	17	29	9	23	3
114 COAL SAMPLER	2	1			1	1	2	1	1	1					1	
115 FAN ATTENDANT			4	1	2		1				1	1	3	2	2	1
116 LABORER	1337	279	1729	574	1862	495	2197	397	918	175	1278	384	1247	246	1474	255
117 ROCKMAN	4	2	17	4	16		16	2	3		16	1	14	1	10	
118 OILER/GREASER	35	21	47	16	62	16	83	39	32	10	80	28	80	33	70	14
119 WELDER	12	7	24	12	38	12	34	4	5	1	21	6	13	4	17	9
122 COAL DUMP OPERATOR	44	15	99	36	110	23	101	24	50	15	78	27	91	30	76	26
123 TRANSIT MAN	6	3	26	7	32	6	29	2	3	1	11		7		6	2
146 ROOF BOLTER			97	80	100	43	104	49			29	17	27	21	25	7
149 LABOR FOREMAN/BULLGA	45	34	281	150	264	115	262	116	71	41	258	121	219	79	165	59
154 BELT CLEANER	176	100	459	157	485	129	418	105	151	90	265	91	230	63	172	50
155 CHAINMAN	3	1	7	1	7	1	11	2	3	2	4	1	2	1	5	1
156 ROCK DRILLER (STOPER			22	9	12	5	6	3			14	8	7	2	8	2
157 PUMPER	130	20	120	29	115	23	125	22	87	13	85	17	74	10	70	15
158 ROCK MACHINE OPERATO	36	24	75	39	69	27	64	34	16	5	46	29	36	16	28	7
159 WATER LINE MAN	11		23	15	23	5	17	3	4		4	1	6	1	7	3
160 SHOPMEN			4	2	6		20	3	1		5	2	11	1	11	2
TOTALS	4373	1317	6390	2250	6275	1713	6634	1640	3354	978	4901	1540	4205	993	4439	1039
TRANSPORTATION EMPLOYEES																
201 BELT/CONVEYOR MAN	238	131	122	45	24	5	32	8	202	70	111	39	60	14	68	18
216 TRACKMAN	258	100	366	101	290	53	267	66	206	104	212	46	143	31	137	24
220 CAGER	11	4	15	3	17	2	6	1	14	7	6	1	2			
221 HOISTMAN	15	4	9	4	9		3		14	1	8	1	1		1	1
240 LOADER HEAD OPR/ROSC	40	25	58	28	34	13	40	15	67	35	76	23	58	14	46	13
250 SHUTTLE CAR OPERATOR			60	47	26	8	25	8	1	1	36	27	20	10	17	8
261 BATTERY STATION OPER	4	1	6	4	6		10	5	12	2	24	7	24	3	27	2
262 BRAKEMAN/ROPE RIDER	29	13	43	18	35	15	28	14	39	15	46	15	44	9	49	20
263 DISPATCHER	16	1	30	2	30	3	41	11	14	4	20	6	10	2	7	
269 MOTORMAN	849	103	849	185	775	161	780	140	1001	134	869	178	663	110	473	97
276 DRIVER	31	19	15	4	1	1	7	4	42	16	33	9	1		1	1
277 BUGGY PUSHER	2	2	1		7	1	16	9			8	5	7	6	16	6
301 CONVEYOR OPERATOR			3						2		2					
[illegible]2 ELECTRICIAN	3	1	1						1		[illegible]					

EMPLOYEE CLASSIFICATION
CONTINUOUS VS CONVENTIONAL

DCC CODE	CONTINUOUS 1971 LAST	VARIED	1972 LAST	VARIED	1973 LAST	VARIED	1974 LAST	VARIED	CONVENTIONAL 1971 LAST	VARIED	1972 LAST	VARIED	1973 LAST	VARIED	1974 LAST	VARIED
TRANSPORTATION EMPLOYEES																
303 ELECTRICIAN HELPER	1															
304 MECHANIC	13	4	2						9	1	19					
305 MECHANIC HELPER											1					
307 BLASTER/SHOTFIRER/SH									2	1	1					
308 MASON	1										1					
309 SUPPLY MAN	4	3	3						1		4					
312 BELT VULCANIZER	2															
313 CLEANUP MAN			1						1							
315 FAN ATTENDANT	2	1														
316 LABORER	4		6						4	1	3					
317 RODMAN									1							
319 WELDER (SHOP)	2	1	2								1					
320 CAGE ATTENDANT/CAGER	1	1									1					
322 COAL DUMP OPERATOR	2	1	2								4					
340 BOOM OPERATOR	2	1							3	1						
360 SHOPMAN	5	1	2								4					
366 WATERBOY									1							
367 COAL SHOVEL OPERATOR	1								1							
368 BULLDOZER OPERATOR	5										1					
373 CAR DROPPER	2		1						2		3					
374 CLEANING PLANT OPERA	6	2	1						1	1	3					
376 COAL TRUCK DRIVER	1		6						2		4					
380 FINE COAL PLANT OPER	2	2							3							
382 HIGHLIFT OPERATOR	2	1	1						2		2					
385 LAMPMAN	5	2	4						1		4					
386 REFUSE TRUCK DRIVER	1								2	1	4					
388 SCALPER-SCREEN OPERA	1															
390 SILO OPERATOR	5															
391 STRIPPING SHOVEL OPE			1													
392 TIPPLE OPERATOR	16	7	2						7	1	6					
393 WEIGHMAN	1										1					
394 CARPENTER											1					
396 WATCHMAN			1													
TOTALS	1583	431	1613	441	1254	262	1255	281	1659	396	1525	357	1033	205	842	190
SUPERVISORY EMPLOYEES																
402 MASTER ELECTRICIAN	28	15	61	20	67	21	53	7	27	9	39	13	33	4	30	6
404 MASTER MECHANIC	14	6	34	11	32	9	39	14	14	5	19	5	11	3	12	3
414 DUST SAMPLER	41	20	51	13	51	8	39	7	24	14	18	1	7		9	1
418 MAINTENANCE FOREMAN	30	14	132	44	141	41	166	49	21	5	71	24	51	11	73	21
423 SURVEYOR	17	11	26	3	32	4	28	4	11	3	9		[illegible]	3	15	
[illegible] ASST MINE FOREMAN/AS	[illegible]	212	483	235	325	[illegible]	356	106	100	66	119	50	83	21	103	[illegible]
[illegible] MINE FOREMAN/MINE MA	[illegible]	167	253	67	265	63	252	63	243	96	226	27	[illegible]	[illegible]	[illegible]	11
[illegible] PRE SHIFT VENT MININ	[illegible]	9	[illegible]	3	29	4	27	2	17	3	18	1	7	1	3	

EMPLOYEE CLASSIFICATION
CONTINUOUS VS CONVENTIONAL

CONTINUOUS

OCC CODE		1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
	SUPERVISORY EMPLOYEES								
462	FIRE BOSS/PRE-SHIFT	169	112	204	41	231	41	220	46
464	INSPECTOR	15	9	20	12	21	9	31	12
481	SUPERINTENDENT	56	8	85	13	95	12	72	14
489	OUTSIDE FOREMAN	1		8	4	5	2	3	2
494	PREPARATION PLANT FO	3		7		4	1	2	
495	SAFETY DIRECTOR	17	8	32	5	37	7	28	6
497	TIMEKEEPER/CLERK	2		16	1	8		6	
590	EDUCATION SPECIALIST					1			
592	MINE SAFETY INSTRUCT			2	1				
593	SAFETY REPRESENTATIV	2	2			1		1	
594	TRAINING SPECIALIST							1	
	TOTALS	1058	600	1447	483	1345	337	1324	334

CONVENTIONAL

OCC CODE		1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
	SUPERVISORY EMPLOYEES								
462	FIRE BOSS/PRE-SHIFT	85	41	132	35	114	33	118	31
464	INSPECTOR	7	2	9	2	5	1	5	1
481	SUPERINTENDENT	46	3	72	14	60	11	64	10
489	OUTSIDE FOREMAN	3	1	7	1	3	1	3	1
494	PREPARATION PLANT FO			2		1	1	1	
495	SAFETY DIRECTOR	7	2	18	4	14		12	1
497	TIMEKEEPER/CLERK	1		9		4	1	2	
590	EDUCATION SPECIALIST								
592	MINE SAFETY INSTRUCT	1							
593	SAFETY REPRESENTATIV					1	1		
594	TRAINING SPECIALIST								
	TOTALS	607	250	764	207	548	122	648	170

EMPLOYEE CLASSIFICATION
DEVELOPMENT VS RETREAT

CODE	FACE EMPLOYEES	DEVELOPMENT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED	RETREAT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
01	BELT/CONVEYOR MAN	580	172	350	157	317	115	425	157	176	45	91	[illegible]	85	32	[illegible]	
02	ELECTRICIAN	469	94	721	249	768	216	675	195	222	47	250	[illegible]	212	57	21?	
03	ELECTRICIAN HELPER	19	1	20	12	12	5	24	15	3	1	6	1	2			
04	MECHANIC	1959	358	2067	580	2058	503	2154	593	805	177	679	176	663	153	6??	
05	MECHANIC HELPER	28	18	78	43	79	30	83	43	13	11	9	6	16	8		
06	ROCK DUSTER	87	19	71	35	40	10	56	16	19	9	6	1	2	1		
07	SHOTFIRERS/SHOOTER/B	1176	264	1326	380	1059	277	921	254	213	51	196	52	152	28	1??	
08	STOPPING BUILDER/VEN	203	115	449	133	341	12?	346	125	34	16	52	18	30	10	[illegible]	
09	SUPPLY MAN	414	55	136	67	100	29	94	39	140	30	73	36	46	22	3?	
10	TIMBERMAN/PROPMAN/JA	1291	310	1461	398	949	280	908	227	533	143	496	162	335	106	29?	[illegible]
11	FIREMAN	15	10	17	10	21	8	16	4	4	2	3	1	2	1		
13	CLEANUP MAN	1	1	126	81	98	28	113	24			46	32	14		[illegible]	
15	FAN ATTENDANT	2	2	4	1	2		2	1								
16	LABORER	859	412	1327	559	1270	348	1548	456	247	133	254	92	303	104	30?	
31	BEATER/SHOTFIRE HELP	2	1	7	4	10	4	8	2	4	2			1	1		
32	BRATTICE MAN	441	98	400	155	323	81	374	114	161	44	111	37	121	26	8?	
33	COAL DRILL HELPER	293	27	72	32	54	25	43	16	48	6	7	2	2	1		
34	COAL DRILL OPERATOR	1140	598	1442	426	1174	313	1121	299	196	114	153	43	93	25	7	
35	CONTINUOUS MINER HEL	1203	478	1521	789	1352	672	1408	663	538	226	494	242	450	209	[illegible]	
36	CONTINUOUS MINER OPE	4582	1332	4255	1300	4654	1375	5280	1589	2012	537	1567	425	1677	434	155?	
37	CUTTING MACHINE HELP	688	232	595	253	448	180	361	143	200	96	127	53	68	27	4	
38	CUTTING MACHINE OPER	3529	957	2940	832	2580	670	2492	698	824	223	482	12?	31?	85	25	
39	HAND LOADERS	731	388	684	190	514	90	429	46	304	167	271	3?	153	3?	11	
40	HEADGATE OPERATOR	19	8	23	17	11	6	8	5	76	41	47	29	6?	3?	[illegible]	
41	JACK SETTER/LONGWALL	65	36	55	33	35	8	47	19	142	94	370	1??	53	18?	[illegible]	
42	LOADING MACHINE HELP	196	309	675	248	567	211	485	178	13?	87	106	9?	7?	2?	[illegible]	
43	LOADING MACHINE OPER	2270	498	2212	611	1894	486	1731	529	617	135	47?	1??	3??	9?	[illegible]	[illegible]
44	LONGWALL SHEAR OPR/P	6	4	18	12	12	3	20	6	101	62	110	[illegible]	15	6?	1?	[illegible]
45	ROCKMAN	24	15	47	27	50	23	56	30	17	15	16		1	7		
46	ROOF BOLTER	4857	999	5704	1515	5302	1365	5480	1533	1749	324	139?	3??	137	314	[illegible]	[illegible]
47	ROOF BOLTER HELPER	101	51	230	110	188	67	282	101	18	9	45	[illegible]	8?	1?	[illegible]	[illegible]
48	ROOF BOLTER MOUNTED	71	47	355	225	337	147	294	103	25	[illegible]	11?	[illegible]	15?	7?	[illegible]	[illegible]
49	SECTION FOREMAN	3?72	445	?711	82?	3645	7??	?332	555	1?47	1??	120	[illegible]	115	[illegible]	[illegible]	[illegible]
50	SHUTTLE CAR OPERATOR	[illegible]	[illegible]	7326	205?	6796	1755	?749	?786	257?	[illegible]	[illegible]	[illegible]	[illegible]	43?	[illegible]	[illegible]
51	STALL DRIVER	2		1		2	2			1		[illegible]		1	1?		[illegible]
52	TAILGATE OPERATOR	1?	2	37	[illegible]	42	[illegible]	4?	[illegible]	7?	[illegible]	[illegible]		1??	57	5	[illegible]
53	UTILITY MAN	[illegible]	35?	29?	44?	112?	3??	134	31?	20?	[illegible]		[illegible]	27	?0		[illegible]
54	SCOOP CAR OPERATOR			[illegible]	[illegible]	5?	[illegible]		119?				[illegible]	7?	[illegible]	[illegible]	[illegible]
55	OFF-SIDE JACKSETTER			[illegible]	2??	3??	[illegible]	[illegible]	[illegible]				[illegible]	10?	8?	[illegible]	[illegible]
	TOTAL	[illegible]		[illegible]	[illegible]	[illegible]						[illegible]		11?	50?		[illegible]

FACE EMPLOYEES

EMPLOYEE CLASSIFICATION
DEVELOPMENT VS RETREAT

OCC CODE	NON FACE EMPLOYEES	DEVELOPMENT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED	RETREAT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
103	ELECTRICIAN HELPER	24	21	53	19	39	16	49	24	46	32	61	27	56	23	60	47
104	MECHANIC	929	252	945	285	951	234	1096	330	2490	448	3263	767	3169	697	3333	716
105	MECHANIC HELPER	32	17	67	33	44	22	73	34	70	51	197	91	180	83	203	102
106	ROCK DUSTER	132	38	195	67	150	36	143	32	336	68	455	131	367	93	327	81
108	STOPPING BUILDER/VEN	386	186	583	166	492	127	478	127	621	338	996	279	1020	244	1003	253
109	SUPPLY MAN	361	117	482	165	431	110	416	117	707	168	758	262	805	277	801	264
110	TIMBERMAN	310	86	507	159	366	100	331	84	391	110	479	211	358	130	339	105
111	WIREMAN	91	41	108	26	84	28	105	31	321	175	460	120	424	70	409	102
112	BELT VULCANIZER	6	2	22	6	26	7	23	2	7	6	53	27	41	13	32	7
113	CLEANUP MAN	90	34	72	22	75	29	51	18	37	19	76	37	52	16	57	25
114	COAL SAMPLER	3	2			1	1	3	1			6	1	5	2	2	1
115	FAN ATTENDANT	2	1	3	1	3	1	1	1	1	1	3	1	3		1	
116	LABORER	1798	303	2478	738	2602	606	3123	542	3520	771	4849	1491	5536	1333	6093	1176
117	RO[illegible]	6	1	30	6	28	1	28	1	23	10	51	6	35	3	32	4
118	OILER/GREASER	44	17	90	35	118	47	129	42	64	39	134	51	147	53	130	54
119	WELDER	13	8	31	9	44	15	49	13	51	25	81	33	121	28	137	36
122	COAL DUMP OPERATOR	56	22	122	43	145	41	123	36	157	26	283	94	288	56	263	51
123	TRANSIT MAN	7	2	30	7	33	6	30	4	28	14	87	25	84	17	78	12
146	ROOF BOLTER			90	65	89	44	87	45			376	280	479	173	456	157
149	LABOR FOREMAN/BULLGA	90	53	442	214	386	167	343	146	94	53	691	377	651	265	802	320
152	BELT CLEANER	244	143	551	213	616	172	538	140	447	238	1039	362	1065	291	1017	229
155	CHAINMAN	3	1	6	1	4	1	6	1	11	6	34	8	57	9	65	6
156	ROCK DRILLER (STOPER			44	18	23	6	16	4			49	28	58	21	66	25
157	PUMPER	129	20	119	25	125	25	126	27	452	75	489	94	441	79	447	90
158	ROCK MACHINE OPERATO	36	18	85	42	81	34	66	25	113	73	191	110	193	66	165	90
159	WATER LINE MAN	14		24	15	23	3	20	5	25		43	22	56	25	55	20
160	SHOPMEN	1		8	3	12	1	22	5	55	42	42	15	54	13	43	10
	TOTALS	5700	1673	8869	2973	8655	2297	9260	2278	11232	2982	17210	5749	17961	4686	18909	4626

OCC CODE	TRANSPORTATION EMPLOYEES	DEVELOPMENT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED	RETREAT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
201	BELT/CONVEYOR MAN	279	115	157	51	52	13	57	20	910	484	422	123	114	31	110	30
216	TRACKMAN	318	146	385	82	301	74	270	56	853	453	1303	329	1197	230	1163	250
220	CAGER	15	12	21	4	17	2	11	4	34	17	34	8	24	3	14	3
221	HOISTMAN	16	3	18	6	15		11	2	47	10	33	5	33	3	40	10
240	LOADER HEAD OPR/RUSC	85	47	98	34	79	24	76	25	54	32	93	30	84	33	81	28
250	SHUTTLE CAR OPERATOR	1	1	76	52	31	15	39	12			64	56	76	40	73	20
261	BATTERY STATION OPER	12	3	29	11	29	3	32	5	8	6	24	7	31	10	25	12
262	BRAKEMAN/ROPE RIDER	50	21	68	21	61	19	61	26	275	165	394	107	349	94	376	124
265	DISPATCHER	16	5	31	6	21	3	25	9	131	24	174	38	177	24	162	24
269	MOTORMAN	1263	189	1229	291	1053	201	824	173	2623	301	3132	500	3080	550	3061	556
276	DRILLER	69	32	52	15	3	1	7	5	16	9	16	5	6	3	8	3
277	BUGGY PUSHER	2	2	9	5	14	7	33	15	5	3	1					
301	CONVEYOR OPERATOR	2		3						7	1	3					
302	ELECTRICIAN	4	1	6						45	2	14					

ELECTRICIAN 1 45 2 14

EMPLOYEE CLASSIFICATION

DEVELOPMENT VS RETREAT

TRANSPORTATION EMPLOYEES	DEVELOPMENT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED	RETREAT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
ELECTRICIAN HELPER									7		3					
MECHANIC	17	4	15						239	72	52	1				
MECHANIC HELPER			1						5		3					
BLASTER/SHOTFIRER/SH	2	1	1													
[illegible]	1		1													
SUPPLY MAN	5	3	8						27	2	14					
BELT VULCANIZER									3		1					
CLEANUP MAN	1		1						12	1	10					
COAL SAMPLER									18		1					
FAN ATTENDANT	1	1							2							
[illegible]	9	1	9						110	35	45					
[illegible]	2								3							
DIPPER/GREASER									14	1	3					
REFILL (SHOP)	1		3						32	5	10					
CAGE ATTENDANT/CAGER	2	2							10		2					
HOIST OPERATOR/HOIST	1		1						11		7					
COAL DUMP OPERATOR	2	1	6						22	4	6					
TRACK [illegible] MAN	1								5	3						
COAL DRILLER	1															
[illegible] OPERATOR	3	1	1						4	3	1					
[illegible]											2					
[illegible] MAN	4	1	6								1					
DISPATCHER											4					
BATTERY [illegible]	1															
[illegible] OPERATOR	2								1							
BULLDOZER OPERATOR	5		1						33	3	14					
[illegible]									1							
[illegible] OPERATOR									5							
[illegible] ATTENDANT									11		3					
CAR DROPPER	4		4						75	4	20					
CLEANING PLANT OPERA	7	3	3						59	3	15	1				
ROAD GRADER OPERATOR									1							
COAL TRUCK DRIVER	2		1						47	1	20					
CRANE OPERATOR/DRAGL									1		1					
DRYER OPERATOR									30	5	3					
FINE COAL PLANT OPER	4	1							15	2	1					
[illegible] OPERATOR	4	1	2						15	1	10					
HIGHWALL DRILL HELPE									2		1					
HIGHWALL DRILL OPERA									2		2					
[illegible] MAN	7	2	7						46	5	31					
REFUSE TRUCK DRIVER	3	1	3						27		11					
ROTARY BUCKET EXCAVA											1					
SCALER/SCREEN OPERA									4		2					
[illegible]	4								7		1					
STRIPPING SHOVEL OPE									5		1					

Appendix E
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EMPLOYEE CLASSIFICATION

DEVELOPMENT VS RETREAT

OCC CODE	DEVELOPMENT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED	RETREAT 1971 LAST	1971 VARIED	1972 LAST	1972 VARIED	1973 LAST	1973 VARIED	1974 LAST	1974 VARIED
TRANSPORTATION EMPLOYEES																
402 TIPPLE OPERATOR	11	2	7						96	12	30					
[illegible] WEIGHMAN	1								12		7					
[illegible] CARPENTER			1						3		7					
[illegible] WATCHMAN			1								2					
TOTALS	2249	602	2265	578	1676	362	1446	352	6026	1669	6055	1290	5171	1021	5133	1060
SUPERVISORY EMPLOYEES																
402 MASTER ELECTRICIAN	39	19	85	29	92	20	81	11	63	22	91	18	112	31	94	22
404 MASTER MECHANIC	18	8	45	12	31	10	45	14	34	14	71	23	69	16	78	19
414 DUST SAMPLER	54	25	43	6	37	4	35	5	64	27	136	26	132	19	148	28
418 MAINTENANCE FOREMAN	42	17	142	53	139	35	188	53	135	80	439	152	554	105	634	118
423 SURVEYOR	22	10	25	7	22	8	32	5	17	7	72	15	75	12	62	9
452 ASST MINE FOREMAN/AS	239	181	443	207	284	89	339	112	242	162	341	186	623	175	663	187
449 MINE FOREMAN/MINE MA	428	193	386	100	346	81	376	102	719	433	501	118	427	110	414	99
456 ENG (ELEC VENT MININ	28	7	37	7	23	3	28	2	125	56	128	27	118	16	127	15
462 FIRE BOSS/PRE-SHIFT	175	107	250	55	273	60	275	64	345	207	527	114	576	120	627	136
[illegible] INSPECTOR	18	10	21	12	22	6	27	11	19	6	72	29	96	10	99	25
[illegible] SUPERINTENDENT	74	6	131	25	131	22	115	17	114	14	183	25	183	26	188	31
[illegible] OUTSIDE FOREMAN	4	1	12	3	6	2	10	3	9	1	27	6	9	2	11	2
[illegible] PREPARATION PLANT FO.	3		7		4	1	2		15	1	23		5	2	8	1
[illegible] SAFETY DIRECTOR	17	9	33	6	31	5	30	5	37	14	81	20	89	11	109	9
[illegible] UNION REPRESENTATIVE													1	1		
[illegible] TIMEKEEPER/CLERK	3		18		10	1	8		19	1	64	6	38	4	23	1
[illegible] EDUCATION SPECIALIST													1			
592 MINE SAFETY INSTRUCT	1		1						3	1	0	3				
593 SAFETY REPRESENTATIV	2	2			2	1	1		1		1		2	1		
594 TRAINING SPECIALIST							1									
TOTALS	1107	595	1579	522	1449	350	1593	409	1961	1016	2985	758	3111	675	3310	702

APPENDIX F

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LIST OF SYMBOLS TO APPENDIX F

Page	Formula	Symbol	Definition
2	1	W	Width of minimum lateral dimension
2	1	T	Seam thickness
2	1	D	Overburden thickness
3	2	B	Minimum width of basis of Arch III
3	2	K_1	Safety Factor
5	5	σ_c	Compressive strength of the roof along the plane
5	5	γ	Unit weight of the roof rock
5	5	h	Height of the dome
5	5	Log	Logarithm base 10
6	11	h_{max}	Maximum height of dome
6	11	B_{max}	Maximum width of pressure arch which cannot exceed width of opening
6		L	Span of opening
6	12	σ_t	Tensile strength of the roof along the plane
6	12	m	Ratio of horizontal to vertical rock pressure
6	13	σ_x^o	Horizontal rock pressure
6	13	σ_z^o	Vertical rock pressure
8	15	n	Poisson's ratio
10	17	W_2	Convergence
10	17	e	Basis of natural logarithm
13	20	S	Pillar strength in Psi
13	20	W	Least lateral dimension
13	20	K_2	Safety factor
13	20	S_p	Compressive strength in Psi of a cube
13	20	d	Edge of cube of material

LIST OF SYMBOLS TO APPENDIX F

(Continued)

Page	Formula	Symbol	Definition
13	20	C_1	Values for most rocks are tabulated
15	25	C_2	Crushing strength in Psi
15	25	H	Height of pillar
15	25	K_2	Safety factor
16	28	C_3	Average strength in Psi of cubical test specimens
16	28	K_3	Safety factor
17	Fig. 4	L	Width of room
17	Fig. 4	d	Width of layered strata
18	30	σ_x	Normal axial stress
18	30	y	Vertical distance of point A from the neutral axis of the beam
18	30	M_x	Moment at the distance x of the point A from the origin of coordinates
18	31	q	Uniform load on the beam per unit length and width
18	32	τ_{xy}	Shear stress
18	33	V_x	Shear on the cross-section
18	34	E_i	Young's modulus of layer i
19	35	σ_{tens}	Maximum tensile stresses
19	35	σ_{comp}	Maximum compressive stresses
20	40	R_o	Modulus of rupture
20	40	K_1	Safety factor
20	44	K_5	Safety factor
20	44	J	Shear strength of rock
21	47	γ	Density of rock
24	50	P	Pneumatic or hydraulic pressure per unit length and width
26	52	S	Average pressure
26	52	W_{max}	Length of the pillar

LIST OF SYMBOLS TO APPENDIX F

(Continued)

Page	Formula	Symbol	Definition
26	53	F	Weight of the column extending from the roof of the pillar to the surface
26	54	A	Area of pillar
28	55	S_o	Compressive strength of coal pillar
28	56	S_t	Compressive strength to crush test cube of diagonal t
28	56	t	Diagonal dimension of test cube
28	58	K_6	Safety factor
31	61	G	Compressive strength in x plane
31	61	A	Moment aim
31	61	A_u	Moment aim for axially unloaded beam
31	62	d	Layer thickness
31	61	Q	Lateral load
32	65	P	G_u or b_e compressive strength
32	65	G_1	Compressive strength for axially loaded beam
32	65	q	Uniform transverse load on the beam
32	66	V	Vertical load at end of beam
32	67	S	Driving force per failure by cracking
32	68	N	Sum of projections of P and V in the rival direction to the crack
34	69	F	Force resisting failure by sliding
34	69	μ	Coefficient of friction along crack
34	70	K_7	Safety factor of the beam due to failure by sliding
34	71	σ_{max}	Maximum compressive stress
34	74	S_{comp}	Compressive strength
34	74	K_q	Safety factor of the beam due to failure by cracking
39	76	L	Room span
39	76	M_{max}	Maximum negative moment at the abutments

LIST OF SYMBOLS TO APPENDIX F
(Continued)

Page	Formula	Symbol	Definition
39	76	q	Uniform transverse load on the beam
39	76	$M_{(x=L/2)}$	Maximum positive moment
39	77	n	Poisson's ratio
39	77	E_p	Modulus of elasticity of the pillar
39	77	E_r	Modulus of elasticity of the roof
39	77	A	Center to center distance between pillars
39	77	d	Roof plate thickness
39	77	H	Height of the pillar
39	77	C	Pillar factor
39	77	I	Multiplication factor #1
39	77	II	Multiplication factor #2
43	77	B	Crosscut width
43	77	$\cancel{M_x}$	Maximum tensile and compressive stress
43	77	K_8	Safety factor - compressive strength
43	77	K_9	Safety factor - tensile strength
43	77	K_{10}	Maximum of K_8 and K_9
50	82	$\overline{E_y}$	Effective Young's modulus
50	82	A^{11}	Area of pillar
50	82	A^{1}	Total area including rooms, crosscuts, and intersections
50	82	E	Young's modulus of the ore
50	82	$\overline{\sigma_{xy}}$	Average stress in the x direction by an applied stress in the y direction
50	83	R	Ratio of pillar area to the total area
55	88	L	Room span
55	88	W_{max}	Maximum lateral dimension of pillar
55	88	e	Extraction ratio
58	88	K_{11}	Safety factor

LIST OF SYMBOLS TO APPENDIX F

(Continued)

Page	Formula	Symbol	Definition
65	90	S_N	Strength of the weakest element in a cubical pillar
65	90	N	Number of flaws
65	90	S_u	Average strength of a unit cube
65	90	N_u	Number of flaws in a unit cube
65	90	$\sigma(S_u)$	Standard deviation of the strength of the weakest element in the cubical pillar
65	90	W_N	Width of the pillar
65	90	W_u	Width of the unit cube
71	93	S	Average strength of the pillar
71	93	W	Minimum lateral dimension of pillar
71	93	H	Pillar height
71	93	C_4	Empirically determined constant
71	93	C_5	Empirically determined constant
72	96	C_6	Empirically determined constant
72	100	C_7	Laboratorally derived constant
73	101	C_8	Empirically determined constant
73	102	C_9	Material constant for coal
73	102	C_{10}	Function of C_9
82	106	C_{11}	Compressive strength of specimens having W/H=1
84	107	R	Radius of the cross section
88	110	x	Distance from surface of pillar to point of concern
90	112	P	Mean vertical pressure
90	112	e	Percent of pillar recovery
95	113	d_{min}	Minimum roof sag observed immediately upon opening of an excavation
95	113	d_{max}	Maximum roof sag allowed to occur at any time

LIST OF SYMBOLS TO APPENDIX F

(Continued)

Page	Formula	Symbol	Description
95	113	d	Thickness of unsupported strata
95	113	L_b	Bolt length
95	113	d_b	Roof sag after bolting
95	113	d_p	Maximum expected sag to occur without bolting prior to roof destruction
95	113	d_u	Roof sag after bolting
97	116	P	Bolt density (bolts/sq.ft.)
97	116	C_{12}	Proportionally constant
104	122	A	Length of lamina
104	122	B	Width of lamina
104	122	n_1	Number of rows of bolts
104	122	n_2	Number of bolts per row
104	122	L_t	Load per bolt
106	123	q_i	Load per unit length
106	123	E_i	Modulus of elasticity
106	123	I_i	Moment of inertia

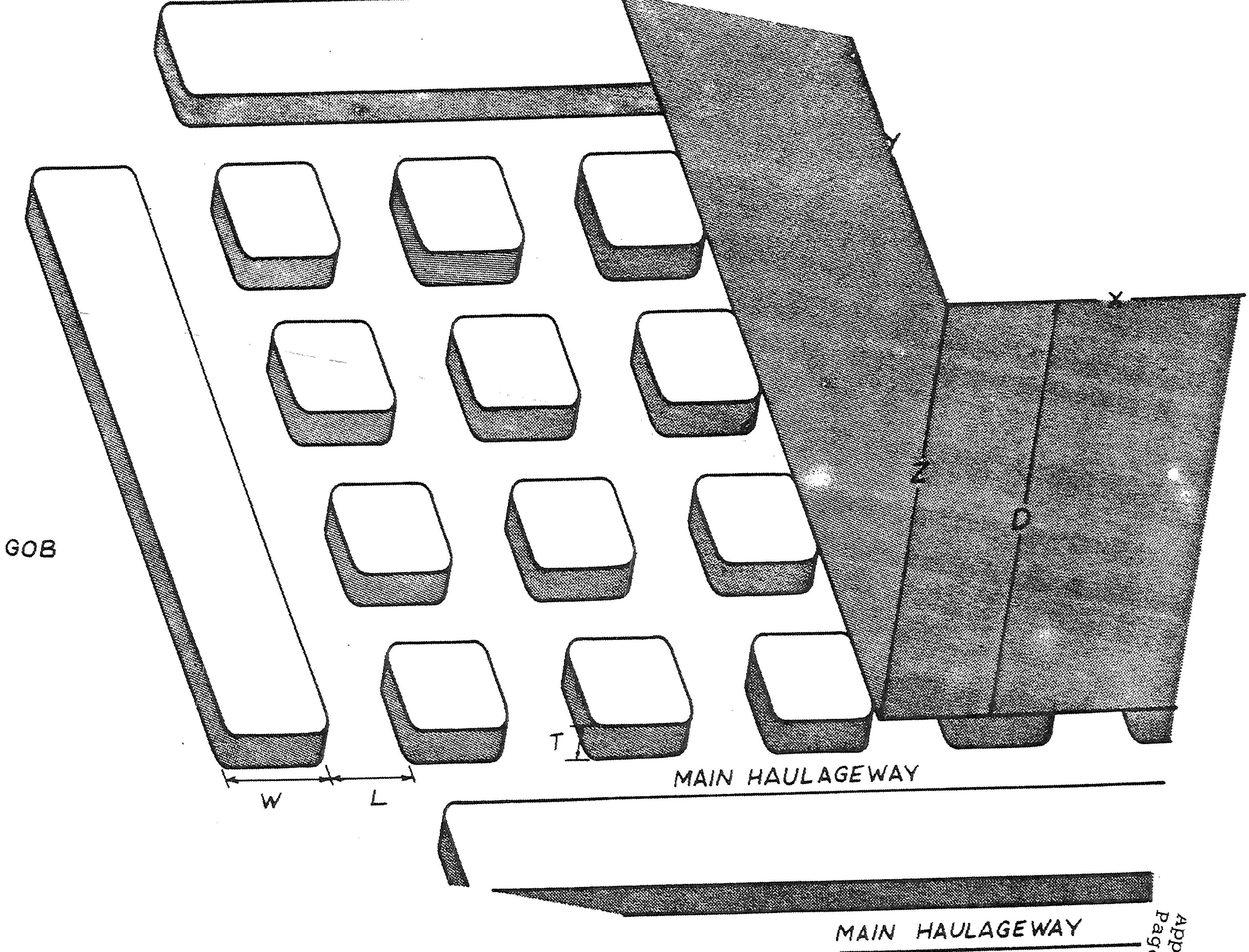
INTRODUCTION

Although the present study is concerned with the optimal design of pillar recovery techniques, the design of room and pillars during the phase of advance mining cannot be overseen. Indeed, it is impossible to safely and efficiently recover a pillar which has been designed erroneously in the first place. Furthermore, the theory of optimal pillar design during the phase of advance mining is very important to the study at hand for a second reason: A quantitative theory of pillar recovery has not been developed yet. Thus, a logical platform for the launching of such a theory is the corresponding theory of advance mining. There is a striking similarity between the goals and the working frames of the two theories. The structural mechanical theory of room and pillar design during the phase of advance mining is concerned with the ground stability within the mine as a whole. The working frame is the entire mine. The stopes are main haulageways, crosscuts, and production rooms. The abutments of unmined ore are the barrier and production pillars. On the other hand, the theory of pillar recovery during the phase of retreat mining is concerned with the ground stability within the partially mined pillar and its immediate environment. The working frame is the pillar and the four stopes defining the pillar. The stopes within the pillar are lifts, such as splits and pockets. The abutments of unmined ore within the pillar are fenders, wings, ribs, stumps, pushouts, etc. In principle, once the working frame of the theory has been redefined, the theory of advance mining should identically hold also for the phase of retreat. This is essentially the thesis of the discussions in this section. However, there is a great difference between the ground stability objectives of the phases of advancement and retreat. The former phase requires permanent stability, whereas the latter phase requires only temporary stability. Indeed, the design of rooms and pillars should guarantee ground stability during the entire life of the mine, which may last many years. On the contrary, the design of pillar recovery techniques should guarantee ground stability during the period of pillar recovery, which may last only a few days or even hours. Thus, tailoring pillar recovery techniques after the principles of pillar design, very likely leads to a limited percent of recovered coal from a pillar. This is good from a safety standpoint, but causes excessive loss of unmined coal in overdesigned abutments. Too conservative a percentage of pillar recovery is also undesirable as hindering spontaneous roof caving after the completion of the pillar recovery (to the decided extent). Uncaved gob constitutes a very hazardous working environment for the adjacent pillars under extraction. Thus, forced caving of the roof and the unmined portions of the pillar will be necessary in a latter phase, causing time expenses and hampering the efficiency of the entire mining cycle.

Everything considered, we still maintain that the adaptation of the advance mining theory to the phase of retreat constitutes a sound first step toward the development and testing of a pillar recovery theory as such. Of course, proper correction factors should be incorporated in each specific application, taking into consideration the difference of the two phases discussed above.

In the present study, the structural mechanical design of underground constructions (excavations and abutments) has been reviewed thoroughly. Of the numerous topics treated quantitatively in the literature, only a few were selected for presentation in the first part of the section which deals with advance mining. The criterion of selection was the extent of applicability and adaptation of these topics to the realities of the phase of retreat mining presented in the second part of this section.

Finally, a qualitative discussion of a number of case problems encountered in U.S. mines visited during the present study is given in Section V. Unfortunately, quantitative treatment of these problems is impossible, in lieu of appropriate theory. Thus, what constitutes safe mining practice is only the educated guess of the author. Therefore, the offered recommendations should be accepted only as hypotheses requiring testing before use.



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OPTIMAL DESIGN OF STOPES AND ABUTMENTS

Although the present study is concerned with the optimal design of pillar recovery techniques, the design of pillars during the mining advancement cannot be overseen. Indeed, it is impossible to safely recover a pillar which has been designed erroneously in the first place. Thus, in the present section we give an account of the state of the art in designing stopes and abutments during both phases of mining development, i.e., advancement and retreat.

I. PHASE OF MINING ADVANCEMENT

The ground stability is upset in underground mining due to the opening of stopes. The support of the overburden is left to the abutments placed between stopes, such as pillars, ribs, fenders, wings, pushouts, etc. Thus, the optimal design of both stopes and abutments is of primary importance for ground control. The stopes are considered presently to be rectangular openings within the seam of coal and no differentiation is made upon their particular function in the mining system, such as, production stopes, transportation stopes, etc. The abutments are distinguished into barrier pillars and production (or working) pillars. The former are interspaced between ingress or egress entries and must be designed much stronger than the latter, because they must last during the entire life of the mine. Possible collapse would close main entries. The production pillars represent 30-70% of the entire coal seam and they are recovered progressively during the phase of retreat.

1. BARRIER PILLARS

The most crucial parameter in pillar design is the minimum lateral dimension, W. A host of empirical formulas has been developed on the basis of extensive observations. Below we present a few of the most widely used.

(1) Ashley or Mine Inspector's Formula[1]

$$W = 20 + 4T + 0.1\,D \tag{1}$$

where

W = width of minimum lateral dimension
T = seam thickness
D = overburden thickness

Limitations: The formula fails to consider the importance of the strength of the coal constituting the pillar

Numerical Example 1

Find the minimum safe pillar width, W, according to the Ashley formula, for a coal seam of 5 ft. thickness with 1000 ft. overburden thickness.

Solution

Given	Sought
D = 1000 ft.	w - ft.
T = 5 ft.	

$$W = 20 + 4.5 + (0.1 \cdot 1000) = 140 \text{ ft.} \tag{1}$$

(2) Pressure-Arch Formulae

The pressure-arch theory advocates that arches (or domes) of differential pressures are developed around a stope (or excavation) according to the details of Fig. 1.[2]

The minimum width of the basis of the Zone III arch (according to one theory)[3-6] is

$$B = 3\left(\frac{D}{20} + 20\right) \tag{2}$$

where

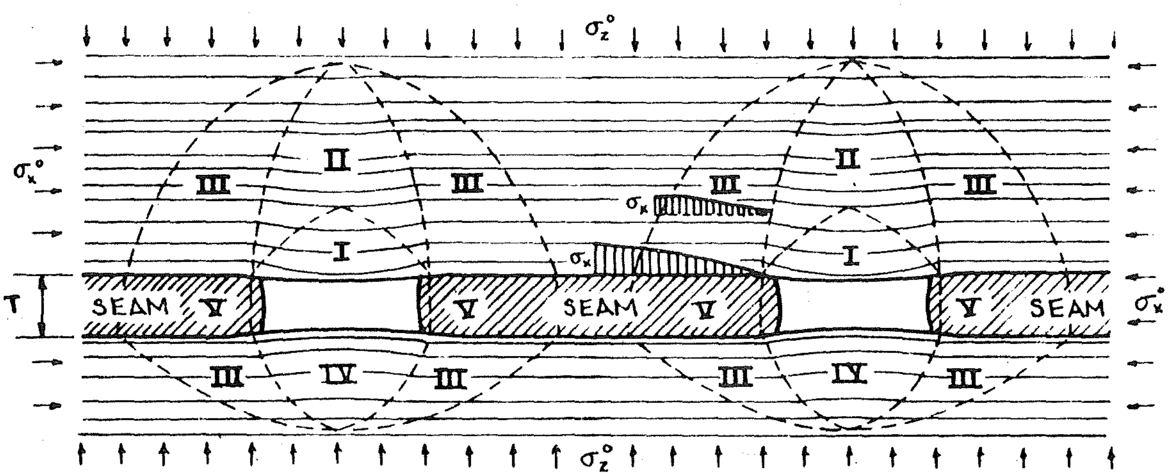
D = overburden thickness

The minimum lateral dimension of the pillar, W, must be

$$W = \frac{B}{2}(1 + K_1) \tag{3}$$

where K_1 is a safety factor

FIG. 3 — DISTURBANCE PRODUCED BY EXCAVATION IN LAYERED STRATA



Zone I — Bed separation occurs due to differential sag and buckling (where horizontal stresses σ_{x_0} are high enough to produce buckling).

Zone II — Layers sag but without bed separation. Gravity acts to develop sag in Zones 1 and 2, and horizontal stresses are reduced compared to σ_{x_0}.

Zone III — Horizontal and vertical pressures build up to their undisturbed values.

Zone IV — Floor heave (uplift) occurs without bed separation — gravity acts to inhibit uplift.

Zone V — Seam expands towards excavation because of release of horizontal stress at the (seam) excavation walls.

Figure 1

$$\frac{1}{2} \leq K_1 \leq \frac{3}{4} \qquad (4)$$

Limitations: The formula ignores both strength of coal and bed thickness

According to another study[7] conducted in South Africa on incohesive rock formations, it was found that the width-height dome relation is

$$B = \left[\frac{8\,\sigma_c D}{\gamma} \left(1 - \frac{h}{D}\right) \log \left(1 - \frac{h}{D}\right) \right]^{1/2} , \qquad (5)$$

where

σ_c = compressive strength of the roof along the plane of Figure 1

γ = Unit weight of the roof rock

h = Height of the dome, at a certain time after the opening of the excavation.

It was also established that

$$h_{max} = 0.63D \text{ for incohesive rock} \qquad (6)$$

Substitution of h_{max} from Eq. 6 in Eq. 5 produces

$$B_{max} = \sqrt{\frac{2.96\,\sigma_c D}{\gamma}} \qquad (7)$$

For cohesive rock, the same study proposes that

$$B = \left[\frac{8\,\sigma_c D}{\gamma} \left(1 - \frac{h}{D}\right) \right]^{1/2} \qquad (8)$$

with

$$h_{max} = 0.5D \text{ for cohesive rock} \qquad (9)$$

and

$$B_{max} = \sqrt{\frac{4\,\sigma_c D}{\gamma}} \qquad (10)$$

This study does not suggest what the safe pillar or room widths are. It simply states that if the span of the excavation (which caused the dome) exceeds B_{max}, the dome fails with the strata precipitation extended all the way to the surface.

A third study[31] theorizes that the dome cross section along the width of the excavation is an ellipse with approximately

$$B_{max} = \frac{1}{2} h_{max} \tag{11}$$

The exact width-height dome relation is

$$B = \frac{4hm}{1-m+\sigma_t/\gamma d} \tag{12}$$

where

σ_t = Tensile strength of the roof rock along the width of the excavation

m = Ratio of horizontal to vertical rock pressure, i.e., according to the notation of Fig. 1

$$m = \frac{\sigma_x^o}{\sigma_z^o} \tag{13}$$

The width-height dome relation in terms of compressive strength (instead of tensile) is

$$B = h(m-1+\sigma_c/\gamma D) \tag{14}$$

This study does not define h_{max}. Thus, B_{max} and the safe pillar and room widths cannot be calculated, unless an appropriate h_{max} has been selected on the basis of another theory.

The numerical value of m depends upon the stress field prevailing around the excavation. Three usual stress fields appear in Fig. 2[8] for m equal to 0, 1/3 and 1. The first case corresponds

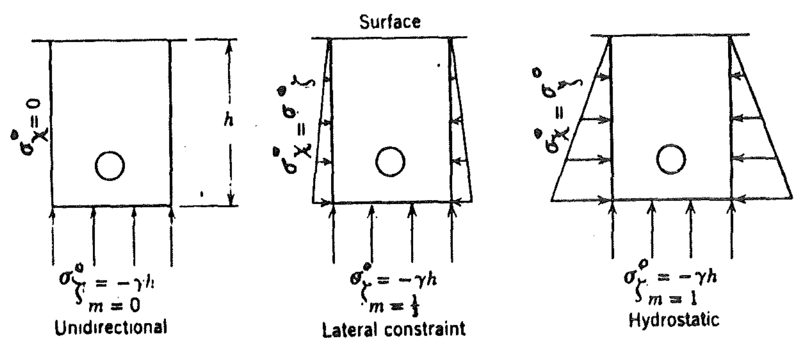


Figure 2

Three stress fields frequently encountered around underground excavations to a unidirectional field which may occur in shallow depths or near vertical free surfaces. The second case occurs over a wide range of depths among different geological counties. The hydrostatic field occurs at great depths and in viscous or plastic rocks. When the rock is completely constrained laterally so that horizontal deformation is not possible, then

$$m = \frac{n}{1 - n} \qquad (15)$$

where

n = Poisson's ratio

<u>Numerical Example 2</u>

Find the minimum safe width of the pillars, W, and the maximum safe span of the rooms, L, within a coal seam under a 1000 ft thick, highly jointed overburden rock, of 0.09 lb/cu.in. mass density, with compressive and tensile strength along the width of the rooms, 300 and 100 psi, respectively; lateral deformation of the seam is not possible and the Poisson's ratio is 1/4.

<u>Solution</u>

<u>Given</u>	<u>Sought</u>
$D = 1000$ ft.	W - ft.
$\sigma_c = 300$ psi	L - ft.
$\sigma_t = 100$ psi	
$\gamma = 0.09$ lb/cu.in.	

According to Eqs. 2 and 3

$$B = 3\left(\frac{1000}{20} + 20\right) = 210 \text{ ft.}$$

$$W = \frac{210}{2}\left(1 + \frac{3}{4}\right) = 184 \text{ ft.}$$

In Eq. 3, K was taken equal to the maximum value it can receive, i.e., 3/4, because the overburden is highly jointed and the maximum necessary safety factor is required.

According to Eq. 7,

$$B_{max} = \left[(2.96 \cdot 300 \cdot 1000)(0.09 \cdot 12)\right]^{1/2} = 906 \text{ ft.}$$

Eq. 7 was preferred over Eq. 10 because highly jointed rock is incohesive. Thus, if the span of the excavation exceeds 906 ft., the overburden will collapse all the way to the surface. However, B_{max} should not be identified with L_{safe}, i.e., the safe room width. The above calculation simply concludes that the room span under no circumstances can exceed 906 ft. It does not suggest, however, that the room span can be even 906 ft.

According to the third pressure-arch theory, h_{max} is not known. However, if we borrow this quantity from Eq. 6 of the second theory, we find

$$h_{max} = 0.63 \cdot 1000 = 630 \text{ ft.}$$

Substitution of the above value in Eq. 11 of the third theory yields

$$B_{max} = \frac{1}{2} \, 630 = 315 \text{ ft.}$$

Since no lateral deformation is possible, Eq. 15 yields,

$$m = \frac{0.25}{1-0.25} = 1/3 \; .$$

Substitution of the above values for h_{max} and m in Eq. 12 of the third theory yields

$$B_{max} = \frac{4 \cdot 630 \cdot (1/3)}{1 - (1/3) + (100 \cdot 12)/(0.09 \cdot 1000)} = 60 \text{ ft.}$$

Substitution of h_{max} and m in Eq. 14 yields

$$B_{max} = 630 \left[\frac{1}{3} - 1 + \frac{300 \cdot 12}{0.09 \cdot 1000}\right] = 24{,}780 \text{ ft.}$$

From the two last pressure-arch theories it was found that the smallest expected B_{max} is 65 ft. Hence, the safe room span is smaller than 65 ft.

Results

$W = 184$ ft

$L_{safe} = 65$ ft.

(3) Holland Formulae

Holland has proposed two alternative formulae:[9]

$$W = 15 \cdot T \qquad (16)$$

where

T = Seam thickness

and

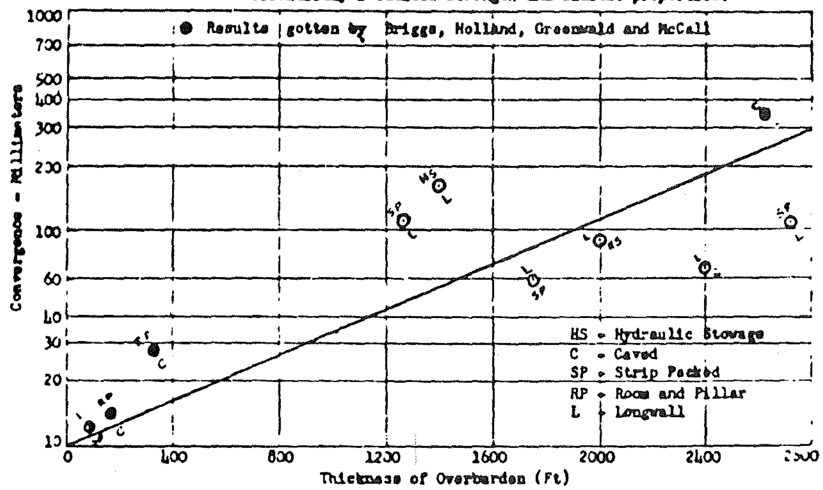
$$W = 5 \left(\frac{\log (2W_2)}{C \cdot \log e}\right) \qquad (17)$$

where

W_2 = convergence in mm read from Fig. 3[10]

e = basis of natural logarithms (=2.7T)

Observed value of W_2 for a seam 7 feet thick. The values represented by the solid dots are for coals that have essentially identical strength and elastic properties.



–Observed value of W_2 for coal seams 7 ft thick and having a crushing strength in the 3-in. cube of 3,000 psi ± 10%.[110]

Figure 3

$$C = \begin{cases} 0.09\text{: free caving} \\ 0.08\text{: building strip packs} \\ 0.07\text{: hydraulic stowage} \\ 0.085\text{: partial pillar recovery with ultimate pillar collapse} \end{cases}$$

Unfortunately, W_2 is given in Fig. 3 only for coal beds of 7 foot thickness. Furthermore, even for this specific case, the two formulae give highly different numerical results.

Numerical Example 3

Find the minimum safe pillar width, W, in a coal seam 7 ft. thick according to the Holland Formulae, for a mining system allowing partial recovery of pillars with ultimate pillar collapse. The overburden thickness is 900 ft.

Solution

Given	Sought
$T = 7$ ft	W
$C = 0.085$	

According to Eq. 16

$$W = 15(7) = 105 \text{ ft.}$$

From Fig. 3 it is found that the convergence, W_2, for 900 ft. overburden thickness is approximately 30. Thus, according to Eq. 17

$$W = 5 \frac{\log 60}{0.085 \cdot \log 2.7} = 242 \text{ ft.}$$

Since we have no criteria to choose between the two formulae, the answer is the one giving the safer pillar width. Thus:

Results

$$W = 242 \text{ ft.}$$

(4) Morrison, Corlett & Rice Formula

The minimum lateral width is

$$W = D/8 \qquad (18)$$

where

D = overburden thickness

and

$$D \leq 4{,}000 \qquad (19)$$

Limitations: The formula ignores both bed thickness and strength of coal

2. PRODUCTION PILLARS

The state of the art primarily consists of empirical formulas. However, these formulas are more sophisticated than the previous ones for barrier pillars in the sense that they do consider the strength of coal. Some of the most successful formulas are presented below. All these formulas are solved with respect to pillar strength, S. However, the width of minimum lateral dimension, W, can be obtained directly by simple rearrangement.

(1) Holland - Gaddy Formula [6,12-16]

$$S = \frac{K_1\sqrt{W}}{T} \cdot C \qquad (20)$$

where

S = pillar strength in psi
W = least lateral dimension
T = seam thickness
K_1 = safety factor

$$1.7 \leq K_1 \leq 2 \qquad (21)$$

and

$$C_1 = S_p \sqrt{d} \tag{22}$$

where

S_p = compressive strength in psi of a cube of the material
d = the edge of the tested cube
C_1 = values for most rocks are tabulated

Field of Application

The following conditions must be met for the valid application of the formula:

1. $T < W$ (23)

2. $\frac{W}{T} < 12$ (24)

When $W/T \geq 12$, the pillar is considered overdesigned, i.e., it is capable of withstanding much more load than will ever be needed.

3. The formula holds even for highly jointed rocks.

Numerical Example 4

Find the necessary width of a pillar within a coal seam 5 ft thick, in order to obtain pillar compressive strength 600 psi, when the compressive strength of a coal cube with 2 in. edge is 1200 psi.

Solution

Given

T = 5 ft
S = 600 psi
Sp = 1200 psi
D = 2 in.

Sought

W

Eq. 22 yields

$$K_1 = 1200\sqrt{2} = 1697$$

We select C=2 for maximum safety. Thus, Eq. 20 gives

$$W = \left(\frac{600 \cdot 5 \cdot 12}{1697.2}\right)^2 \cong 112 \text{ in.} \cong 10 \text{ ft.}$$

This pillar width is sufficient to achieve the desired pillar strength of 600 psi. It is not the minimum safe pillar width, unless it has been established that 600 psi is the minimum necessary pillar strength to keep the pillar in tact for as long as it is necessary.

Results

$$W = 10 \text{ ft.}$$

(2) Morrison, Corlett & Rice Formula (17)

$$S^2 = K_2\sqrt{\frac{W}{T}}\, C_2 \qquad (25)$$

where

C_2= crushing strength in psi

K_2= safety factor equal to 4 or 5

Field of Application

The following conditions must hold if the formula is to be valid.

1. $\frac{W}{T} > 1$ (26)

2. $\frac{W}{H} < 8$ (27)

where

H = height of pillar. Usually H = T

3. The formula holds even for highly jointed rocks

(3) Obert Formula Modified by Holland[10]

$$S = C_3(0.778 + 0.222 \frac{W}{H}) \cdot K_3 \qquad (28)$$

where

C_3= average strength in psi of cubical test specimens

K_3= 4.5 (safety factor)

Field of Application

The formula holds only when:

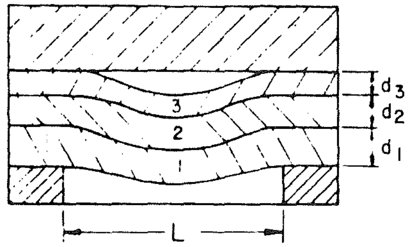
1. $0.25 \leq \frac{W}{H} \leq 4.0 \qquad (29)$

2. It is not valid for highly jointed rocks

3. CALCULATION OF THE MAXIMUM SAFE SPAN OF ROOMS

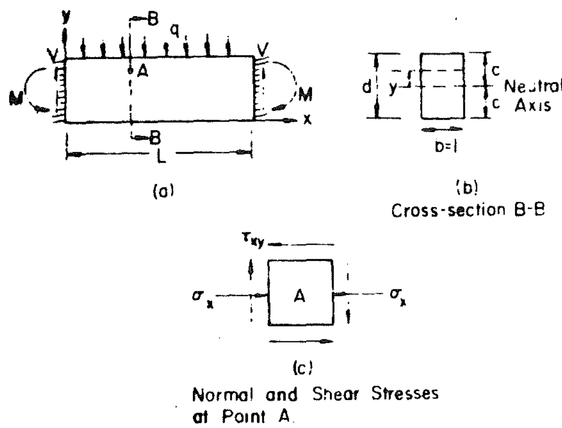
Most of the stopes (production and transportation) are rectangular excavations of height equal to the bed thickness. The length and width vary, but the minimum lateral dimension of them, as in the case of the pillars, is the crucial one for ground stability. The maximum allowable width of a stope can be determined from beam theory, provided that the stope is not intersected by another stope. In the latter case, plate theory is required.

The roof of coal seams usually consists of laminated formations fairly smooth and flat, with weak bonding between successive beds. Thus, slabs or plates are detached from the rock above, as in Fig. 4.[18] A slab is assumed to behave as a series of parallel beams of unit width, with the ends of the beam fixed on each side of the room, on a pillar, and with spans equal to the width of the room.



—Immediate roof with multiple beds.

Figure 4



—Stresses at a point in a beam.

Figure 5

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The normal axial stress, σ_x, (known also as a fiber stress) at any point A (Fig. 5) is:

$$\sigma_x = \frac{12\, M_x\, y}{d^3} \tag{30}$$

where

y = vertical distance of point A from the neutral axis of the beam (disecting horizontally the cross-section)

M_x = moment at the distance x of the point A from the origin of coordinates (left end of the beam) given by

$$M_x = \frac{q}{12}\,(6Lx - 6x^2 - L^2) \tag{31}$$

The shear stress, τ_{xy}, of the same point A, Fig. 5), is

$$\tau_{xy} = \frac{3V_x}{2}\left(\frac{d^2 - 4y^2}{d^3}\right) \tag{32}$$

where

V_x = shear on the cross-section through A given by

$$V_x = \frac{qL}{2}\left(1 - \frac{2x}{L}\right) \tag{33}$$

In Eqs. 31 and 33 , q stands for the uniform load on the beam per unit length, per unit width. If the detached immediate roof consists of a single layer, q is calculated from the mass density of this layer. If more than one layer constitutes the detached immediate roof, (Fig 4), the uniform load on the lower layer of thickness, d_1, consists of the weight of the lower layer itself, plus the weight of the remaining detached layers resting on the lower layer. The total load on the lower layer, q, is

$$q = \frac{E_1 d_1^3\,(q_1 d_1 + q_2 d_2 + - - + q_n d_n)}{E_1 d_1^3 + E_2 d_2^3 + - - - E_n d_n^3} \tag{34}$$

where

q_n = the uniform load per unit length, per unit width, on the layer (or beam) n, due to its own weight

d_n = thickness (or vertical width) of layer n

E_n = Young modules of layer n.

The maximum tensile and compressive stresses on the cross section of the examined point A, (Fig.5), occur at the top and bottom of the beam, at a distance $\pm$ C = $\pm$ d/2 from the neutral axis, respectively. Substitution of y by $\pm$ d/2 in Eq.30 yields for the maximum stresses on a given cross section at distance x from the origin,

$$\sigma_{\binom{tens}{compr}} = \pm \frac{6M_x}{d^2} \tag{35}$$

The maximum vertical and horizontal shear stresses occur at the neutral axis. Substitution of Y by O in Eq. 32 yields

$$\tau_{xy} = \frac{3V_x}{2d} \tag{36}$$

From Eq. 31 becomes apparent that the maximum M_x occurs at x = O and x = L, i.e. at the ends of the beam, where

$$M_{max} = -\frac{qL^2}{12} \tag{37}$$

From Eq. 33 becomes apparent that the maximum V_x occurs also at x = O and x = L, where

$$V_{max} = \pm \frac{qL}{2} \tag{38}$$

From Eq. 35 it is obvious that the maximum tensile and compressive stresses throughout the beam occur at the top and bottom of the particular cross-sections at which the maximum M_x occurs.

Thus, according to the foregoing, the maximum tensile and compressive stresses occur at the ends of the beam and substitution of M_{max} from Eq.37 in Eq.35 yields

$$\sigma_{\binom{tens}{compr}} \max = \mp \frac{qL^2}{2d^2} \qquad (39)$$

It is well known that the strength of any rock in tension is much weaker than in compression. The maximum safe tensile stress a rock can withstand is

$$\sigma_{(tens)} \text{ safe} = \frac{R_o}{K_4} \qquad (40)$$

where

R_o = modulus of rupture of the rock
K_4 = safety factor

It is obvious that for roof stability

$$\sigma_{(tens)} \text{safe} = \frac{R_o}{K_4} \geq \sigma_{\binom{tens}{compr}} \max = \mp \frac{qL^2}{2\ d^2} \qquad (41)$$

Solution of Eq. 41 with respect to L yields

$$L \leq d \sqrt{\frac{2R_o}{q\ \ K_4}} \qquad (42)$$

From Eq.36 follows directly that the maximum J_{xy} throughout the beam occurs at the neutral axis of the particular cross-sections at which the maximum V_x occurs. Thus, according to the foregoing, the maximum shear stress, J_{xy}, occurs at the ends of the beam and substitution of V_{max} from Eq.38 in Eq.36 yields

$$J_{xy,\ \max} = \pm \frac{3qL}{4d} \qquad (43)$$

The maximum safe shear stress a rock can withstand is,

$$J_{xy,safe} = \frac{J}{K_5} \qquad (44)$$

where

J = shear strength of the rock

K_5 = Safety factor

It is obvious that for roof stability

$$J_{xy,safe} = \frac{J}{K_2} \geq J_{xy,max} = \frac{3qL}{4d} \qquad (45)$$

Solution of Eq. 45 with respect to L yields

$$L \leq \frac{4dJ}{3qK_2} \qquad (46)$$

The safe room span for both, shear stress, and tensile-compressive stress, is the smaller of the two L's calculated through Eqs. 42 and 46 .

It is important to draw attention to the fact that q is the weight of a column of unit length and width, but height equal to the layer thickness, d. If instead of q the unit weight or density of the rock, γ, is to be used, (lb/cu.ft.), then

$$q = d\gamma \qquad (47)$$

and Eqs. 42 and 46 may be rewritten

$$L \leq \sqrt{\frac{2dR_o}{\gamma K_4}} \qquad (48)$$

and

$$L \leq \frac{4J}{3\gamma K_5} \qquad (49)$$

Since for the majority of rocks the tensile strength is weaker than the shear strength, Eq. 48 furnishes the maximum safe room span in most cases. The maximum safe room span has been calculated on the basis of Eq. 48 for a number of critical stresses, R_o/K_4, over a wide range of layer thickness, d, for each R_o/K_4. The results are tabulated in Fig. 6.

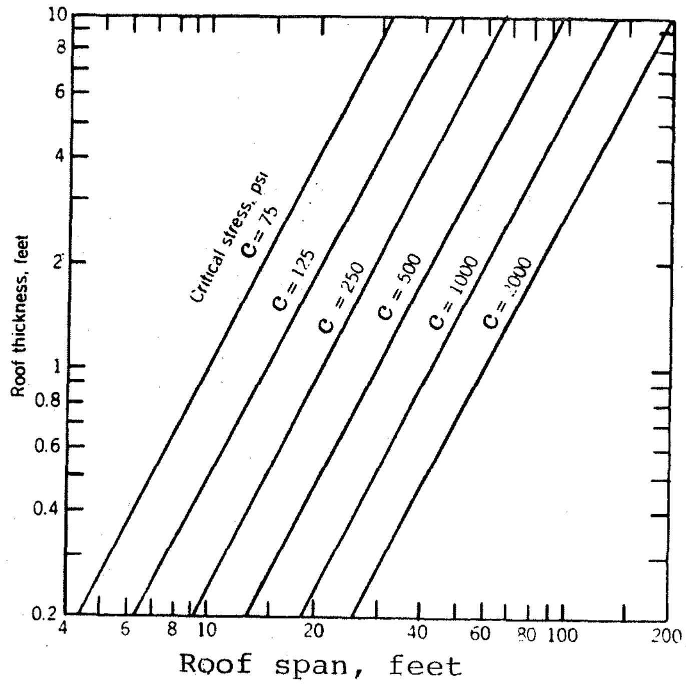


Figure 6

The maximum safe roof span as a function of the roof thickness and the critical rock stress

$$c = \frac{R_O}{K_4} \quad \text{(psi)}$$

Numerical Example 5

Find the maximum safe room span in a coal seam of critical tensile stress 250 psi, with an immediate roof of 5 ft. thickness and 0.085 lb/cu.in. mass density. The shear strength of the immediate roof is 750 psi and the safety factor to shear stress is 4.

Solution

Given		Sought
$\frac{Ro}{K_1}$ =	250 psi	L-ft.
d =	5 ft	
γ =	0.085 lb/cu.in.	
J =	750 psi	
K_5 =	4	

According to Eq. 48

$$L \leq \left[\frac{2 \cdot 5 \cdot 250 \cdot 12}{0.085}\right]^{1/2} = 594 \text{ in.} = 49 \text{ ft.}$$

$L \leq$ 48 ft.

We see that the agreement is very good although Fig. 6 does not provide for the effect of γ. This omission does not affect substantially the effectiveness of Fig. 6, because all coal overburden rock formations have a γ value between 0.08 and 0.10 and Fig. 6 has been drawn on this assumption.

According to Eq. 49

$$L \leq \frac{4 \cdot 750}{3 \cdot 0.085 \cdot 4} = 2941 \text{ in.} = 245 \text{ ft.}$$

We see that the design with the shear strength for criterion allows too large a room span for the tensile strength. Thus:

Results

$$L \leq 48 \text{ ft.}$$

4. CALCULATION OF THE MAXIMUM SAFE SPAN OF STOPES UNDER PNEUMATIC OR HYDRAULIC PRESSURE

Frequently a coal seam is extended under an underground or surface water horizon or a pressurized gas reservoir. If the unit weight of the overburden is γ and the pneumatic or hydraulic pressure load per unit width and per unit length is p, the maximum tensile stress is

$$\sigma_{max} = \frac{L^2}{2d} + \frac{PL^2}{2d^2} \tag{50}$$

Following a derivation similar to the one of the previous section, the maximum safe room span, L, is found to be,

$$L = \sqrt{\frac{2R_o d}{(\gamma + P/d)\ K_1}} \tag{51}$$

It may be seen from numerical examples that even very small pneumatic or hydraulic pressures can double or triple the maximum tensile stress and, therefore, cut down the maximum safe room span to a small fraction of what it could be without the pneumatic or hydraulic pressure. Thus, relief of the coal seam from the additional load by gas depressurization or water drainage, is highly profitable, and it may make the difference between rendering the coal mining economically permissible or prohibitive.

Numerical Example 6

Find the maximum safe room span in the coal seam of Example 5, with the addition of a hydraulic pressure of 2 psi on the immediate roof due to the existance of a water horizon.

Solution

According to Eq. 51

$$L = \left[\frac{2 \cdot 250 \cdot 5 \cdot 12}{0.085 + 2/5}\right]^{1/2} = 247 \text{ in.} = 21 \text{ ft.}$$

We see that hydraulic pressure of only 2 psi cuts the maximum safe span from 49 ft. to 21 ft.

Results

L = 21 ft.

5. RELATION OF PILLAR WIDTH TO ROOM SPAN

In designing a room and pillar mine, the maximum safe span of rooms and haulageways is to be determined first, on the grounds presented in the previous sections. The next critical dimension to be determined is the minimum safe width of the production pillars. This can be done in terms of structural mechanics, on the basis of the following assumptions:

1. The load on the pillar is considered to be constant and independent of the deformation of the pillar, roof, and floor, during the lifetime of the pillar.

2. It is assumed that the pillar will collapse once a critical average compression is reached.

In reality the load on the pillar decreases rapidly and proportionally to the convergency (strata deformation). Furthermore, in reality, the pillar can withstand the critical compression for quite a long time before it collapses. Thus, both the above assumptions lead to overdesign of the width of the pillar. This reduces the productivity during the phase of advancement, but assures safe roof support. If phase of retreat does not exist (the pillars are not recovered) the problem of pillar overdesign causes loss of large amounts of coal and must be avoided. However, if the phase of retreat is to take place with recovery of the largest portion of the production pillars, overdesign of pillars does not cause any coal losses. Of course it causes cash flow problems for the operator because it ties up products which cannot be utilized before the phase of retreat. Everything being considered, these assumptions have broadly been employed in design of room and pillar layouts and the following relation of room span-pillar width can be obtained.

Let us consider the situation of Fig. 7.[19] The roofs of the rooms JK and LM are beams with both their ends supported, thus each pillar at their sides carries half of their weight. If the average density of the overburden is assumed to be approximately 160 lb/cu.ft. or 1.1 psi per ft. depth, the average pressure, S, at depth D is

$$S = 1.1D \text{ psi} \tag{52}$$

If the length of each room is W_{max}, the total load carried by the middle pillar, F, is the weight of the column extended from the roof of the rooms to the surface and spanning the pillar to the midpoint of each roof,

$$F = 1.1D\ (W\ \ + L)\ W_{max} \tag{53}$$

The middle pillar supports the above weight through a surface,

$$A = W_{max}\ .\ W \tag{54}$$

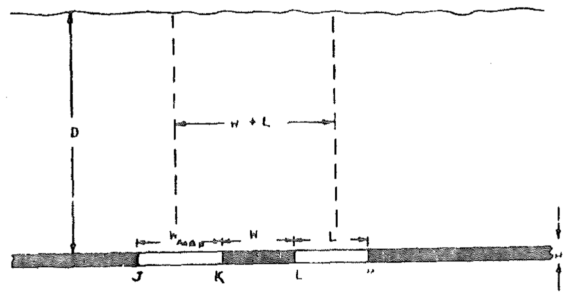


Figure 7

Elevation of room and pillar layout

Thus, on the basis of the first assumption, the average pillar pressure is

$$S = \frac{F}{A} = 1.1D \left(\frac{W + L}{L}\right) \qquad (55)$$

The compressive strength, So (psi), of the coal constituting the pillars is determined laboratorily by subjecting a cylindrical coal specimen (with length to diameter ratio equal to 2 and diameter equal to 2 in) to compression until disintegration occurs. If the diameter is t and the compressive force achieving disintegration is S_t, then,

$$S_o = S_t / \left(\frac{\pi}{4} t^2\right) \qquad (56)$$

Experiments have proven that the actual strength of a full pillar, S_p, is a fraction of S_o. Thus,

$$S_p = S_o / K_6 \qquad (57)$$

where

K_6 = safety factor, empirically found to be

$$4 \leq K_6 \leq 6 \qquad (58)$$

On the basis of the second assumption, the average pressure on the pillar, S, must be smaller than its compressive strength, Sp,

$$S = 1.1D \left(\frac{L + W}{W}\right) < S_p = \frac{S_o}{K_6} \qquad (59)$$

or

$$L > \frac{W}{\dfrac{S_o}{1.1K_6 D} - 1} \qquad (60)$$

Numerical Example 7

Find the minimum safe pillar width in a coal seam of 10,000 psi compressive strength, with 1000 ft. overburden thickness and room span 16 ft, when the required safety coefficient is 4.

Solution

Given	Sought
$D = 1000$ ft.	L - ft.
$S_o = 10{,}000$ psi	
$K_6 = 4$	
$W = 16$ ft.	

According to Eq. 60

$$L \geq \frac{1.1 \cdot 4 \cdot 1000 \cdot 16}{10{,}000 - 1.1 \cdot 4 \cdot 1000} = 13 \text{ ft.}$$

6. SAFETY FACTOR OF A FRAGMENTED ROOF

Let us consider the fragmented roof of Fig. 8.[18] This roof may fail due to three reasons:

a. The two pieces may slide down if sufficient horizontal compressive force, G, does not thrust them securely against each other.

b. The rock may be fragmented further along the crack, at areas of high compressive stress. The newly created small fragments may rotate or shift position, creating open space between the two halves. From this point, the situation is reduced to the previous one.

c. Elastic or plastic buckling allows rotation or shifting of blocks without actual further fragmentation occuring. Again, the different parts of the roof may not find sufficient support from the rest. Thus, cracking and downward sliding will occur simultaneously, after a certain period of inbalance.

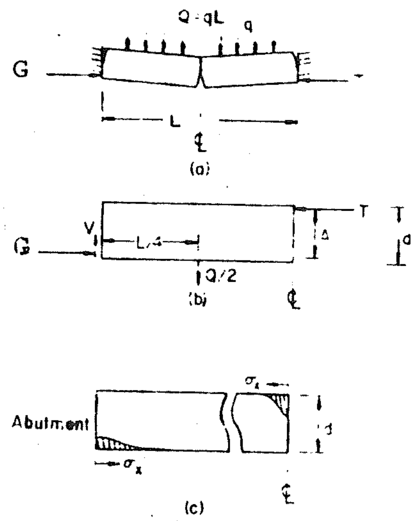


Figure 8

a. Forces acting on a cracked beam
b. Left half of the beam
c. Stress distribution on the left half

If it is assumed that the crack divides the beam into two equal parts, it becomes obvious from Fig. 8 that the left half will not fail only when the moments acting on this half counter-balance each other, i.e.,

$$\frac{Q}{2} \cdot \frac{L}{4} = G \cdot A \tag{61}$$

where

Q = Lateral Load

A = Moment arm of the axial force, G, to be determined

For the special case of a beam supported by two abutments which do not move outward, but do not apply any prior to cracking axial force, T, it has been found that

$$A_u = 0.91d - \frac{0.44d^2}{L} \tag{62}$$

where

L = beam length (Room span)

d = beam thickness

The subscript u stands for axially unloaded beam.

For this particular case it is obvious that the total axial force, G, is a result of the cracking itself and it can be calculated in terms of A from Eq.61, i.e.,

$$G_u = \frac{LQ}{8\ A_u} \tag{63}$$

where A_u is calculated through Eq. 62.

For the more complex case of a horizontal stress field subjecting the beam to an axial compressive force, P_o, the total axial thrust after cracking is

$$G_L = \frac{P_o d}{4A_u} + \left[\frac{P_o^2\ (8A_u^2 - 4A_u d + d^2)}{8A_u^2} + \frac{QL\ (QL-4P_o\ d)}{64A_u^2}\right]^{1/2} \tag{64}$$

where

P_o = axial compressive force acting on the beam before cracking

The subscript 1 stands for axially loaded beam prior to cracking.

Let us symbolize the axial and lateral loads acting on a beam by P and Q, respectively.

$$P = \begin{cases} G_u : \text{(Eq.63) in absence of axial stress field prior to cracking} \\ G_1 : \text{(Eq.64) in presence of axial stress field prior to cracking} \end{cases}$$

$$Q = L \cdot q \tag{65}$$

where

L = beam length - in

q = uniform transverse load on the beam per unit width, per unit length - psi

These loads can be converted into forces acting parallel and normally to the direction of a crack forming any angle with the axis of the beam (Fig. 9).[20]

The vertical load on each abutment at the ends of the beam is

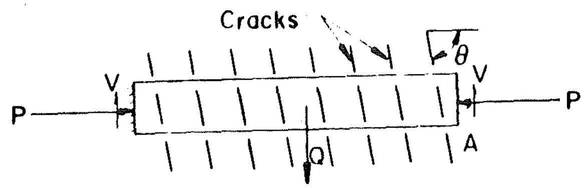
$$V = \frac{Q}{2} \tag{66}$$

The driving force for failure by sliding, S, is the algebraic sum of the projections of P and V on the direction of the crack,

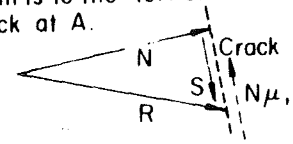
$$S = P\cos\theta + V\sin\theta \tag{67}$$

The algebraic sum of the projections of P and V on the normal direction to the crack is

$$N = P\sin\theta - V\cos\theta \tag{68}$$



A Forces on a cracked beam



$V = Q/2$

$S = P\cos\theta + V\sin\theta$
$N = P\sin\theta - V\cos\theta$

B Forces acting on the surface of a crack

Driving force = S
Resisting force = $N\mu$

Figure 9

Conversion of axial and lateral loads acting on a beam into parallel and normal forces to the direction of a crack

The force resisting failure by sliding is

$$F = N\mu \tag{69}$$

where

μ = Coefficient of friction along the surface of the crack

The safety factor of the fractured beam to failure by sliding is

$$K_7 \quad \frac{\text{Absolute value of F, resisting force}}{\text{Absolute value of S, driving force}} \tag{70}$$

The above discussion pertains to failure of a fragmented roof due to the first of the three reasons mentioned at the beginning of the present section. Failure due to the second reason, namely, further fragmentation or crushing, is the result of development of excessive compressive stresses. If triangular stress distribution is assumed instead of the actual sigmoidal, (Fig. 8.C), the maximum compressive stress in absence of horizontal load, P, prior to cracking, is

$$\sigma_{max} = \frac{2\ G_u}{d - A_u} \tag{71}$$

In case of existance of horizontal load, P, prior to cracking,

$$\sigma_{max} = \frac{2\ T_1}{d - A_1} \tag{72}$$

where

$$A_1 = \frac{LQ}{8}T_1 \tag{73}$$

The safety factor of the fractured beam to failure by further cracking is

$$K_8 = \frac{S_{comp}}{6\ \sigma_{max}} \tag{74}$$

where

S_{comp} = compressive strength of beam in transverse load-psi

According to Eq. 67

$$S = 1,156.57 \cdot 0.77 + 1,451.50 \cdot 0.64 = 1,819.52 \text{ lb/in.}$$

According to Eq. 68

$$N = 1,156.57 \cdot 0.64 - 1,451.50 \cdot 0.77 = -377.46 \text{ lb/in.}$$

According to Eq. 69

$$F = -377.46 \cdot 0.6 = -226.47 \text{ lb/in}$$

According to Eq. 70

$$K_7 = \frac{226.47}{1,819.52} = 0.12$$

According to Eq. 71

$$\sigma_{max} = \frac{2 \cdot 1,156.57}{(7 - 5.02) \cdot 12} = 97.35 \text{ psi}$$

According to Eq. 74

$$K_8 = \frac{1500}{6 \cdot 97.35} = 2.57$$

We see that $K_7 < K_8$, thus:

$$K = K_7 = 0.12$$

It is obvious that the examined excavation is structurally unstable and the roof will fail by sliding without artificial support.

<u>Case b. (S_h = 12 psi)</u>

$$P = G_1 = 1,390.36 \text{ lb/in.}$$

According to Eq. 67

$$S = 1,390.36 \cdot 0.77 + 1,451.50 \cdot 0.64 = 1,999.54 \text{ lb/in.}$$

The coefficient 6 in the denominator indicates that Eq. 71 and 72 have not been tested and verified sufficiently. Thus, their results may underestimate the actual maximum compressive stress by a much as 6 times or even more.

Analytical treatment of the failure of a fractured beam due to the third reason, i.e., buckling, has not yet been established. The overall safety factor of a fractured beam is the smaller of the two numerical values obtained through Eqs. 70 and 74 .

Numerical Example 8

Find the safety factor of a fragmented roof with cracks forming 40 degree angle with the horizontal. The compressive strength of the roof rock is 1,500 psi, the room span is 16 ft., the immediate roof thickness is 7 ft., the unit weight of the roof rock is 0.18 lb/cu.in., the coefficient of friction along the surface of the cracks is 0.6, no axial compressive stress was acting on the roof prior to the formation of cracks. What would the safety factor be if an axial compressive stress of 12 psi was acting on the roof prior to cracking?

Solution

Given

θ = 40°

S_{comp} = 1,500 psi

L = 16 ft.

d = 7 ft.

γ = 0.18 lb/cu.in.

μ = 0.6

$$S_h = \begin{cases} a: = 0 \\ b: = 12 \text{ psi} \end{cases}$$

Sought

K (for S_h = 0)

K (for S_h = 12 psi)

According to Eq. 62

$$A_u = 0.91 \cdot 7 - \frac{0.44 \cdot 7^2}{16} = 5.02 \text{ ft.}$$

According to Eq. 47

$$q = 0.18 \cdot 7 \cdot 12 = 15.12 \text{ psi}$$

According to Eq. 65

$$Q = 16 \cdot 15.12 \cdot 12 = 2,903 \text{ lb/in.}$$

According to Eq. 63

$$G_u = \frac{16 \cdot 2,903}{8 \cdot 5.02} = 1,156.57 \text{ lb/in.}$$

The axial compressive force, P_o, can be calculated from the axial compressive stress, S_h, and the thickness of the roof, d, according to

$$P_o = S_h \cdot d \tag{75}$$

Thus, in the present numerical example,

$$P_o = 12 \cdot 7 \cdot 12 = 1,008 \text{ lb/in.}$$

According to Eq. 64

$$T_L = \frac{1,008 \cdot 7}{4 \cdot 5.02} + \left[\frac{1,008^2 (8 \cdot 5.02^2 - 4 \cdot 5.02 \cdot 7 + 7^2}{8 \cdot 5.02^2} + \frac{2,903 \cdot 16 \cdot 12 (2,903 \cdot 16 \cdot 12 - 4 \cdot 1,009 \cdot 7 \cdot 12)}{64 \cdot 5.02^2 \cdot 12^2}\right]^{1/2} =$$

$$= 1,390.36 \text{ lb/in.}$$

According to Eq. 66

$$V = 2,903/2 = 1,451.50 \text{ lb/in.}$$

Case a. (S_h = o)

$$P = G_u = 1,156.57 \text{ lb/in.}$$

According to Eq. 67

$S = 1{,}156.57 \cdot 0.77 + 1{,}451.50 \cdot 0.64 = 1{,}819.52$ lb/in.

According to Eq. 68

$N = 1{,}156.57 \cdot 0.64 - 1{,}451.50 \cdot 0.77 = -377.46$ lb/in.

According to Eq. 69

$F = -377.46 \cdot 0.6 = -226.47$ lb/in

According to Eq. 70

$$K_7 = \frac{226.47}{1{,}819.52} = 0.12$$

According to Eq. 71

$$\sigma_{max} = \frac{2 \cdot 1{,}156.57}{(7 - 5.02) \cdot 12} = 97.35 \text{ psi}$$

According to Eq. 74

$$K_8 = \frac{1500}{6 \cdot 97.35} = 2.57$$

We see that $K_7 < K_8$, thus:

$K = K_7 = 0.12$

It is obvious that the examined excavation is structurally unstable and the roof will fail by sliding without artificial support.

<u>Case b. (S_h = 12 psi)</u>

$P = G_1 = 1{,}390.36$ lb/in.

According to Eq. 67

$S = 1{,}390.36 \cdot 0.77 + 1{,}451.50 \cdot 0.64 = 1{,}999.54$ lb/in.

According to Eq. 68

$$N = (1,390.36 \cdot 0.64) - (1,451.50 \cdot 0.77) = -227.83 \text{ lb/in.}$$

According to Eq. 69

$$F = -227.83 \cdot 0.6 = -136.70 \text{ lb/in.}$$

According to Eq. 70

$$K_7 = \frac{136.70}{1,999.54} = 0.07$$

According to Eq. 73

$$A_1 = \frac{16 \cdot 12 \cdot 2,903}{8 \cdot 1,390.36} = 50.11 \text{ in.}$$

According to Eq. 72

$$\sigma_{max} = \frac{2 \cdot 1,390.36}{7.12 - 50.11} = 82.05 \text{ psi}$$

According to Eq. 74

$$K_8 = \frac{1500}{6 \cdot 82.05} = 3.05$$

We see that $K_s < K_c$, thus:

$$K = K_7 = 0.07$$

The structure is less stable than in the case of absence of horizontal stress prior to cracking.

7. CALCULATION OF ROOF TENSILE AND COMPRESSIVE STRESSES AT INTERSECTIONS

Plate theory implemented with finite difference method is the usual analytic approach to the roof stability at intersections of underground openings. On the basis of the analytic results of such an approach[21], the following graphic method for the calculation of tensile and compressive stresses has been postulated.

(1) The maximum negative moment at the abutments (the ends of unit beam) is calculated through Eq.37 for the existing room span, L, and the uniform load on the beam, per unit width, per unit length, q.

(2) The maximum positive moment at the center of the uniformly loaded unit beam is calculated through the equation,

$$M_{(x=L/2)} = \frac{1}{24} qL^2 \qquad (76)$$

which is derived from Eq.31 for $x = \frac{L}{2}$

(3) A pillar factor, C, is defined and calculated according to the expression

$$C = \frac{3}{4} (1 - n^2) \left(\frac{E_P}{E_N}\right) \left(\frac{A^4}{Hd^3}\right) \qquad (77)$$

where

n = Poisson ratio of the roof plate in the horizontal direction

E_P = Modulus of elasticity of the pillar in the vertical direction

E_N = Modulus of elasticity of the roof in the horizontal direction

H = Height of the pillar

d = Roof plate thickness

A = Center-to-center distance between pillars, (Fig 10) (18)

(4) From Fig.11 (18) the multiplication factor, I, can be calculated for the pillar factor, C, calculated above. Two values of I must be calculated, one for the maximum positive moment at the center of the unit beam and one for the maximum negative moment at the ends of the beam (abutments). If the pillar pattern is regular, the curves R should be used and if it is staggered, the curves S provide the I's. The sign (+) stands for the positive moment and the sign (-) for the negative moment.

(5) The ratio of the room span L (width of main haulageways) to the center-to-center distance between pillars, A, must be calculated. From Fig.12 (18) a second multiplication factor, II, can be calculated for the existing L/A.

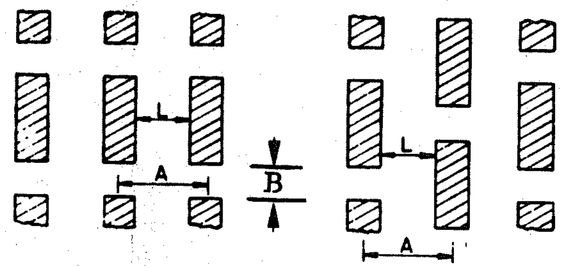


Figure 10

PILLAR PATTERNS

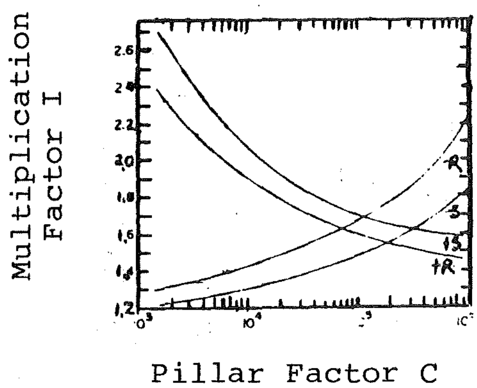


Figure 11

Calculation of Multiplication Factor I
as a Function of the Pillar Factor C,
and the extreme Moments

+R : Maximum positive moment at the center of a unit beam for regular pillar pattern

-R : Maximum negative moment at the abutments of a unit beam for regular pillar pattern

+S : Maximum positive moment for staggered pillar pattern

-S : Maximum negative moment for staggered pillar pattern

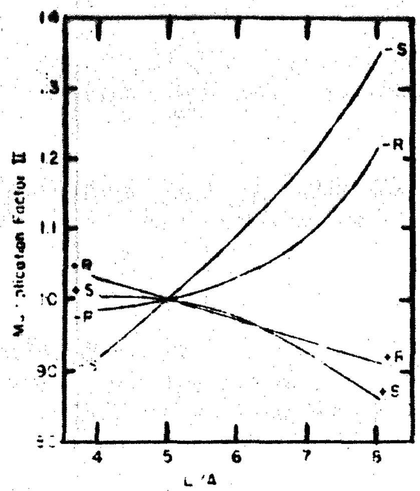


Figure 12

Calculation of Multiplication Factor II as a Function of Room Width, L, to Center-to-Center Distance, A, Ratio, and the extreme Moments (+R, -R, +S, -S, same as in caption of Fig.11)

(6) The ratio of the crosscut width, B, to the main room width, L, is calculated and the multiplication factor III is determined from Fig. 13 .[18]

(7) The maximum negative moment (as calculated in Step 1) is multiplied with the multiplication factors, I, II, and III (as calculated in steps 4,5, and 6 from -R or -S curves).

(8) The maximum positive moment (as calculated in Step 2) is multiplied with the multiplication factors I, II, and III (as calculated in steps 4,5, and 6 from +R or +S curves).

(9) The maximum tensile and compressive stresses are calculated from Eq.35 where M_x is the greater in absolute value of the two extreme moments calculated in Steps 7 and 8. For a regular pillar pattern the maximum negative moment (calculated in Step 7) has always greater absolute value than the positive maximum moment (calculated in Step 8).

(10) The ratio of the compressive strength of the roof rock to the compressive stress (as calculated in Step 9) is determined.

(11) The ratio of the tensile strength of the roof rock to the tensile stress (as calculated in Step 9) is determined.

(12) The minimum of the two ratios calculated in steps 10 and 11 is the safety factor of the roof at the intersection.

The above twelve-step calculation allows determination of the safety factor of the roof at the intersection, for the given room span, L, crosscut span, B, the center-to-center distance, A, and the pattern of the pillars (regular or staggered). Thus, two alternative room-and-pillar designs under consideration can easily be comparatively evaluated in terms of safety factors at the intersections. Obviously, the pattern with safety factors between 4 and 6 is the better one, because it does not overdesign or underdesign the intersections.

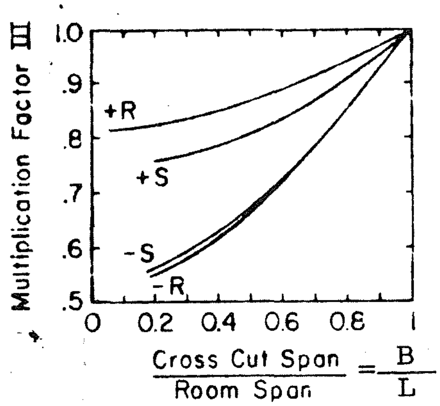


Figure 13

Calculation of the Multiplication Factor III as a function of B/L and the Extreme Moments (+R, +S, -R, -S, as in caption of Fig. 11)

Another very valuable utilization of the above twelve-step calculation is the determination of one parameter of the L,A,B, L/A, or B/L, when all the rest are given, in order for the safety factor to assume a desired given value, e.g.5. In this case, a computer algorithm should be built allowing determination of the sought parameter by the trial-and-error, i.e., spanning the variable of concern over a spectrum of numerical values (everything else being constant) until the desired safety factor is achieved.

Numerical Example 9

Find which of the following two intersections is safer: The first is the intersection of a main haulageway 16 ft wide, and a crosscut 12 ft. wide. The pillar pattern is regular and the center-to-center distance is 50 ft. The height of the pillars is 6 ft. and the immediate roof plate is 5 ft. thick. The second intersection belongs to a staggered pillar pattern with main haulageways 18 ft. wide, crosscuts 10 ft. wide, and center-to-center distance of pillars 60 ft. Obviously the sides of the regular pillars along the main haulageways are 38 ft. long, and the sides along the crosscuts are 34 ft. long. The sides of the staggered pillars along the mains are 50 ft. long and the sides along the crosscuts are 42 ft. long. The two intersections are located within the same coal seam, which has modules of elasticity in the vertical direction equal to 5.10^5 psi. The roof has Poisson ratio in the horizontal direction equal 0.33, modules of elasticity in the same direction equal to $1 \cdot 10^6$ psi, and transverse unit load equal to 5 psi. The compressive and tensile strength of coal are 150 psi and 80 psi, respectively.

Solution

Given	Sought
$L_R = 16$ ft.	K_R
$B_R = 12$ ft	K_S

Given

$A_R = 50$ ft.
$H_R = H_S = 6$ ft.
$d_R = d_S = 5$ ft.
$L_S = 18$ ft.
$B_S = 10$ ft.
$A_S = 60$ ft.
$E_{p,R} = E_{p,S} = 5 \cdot 10^5$ psi
$n_R = n_S = 0.33$
$E_{N,R} = E_{R,S} = 1 \cdot 10^6$ psi
$q_R = q_S = 5$ psi
$S_{comp} = 150$ psi
$S_{tens} = 80$ psi

Sought

K_R, K_s

where subscripted R and S stand for regular and staggered pillar patterns, respectively

(1) According to Eq. 37

$$M_{max,R} = -\frac{1}{12} \cdot 5 \cdot 16^2 \cdot 12^2 = -15{,}360 \text{ in-lb/in}$$

$$M_{max,S} = -\frac{1}{12} \cdot 5 \cdot 18^2 \cdot 12^2 = -19{,}440 \text{ in-lb/in}$$

(2) According to Eq. 76

$$M_{(x=L/2),R} = \frac{1}{24} \cdot 5 \cdot 16^2 \cdot 12^2 = 7{,}680 \text{ in-lb/in}$$

$$M_{(x=L/2),S} = \frac{1}{24} \cdot 5 \cdot 18^2 \cdot 12^2 = 9{,}720 \text{ in-lb/in}$$

(3) According to Eq. 77

$$C_R = \frac{3}{4}(1 - 0.33^2)\left(\frac{5 \cdot 10^5}{1 \cdot 10^6}\right)\left(\frac{50^4}{6 \cdot 5^3}\right) = 2.78 \times 10^3$$

$$C_S = \frac{3}{4}(1-0.33^2)\left(\frac{5 \cdot 10^5}{1 \cdot 10^6}\right)\left(\frac{60^4}{6 \cdot 5^3}\right) = 5.77 \times 10^3$$

(4) From Fig. 11, for $C_R \approx 3 \cdot 10^3$ and $C_S \approx 6 \cdot 10^3$ we find

$I_R + \approx 2.20$

$I_R - \approx 1.34$

$I_S + \approx 2.24$

$I_S - \approx 1.25$

(5) $\dfrac{L_R}{A_R} = \dfrac{16}{50} = 0.32$

$\dfrac{L_S}{A_S} = \dfrac{18}{60} = 0.30$

From Fig. 12 for $\dfrac{L}{A} \approx 0.35$ we find

$II_{R+} \approx 1.04$

$II_{R-} \approx 0.98$

From Fig. 12 for $\dfrac{L}{A} \approx 0.30$ we find

$II_{S+} \approx 1.0$

$II_{S-} \approx 0.9$

(6) $\dfrac{B_R}{L_R} = \dfrac{12}{16} = 0.75$

$\dfrac{B_S}{L_S} = \dfrac{10}{18} = 0.56$

From Fig. 13 for $\dfrac{B}{L} = 0.75$ we find

$III_{R+} \approx 0.93$

$III_{R-} \approx 0.81$

From Fig. 13 for $\dfrac{B}{L} \approx 0.55$

$III_{S+} \approx 0.82$

$III_{S-} \approx 0.70$

(7) $\overset{\not{-}}{M}_{max,R} = M_{max,R} \cdot I_{R-} \cdot II_{R-} \cdot III_{R-}$ where $\not{-}$ stands for intersection (78)

According to Eq. 78

$$\overset{\not{-}}{M}_{max,R} = -15,360 \cdot 1.34 \cdot 0.98 \cdot 0.81 = -16,338$$

Similarly

$$\overset{\not{-}}{M}_{max,S} = M_{max,S} \cdot I_{S-} \cdot II_{S-} \cdot III_{S-} \qquad (79)$$

According to Eq. 79

$$\overset{\not{-}}{M}_{max,S} = -19,440 \cdot 1.25 \cdot 0.9 \cdot 0.70 = -15,309$$

(8) $\overset{\not{-}}{M}_{(x=L/2),R} = M_{(x=L/2),R} \cdot I_{R+} \cdot II_{R+} \quad III_{R+}$ (80)

According to Eq. 80

$$\overset{\not{-}}{M}_{(x=L/2),R} = 7,680 \cdot 2.20 \cdot 1.04 \cdot 0.93 = 16,342$$

$$\overset{\not{-}}{M}_{(x=L/2),S} = M_{(x=L/2),S} \cdot I_{S+} \cdot II_{S+} \cdot III_{S+} \qquad (81)$$

According to Eq. 81

$$\overset{\not{-}}{M}_{(x=L/2),S} = 9,720 \cdot 2.24 \cdot 1.0 \cdot 0.82 = 17,854$$

(9) $\left|\overset{\not{-}}{M}_{max,R}\right| = 16,338 < \left|\overset{\not{-}}{M}_{(x=L/2),R}\right| = 16,342$

$\therefore$

$$\sigma_{\substack{(tens)\\(comp)}R} = \pm \frac{6 \cdot 16,342}{(5 \cdot 12)^2} = \pm 27.24 \text{ psi}$$

$$\left|\overset{\not{-}}{M}_{max,S}\right| = 15,309 < \left|\overset{\not{-}}{M}_{(x=L/2)S}\right| = 17,854$$

$\therefore$

$$\sigma_{\substack{(tens)\\(comp)}S} = \pm \frac{6 \cdot 17,854}{(5 \cdot 12)^2} = \pm 29.76 \text{ psi}$$

(10) $K_{8,R} = \frac{-150}{-27.24} = 5.51$

$K_{8,S} = \frac{-150}{-29.76} = 5.04$

(11) $K_{9,R} = \frac{80}{27.24} = 2.94$

$K_{9,S} = \frac{80}{29.76} = 2.69$

(12) $K_{8,R} = 5.51 > K_{9,R} = 2.94$

$\therefore$

$K_{10} = 2.94$

$K_{8,S} = 5.04 > K_{9,S} = 2.69$

$\therefore$

$K_{10,S} = 2.69$

$K_{10,R} = 2.94 > K_{10,S} = 2.69$

$\therefore$

The intersection within the regular pillar pattern is safer than the intersection within the staggered pillar pattern, but not by much. Both intersections need artificial roof support.

8. AVERAGE PILLAR STRESS AND PERCENTAGE OF OVERBURDEN WEIGHT CARRIED BY A PILLAR AS FUNCTIONS OF PILLAR DIMENSIONS AND PERCENTAGE OF SEAM EXTRACTION

The effective or average Young's modulus in the direction of the pillar length, $\bar{E}y$, is a function of the percentage of recovery. Specifically:

$$\bar{E}_y = \frac{A''}{A'} E \tag{82}$$

where

$\bar{E}_y$ = effective Young's modulus in the direction of pillar length

A" = shaded area in Fig. 14 [22]

A' = total area enclosed by the dotted line in Fig. 14

E = Young's modulus of the ore composing the pillar in the direction of pillar length.

Eq. 82 may be rewritten as:

$$\frac{A''}{A'} = \frac{\bar{E}}{E}y = R \tag{83}$$

The average stress ratio in the y direction is determined from the minimum lateral dimension of the pillar, W, and the bed thickness or pillar height, H. In Fig. 14, $W_x < W_y$. Hence, $W_x = W$. Let us consider an area A compressed with an overburden pressure S_y,

$$S_y = D \cdot \gamma \tag{84}$$

D = overburden thickness (in inches)

γ = mass density of the overburden (lb/cu.in.)

Let us also define by $\overline{\sigma}_{yy}$, the average stress in the direction of the first subscript produced by an applied stress in the

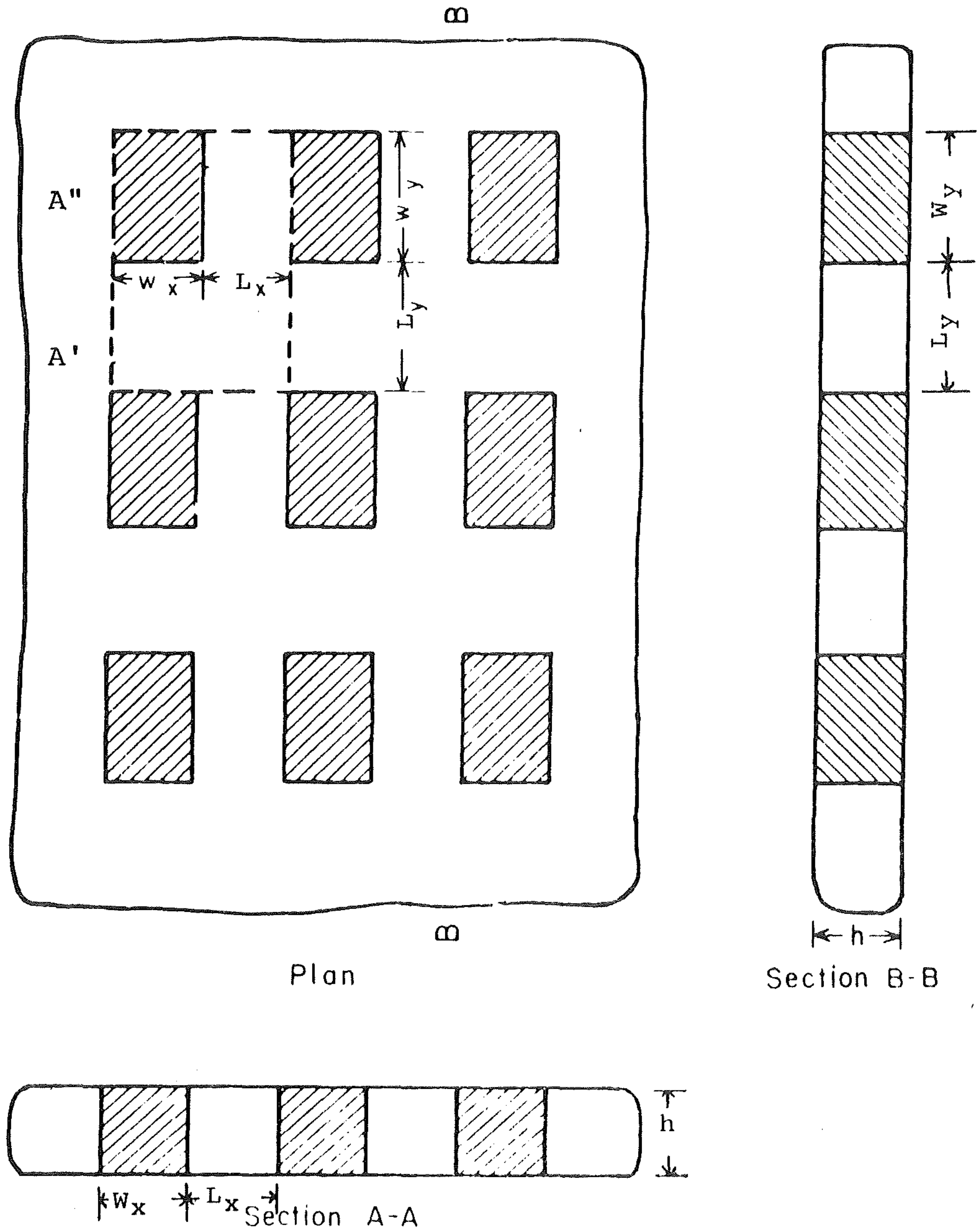


Figure 14

A simplified pattern of pillar recovery employed for the derivation of average stresses.

direction of the second subscript. On the basis of structural mechanics we may show that:

$$\frac{\overline{\sigma}_{yy}}{S_y} = \frac{2W + H}{2W + H/R} \tag{85}$$

and also that the ratio of the average pillar stress, $\overline{\sigma}_{py}$, to the overburden pressure, S_y, is

$$\frac{\overline{\sigma}_{py}}{S_y} = \frac{\overline{\sigma}_{yy}}{S_y} \cdot \frac{E}{\overline{E}_y} \tag{86}$$

Substituting $\overline{\sigma}_{yy}/S_y$ and $E/\overline{E}_y$ in eq.86 from eq.85 and 83, respectively, we find

$$\frac{\overline{\sigma}_{py}}{S_y} = \frac{2W + H}{2W + H/R} \cdot \frac{A'}{A''} = \frac{2W/H + 1}{2W/H + A'/A''} \cdot \frac{A'}{A''} \tag{87}$$

Thus, it is apparent that $\overline{\sigma}_{py}/S_y$ is a function of the W/H ratio and the percentage of remaining coal, A"/A'. By assigning values to A"/A' from 0.10 to 0.50 and values to W/H from 2 to 100 and plotting the results of eq.87 in semi-logarithmic scale, the plots of Fig. 15[22] are obtained. These plots allow the calculation of the average pillar stress, $\overline{\sigma}_{py}$, from the knowledge of the mass density of the overburden, γ, the overburden thickness, D, the minimum lateral dimension of the pillar, W, the height of the pillar or the bed thickness, H , and the percentage of recovery, 1 - A"/A'. Similarly, if the maximum allowed pillar stress is given, the maximum percentage of recovery can be found, for a given W. Furthermore, if maximum allowed pillar stress is given, and a certain percentage of recovery is desired, the minimum pillar width needed, can be calculated.

Since the concept of overburden weight to be supported is easier to comprehend and deal with than the concept of average

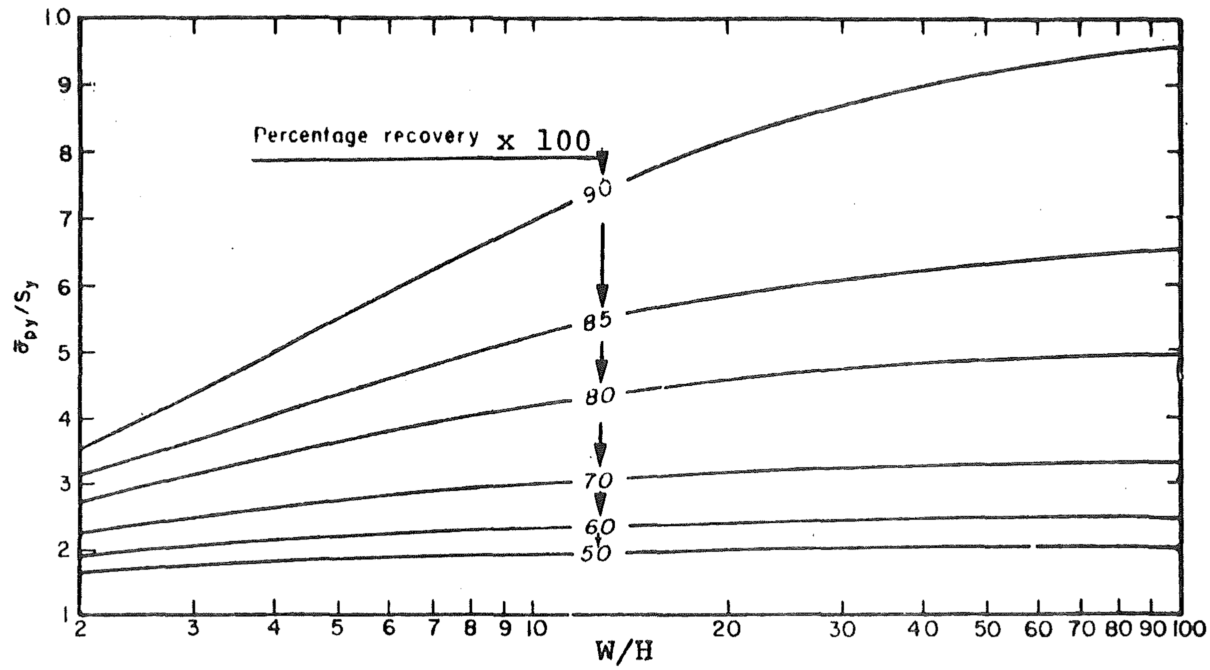


Figure 15

The ratio of the average pillar stress, $\overline{\sigma_{py}}$ to the overburden pressure, S_y, as a function of the percentage of recovery, $(1-A''/A')\cdot 100$, and the ratio of minimum lateral dimension of pillar, W, to the bed thickness, H.

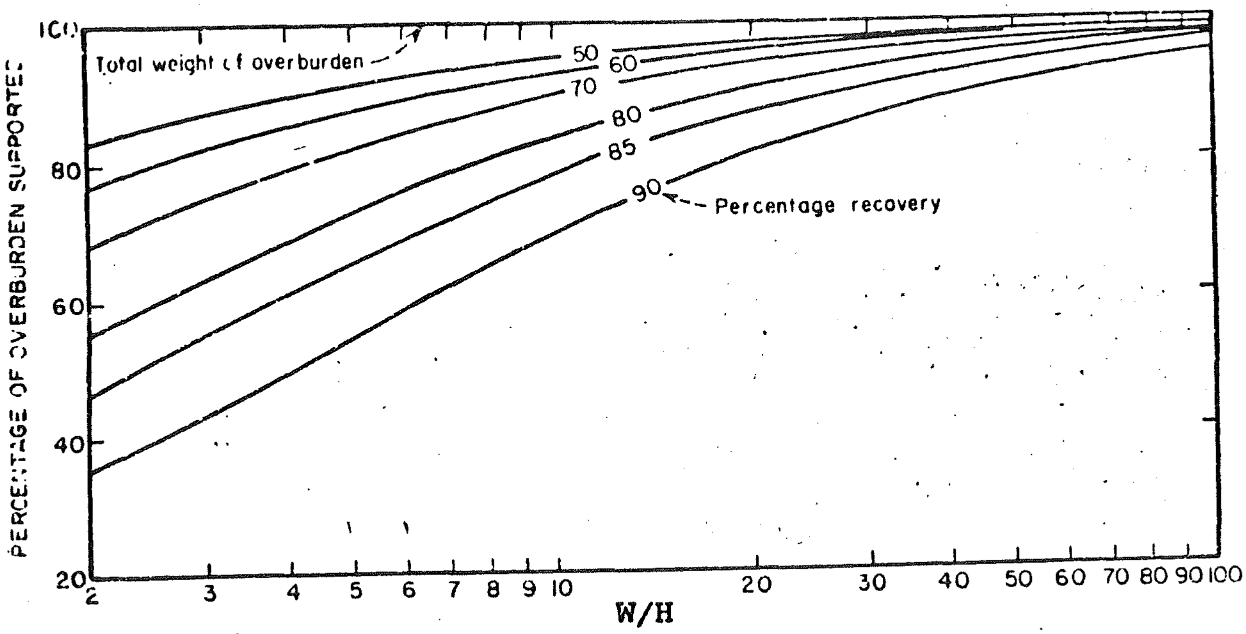


Figure 16

The actual percentage of overburden weight carried by a pillar as a function of the percentage of recovery and the W/H ratio.

stress due to the overburden weight, Fig. 16[22] is introduced. In this figure the actual percentage of overburden weight carried by a pillar is given as a function of the percentage of recovery and the W/H ratio.

The above calculated average pillar stress has been derived on the basis of finite-element theory. The same property, however, can be calculated also empirically on the basis of statistical correlation of a large number of observations. This has been done for coal mines in South Africa.[23] The obtained formula is

$$\bar{\sigma}_{py} = \frac{1.1\ D}{1 - e} = \frac{1.1\ (W + L)(W_{max} + L)D}{W \quad W_{max}} \text{(psi)} \tag{88}$$

where

L = Room width (or roof span) -ft (where $L_x = L$)
W_{max} = Pillar length (maximum lateral dimension) -ft max/ W_x, W_y/
D = Overburden thickness - ft
e = Percentage of recovery (extraction ratio) $(1- \frac{A''}{A'})$

Attention must be drawn to the fact that e is not an independent parameter but a function of L, W, and W_{max}. Indeed, if the entire coal seam is divided into unit surface segments such as the one bordered by the broken line in Fig.17,[24] then the extraction ratio within this segment is

$$e = \frac{(W_{max} + L) \cdot (W + L) - (W_{max} \cdot W)}{W_{max} + L) \cdot (W + L)} \tag{89}$$

Obviously, this is also the extraction ratio within the entire coal seam. Substitution of e from eq.89 into the second part of Eq. 88 produces the third part.

Whether or not eq.88 is valid for a given North American

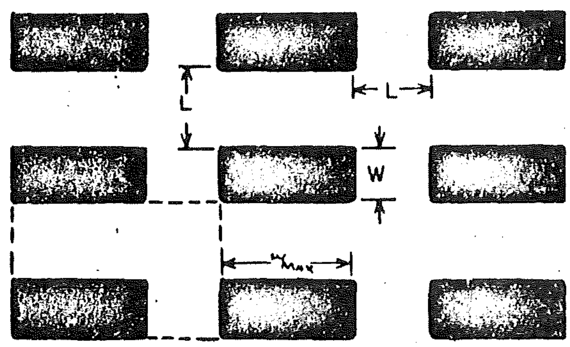


Figure 17

Expression of the extraction ratio in terms of the design parameters coal seam depends upon the similarity of the stress field of this seam with the corresponding stress field in South Africa. A discussion about stress fields was made in pages 6-8.

Numerical Example 10

Find the safety factor of the pillars in a room and pillar mine within a coal seam 6 ft. thick, under an overburden 500 ft. thick of 0.11 lb/cu.in. mass density, when the compressive strength of the coal in the vertical direction is 2,500 psi, the minimum pillar width is 50 ft., and the percentage recovery during advancement is 60% (the remaining coal in the pillars represents 40% of the total initial coal in the seam).

Solution

Given	Sought
γ = 0.11 lb/cu.in.	Kp
W = 50 ft.	
H = 6 ft.	
D = 500 ft.	
S_{comp} = 2,500 psi	
$\frac{A''}{A'}$ = 0.4	

According to eq.84

$$S_y = 500 \cdot 12 \cdot 0.11 = 660 \text{ psi}$$

According to eq.87

$$\frac{\overline{\sigma_{py}}}{660} = \frac{2 \cdot 50/6 + 1}{2 \cdot 50/6 + 1/0.4} \cdot \frac{1}{0.4} = 2.30$$

$$\frac{W}{H} = \frac{50}{6} = 8.33$$

From Fig. 15 for 60 percent recovery and W/h = 8.33 we find

$$\frac{\overline{\sigma}_{py}}{S_y} \simeq 2.33$$

The agreement is very well within the limits of the accuracy of graphic representation of analytic formulae.

From the analytic calculation performed above, we find

$$\overline{\sigma}_{py} = 2.30 \cdot 660 = 1518 \text{ psi}$$

The safety factor K_{11} is

$$K_{11} = \frac{S_{comp}}{\overline{\sigma}_{py}} = \frac{2,500}{1,518} = 1.65$$

Numerical Example 11

Find the minimum allowed width of pillars in the coal seam of example 10, if the desired safety factor K, and percentage of coal recovery e, during advancement are 2 and 0.5, respectively.

Solution

Given	Sought
γ = 0.11 lb/cu.in	W - ft
h = 6 ft.	
D = 500 ft.	
S_{comp} = 2,500 psi	
$e = 1 - \frac{A''}{A'} = 1 - .5 = .5$	
K = 2.0	

It was found in example 10 that

S_y = 660 psi ·

$$K_{11} = \frac{S_{comp}}{\bar{\sigma}_{py}} \qquad \bar{\sigma}_{py} = \frac{S_{comp}}{K_{11}} = \frac{2,500}{2.0} = 1,250 \text{ psi} \cdot$$

$$\frac{\bar{\sigma}_{py}}{S_y} = \frac{1,250}{660} = 1.89$$

According to eq.87

$$1.89 = \frac{2W/6 + 1}{2W/6 + 1/0.5} \cdot \frac{1}{0.5} \qquad \text{or}$$

$$0.95 = \frac{0.33W + 1}{0.33W + 2} \blacktriangleright W = 45 \text{ ft.}$$

$$\frac{W}{H} = \frac{45}{6} = 7.50 \cdot$$

From Fig. 15 we also find that for $\bar{\sigma}_{py}/S_y$ equal to 1.9 and percentage recovery equal to 50%,W/H is equal to 7.3 which agrees very well with the analytic result.

Numerical Example 12

Find the maximum allowed percentage of coal recovery during advancement, if the desired minimum pillar width and the safety factor in the coal seam of examples 10 and 11 are 40 ft. and 2.5, respectively.

Solution

Given	Sought
γ = 0.11 lb/cu.in.	$\frac{A''}{A'}$
H = 6 ft.	
D = 500 ft.	
S_{comp} = 2,500 psi	
W = 40 ft.	
K_{11} = 2.5	

It has already been established in example 10 that

$S_y = 660$ psi .

It was also shown in example 11 that

$$\overline{\sigma}_{py} = \frac{S_{comp}}{K_{11}} = \frac{2,500}{2.5} = 1,000 \text{ psi} .$$

$$\frac{\overline{\sigma}_{py}}{S_y} = \frac{1,000}{660} = 1.52$$

According to eq.87

$$1.52 = \frac{2 \cdot 40/6 + 1}{2 \cdot 40/6 + A'/A''} \cdot \frac{A'}{A''} \rightarrow \frac{A'}{A''} = 1.58 \rightarrow \frac{A''}{A'} = 0.63$$

The maximum allowed percentage of recovery is

$(1 - 0.63) \cdot 100 = 37\%$.

Unfortunately, Fig. 15 does not include curves for % of recovery less than 50, thus, comparison with the graphic solution is not possible.

Numerical Example 13

Find the minimum allowed pillar width in the coal seam of examples 10-12, if the desired percentage of recovery during advancement and the percentage of overburden weight supported by the pillars are 60% and 90%, respectively.

Solution

Given

$H = 6$ ft.

$\frac{A''}{A'} = 0.40$

% of overburden weight supported = 90%

Sought

W - ft.

From Fig. 16 we find that for 90% of overburden weight supported and 60% recovery, the W/H ratio required is

$$\frac{W}{H} = 6.5 \Rightarrow W = 6.5 \cdot 6 = 39 \text{ ft.}$$

Numerical Example 14

Find the maximum allowed percentage of recovery during advancement in the coal seam of examples 10-13 if the desired safety factor is 5 and the minimum pillar width is 35 ft.

Solution

Given

$H = 6$ ft.

$W = 35$ ft.

$K_{11} = 5$

$D = 500$ ft.

$\gamma = 0.11$ lb/cu.in.

$S_{comp} = 2{,}500$ psi

Sought

$\frac{A''}{A'}$

It was found in example 10 that the pressure due to the overburden weight is

$$S_y = 660 \text{ psi}$$

The maximum allowed pressure on the pillars is

$$S_{max} = \frac{S_{comp}}{K_{11}} = \frac{2{,}500}{5} = 500 \text{ psi.}$$

The percentage of overburden pressure allowed to be conveyed by the pillars is

$$\frac{S_{max}}{S_y} = \frac{500}{660} = 0.76 = 76\%$$

$$\frac{W}{H} = \frac{35}{6} = 5.83$$

From Fig. 16 we find that for $W/H = 5.8$ and percentage of overburden supported equal to 0.75, the maximum allowed percentage of recovery is between 80% and 85%, and approximately 81%. Thus, for safety, the answer is

$$\frac{A''}{A'} = 100 - 80 = 20\%$$

Numerical Example 15

Find the prevailing safety factor of the pillars in the coal seam of examples 10-14, if the pillar width is 30 ft., and the percentage of coal recovery during advancement is 70%.

Solution

Given	Sought
$W = 30$ ft.	K_{11}
$h = 6$ ft.	
$S_{comp} = 2{,}500$ psi	
$D = 500$ ft.	
$\gamma = 0.11$ lb/cu.in.	
$\frac{A''}{A'} = 0.30$	

$$\frac{W}{H} = \frac{30}{6} = 5$$

From Fig. 16 we see that for $W/H = 5$ and percentage of recovery equal to 70, the percentage overburden supported by the pillars is 82%.

We have found in example 10 that $S_y = 660$ psi. Thus, the pressure carried by the pillars is

$$S_{max} = 660 \cdot 0.822 = 542.52 \text{ psi}$$

The safety factor is

$$K_{11} = \frac{S_{comp}}{S_{max}} = \frac{2,500}{542.52} = 4.61$$

<u>Numerical Example 16</u>

Find the safety factor for the pillars in the coal seam of the examples 10-15, when its stress field is similar to the South African case, the extraction ratio during advancement is 70%, the minimum pillar width is 40 ft. and the pillar length is 70 ft.

<u>Solution</u>

Given	Sought
$e = 0.70$	K_{11}
$W = 40$ ft.	
$W_{max} = 70$ ft.	
$S_{comp} = 2,500$ psi	
$D = 500$ ft.	
$H = 6$ ft.	
$\gamma = 0.11$ lb./cu.in.	

We solve eq.89 with respect to L:

$$0.70 = \frac{(70 + L)(40 + L) - 70 \cdot 40}{(70 + L) \cdot (40 + L)}$$

$$0.3L^2 + 33L - 1960 = 0$$

$$L = 42.77 \text{ ft.}$$

According to the first part of eq.88

$$\overline{\sigma}_{py} = \frac{1.1 \quad 500}{1 - 0.70} = \frac{550}{0.30} = 1,833 \text{ psi.}$$

According to the second part of eq.88

$$\overline{\sigma}_{py} = \frac{1.1\ (40 + 42.77)(70 + 42.77)}{40.77} = 1{,}833.46 \text{ psi}$$

The safety factor is

$$K_{11} = \frac{S_{comp}}{\overline{\sigma}_{py}} = \frac{2{,}500}{1{,}833} = 1.36$$

For comparison purposes, we calculate $\overline{\sigma}_{py}$ also through Fig. 15.

$$\frac{W}{H} = \frac{40}{6} = 6.67 \text{ ft.}$$

$$S_y = D \cdot \gamma = (500)\ (1.1)\ (12) = 660 \text{ psi}$$

Fig. 15 shows that for W/ = 6.7 and percentage recovery = 70%,

$$\frac{\overline{\sigma}_{py}}{S_y} = 2.79 \rightarrow \overline{\sigma}_{py} = 2.79 \cdot 660 = 1{,}841 \text{ psi}$$

We see that the two formulae (eq.87 and 88) agree exceptionally well, despite their very different origins (theoretical and empirical, respectively).

9. AVERAGE PILLAR STRENGTH AS A FUNCTION OF THE PILLAR SHAPE

(1) Cubical Pillars

The strength of cubical pillars has been treated statistically on the basis of the weakest link theory.(25) This theory assumes a normal (Gausian) distribution of the frequency of destruction of a pillar at its weakest point (usually the most serious flaw point) over the field of compressive strength values of the pillar (Fig. 18).(25) Flaws can be weaknesses such as cracks, joint sets, foreign body inclusions within the coal, etc. The strength of the weakest element in a cubical pillar, S_N, is given by the theory as

$$S_N = S_u - \sigma(S_u)\ \left[(2\ \log N)^{1/2} - 1/2\left\{\log\ (\ \log N\ ) + \log\ (4\pi)\right\} \cdot (2\ \log N)^{-1/2}\right] \qquad (90)$$

where

N = the number of flaws in the cubical pillar of one cubic inch dimension

S_u = the average strength of a unit cube of the coal constituting the pillar, containing N_u flaws

$\sigma(S_u)$ = the standard deviation of the strength of the unit cube, S_u.

The standard deviation of the strength of the weakest element in the cubical pillar, S_N, is

$$\sigma(S_N) = \sigma(S_u) \cdot \pi \cdot (12\ \log N)^{-1/2}. \qquad (91)$$

Eq. 90 may be rewritten as

$$\frac{S_N}{S_u} = 1 - \frac{\sigma(S_u)}{S_u} \cdot F(N), \qquad (92)$$

where F(N) is the polynomial in the bracket at the right hand side of Eq. 90. For N_u = 55, the reduced pillar strength S_N/S_u has been calculated for a number of $\sigma(S_u)/S_u$ values, according to Eq. 92. The results are plotted in Fig. 19,[25] for pillars of cube side 1-18 times the unit cube size. The dependence of SN/S_u on W_N/W_u can be understood if the dependence of N on W_N/W_u where W_N equals width of the pillar and W_u equals width of the unit cube, is recalled. The curves of Fig. 19 can be used for any coal seam of any flaw density, if the unit cube for this seam has been defined accordingly. For example, for a coal seam containing 18,000

flaws per cubic foot, a volume of 55/18,000 cu.ft. contains 55 flaws and this volume should be selected as the volume of the unit cube in order for Fig. 19 to be applicable to the given seam. The side length of the unit cube should be 1.74 inches. It has been shown[25] that all empirical Eqs. 93-97 can be fit by Eq. 92 with excellent precision for properly chosen N_u and $\sigma(S_u)/S_u$ values. These parameters can be determined very precisely through laboratory tests and in-situ observations. Indeed, N can be determined through observations. S_u can be measured through compressive tests. $\sigma(S_u)$ is an index of the scattering of flaw sizes and orientations around a mean size and a mean orientation, respectively.

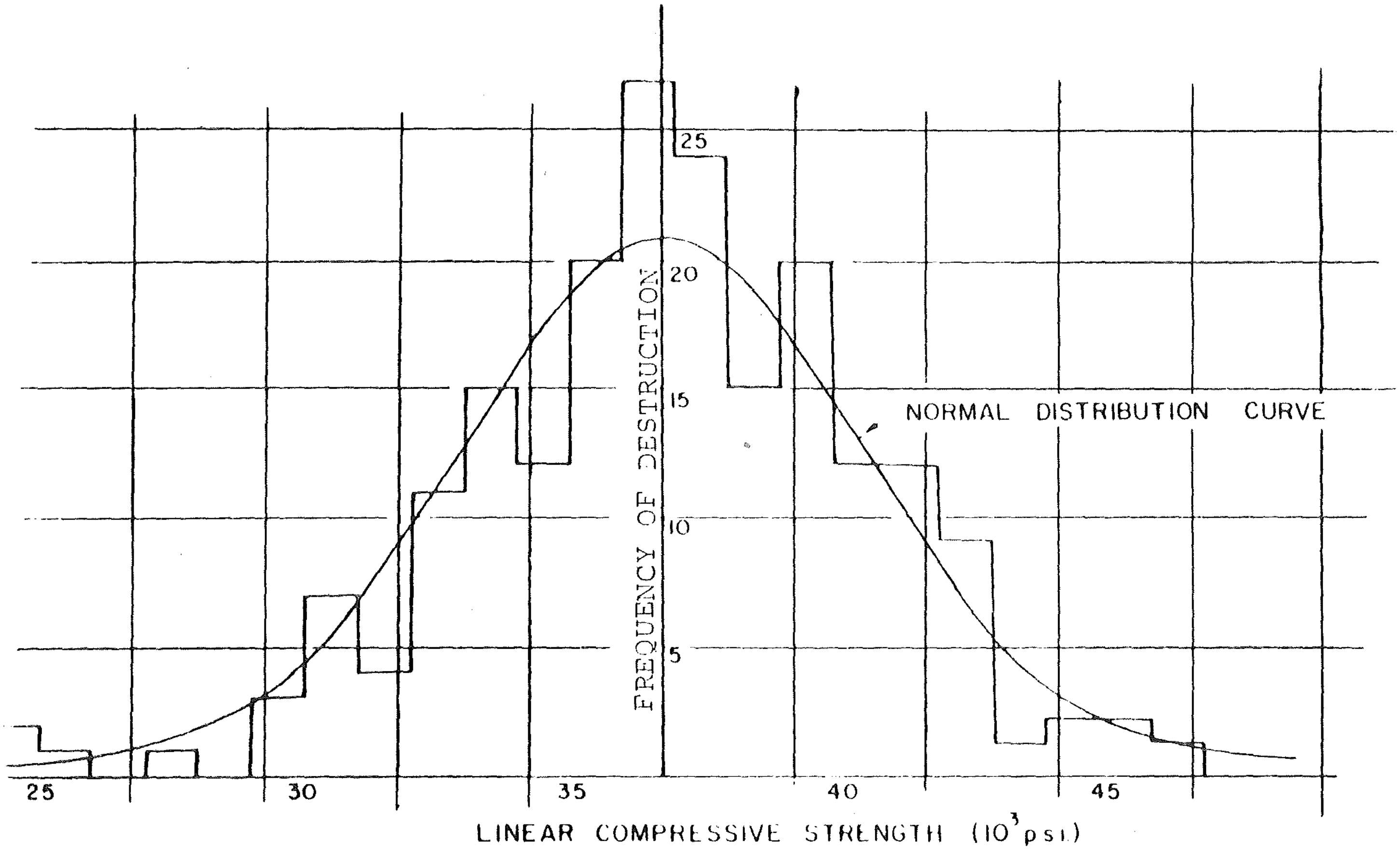


Figure 18

Typical Frequency Distribution of Linear
Compression Tests to Destruction

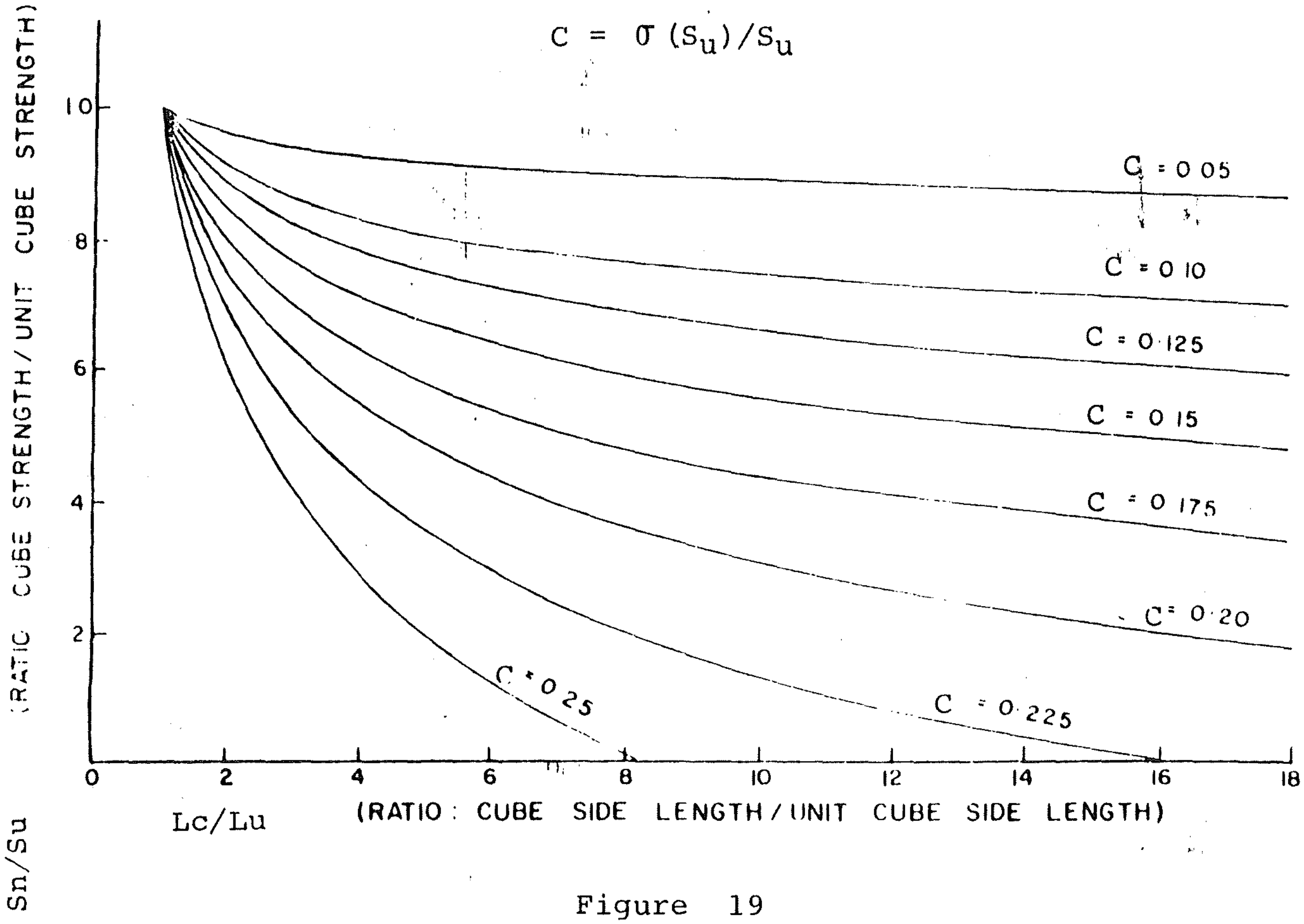


Figure 19

Pillar Strength Reduction With Increasing Pillar Dimensions

Numerical Example 17

Find the average strength of a cubical pillar, with 14 ft. cube side length, and 110 flaws/cu. ft. density of weaknesses. The compressive strength of a coal cube of 9.52 in cube side length is 15,000 psi. The standard deviation of this strength is 2,250 psi.

Solution

Given		Sought
W_N	= 14 ft.	S_N -psi
S_u	= 15,000 psi	
$\sigma(S_u)$	= 2,250 psi	
W_u	= 9.52 in.	
N_u/W_u^3	= 110 flaws/cu. ft.	

$$\frac{\sigma(S_u)}{S_u} = \frac{2,250}{15,000} = 0.15$$

$$N = 110 \cdot 14^3 = 301,840 \text{ flaws}$$

$\log N = 5.48$

$\log(\log N) = 0.74$

$(2 \log N)^{\frac{1}{2}} = 3.31$

$\log(4\pi) = 1.10$

$(2 \log N)^{-\frac{1}{2}} = 0.30$

$F(N) = 3.31 - \frac{1}{2}\, 0.74 + 1.10\ 0.30 = 3.03$

According to Eq. 92,

$$\frac{S_N}{S_u} = 1 - 0.15 \cdot 3.03 = 0.55$$

$$S_N = 0.55 \cdot 15,000 = 8,250 \text{ psi}$$

In order to be able to calculate S_N from Fig. 19, the unit cube must have size such that $N_u = 55$. But the size of the unit cube is automatically fixed once an S_u value has been used. In the present case $W_u = 9.52$, because $S_u = 15,000$ psi corresponds to this size. In order for N_0 to be equal to 55, the unit cube must have volume

$$W_u^3 = \frac{55 \text{ flaws/unit cube}}{110 \text{ flaws/cu. ft.}} = 0.50 \frac{\text{cu. ft.}}{\text{unit cube}} =$$

$$864 \text{ cu. in./unit cube} \Rightarrow$$

$$W_u = (864)^{1/3} = 9.52 \text{ in.}$$

We see that the reading of the problem has been set in such a way that the unit cube corresponding to the given S_u and the unit cube required for use of Fig. 19 with the given flaw density, are identical. This allows calculation of S_N from Fig. 19 and comparison of the two results.

$$\frac{W_N}{W_u} = \frac{14 \cdot 12}{9.52} = 17.65$$

For $W_N/W_u = 17.65$ and $\sigma(S_u)/S_u = 0.15$ we find from Fig. 19 that

$$S_n/S_u \cong 0.481 \Rightarrow$$

$$S_n = 0.481 \cdot 15{,}000 = 7{,}215$$

We see that the accuracy of Fig. 19 is limited. Also the applicability of Fig. 19 is limited because W_N/W_u is much larger than 18 for most practical cases. Finally, very seldom will we know S_u for the particular W_u which has $N_u = 55$. Thus, Eq. 92 is a much stronger tool of calculating S_N than Fig. 19.

(2) Rectangular Pillars

In the same South African study, which produced eq.88[23] the average pillar strength was also determined empirically. The developed equation is

$$S = S_u \left(\frac{WH}{W_u H_u}\right)^{C_3} \left(\frac{W}{H}\right)^{C_4} \approx 1300 \ (WH)^{-0.1} \ \left(\frac{W}{H}\right)^{0.5} \qquad (93)$$

where

S = Average strength of the pillar in question - psi
W = Minimum lateral dimension of the pillar in question - ft.
H = Height of the pillar in question - ft.
C_3, C_4 = Empirically determined constants

Subscripted quantities correspond to a reference pillar, the strength of which is known.

The general case of Eq. 93 with undetermined a and b is expected to hold in American coal mines. It can be useful when a and b have been determined for the particular mine or coal field where the formula is to be used. Whether or not the approximate formula with the given numerical constants is valid must be checked in each case before it is used.

The pillar strength, once known, contributes an important input to the use of the graphs of Figs. 15 and 16.

An alternative expression to Eq. 93 describing also the South African observations [23] is:

$$S = C_5 W^{0.46} H^{-0.66} \qquad (94)$$

where

C_5 = Empirical constant

Coal-pillar jacking tests have suggested[26] a similar expression for pillar strength. This is,

$$S = 2,800 \ W^{0.5} \ H^{-0.83} \qquad (95)$$

With W and h expressed in inches. Graphical illustration of the formula is given in Fig. 20.[29]

Laboratory tests on West Virginia coals have also demonstrated[12] validity of the relation,

$$S = C_6 W^{0.5} H^{-1.0} \tag{96}$$

where

C_6 = Laboratorily determined constant

Eq. 96 has also been proposed by a South African study.[27] The implications of Eq. 96 can be summarized through the following three rules:

1. For pillars with square cross-section of the same width but different height, it holds that:

$$\frac{S_1}{S_2} = \frac{H_2}{H_1} \tag{97}$$

   This relationship is illustrated in Fig. 21[27]

2. For pillars with square cross-section of the same height but different width, it holds that:

$$\frac{S_1}{S_2} = \left(\frac{W_1}{W_2}\right)^{0.5} \tag{98}$$

   This relationship is illustrated in Fig. 22[27]

3. For cubical pillars with sidelengths W_{max} and W_{max2}, it holds that:

$$\frac{S_1}{S_2} = \left(\frac{W_{max2}}{W_{max1}}\right) \tag{99}$$

   This relationship is illustrated in Fig. 23[27]

Finally, an alternative relation derived[28] from laboratory tests on specimen compression to destruction is

$$S = C_7 W^{0.16} H^{-0.55} \tag{100}$$

where

C_8 = Experimentally determined constant being a function of the nature of the coal specimen

Finite element theory has produced[25] for the average strength of rectangular pillars the expression,

$$S = \frac{S_u \left\{ C_9^{\bar{W}} - 1 + \frac{1}{2} C_9^{\bar{W}} (\bar{W}_{max} - \bar{W}) C_{10} - \frac{1}{2} C_{10} (\bar{W}_{max} + \bar{W}) + \bar{W}_{max} W C_{10} (C_{10} - \frac{1}{2}) \right\}}{\bar{W}_{max} \bar{W} \left\{ C_9 + C_{10}^2 (C_{10} - \frac{1}{2}) - C_{10} - 1 \right\}} \quad (101)$$

where

S_u = the average strength of a unit cube of the coal constituting the pillar with side length equal to the height of the pillar

$\bar{W}_{max}$ = the length of the pillar

$\bar{W}$ = the width of the pillar (the smaller lateral dimension)

$$C_{10} = \ln C_4 = 2.303 \log C_4 \quad (102)$$

C_9 is a material constant of the coal of concern.

$$\bar{W} = W/H \quad (103)$$

and

$$\bar{W}_{max} = W_{max}/H \quad (104)$$

where

H = Height of pillar (usually equal to the thickness of the coal seam)

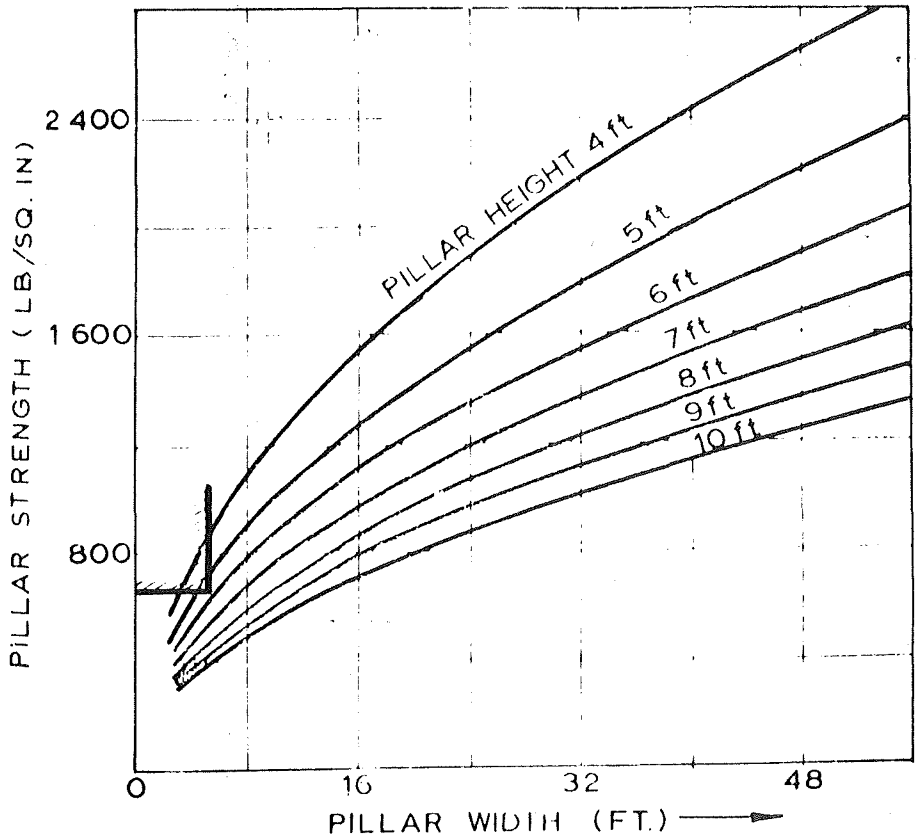


Figure 20

Graphical illustration of Eq. 25. It is important to realize that the formula is based on observations of pillars 2.4 to 5.3 ft. tall and 1.0 to 5.3 ft. wide. Thus, the graph is subjected to the uncertainties of extensive extrapolation.

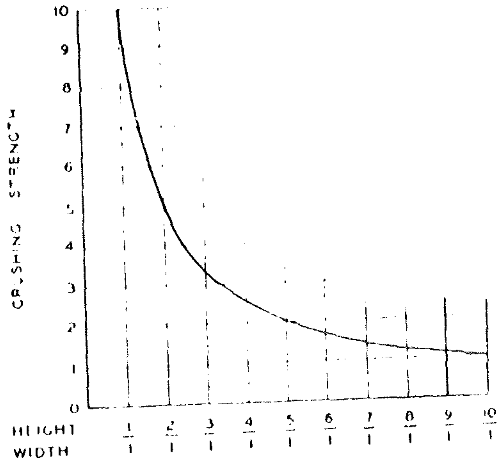


Figure 21

Graphical illustration of Eq. 97

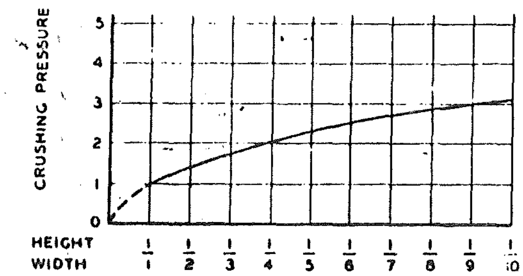


Figure 22

Graphical Illustration of eq. 98

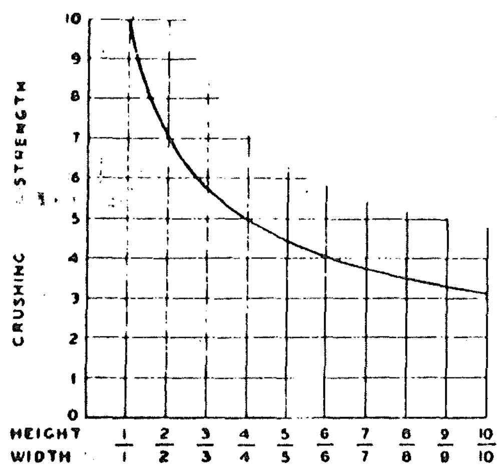


Figure 23

Graphical Illustration of eq. 99

Numerical Example 18

Find the average pillar strength of a pillar 50 feet wide, 6 feet tall.

Solution

Given	Sought
W = 50 feet	S - psi
H = 6 feet	

According to eq. 93

$$S \approx 1300 \frac{1}{(50 \cdot 6)^{0.1}} \left(\frac{50}{6} \right)^{0.5} = 2{,}108 \text{ psi}$$

According to eq. 95

$$S = 2{,}800\ (50 \cdot 12)^{0.5} (6 \cdot 12)^{-0.83} =$$

$$\frac{2{,}800 \cdot 24.49}{34.80} = 1{,}970 \text{ psi}$$

The agreement is very good.

Numerical Example 19

Find the average strength of a pillar 36 in. tall, 51 in. long, 46 in. wide, with material constant C_9 equal to 1.62 when the flaws density is 92 flaws/cu. ft. The average strength of a unit cube of the coal constituting the pillar is 1580 psi. The standard deviation of the strength of the unit cube is 258 psi.

Solution

Given		Sought
H	= 36 in.	S - psi
W_{max}	= 51 in.	
W	= 46 in.	
C_9	= 1.62	
N_{ft^3}	= 92 flaws/cu. ft.	
S_N	= 1580 psi	
$\sigma(S_N)$	= 258 psi	

We first calculate S_u according to eq. 90

$N = 92 \cdot 3^3 = 2{,}484$

$\ln N = 7.817$

$\ln(\ln N) = 2.056$

$(2 \ln N)^{1/2} = 3.954$

$(2\ln N)^{-1/2} = 0.253$

$\ln(4\pi) = 2.531$

Eq. 90 gives

$$S_u = 1580 - 258\,[3.954 - 1/2 \{2.056 + 2.531\}\, 0.253] = 710$$

We now calculate S according to Eq. 101:

$C_{10} = \ln 1.62 = 0.482$

$\bar{W} = 46/36 = 1.28$

$\bar{W}_{max} = 51/36 = 1.42$

$C_9{}^{\bar{W}} = (1.62)^{1.28} = 1.854$

$$0.5\ C_9^{\bar{W}}\ (W_{m\bar{a}x} - \bar{W})\ C_{10} = 0.5 \cdot 1.854 \cdot (1.42 - 1.28) \cdot 0.482 = 0.063$$

$$0.5\ C_{10}\ (W_{m\bar{a}x} + \bar{W}) = 0.5 \cdot 0.482 \cdot (1.42 + 1.28) = 0.651$$

$$C_{10}^2\ (C_{10} - 1/2) = (0.482)^2\ (0.482 - 0.5) = -0.004$$

$$W_{m\bar{a}x}\ \bar{W}\ C_{10}^2 (C_{10} - 1/2) = 1.42 \cdot 1.28 \cdot (-0.004) = -0.007$$

The numerator, I, of Eq. 101 becomes

$$I = 710 \cdot (1.854 - 1 + 0.063 - 0.651 - 0.007) = 183$$

The denominator, II, of Eq. 101 becomes

$$II = 1.42 \cdot 1.28\ (1.62 - 0.004 - 0.482 - 1) = 0.243$$

Thus,

$$S = \frac{I}{II} = 753 \text{ psi}$$

(3) <u>Pillars With Square Cross Section</u>

The same finite element analysis[25] has also derived the following expression for the average strength of pillars with square cross-section:

$$S = \frac{S_u \left\{ C_9^{\bar{W}} + \bar{W}^2 C_{10}\ (C_{10} - 1/2) - \bar{W} C_{10} - 1 \right\}}{\bar{W}^2 \left\{ C_9 + C_{10}^2\ (C_{10} - 1/2) - C_{10} - 1 \right\}} \tag{105}$$

This is the general expression when C_9 1. In the special case of $C_9 = 1$, Eq. 105 is reduced to

$$S = \frac{S_u\ (6+\bar{W})}{7} \tag{106}$$

Numerical results of Eq. 105 for different C_2 values appear in Fig. 24[25]. In this graph S depends on H directly. In Eq. 105 S depends on h indirectly through S_u and $\bar{W}$. Indeed, S_u is the strength of a cubical pillar of sidelength H, whereas $\bar{W}$ is the reduced width of the pillar by its height, i.e., $\bar{W} = W/h$. C_{10} is defined in Eq. 102. S_u is calculated through Eq. 90.

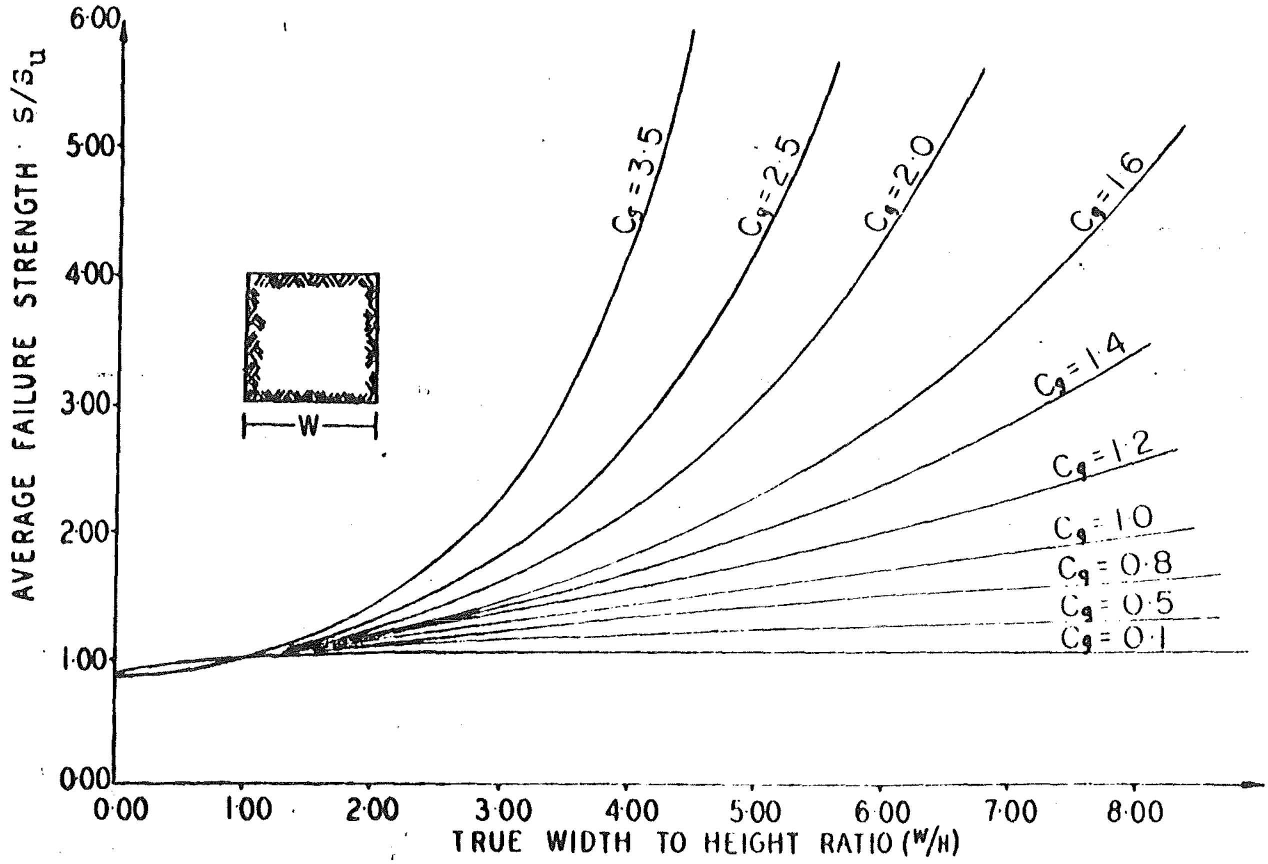


Figure 24

The Average Strength of Pillars
With Square Cross-Section For
Different Material Constants C9.

Numerical Example 20

Find the average strength of a pillar 30 inches tall, 300 inches long, 300 inches wide, with material constant C_9 equal to 1.096 when the falws density is 15,600 flaws/cu.in. The average strength of a unit cube of the coal constituting the pillar is 25,403 psi. The standard deviation of the strength of the unit cube is 2,994 psi.

Solution

Given	Sought
h = 30in. = 2.5 ft.	S - psi
W_{max} = 300 inches	
W = 300 inches	
C_9 = 1.110	
N_{ft^3} = 1,950 flaws/cu.ft.	
S_N = 25,403	
$\sigma(S_N)$ = 2,994	

We first calculate S_u according to Eq. 90.

$N = 1{,}950 \cdot (2.5)^3 = 30{,}469$ flaws

$\ln N = 10.324$

$\ln(\ln N) = 2.334$

$(2 \ln N)^{1/2} = 4.544$

$(2 \ln N)^{-1/2} = 0.220$

$\ln(4\pi) = 2.531$

Eq. 90 gives

$$S_u = 25{,}403 - 2{,}994 \left[4.544 - 0.5 \left\{2.334 + 2.531\right\} 0.220\right] = 13{,}400 \text{ psi}$$

We now calculate S according to Eq. 105.

$C_{10} = \ln 1.110 = 0.104$

$\bar{W} = 300/30 = 10$

$C_9^{\bar{W}} = 1.110^{10} = 2.839$

$\bar{W}^2 C_{10}^2 (C_{10} - 1/2) = 10^2(0.104)^2(0.104-0.5) = -0.428$

$\bar{W} C_{10} = 1.040$

The numerator, I, of Eq. 105 becomes

$I = 13,400 \{2.839 - 0.428 - 1.040 - 1\} = 4,971$ psi
$\bar{W}^2 = 10^2 = 100$
$C_{10}^2 (C_{10} - 1/2) = 0.104^2 (0.104 - 0.5) = -0.004$

The denominator, II of Eq. 105 becomes

$II = 100 \{1.110 - 0.004 - 0.104 - 1\} = 0.200$

Thus,

$$S = \frac{4,971}{0.2} = 24,857 \text{ psi}$$

(4) Barrier Pillars

The previous finite element theory[25] has also developed the following expression:

$$S = \frac{S_u \{1/2\ C_{10}\ C_9 \bar{W} - 1/2\ C_{10} + W C_{10}^2\ (C_{10} - 1/2)\}}{\bar{W} \{C_9 + C_{10}^2\ (C_{10} - 1/2) - C_{10} - 1\}} \qquad (107)$$

Barrier pillars are rectangular pillars, which are much longer than wide. Numerical results of Eq. 107 for different C_9 values appear in Fig. 25[25].

On the basis of laboratory observations,[8] the following empirical expression has also been proposed for barrier pillars:

$$S = C_{11} \left[0.778 + 0.222 \left(\frac{W}{h}\right)\right] \qquad (108)$$

where

C_{11} = Compressive strength of specimens having $\frac{W}{h} = 1$

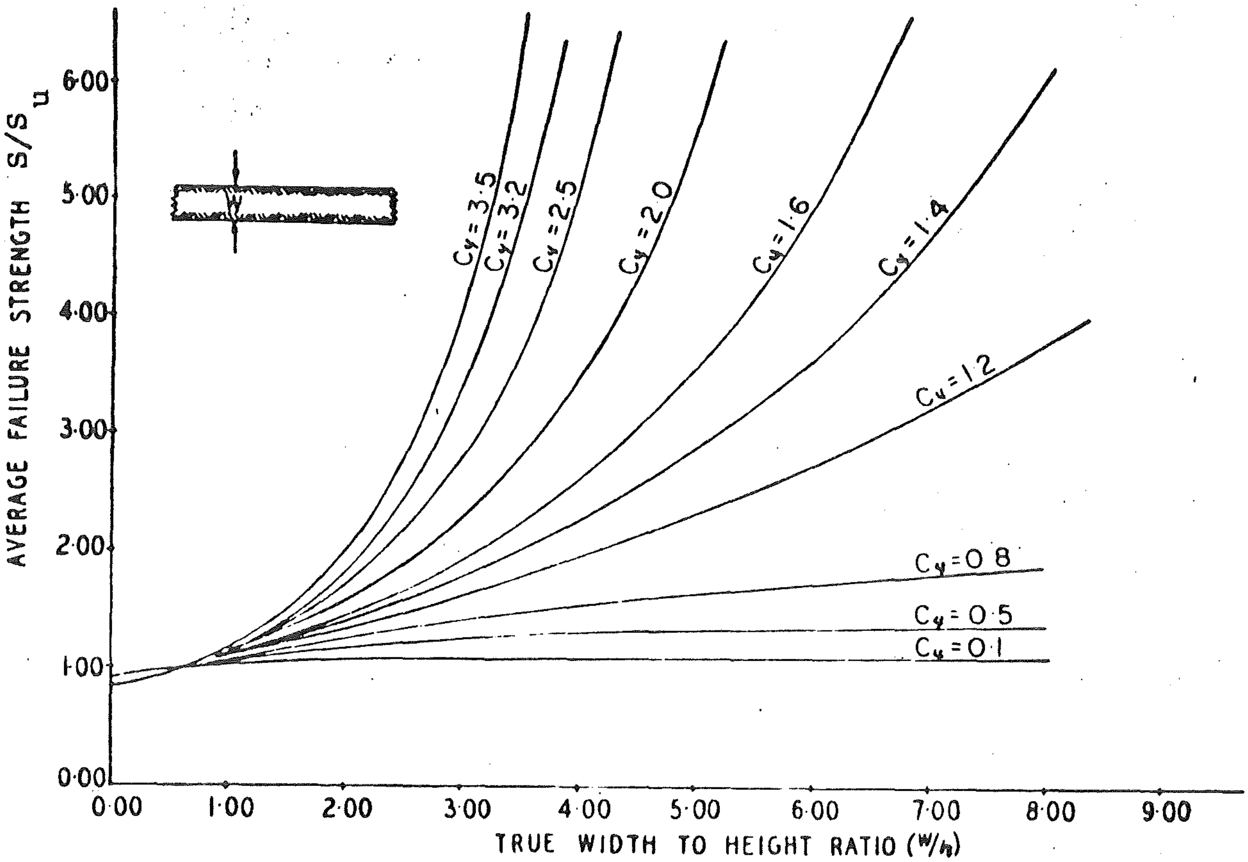


Figure 25

The Average Strength of Barrier Pillars for Different Material Constants, C_4.

Numerical Example 21

Find the average strength of a pillar 5 feet tall, 35 feet wide, and infinitely long, with material constant C_9 equal to 1.4, when the flaws density is 200 flaws/cu.ft. The average strength of a unit cube of the coal is 2,000 psi. The standard deviation of the strength of the unit cube is 250 psi.

Solution

Given	Sought
$h = 5$ feet	S - psi
$W_{max} =$	
$W = 35$ feet	
$C_9 = 1.4$	
$N_{ft^3} = 200$ flaws/cu.ft.	
$S_N = 2{,}000$ psi	
$\sigma(S_N) = 250$ psi	

We first calculate S_u according to Eq. 90.

$N = 200 \cdot 5^3 = 25{,}000$ flaws
$\ln N = 10.127$
$\ln(\ln N) = 2.315$
$(2\ln N)^{1/2} = 4.500$
$(2\ln N)^{-1/2} = 0.222$
$\ln(4\pi) = 2.531$

Eq. 30 gives

$$S_u = 2{,}000 - 250\left[4.5 - 0.5\left\{2.315 + 2.531\right\}0.222\right] = 1{,}009 \text{ psi}$$

We now calculate S according to Eq. 101

$C_{10} = \ln 1.4 = 0.336$
$\bar{W} = 35/5 = 7$
$1/2\ C_{10}C_9^{\bar{W}} = 0.5 \cdot 0.336 \cdot 1.4^7 = 1.771$

$$\bar{W}\, C_{10}^2\ (C_{10} - 1/2) = 7.0336^2\ (0.336 - 0.5) = -0.130$$

The numerator, I, of Eq. 107 becomes

$$I = 1{,}009 \left\{ 1.771 - 0.168 - 0.130 \right\} = 1{,}486 \text{ psi}$$
$$C_{10}^2\ (C_{10} - 1/2) = 0.336^2\ (0.336-0.5) = -0.018$$

The denominator, II, of Eq. 107 becomes

$$II = 7 \left\{ 1.4 - 0.018 - 0.336 - 1 \right\} = 0.322$$

Thus

$$S = \frac{1{,}486}{0.322} = 4{,}615 \text{ psi}$$

Hence

$$S/S_u = 4{,}615/1{,}009 = 4.574$$

From Fig. 25 for W/h = 7 and $C_9 = 1.4$ we find that $S/S_u = 4.667$

The agreement is very good.

Eq. 108 gives

$$S = 2{,}000 \left[0.778 + 0.222 \cdot 7 \right] = 4{,}664 \text{ psi}$$

The agreement between Eqs. 107 and 108 is indeed excellent, considering the drastic difference between their origins.

(5) Cylindrical Pillars

According to Ref. 25, the strength of cylindrical pillars is:

$$S = \frac{S_u \left\{ C_9 2\bar{R} + 4\bar{R}^2\, C_{10}^2\ (C_{10} - 1/2) - 2\bar{R}\ C_{10-1} \right\}}{4\bar{R}^2 \left\{ C_9 + C_{10}^2\ (C_{10} - 1/2) - C_{10} - 1 \right\}} \tag{109}$$

Where,

$$\bar{R} = R/h \tag{110}$$

R is the radius of the cross section and h is the height of the pillar. S_u and C_{10} are defined through Eqs. 90 and 102 respectively.

Numerical Example 22

Find the average strength of a cylindrical pillar 5 feet tall, with radius 30 feet, C_4 equal to 1.2, and flaws density equal to 150 flaws/cu.ft. The average strength of a unit cube of the coal is 2500 psi. The standard deviation of the strength of the unit cube is 300 psi.

Solution

Given		Sought
h	= 5 feet	S - psi
R	= 30 feet.	
C_9	= 1.2	
N_{ft^3}	= 150 flaws/cu.ft.	
S_N	= 2,500 psi	
$\sigma(S_N)$	= 300 psi	

We first calculate S_u according to Eq. 90.

$N = 150 \cdot 5^3 = 18{,}750$ flaws

$\ln N = 9.839$

$\ln(\ln N) = 2.286$

$(2\ln N)^{1/2} = 4.436$

$(2\ln N)^{-1/2} = 0.225$

$\ln(4\pi) = 2.531$

Eq. 90 gives

$$S_u = 2{,}500 - 300\left[4.436 - 0.5\left\{2.286 + 2.531\right\} \cdot 0.225\right] = 1{,}332$$

We now calculate S according to Eq. 109

$C_{10} = \ln 1.2 = 0.182$

$\bar{R} = 30/5 = 6$

$C_9{}^{2\bar{R}} = 1.2^{12} = 8.916$

$4\bar{R}^2\, C_{10}\, (C_{10} - 1/2) = 4 \cdot 6^2 \cdot (0.182)^2 (0.182 - 0.5) = -1.517$

$2\bar{R}\, C_{10} = 2 \cdot 6 \cdot 0.182 = 2.184$

The numerator, I, of Eq. 109 is

$I = 1{,}332 \left\{ 8.916 - 1.517 - 2.184 - 1 \right\} = 5{,}614$

$C_{10}{}^2\, (C_{10} - 1/2) = 0.182^2\, (0.182 - 0.5) = -0.010$

The denominator, II, of Eq. 109 is

$II = 4 \cdot 6^2 \left\{ 1.2 - 0.01 - 0.182 - 1 \right\} = 1.152$

Thus

$$S = \frac{5{,}614}{1.152} = 4{,}873 \text{ psi}$$

(6) Comparison of the Strength of Pillars With Different Shapes

The relative strength of the examined shapes may be evaluated on the basis of the least lateral dimension of the pillar, W, or on the basis of the cross-section area of the pillar. Results on the basis of the first criterion appear in Fig. 26.[25] The ratio $W_{max} = \infty$ corresponds to barrier pillars, the ratio $W_{max}/W = 1.0$ to square pillars and the rest ratios to rectangular pillars. All pillars have the same material constant $C_9 = 1.4$. The information presented in this figure should not be misinterpreted. The strength of a barrier pillar having the same height and width with a square pillar is greater than the strength of the square pillar. However, the length of the barrier pillar is greater than the length of the square pillar and thus, the additional strength of the barrier pillar requires additional loss of coal. In other words, the compared pillars in Fig. 26 allow different percentages of coal recovery and the comparison lacks a common denominator.

The strength comparison among pillars of different shapes on the basis of their cross-section area is more meaningful. The comparison is performed on a common basis by determining an equivalent square pillar of equal cross-section area, for each of the pillars to be compared. A cylindrical pillar of radius R has an equivalent square pillar of side

$$W = (\pi R^2)^{1/2}. \tag{108}$$

A rectangular pillar of length W_{max} and width W has an equivalent square pillar of side

$$W_{sq} = (W_{max} \cdot W)^{1/2} \tag{109}$$

Strength comparison on the basis of equivalent square pillars is presented in Fig. 27.[25]

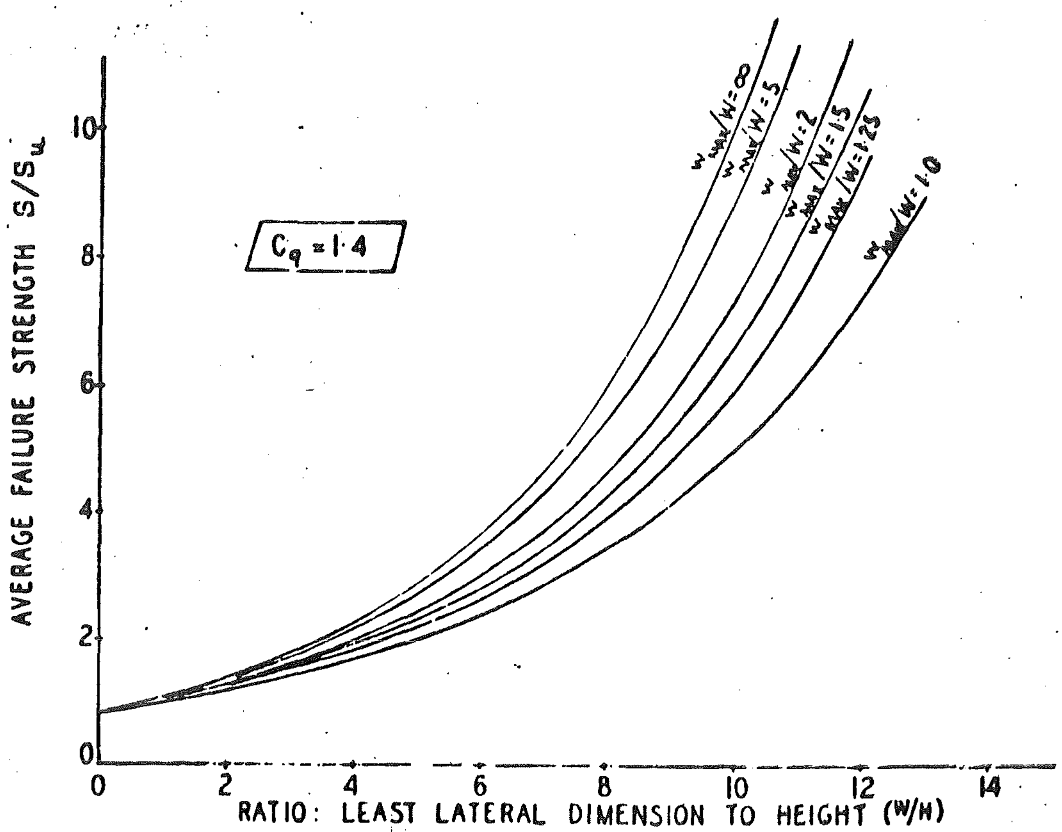


Figure 26

Comparison of the Strength of
Pillars of Different Shapes
on the Basis of Their
Length to Width Ratio

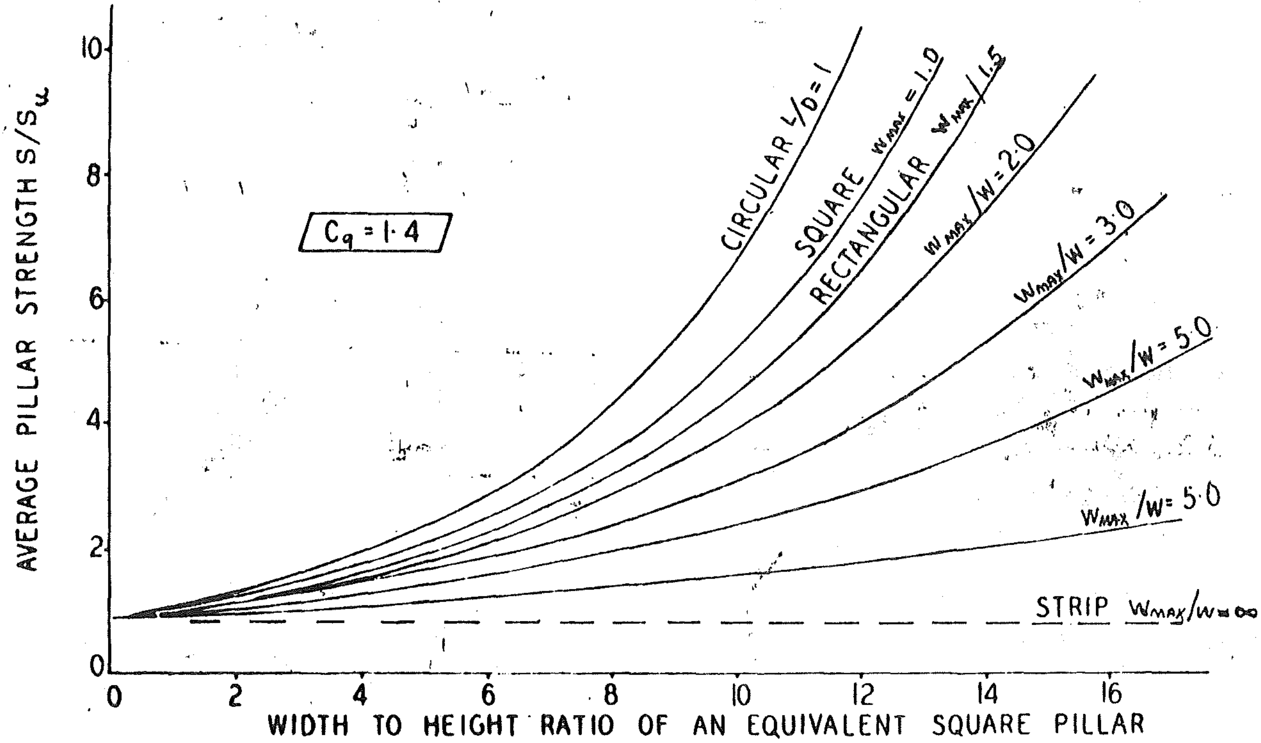


Figure 27

Comparison of the Strength of Pillars of Different Shapes on the Basis of Their Cross-Section Area

(7) The Strength of Points Within Pillars as a Function of Their Distance From the Nearest Free Surface (Pillar Side)

Finite element analysis[25] has been used to derive the strength expression

$$\frac{1/2\ C_{10}^{2}\ S_u\ (C_9^{2x} + 2C_{10} - 1)}{C_9 + C^2_{10}(C_{10} - 1/2) - C_{10} - 1} \tag{110}$$

where

x = the distance of the point of concern from its nearest pillar free surface

S_u = the uniaxial compressive strength of a cube of the coal substituting the pillar with side length equal to the pillar height

At the limiting case of $x = 0$, i.e., on the four sides (or the cylindrical surface) of the pillar, eq. 110 becomes

$$\frac{C_{10}^{3}\ S_u}{C_9 + C_{10}^{2}(C_{10} - 1/2) - C_{10} - 1} \tag{111}$$

Numerical results of eq. 103 for different C_9 values and for different reduced distances x/H (where H is the height of the pillar) are plotted in Fig. 28.[25]

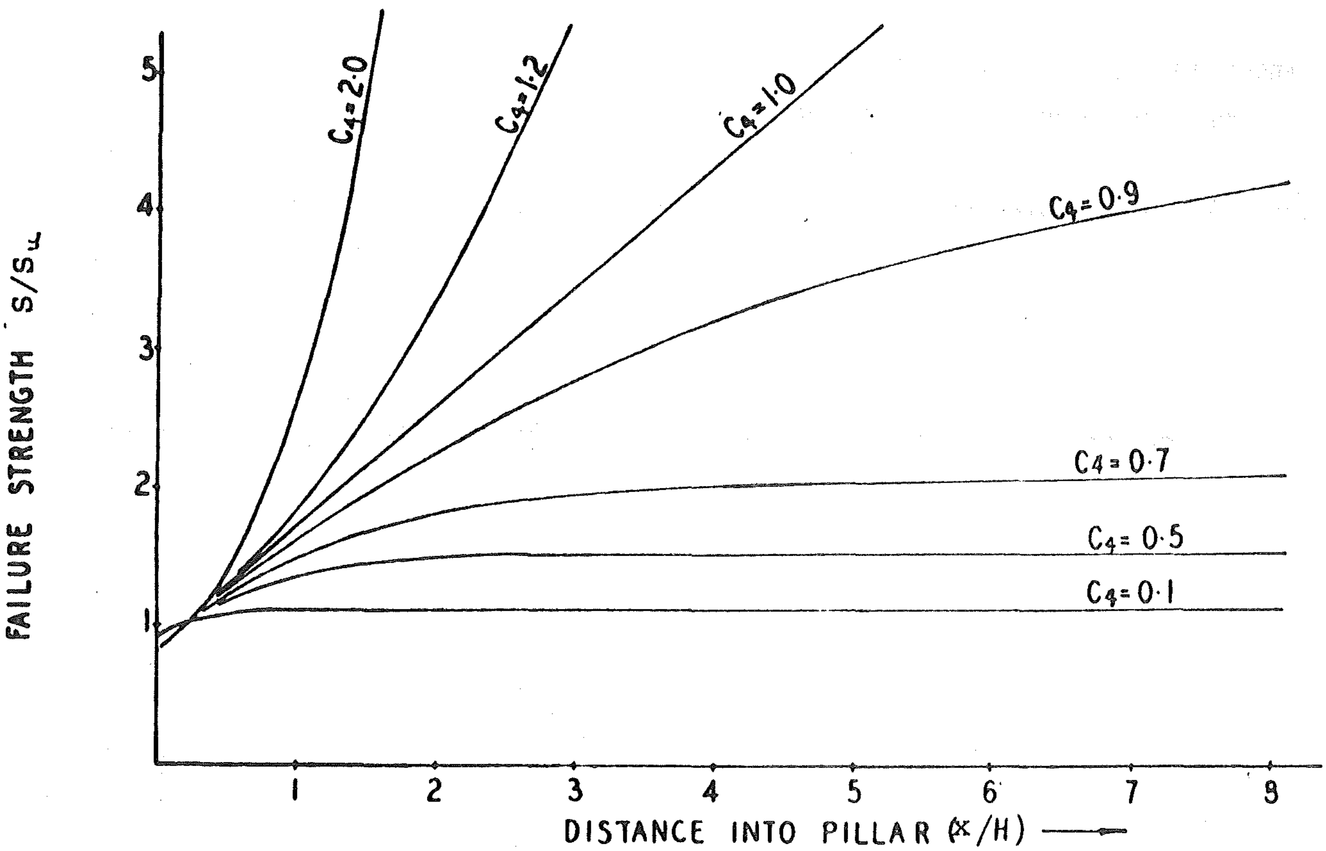


Figure 28

The Pillar Strength At
Different Depths From the
Pillar Surface, as a Function
of the Material Constant C_4

10. MEAN VERTICAL PRESSURE INCREASE ON A PILLAR AS A FUNCTION OF THE PERCENT OF PILLAR RECOVERY

It has been determined[30] experimentally that the mean vertical pressure increase, ΔP, in a pillar is approximately a parabolic function of the percent of pillar recovery, e,

$$\Delta P = -0.11e + 0.356e^2 \qquad (112)$$

Graphic presentation of this equation is given in Fig. 29.

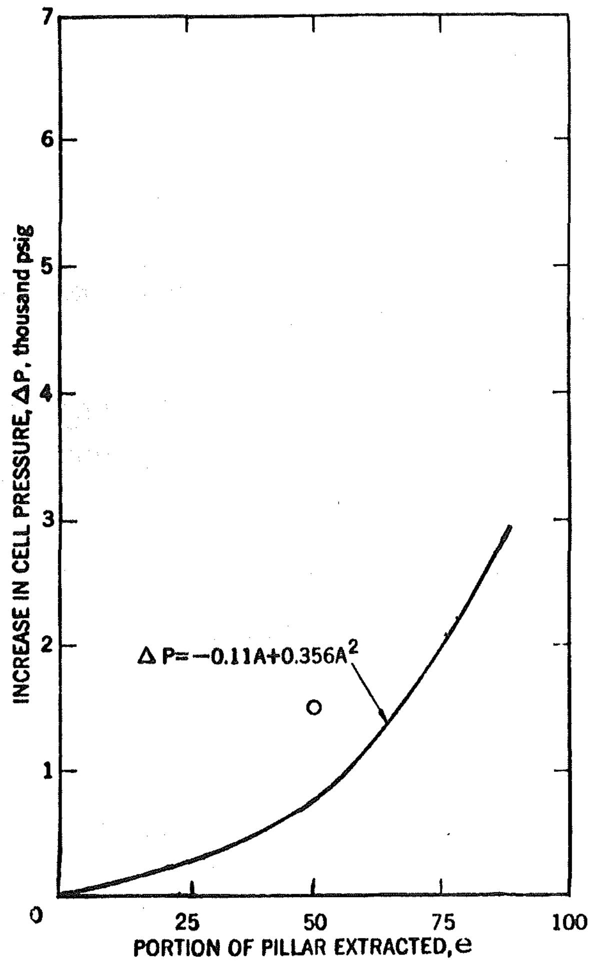


Figure 29

Mean Vertical Pressure Increase
On a Pillar As a Function of
the Percent of Pillar Recovery, e.

II. OPTIMAL DESIGN OF ROOF BOLTING

1. SELECTION BETWEEN TWO ALTERNATIVE BOLTING PATTERNS IN A MANNER MINIMIZING TOTAL STEEL CONSUMPTION FOR A GIVEN ROOF SAG

Let us introduce the following notation:

d_{min} = minimum roof sag observed immediately upon the opening of an excavation

d_{max} = maximum roof sag allowed to occur at any time after bolting

d = thickness of unsupported strata over the roof due to detachment from the higher layers of overburden. In Fig. 1, h is equal to $d_1 + d_2 + d_3$

L_b = bolt length

d_b = roof sag after bolting, expected to occur due to the weight of the bolted layers. In Fig.30, d_b is due to the weight of layers 1 and 2, since the bolt length happens to be equal to $d_1 + d_2$

d_p = maximum roof sag expected to occur without bolting prior to roof destruction. In other words, d_p is the maximum roof sag to be calmly acquired before cracks or violent burstings start to occur

d_u = roof sag after bolting, expected to occur due to the weight of the part of the unsupported layers which remain unbolted. In Fig.30, d_u is due to weight of layer 3

Obviously,

$$d_{max} = d_b + d_u \tag{113}$$

It has been found[2] that

$$d_b = d_{min} + (d_{max} - d_{min}) \frac{Lb}{d} \tag{114}$$

The normalized deflection, $\bar{d}_b$, is defined as

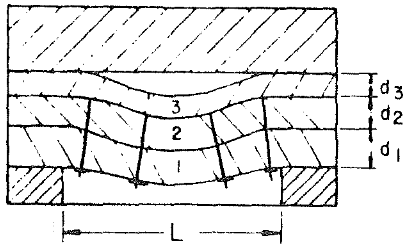


Figure 30

Support of detached layers over the roof through reinforcement action of bolts penetrating only a part of the thickness of the detached layers.

$$\overline{d}_b = \frac{d_p - d_b}{d_b - d_{min}} \tag{115}$$

It has been observed[2] that the normalized deflection, $\overline{d}_b$, exhibited in a given bolting design is proportional to the bolting density, P (bolts/ sq.ft.), of this design, i.e.,

$$\overline{d}_b = C_{12}\, P \tag{116}$$

where

C = proportionality constant independent of bolting design

Thus, for two different bolting designs, it holds that,

$$P_2 = P_1 \frac{\overline{d}_{b,2}}{\overline{d}_{b,1}} \tag{117}$$

The above equations allow comparative evaluation of the effectiveness of two alternative bolting designs under consideration, in many aspects. Demonstration of alternative utilizations of the theory of this section is given in the numerical examples 23-25.

Numerical Example 23

A 3.5 x 3.5 ft. grouted bolt pattern has been employed successfully with 5 ft.-long bolts. It is known that bed separation is extended up to 10 ft. above the opening. Roof sag of 0.7 in. occurs immediately upon opening the stope. At "infinite" time after bolting, the sag becomes 3.5 in. In similar unbolted stopes, the sag became 6 in. after "infinite" time upon opening the stope. Find the bolt density of an alternative bolting pattern utilizing 7 ft. long bolts, which allows only 2 in. maximum sag at any time after bolting. Comparatively evaluate the two patterns.

Solution

Given	Sought
$d_{min} = 0.7$ in.	P_2 - bolts/sq. ft.
$d_{max,1} = 3.5$ in.	
$d_p = 6$ in.	
$d = 10$ ft.	
$L_{b,1} = 5$ ft.	
$L_{b,2} = 7$ ft.	
$d_{max,2} = 2$ in.	
$P_1 = 1/(3.5 \times 3.5)$ bolts/sq. ft.	

According to eq. 114

$$d_{b,1} = 0.7 + (3.5 - 0.7)\frac{5}{10} = 2.1 \text{ in.}$$

$$d_{b,2} = 0.7 + (2 - 0.7)\frac{7}{10} = 1.61 \text{ in.}$$

According to eq. 115

$$\bar{d}_{b,1} = \frac{6 - 2.1}{2.1 - 0.7} = 2.79$$

$$\bar{d}_{b,2} = \frac{6 - 1.61}{1.61 - 0.7} = 4.82$$

The bolt density of the first pattern is

$$P_1 = \frac{1}{3.5 \times 3.5} = \frac{1}{12.25} \text{ bolts/sq. ft.}$$

Thus, according to eq. 117

$$P_2 = \frac{1}{12.25} \cdot \frac{4.82}{2.79} = \frac{1}{7.09} \quad \frac{1}{2 \times 3.5} \text{ bolts/sq. ft.}$$

We see that a pattern achieving the required results with 7 ft. long bolts is a 2 x 3.5 pattern.

In order to roof bolt an area of 100 sq. ft., we need

$$N_1 = \frac{100}{12.25} \approx 8 \text{ bolts.}$$

$$N_2 = \frac{100}{7.09} \approx 14 \text{ bolts.}$$

or

8 bolts x 5 ft/bolt = 40 ft. of steel for bolts

14 bolts x 7 ft/bolt = 98 ft. of steel for bolts

The second pattern requires almost two and a half times the amount of steel used by the first pattern. Obviously, this additional expense is needed for restricting the maximum sag after bolting from 3.5 in. to 2 in. Since the first pattern has been proven successful, the justification of the additional expense is questionable. Thus, it is of interest to find out what the second pattern would be for the same maximum sag after bolting. Now we have,

$$d_{max,2} = 3.5 \text{ in.}$$

Thus,

$$d_{b,2} = 0.7 + (3.5 - 0.7)\frac{7}{10} = 2.66 \text{ in.}$$

According to eq. 115

$$\bar{d}_{b,2} = \frac{6 - 2.66}{2.66 - 0.7} = 1.70$$

and according to eq. 117

$$P_2 = \frac{1}{12.25} \cdot \frac{1.70}{2.79} = \frac{1}{20.10} \approx \frac{1}{4 \times 5} \text{ bolts/sq. ft.}$$

We see that a possible pattern satisfying the new conditions would have the 7 ft. long bolts in 4 ft. x 5 ft. positions. In order to roof bolt an area of 100 sq. ft., we need

$$N_2 = \frac{100}{20.10} \approx 5 \text{ bolts}$$

Thus

5 bolts x 7 ft/bolt = 35 ft. of steel for bolts

We see that the second pattern saves 13% of the steel consumed by the first pattern and achieves identical results. Thus, it is preferable.

Numerical Example 24

In the case of the numerical example 23, the second pattern allows the same maximum sag with the first pattern and its bolts are placed in 5 ft. x 5 ft. distances. Find the required bolt length for the second pattern and evaluate it comparatively to the first.

Solution

Given

$d_{min} = 0.7$ in.

$d_{max,1} = 3.5$ in.

$d_p = 6$ in.

$d = 10$ ft.

$L_{b,1} = 5$ ft.

$d_{max,2} = 3.5$

$P_1 = 1/(3.5 \times 3.5)$ bolts/sq. ft.

$P_2 = 1/(5 \times 5)$ bolts/sq. ft.

Sought

$L_{b,2}$ - ft

By rearranging eq. 117 we obtain

$$\bar{d}_{b,2} = \bar{d}_{b,1} \cdot P_2/P_1 \tag{118}$$

Thus,

$$\bar{d}_{b,2} = 2.79 \cdot \frac{1}{25}/\frac{1}{12.25} = 1.37$$

By solving eq. 115 with respect to d_b we obtain

$$d_b = \frac{\bar{d}_b \cdot d_{min} + d_p}{\bar{d}_b + 1} \tag{119}$$

Thus,

$$d_{b,2} = \frac{1.37 \cdot 0.7 + 6}{1.37 + 1} = 2.94 \text{ in.}$$

By solving eq. 114 with respect to h_b we obtain

$$L_b = (d_b - d_{min})\ h/(d_{max} - d_{min}) \tag{120}$$

Thus,

$$L_{b,2} = (2.94 - 0.7)\cdot 10/(3.5 - 0.7) = 7.98 \approx 8 \text{ ft.}$$

An area 100 sq. ft. needs

$$N_2 = \frac{100}{25} = 4 \text{ bolts}$$

Consequently, the total length of steel required in the second pattern is

4 bolts x 8 ft/bolt = 32 ft.

We see that there is 20% savings in steel consumption by selecting the second pattern.

Numerical Example 25

In the past, 5 ft. long bolts have been used successfully. It is desirable to evaluate the shortening of the bolts to 3 ft. Evaluate the safety impact of this action, when the maximum safe roof sag is 5 in.

Solution

Given | Sought

$d_{min} = 0.7$ in. | $d_{max,2}$-ft.

$d_{max,1} = 3.5$ in.

$d_p = 6$ in.

$d = 10$ ft.

$L_{b,1} = 5$ ft.

$P_1 = P_2 = 1/(3.5 \times 3.5)$ bolts/sq. ft.

$L_{b,2} = 3$ ft.

According to eq. 118

$\bar{d}_{b,2} = \bar{d}_{b,1} = 2.1$ in.

According to eq. 119

$$d_{b,2} = \frac{2.1 \cdot 0.7 + 6}{2.1 + 1} = 2.41 \text{ in.}$$

By solving eq. 114 with respect to d_{max}, we obtain

$$d_{max} = d_{min} + (d_b - d_{min})\frac{d}{L_b} \qquad (121)$$

Thus,

$$d_{max,2} = 0.7 + (2.41 - 0.7)\frac{10}{3} = 6.40 \text{ in.}$$

We see that reducing the length of the bolts by 2 ft. causes a 2.9 in. increase of the roof sag. Since 5 in. is the maximum safe roof sag, this length reduction is not permissible.

2. LOAD PER BOLT IN A HORIZONTALLY LAMINATED ROOF

An important practical information for the optimal design of a roof bolting pattern is the load carried by each bolt of the pattern. We distinguish three cases.

(1) Completely Suspended Rock Lamina

The case is illustrated in Fig.31a[8]. A lamina of length A, width B, thickness d, and unit weight γ, is suspended from the overlying roof through the supporting action of belts exclusively. There are n_1 rows of bolts containing n_2 bolts per row. It has been shown[8] that the load per bolt L_t is given by

$$L_t = \frac{\gamma\, dBA}{(m_1 + 1)\ (m_2 + 1)} \tag{122}$$

(2) Rock Lamina With Clamped Ends and Bolt Suspensions

This is the case of a clamped beam supported throughout its length by bolts. The case is illustrated in Fig.31b[8]. It has been shown [8] that the bolt load in this case is identical with the one of the previous case, i.e., is given by eq.(122).

(3) Two Detached Laminae With Clamped Ends, Held Together By Bolts

If the upper lamina is thinner than the lower one, (Fig. 31c[8]), or in more scientific terms, if the ratio of the load per unit length to the flexural rigidity (E·l) of the upper lamina is smaller than that for the lower lamina, the upper

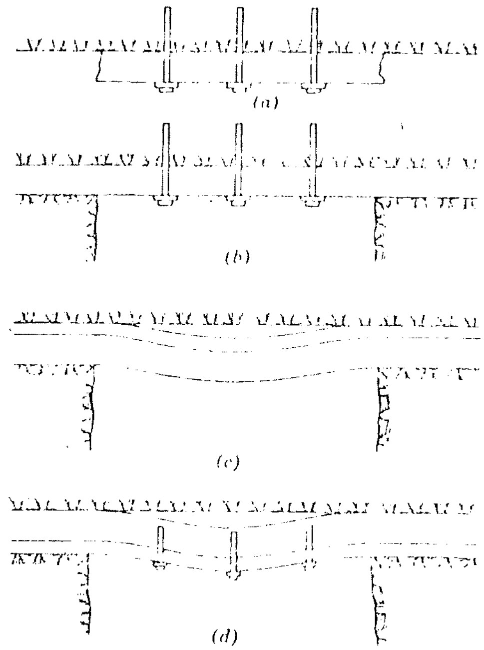


Figure 31

(a) Completely suspended rock lamina

(b) Rock lamina supported by bolts and by fixed cuts

(c) Two detached laminae with the upper one resting on the lower one

(d) Two detached laminae, with load transfer through bolts lamina is resting on the lower one. The load per unit length transferred from the upper to the lower lamina, q, is given by

$$q = \frac{q_2E_1I_1-q_1E_2I_2}{E_1I_1+E_2I_2} \tag{123}$$

Where E_i, I_i, and q_i stand for moduli of elasticity, moments of inertia, and loads per unit length, respectively. In this case, rock bolting cannot affect the load transfer--suspension effect due to bolting is nil --therefore, rock bolting is totally unnecessary.

If the upper lamina is thicker than the lower one, (Fig. 31d[8]) or

$$q_u/E_uI_u \leq q_l/E_lI_l$$

where u and l stand for upper and lower lamina, respectively, then without rock bolting the two laminae will flex to a different extent and will be detached from each other. However, if a sufficient number of bolts is installed, the two laminae can be kept in frictionless contact with uniform transfer of load along length and width of the laminae, according to eq.123. The difference from the previous case of

$$q_u/E_uI_u \geq q_l/E_lI_l$$

is that the load transfer is made through the bolts now. The load per bolt, when n bolts have been installed per row, is given by

$$L_t = \frac{qA}{n} \tag{12}$$

where q is given by eq. 123

Numerical Example

A given clamped roof lamina has been roof bolted by 3/4 in. diameter mild steel bolts in a pattern consisting of two rows of bolts with 9 bolts per row. The yield strength of the above bolts is 11,900 pounds. It has been found that the load per bolt in the above pattern is 11,200 pounds. Find whether or not an alternative pattern with 3 rows of bolts having 6 bolts per row of the same kind of bolts as before improves the bolting effect.

Solution

Given

$n_{1,1}$ = 2 rows

$n_{2,1}$ = 9 bolts/row

$n_{1,2}$ = 3 rows

$n_{2,2}$ = 6 bolts/row

L_{t_1} = 11,200 lb.

L_{max} = 11,900 lb.

Sought

L_t - lb.

According to eq.122 and the given data

$$L_{t_1} = \frac{d \cdot B \cdot A}{(2+1)(9+1)}$$

and

$$L_{t_2} = \frac{d \cdot B \cdot A}{(3+1)(6+1)}$$

Division by parts produces

$$\frac{L_{t_2}}{L_{t_1}} = \frac{30}{28} = 1.07 \quad \text{or}$$

$$L_{t_2} = 11,984 \text{ lb.}$$

We see that the second pattern is not only worse than the first pattern, but it is also prohibited since it loads the bolts beyond endurance.

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