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PERFORMANCE EVALUATION OF AUTOMATIC EXTRACTION SYSTEM

VOLUME II - PERFORMANCE EVALUATION OF THE AES

FINAL TECHNICAL REPORT

CONTRACTOR - THE DEPARTMENT OF MINERAL ENGINEERING
THE PENNSYLVANIA STATE UNIVERSITY

DATE PUBLISHED - JULY 1980

Contract No. - U.S. D.O.E. ET-77-C-01-9144
(formerly U.S.B.M. J0377030)

U. S. Department of Energy
Assistant Secretary for Energy Technology
Division of Fossil Fuel Extraction
Mining Research and Development

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This report represents work on a program that was originated in the Interior Department's Bureau of Mines and was transferred to the Department of Energy on October 1, 1977.

FE-9144-1-Vol. 2
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PERFORMANCE EVALUATION
OF
AUTOMATIC EXTRACTION SYSTEM
VOLUME II PERFORMANCE EVALUATION OF THE AES

FINAL TECHNICAL REPORT

AS OF

31 JULY, 1980

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U.S. DEPARTMENT OF ENERGY

Assistant Secretary for Energy Technology

Solid Fuels Mining & Preparation

PREFACE

This report was prepared by the Department of Mineral Engineering of The Pennsylvania State University under U.S. Department of Energy (DOE) Contract No. ET-77-C-01-9144 (formerly U.S. Bureau of Mines (USBM) Contract No. J0377030). The contract was initiated under the Automated Remote Control Continuous Mining (ARCCM) Program. Richard Farrar was Technical Project Officer and Patrick Neary was the Contract Administrator for the U.S.B.M. Contract. When transferred to DOE Michael Mahaffey became Technical Project Officer and Eula L. Irick became Contract Administrator. This report is a summary of the work completed as part of this contract during the period June 20, 1977 to July 31, 1980. This report was submitted by the authors on June 23, 1980.

The purpose of the Performance Evaluation of the Automatic Extraction System (AES) is to independently and thoroughly document the experimental mine trial of a pioneer mining system incorporating many interesting features that should improve safety, production, and profits in future underground coal mining. In addition to a detailed documentation for the operating, maintenance, and special function parameters of the AES certain other studies are performed including recommendations for training the AES operating personnel; a maintenance and operating plan that will efficiently utilize the AES; and recommendations for improvements necessary to enhance the operation of the AES.

The voluminous amount of data collected, the extensive data collection system employed, and the diverse training and utilization plans advanced suggested a series of reports of the AES in-mine experiment for specific reader interest. These reports are:

I. Executive Summary

II. Performance Evaluation of the AES

III. AES Field Documentation

IV. Recommended Operating, Maintenance, and Training Plans

V. Geotechnical Investigations of the Roof Conditions in the Area Mined by the AES Machine

VI. An Assessment of AES Operation on Mine Strata

This series of documents is arranged to provide the designers and developers of future innovative underground mining machinery with the experimental results of the prototype AES machine; a comprehensive review of the recommended training, operating, maintenance, and improvement plans; plus a suggested format for documenting future innovative mining equipment experiments.

Weekly and monthly reports of the AES activities were supplied to all participants in the project during the mine trial period. These reports are available if more detailed examination is desired.

The AES machine is a direct outgrowth of an innovative mining systems concept developed by the Lee Engineering Division of Consolidation Coal Co. (Consol). National Mine Service Company (NMSC) personnel, who designed and built the AES, contributed a significant amount of information to this report. The personnel of Consol's McElroy Mine and Lee Engineering also supplied valuable data. The technical project officers from the funding agencies, first the USBM and then DOE contributed to the overall organization and final reporting of this extensive evaluation project. We wish to acknowledge all their cooperation and are especially grateful to the following individuals who contributed to the data presented herein:

Mr. Michael Mahaffey - U.S. DOE
Mr. Richard Farrar - U.S. DOE
Mr. Howard Parkinson - U.S. DOE

Mr. Norris Brooks - Consol
Mr. Greyson Heard - Consol
Mr. James Chuckery - Consol

Mr. Jerry Karlovsky - NMSC
Mr. Donald L. Freed - NMSC
Mr. Dale Blohm - NMSC
Mr. George Lancaster - NMSC

The Penn State Audit Team referred to in this series of reports are:

- Professor Robert L. Frantz - Project Director
- Dr. Robert H. King - Project Engineer
- Mr. James D. Bennett - Project Engineer
- Mr. David L. Bartsch - Graduate Student, Investigator
- Mr. Charles R. Bickerton - Graduate Student, Investigator
- Mr. Willard E. McClain - Faculty Associate, Maintenance
- Dr. Lee W. Saperstein - Faculty Associate, Noise and Training
- Dr. H. Reginald Hardy - Faculty Associate, Roof Control
- Professor Jesse Core - Consultant
- Mr. Joseph Bacci - Consultant
- Mr. Michael Cappozoli - Consultant

The Geotechnical Investigation personnel are:

- Dr. Z. T. Bieniawski - Principal Investigator, Geotechnical Investigation
- Mr. Farzan Rafia - Graduate Student, Investigator
- Mr. David Newman - Graduate Student, Investigator

The AES is designed to be a continuous mining system that combines the cutting and loading functions with complete roof control, auxiliary ventilation, and environmental controls. The AES machine uses the framework of a Marietta 3080 Drum Miner to combine many face functions into one machine. The gathering head uses four ribbed discs to load coal into the armored chain conveyor. Temporary roof support is accomplished with overhead beams mounted on ten hydraulic cylinders. The AES loading conveyor is extensible to help maintain a stable roof bolting platform. Auxiliary ventilation is provided by two 3000 cfm fans located in the middle of the hollow inner roof beam that is used as an air duct. The AES had a feature called Automatic Control System (ACS) that can operate the machine automatically throughout its cutting cycle. Permanent roof support is afforded by four roof bolting units mounted at four-foot intervals across the rear of the machine that are used to install a row of five-foot resin roof bolts concurrent with each four feet of face advance.

Field testing of the AES was performed at Consol's McElroy Mine located in West Virginia's northern panhandle. The AES was located in one South, a six-entry sub-main development section. The in-mine trial was originally scheduled for a five month operating period. However, special circumstances caused the trial to last nine months including a three month miner strike interval. During the in-mine trial beginning in August 1977 the full time audit team observers documented all events on every production shift and on 94 percent of the maintenance shifts in a Comprehensive Field Documentation (CFD) program. Data on the eight maintenance shifts that were not monitored by the audit team observers was supplied by either NMSC or Consol personnel.

Any conclusions and observations to date regarding the operating performance on machine capabilities of the AES first generation experimental model must always be tempered with the full knowledge that the machine operated on only 21 of 71 available production days because of a combination of machine breakdowns, modifications, and adverse geological conditions. A substantial amount of additional in-mine experimentation and production is required before any degree of meaningful validation can be attributed to any of the AES components by performance evaluation studies. However, it is believed that certain strengths and weaknesses of the particular mining concept and specific machine construction can be postulated on the basis of the very limited but factual McElroy Mine observations.

This report, which is the second in a series of six, presents the performance evaluation of the automatic extraction system (AES). Chapter 1 includes a discussion of the AES and the McElroy Mine test site. The in-mine trial documentation is presented next (Chapter 2) with an abstract of the events, a list of the tests performed and not performed, the Full-Time AES Audit, the Worker Questionnaire Audit, and each Special Audit. The next chapter (3) is an analysis of the data presented in the previous chapter. Then, following a bibliography (chapter 4), the appendices are presented as chapter 5. The first appendix is a detailed commentary of the in-mine trial. The second is a correlation of operating and maintenance parameters. The third is a discussion of the failed AES parts returned to manufacturers for analysis and repairs. The fourth presents the AES specification in detail. The fifth is the state of the art of equipment evaluation.

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LIST OF ABBREVIATIONS

ACS = Automatic Control System

AES = Automatic Extraction System

ARCCM = Automated Remote Controlled Continuous Miner (Research Program)

CFD = Comprehensive Field Documentation

cfm = Cubic feet per miniute

Consol = Consolidation Coal Company

DOE = U.S. Department of Energy

fpm = Feet per minute

mg/cm^3 = Milligrams per cubic centimeter

NMSC = National Mine Service Company

OJT = On-the-Job Training

tpm = Tons per minute

USBM = United States Bureau of Mines

6 CM = Joy 6 CM Continuous Miner

MH/H = Mean manhours per operating hour

MH/T = Mean manhours per ton

MSHA = Mine Safety and Health Administration

Q = Quantity

ROM = Run-of-Mine

V_{max} = Maximum velocity

r = Correlation Coefficient

DSC = Degree of Significance of Correlation

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12. Sponsoring Organization Name and Address U.S. Department of Energy Washington, DC 20241		13. Type of Report Final Technical Report
		14.
15. Supplementary Notes		

16. Abstract

The full-time reporting of an in-mine experimental mine trial is a comprehensive subject requiring in-depth objective evaluation of the relationship between men, machinery and geological conditions. The quantitative performance assessment results from objective tests developed to evaluate anticipated advantages and problems of the AES operation. The study documents all facets of machine experimental field operation to inform present and future researchers of the mining system capability in its applied environment.

The recommended methods, techniques and alternatives for selecting and applying various documentation techniques are based on a review and critical evaluation of the methods reported in the literature and applied in the field. They are presented in a series of six reports which are suggested as a user's manual for comprehensive field documentation of underground innovative mining machine applications.

This report, which is the second of a series of six, presents the performance evaluation of the Automatic Extraction System (AES). Three major areas are addressed. First, the AES and the McElroy Mine test are described. Second, the comprehensive engineering documentation of the AES in-mine trial are presented. Third, the analysis of the results of the full-time AES audit is reviewed. However, limited machine operating times must be considered in any evaluation.

This report was submitted in partial fulfillment of the U.S. Department of Energy Contract No. ET-77-01-9144 by the Department of Mineral Engineering of the Pennsylvania State University.

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CHAPTER 1

AES AND McELROY MINE TEST-SITE DESCRIPTIONS

1.1 DESCRIPTION OF THE AES

The National Mine Service Company's Automatic Extraction System (AES) was the first hardware development of Automatic Remote Controlled Continuous Miner (ARCCM) researchers. The AES was designed to be a continuous mining system that combined the cutting and loading functions with complete roof control, auxiliary ventilation, and environmental controls.

The AES machine (Figures 1 through 4) used the well developed framework of a Marietta 3080 Drum Miner to combine many face functions into one machine. It weighed 75 tons and was 38.3 feet long. The AES cut an entry 15 feet wide, using a 3-foot diameter drum mounted with carbide-tipped bits (Figure 5). It retracted to 13.3 feet for clearance when tramming and was capable of cutting an entry ten feet high. The cutting drum was tapered on the ends so that the corners of the entry were rounded to increase roof support. The drum, turning at a rate of 57 rpm, produced a bit tip speed of 535 fpm.

The gathering head (Figure 6) used four ribbed-discs to load coal into the armored chain conveyor. According to the National Mine Service Company (NMSC), the advantages of the disc-type head were reduced noise and smoother "flow" of the coal onto the conveyor causing less spillage.

Temporary roof support was accomplished with overhead metal beams (Figure 7) mounted on ten hydraulic cylinders. The roof was supported when these cylinders were extended and the inner, outer, and rear beams were pressed against it. The inner beam connected to the tractor frame by four cylinders, the two outer beams connected to floor beams by two cylinders each, and the rear beam connected to the rear operators' platform by two cylinders. Permanent support was afforded by four roof-bolting units mounted at four-foot intervals across the machine (Figure 8). Located about 19 feet from the front of the AES, the units were used to install a row of five-foot resin roof bolts with each four feet of advance.

The AES' loading conveyor was able to turn 60 degrees from center to either side (Figure 9) to facilitate loading, and was also extensible to help maintain a stable bolting platform. (Figure 9 shows the extensible section; Figure 10, the takeup assembly.)

Auxiliary ventilation was provided by two 3000 cfm fans located in the middle of the hollow inner roof beam (Figure 11) that was used as an air duct (Figure 12). Air entered the front of the beam 81 in. from the coal face, passed through the fans, and then through an extensible duct that connected to the rear beam. This air, which flowed

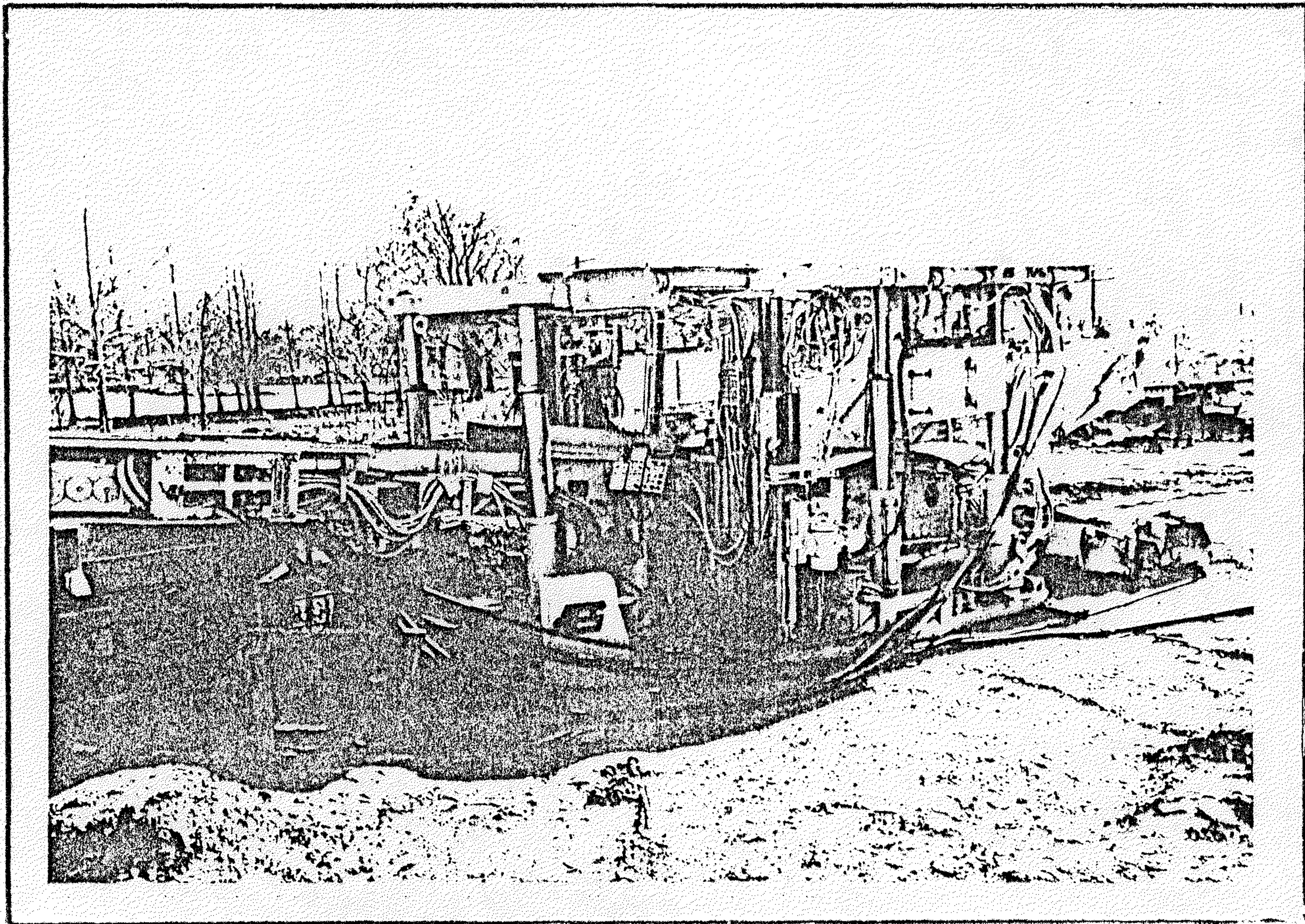


Figure 1

Automated Extraction System
View is from right rear showing the conveyor to the extreme left, the operator's station in the center, and the gathering head and conveyor at the extreme right.

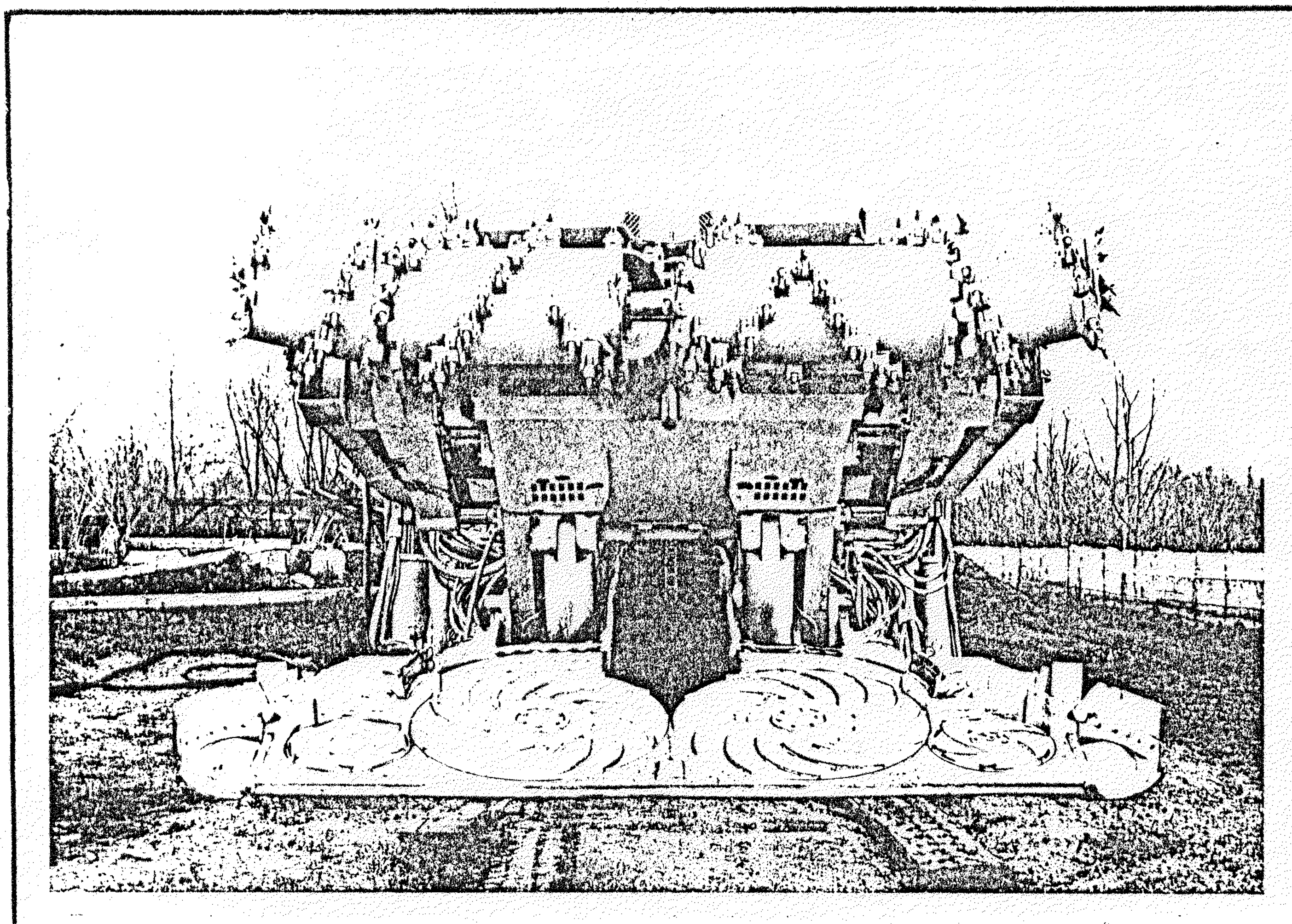


Figure 2

Automated Extraction System

View is from front showing the ventilation-duct intakes above the cutter head which is extended, the intakes to the dust scrubbers in the center of the picture, and the disc gathering-head and conveyor at the bottom.

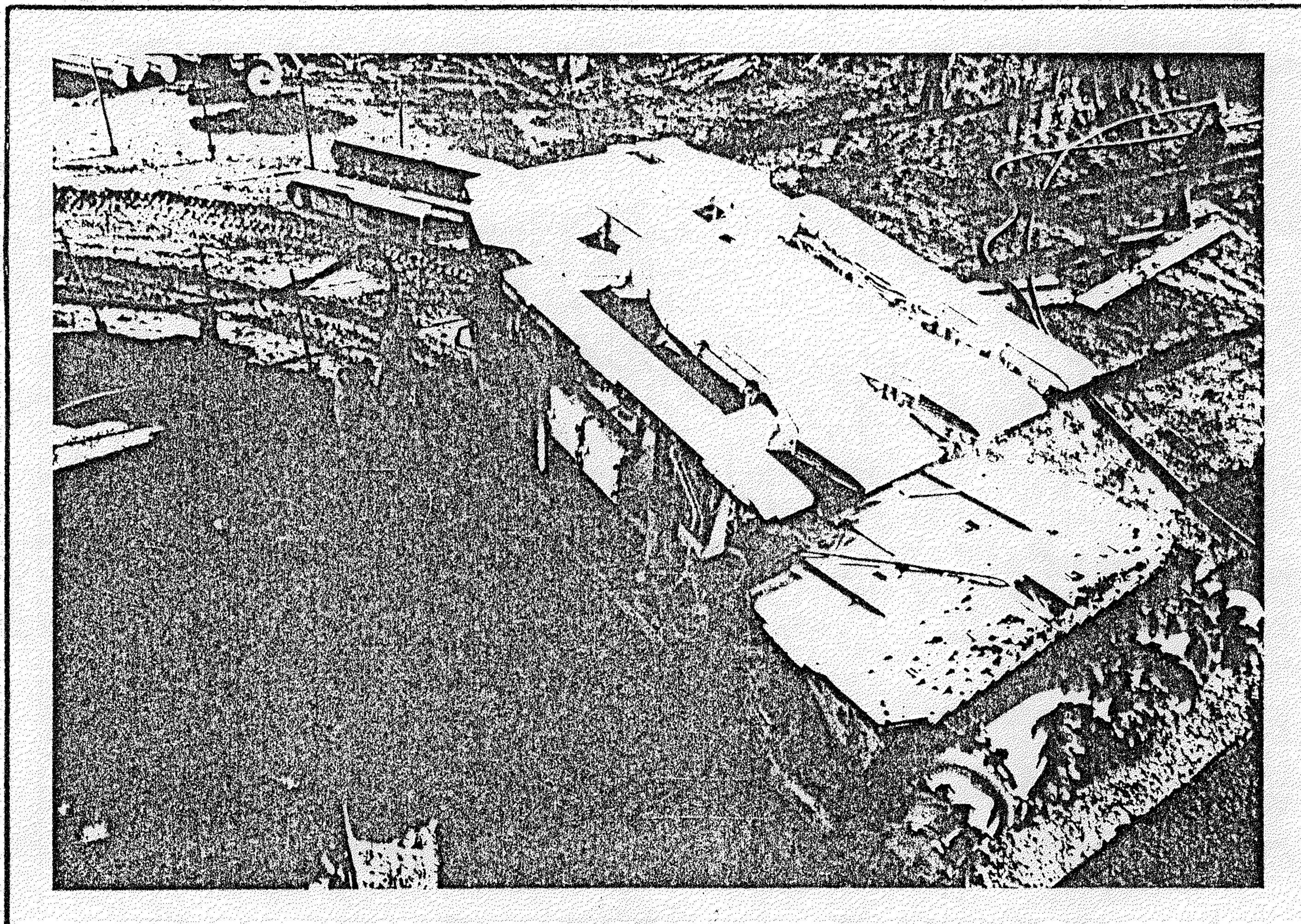


Figure 3

Automated Extraction System

View is from top showing the roof-beam assembly, the conveyor (turned to right of the AES) at the top, and the cutter head at the lower right of the photo.

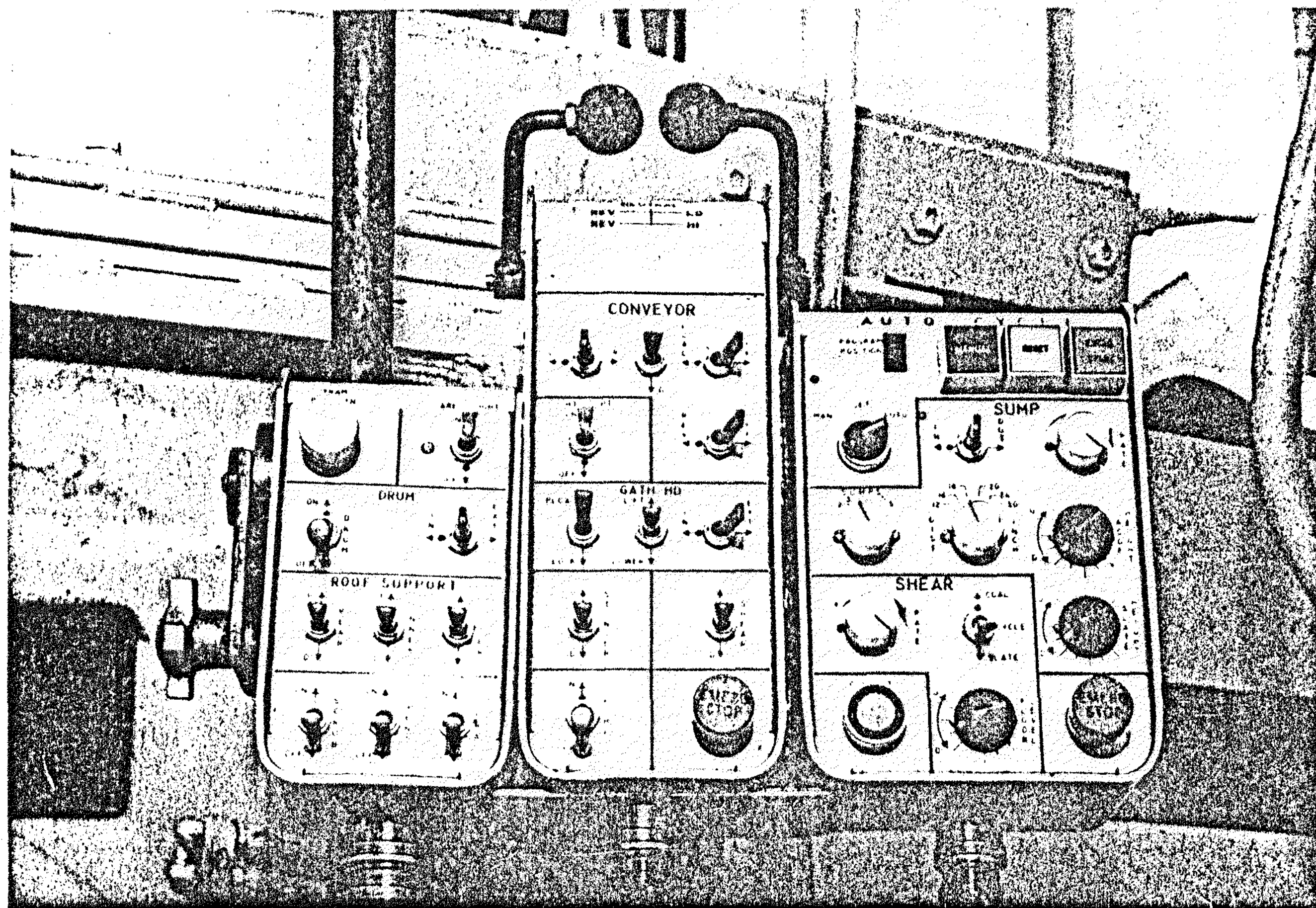


Figure 4

AES Operator's Control Panel

The tram handles are at the top, and the center section was removable to be used from an umbilical cord. The section to the right controlled the ACS, and the left section controlled the roof supports, lights, and dust-suppression system.

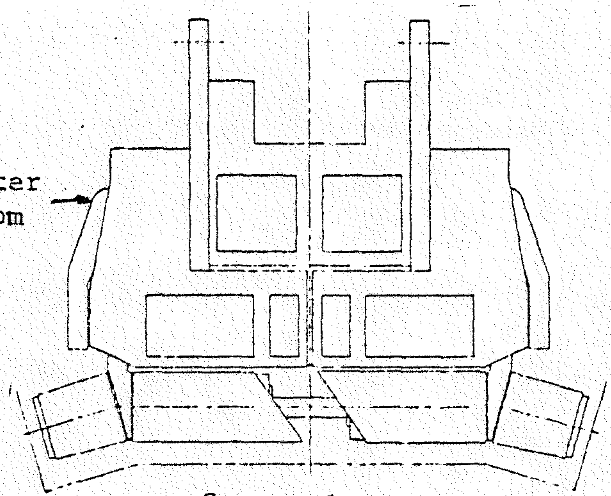


Figure 5

Cutter Boom

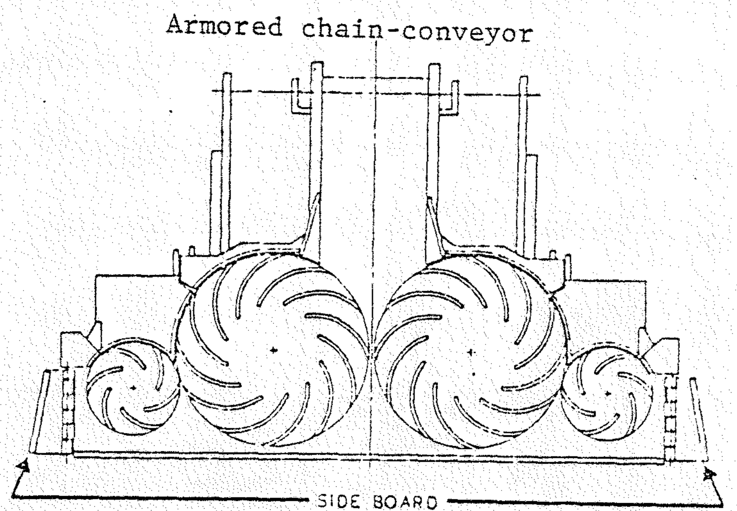


Figure 6

Gathering Head

Courtesy of National
Mine Service Company

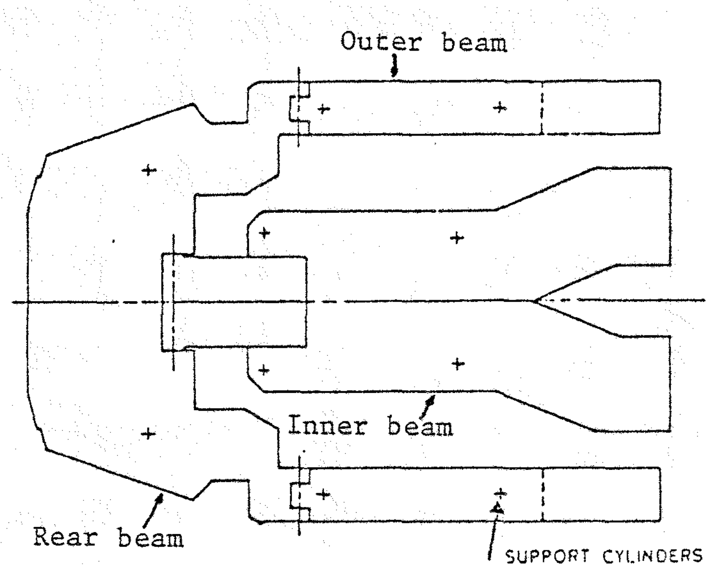


Figure 7

Roof-Beam Assembly

Courtesy of National
Mine Service Company

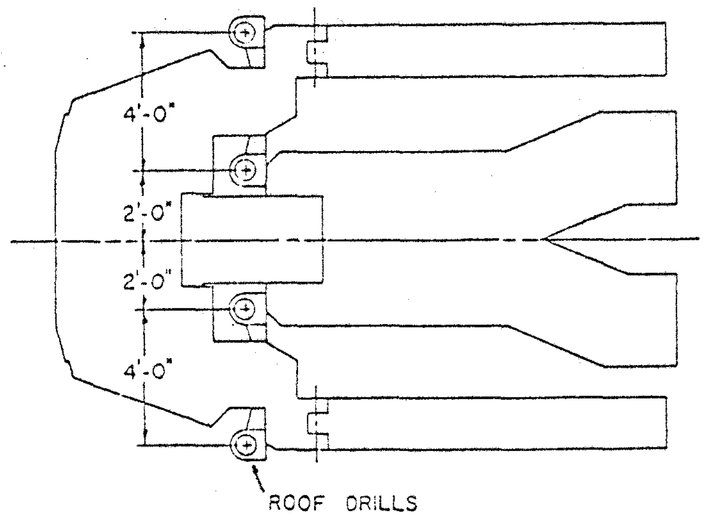


Figure 8

Roof Drills

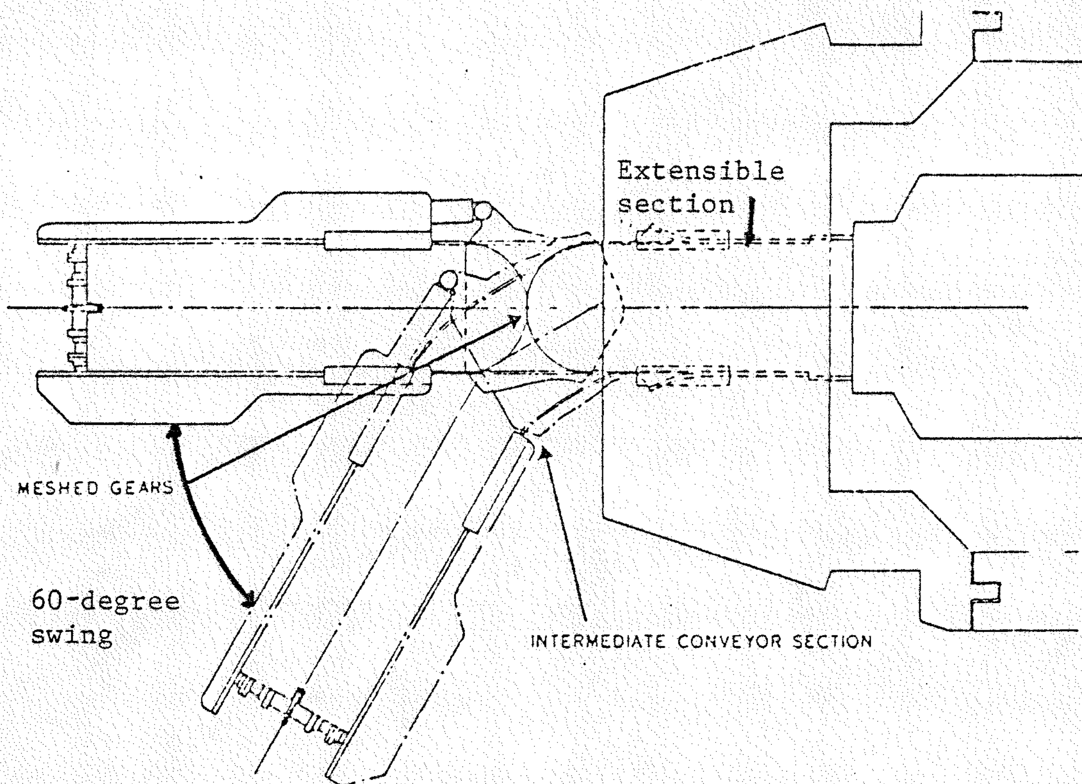


Figure 9

Conveyor Boom

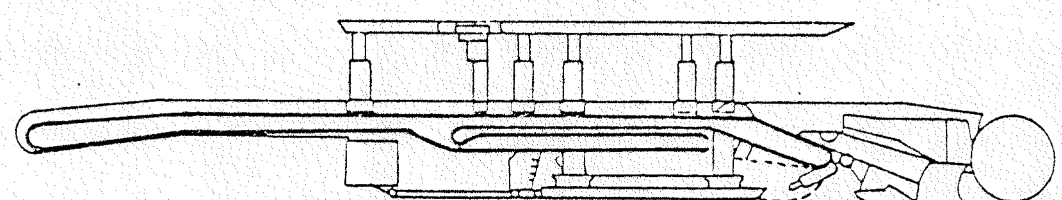


Figure 10

Extensible-Conveyor-Takeup Assembly

Courtesy of National
Mine Service Company

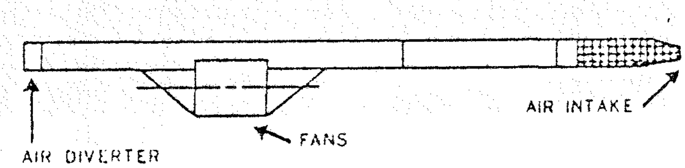


Figure 11

Ventilation Fans and Duct

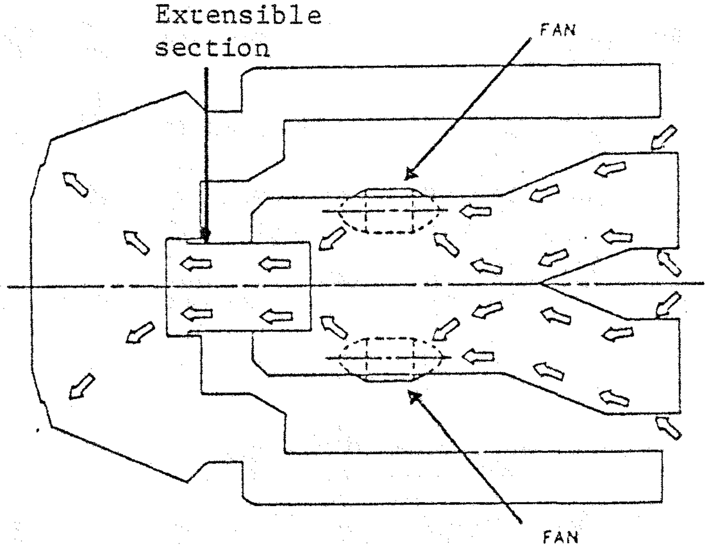


Figure 12

Airflow Through Fans and Ducts

Courtesy of National Mine Service Company

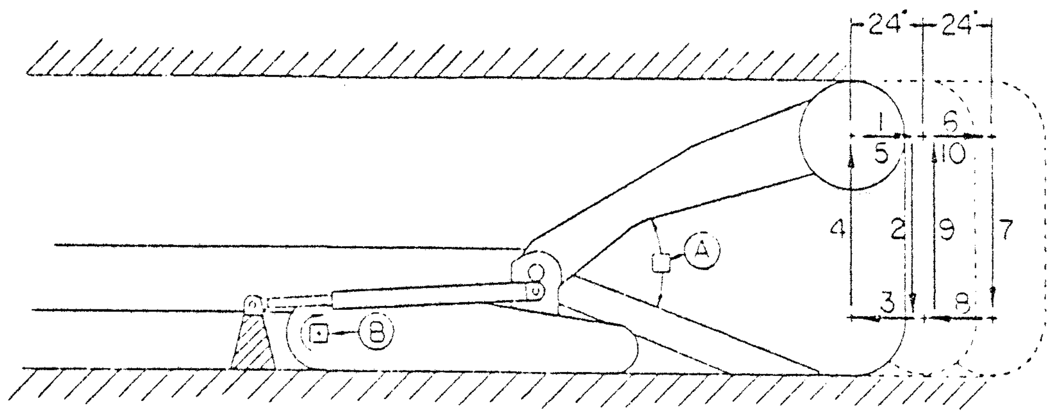


Figure 13

Typical Face Cycle for The Automatic Control System into inflatable tubing hung behind the AES, eliminated most of the methane gas liberated and respirable dust produced during mining. Water sprays and scrubber units helped suppress respirable dust at the face.

The AES had a feature called, Automatic Control System (ACS), that could operate the machine automatically throughout its cutting cycle (Figure 13). The operator selected the cutting height, depth, and number of cycles to be performed. The control cycle was started when the operator pushed a consent button. The rear and outer roof beams and the floor beams were automatically locked into place against the roof and floor, respectively. Thus, a stable platform for simultaneously cutting, loading, and bolting was to be provided (according to NMSC). The cutter drum was then sumped into the face (Figure 13, step 1) by hydraulic sump-cylinders, sheared down (step 2), retracted (step 3), raised (step 4), and advanced to the next face (step 5). The AES then automatically repeated the cycle until the present number was attained. The inner beam and tractor frame moved with the drum to maintain ventilation within ten feet of the face, as is mandatory by law. The rear bumper remained stationary during the cycles (because of the extensible conveyor and duct) to facilitate simultaneous cutting, loading, and bolting. After all the cycles were completed, the outer and rear roof-cylinders were retracted and the inner beams were locked into place against the roof. The outer beams, the rear beams, and the rear bumper were then pulled forward hydraulically and when they were again locked against the roof, the cycle was repeated.

1.2 McELROY MINE TEST-SITE DESCRIPTION

Field testing of the AES was performed at the Consolidation Coal Company's McElroy Mine. The mine is located in West Virginia's northern panhandle, 20 miles south of Wheeling. Mining is in the Pittsburgh Seam. The main roof was composed of Redstone limestone overlying a layer of limy shale (see Figure 14). The immediate roof was composed of a layer of claystone overlying a layer of bone and roof coal. The mining height included a layer of shale (called draw slate) which varied in thickness from one to three feet, approximately five feet of coal, and several thin bands of impurities (shales and pyrites). The floor consisted of soft fireclay.

The AES was located in 1 South, a sub-main development section with six entries as shown in Figure 15. The fall area shown in Entry 1 was believed to be adversely influenced by a stream flowing on the surface above the entry. The roof was also poor in some other areas, especially near the faces--believed to be caused in part by the section being idle for nine months prior to the start-up date of the in-mine trial (August 29, 1977). In addition, slickensides in the boundaries of various layers of the roof rock were believed to have had some deleterious effects. The coal ribs underwent considerable sloughage, and natural water had collected at the faces of Entries 1, 2 and 3 before mining with the AES had begun.

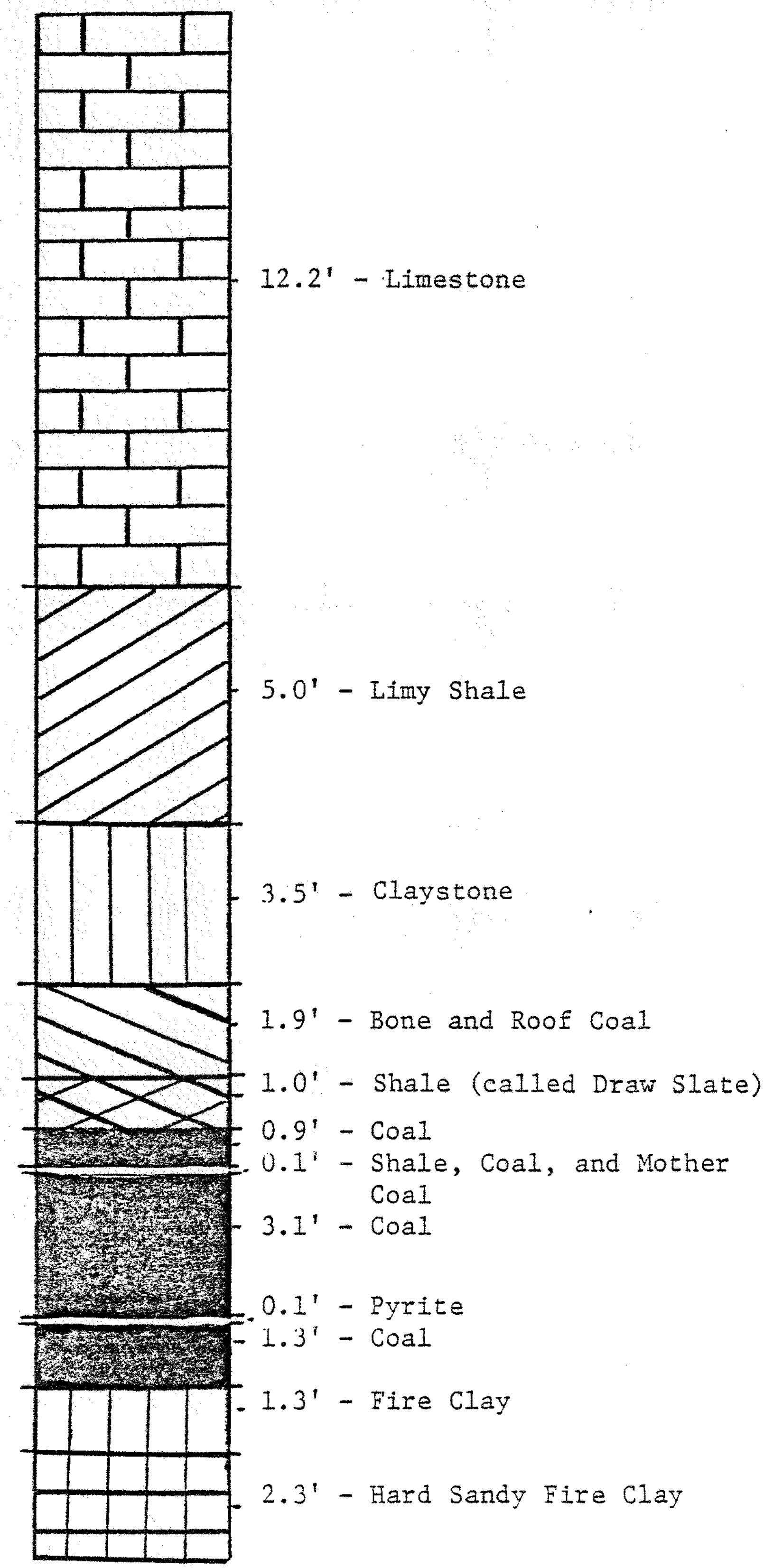


Figure 14

Stratographic Column of McElroy Mine

Courtesy of
Consol

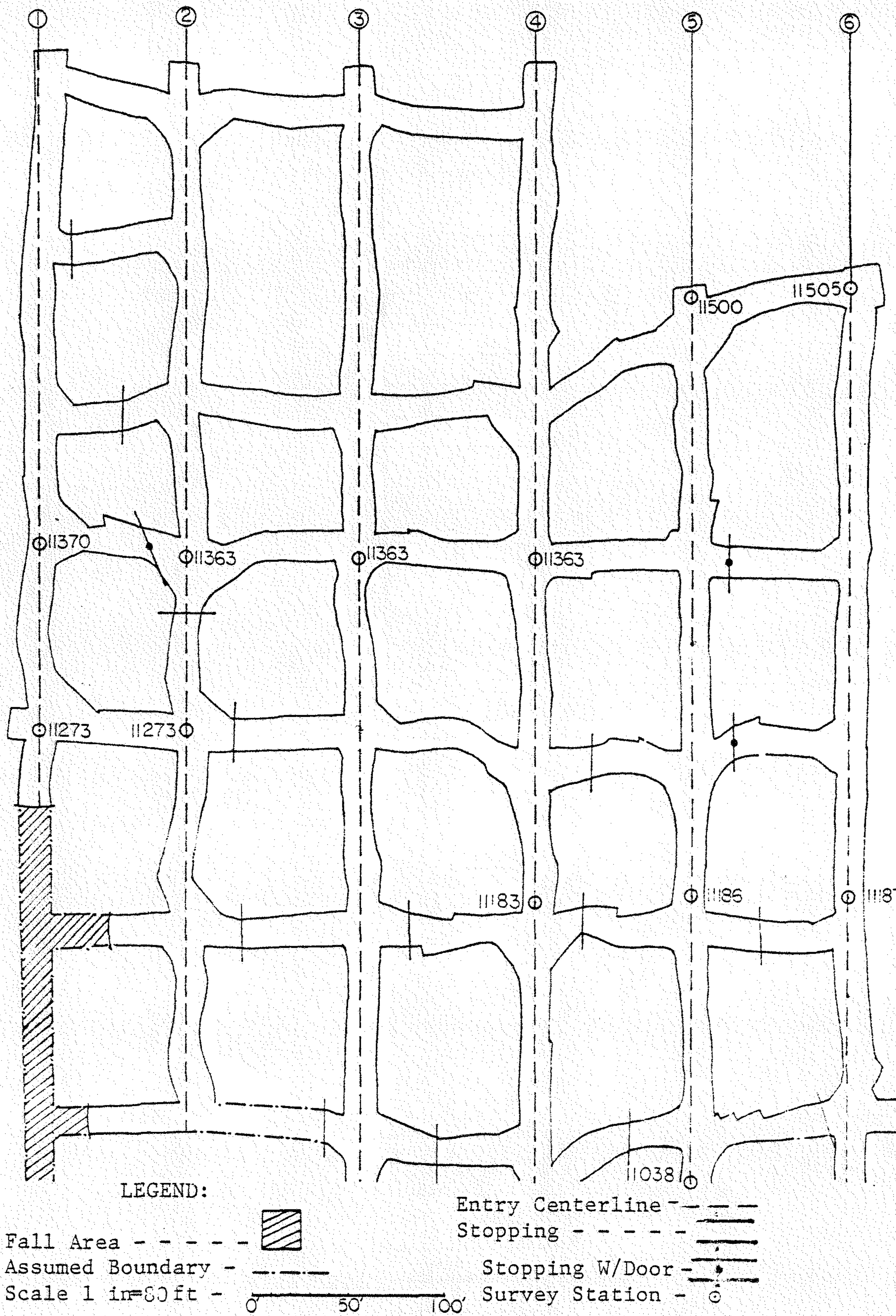


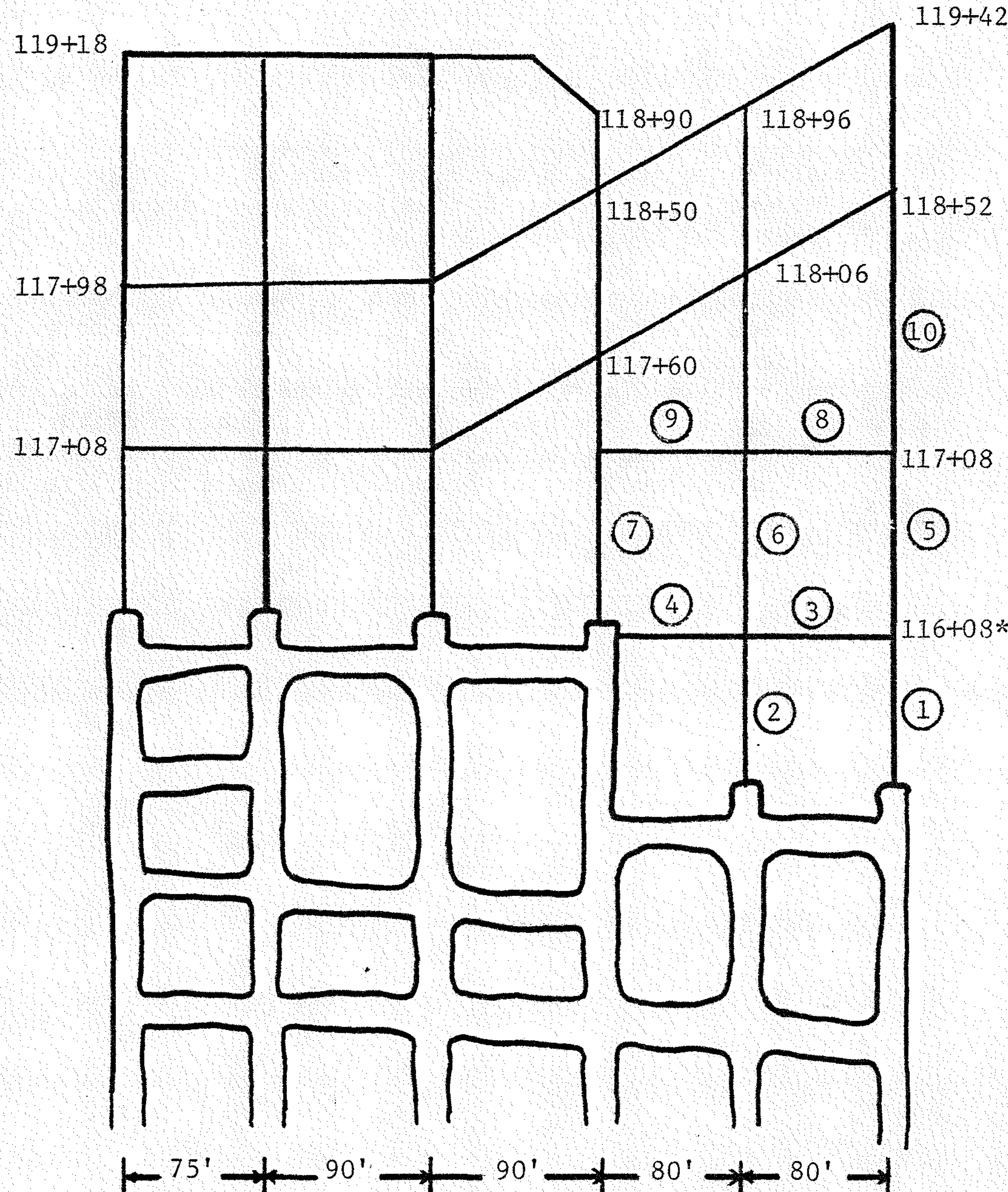
Figure 15

McElroy Mine-1 South Faces Prior to AES Mining.

In addition to the AES, the section equipment included a Joy 14 BU loader with a roof drill mounted for spot bolting (during the first production trial); a spare Joy 6 CM continuous miner; a spot bolter (during the second production trial); two NMSC shuttle cars, each with a rated ten-ton payload; an auxiliary fan; a section pod-type rock duster; and a 36-inch wide conveyor belt. The equipment was in fair condition.

The six-entry cut plan shown in Figure 16 was projected for the AES. Additional projections of this section would require from six to eight months of mining to complete.

The machine production performance presented in this report was observed in the McElroy Mine during 21 designated production shifts out of 71 planned shifts. It is recognized that a substantial additional amount of in-mine experimentation and production is required before any degree of reliable performance evaluation of the AES can be made. However, one major aspect of this study is to establish and test those machine evaluation parameters that are realistic and beneficial in an in-mine performance validation test.



Scale 1" = 100'

◯ = Cut Number

* = Distance from mouth of the section (in feet).

Figure 16

Projected Mining Plan
McElroy Mine - 1 South Face

CHAPTER 2

AES IN-MINE TRIAL DOCUMENTATION

2.1 INTRODUCTION

This chapter presents the comprehensive engineering documentation of the AES in-mine trial at McElroy Mine. Included here are an overview of the performance of the machine and a discussion of the proposed underground testing that was actually carried out. Also included are the data collected during the Full-Time AES Audit, the Worker Opinion Questionnaire Audit, and the Special Audits. In addition to the information presented in this chapter, a detailed commentary of the events of each week is presented in Appendix 5-1. The narratives were taken from the weekly reports presented to the Department of Energy (DOE) Consolidation Coal Co. (Consol) and National Mine Service Company (NMSC). Short synopses are presented in Section 2.4.1 of this chapter.

The in-mine trial was originally scheduled for a five-month operating period. However, special circumstances caused the trial to last nine months, including a three-month miner's strike interval. The events of the in-mine trial have been divided into periods as follows in order to simplify the description of the activities; the dates are inclusive:

- Assembly - April 26 - August 28, 1977.
- First Production Trial - August 29 - October 20, 1977.
- Modification - October 21 - December 6, 1977.
- Strike Interval - December 7, 1977 - April 4, 1978.
- Modification - April 5 - April 10, 1978.
- Second Production Trial - April 11 - May 24, 1978.
- Disassembly - May 25 - September 2, 1978.

Figure 17 is a graphical representation of these periods.

Figure 18 includes additional information regarding each period. Production days are represented by the dots in the production intervals. The accomplishments of the AES during each production trial and the major modifications to the AES are listed on the figure.

Figure 19 shows the advance of the section during the in-mine trial by the AES and a Joy 6 CM which was used during extended AES downtimes. The four areas mined by the AES are designated as the First, Second, Third, and Fourth Cuts on the diagram. Table I lists the activities of the AES at each location on Figure 20 (five pages), which documents the movement of the machine through the section during the in-mine trial.

Table II shows the section equipment and personnel used during the in-mine trial. In addition to these people, there was usually a project engineer from Consol, one or two engineers and a serviceman

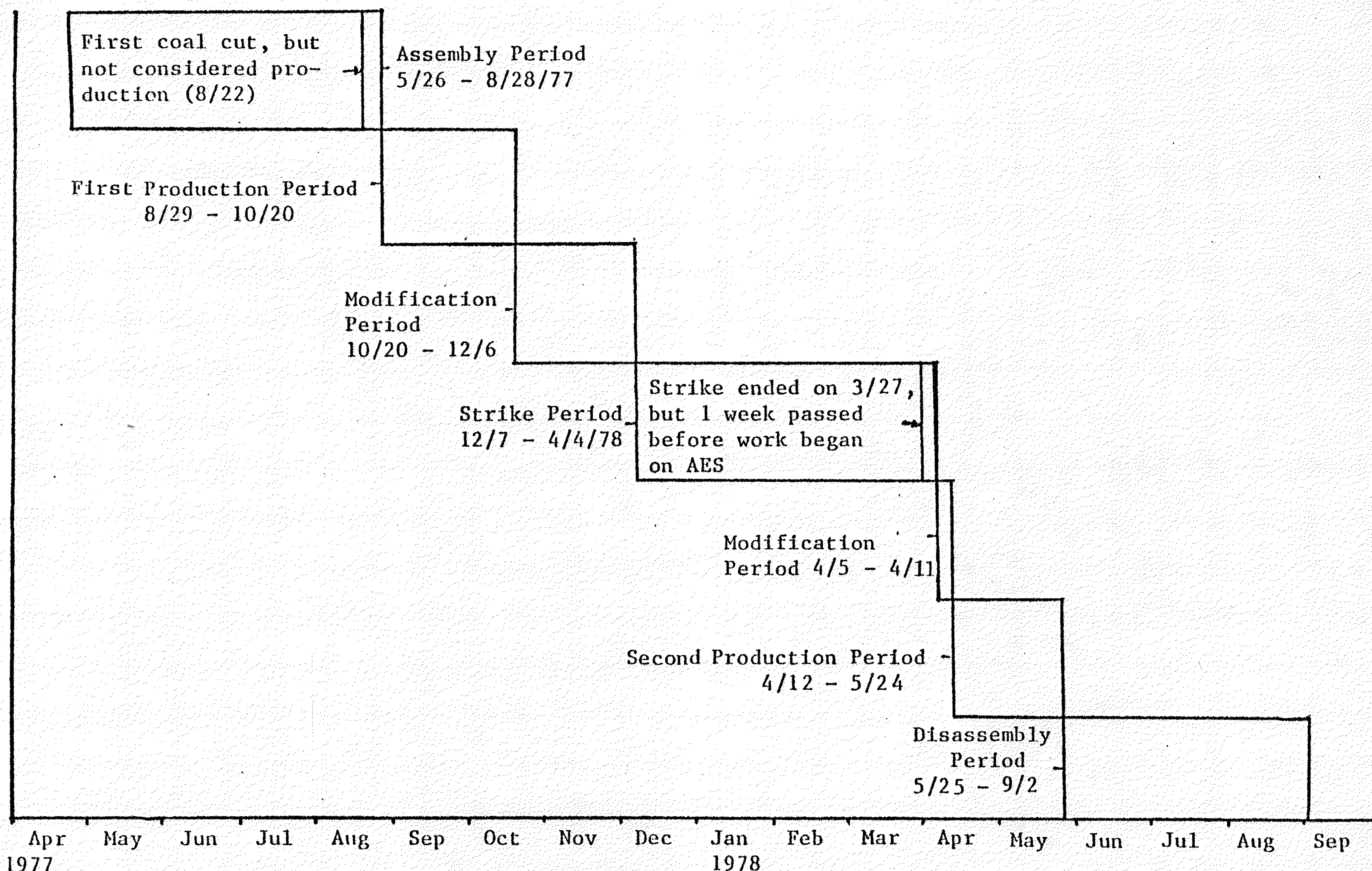


Figure 17
AES McElroy In-Mine Experiment
Time Chart

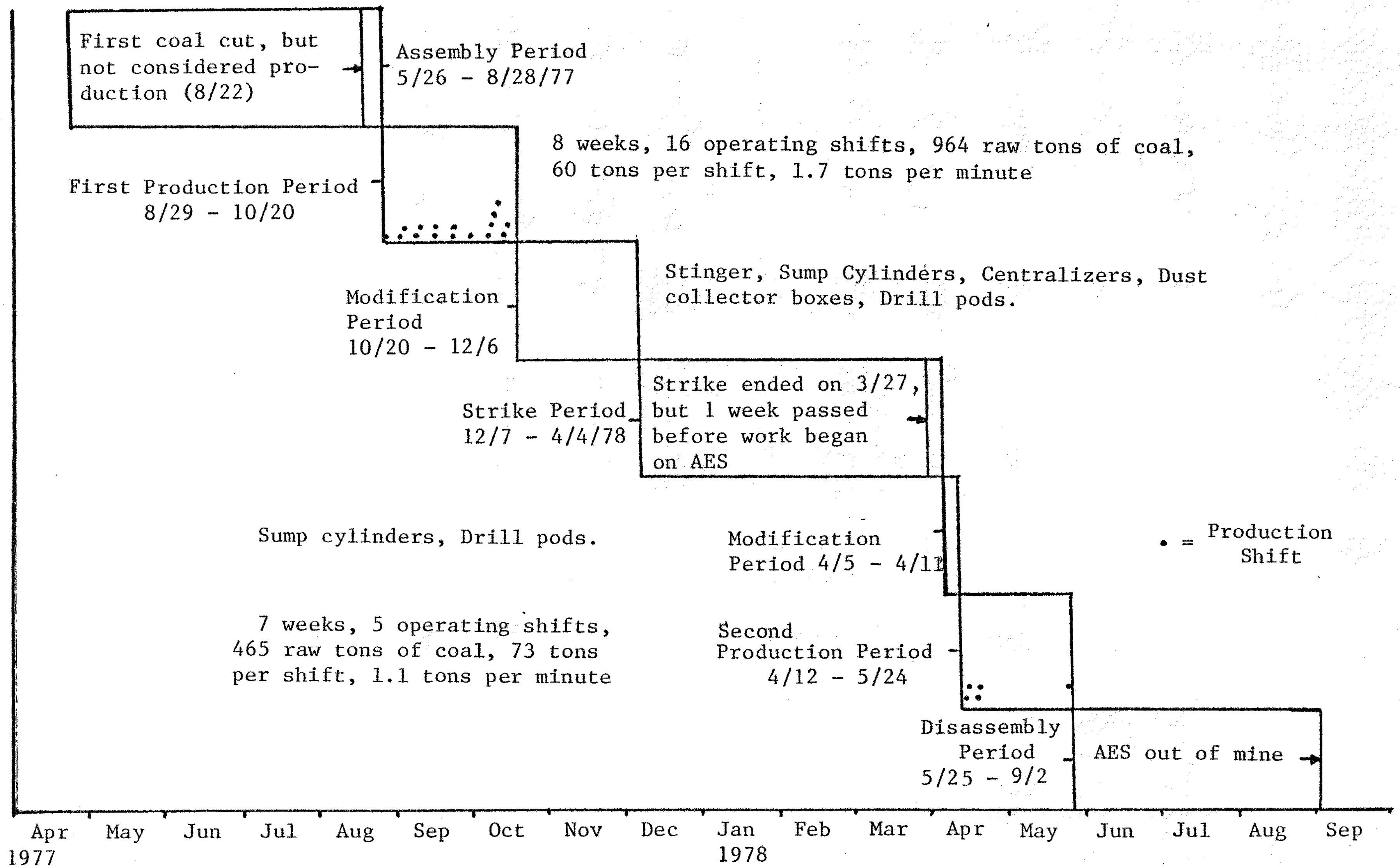


Figure 18
AES McElroy In-Mine Experiment
General Data

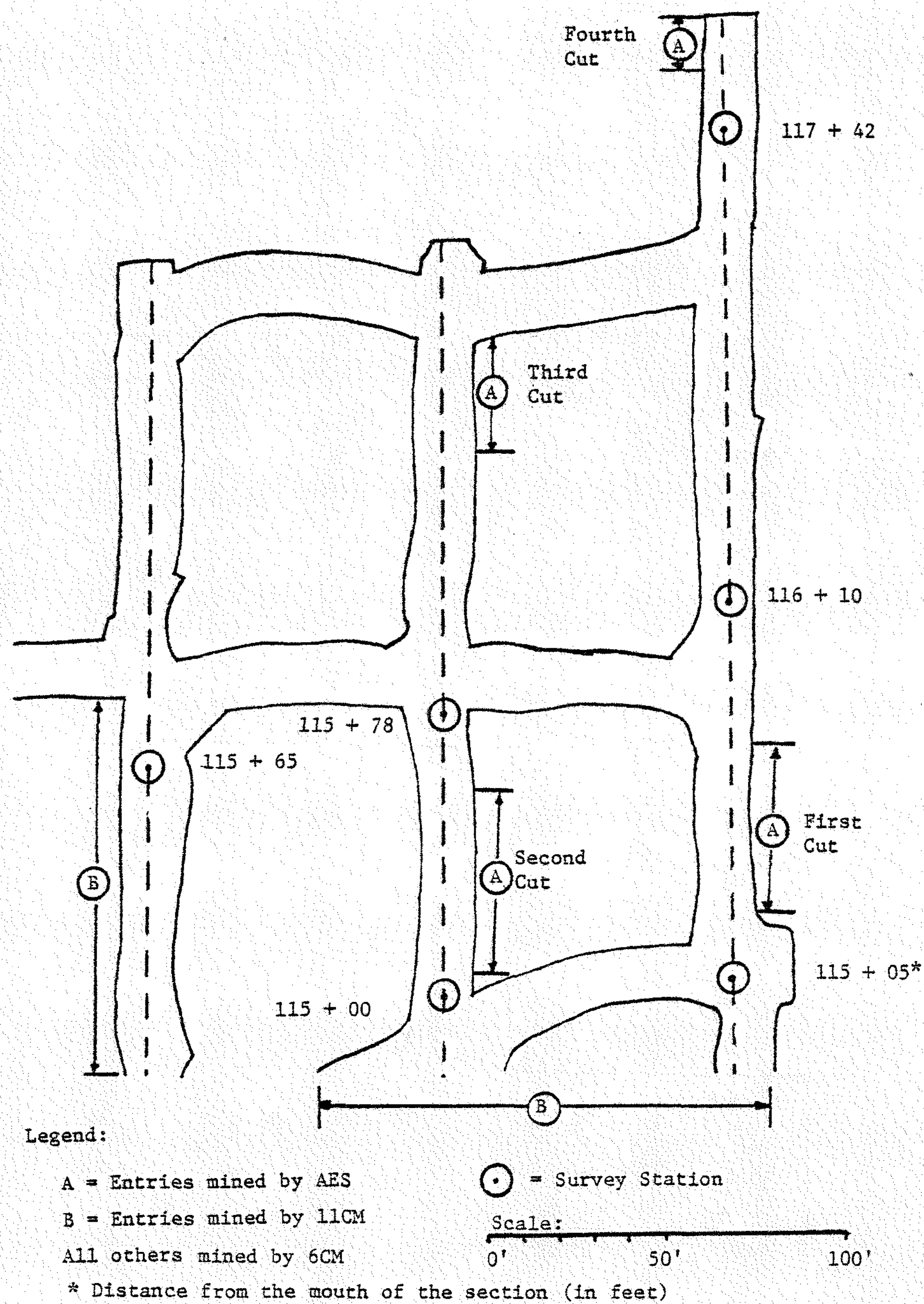


Figure 19

Entries Mined During the In-Mine Trial.

TABLE I

Movement of the AES Through 1 South Face

During the In-Mine Trial
(Refers to Figure 20)

A = ASSEMBLY LOCATION

B = ASSEMBLY AND TESTING LOCATION (Roof Drills and Dust Collector tests)

C = TESTING LOCATION (Automatic Control System tests)

D = READY TO MINE FIRST CUT

E = END OF FIRST CUT

F = READY TO MINE SECOND CUT (1st CUT NOT COMPLETED)

* E - F = PLACE CHANGE

G = END OF SECOND CUT

H = MODIFICATION LOCATION, STRIKE LOCATION

I = READY TO MINE THIRD CUT

J = END OF THIRD CUT

K = WAIT FOR SECTION ADVANCE, PUMP DRIVE GEAR CASE FAILURE

L = READY TO MINE FOURTH CUT

M = END OF FOURTH CUT

N = DISASSEMBLY LOCATION I

O = DISASSEMBLY LOCATION II

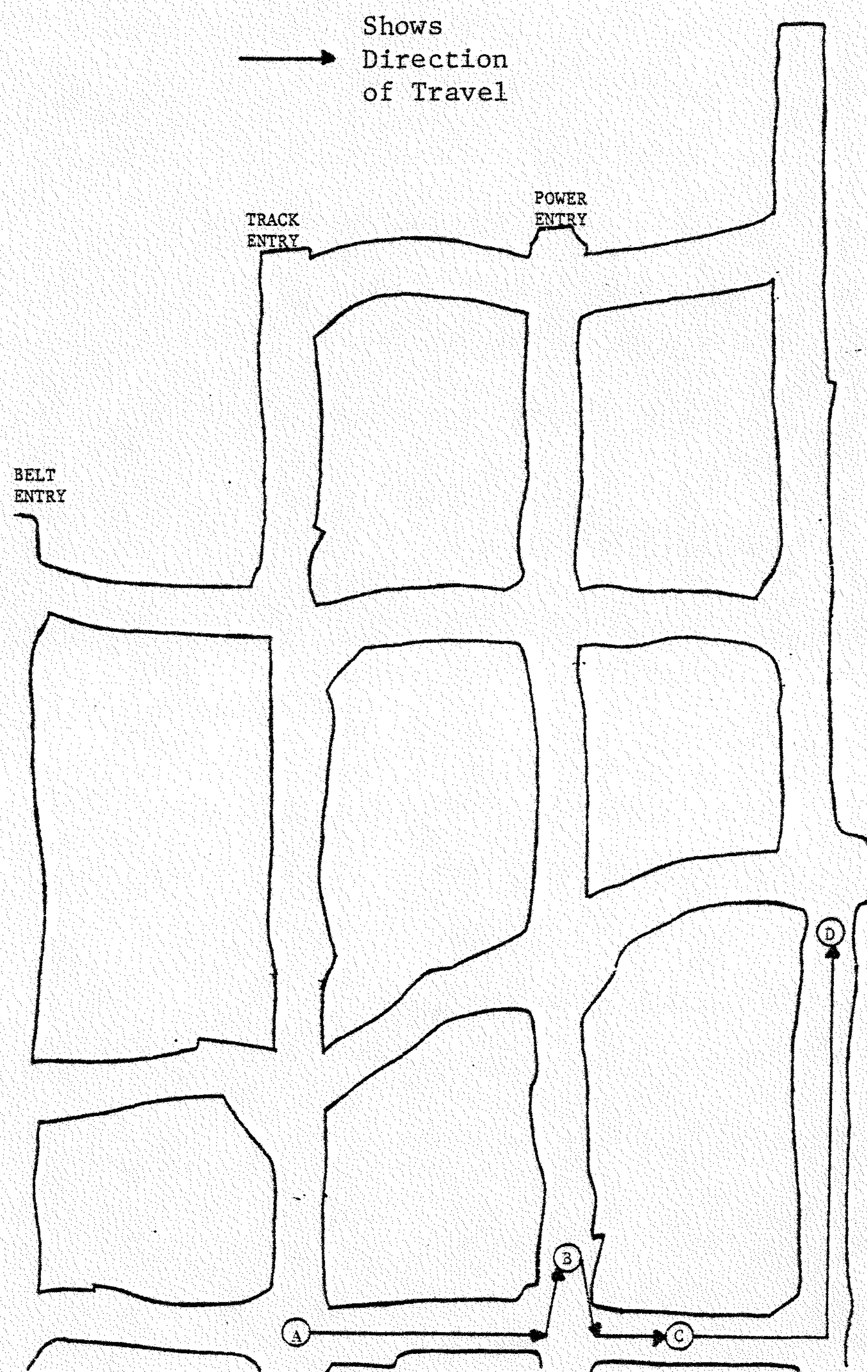


Figure 20

Movement of AES During Assembly Period
(April 26 - August 28, 1977)

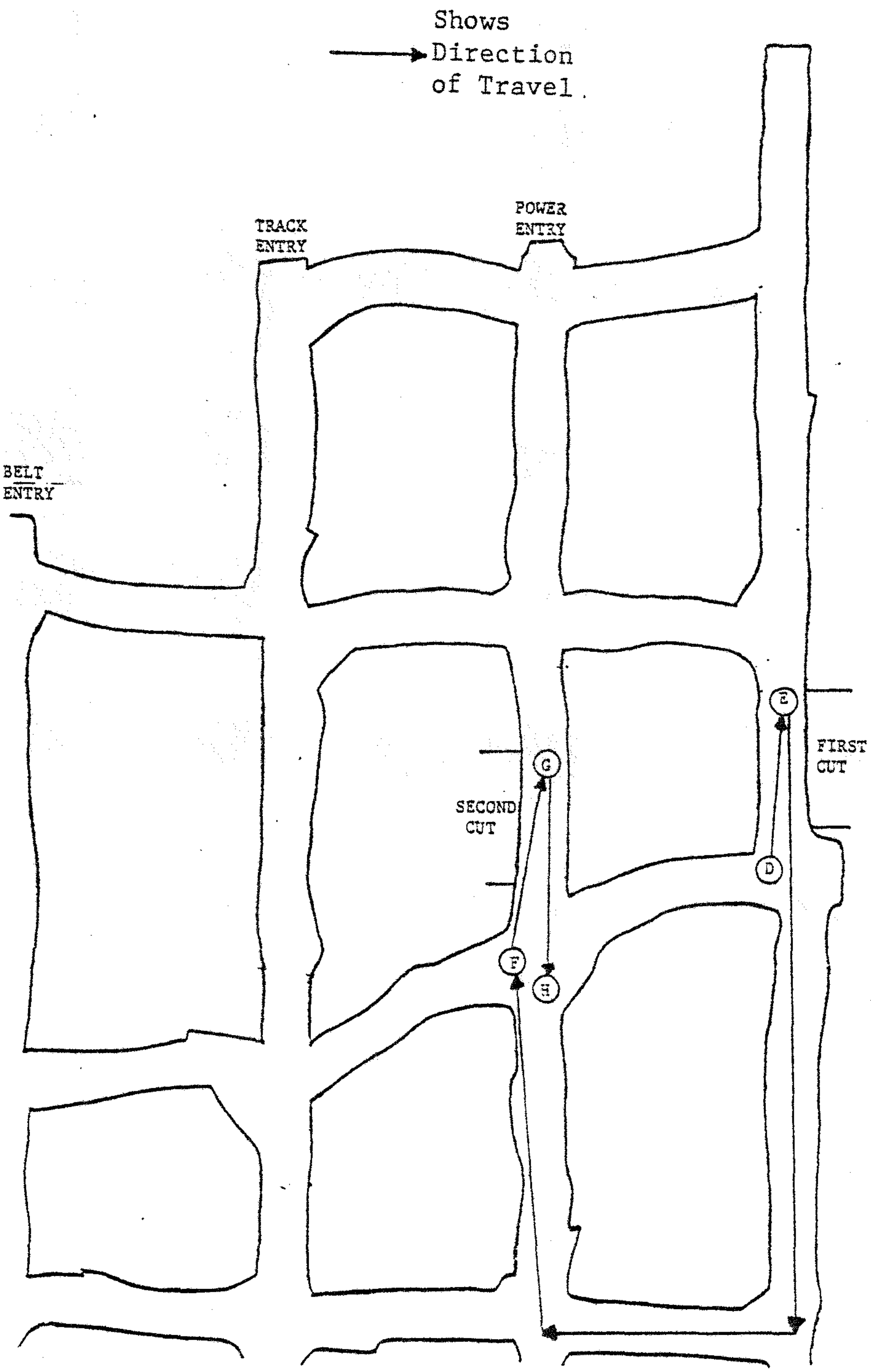


Figure 20 (Continued)
Movement of AES During First Production
Period (August 29 - October 20, 1977)

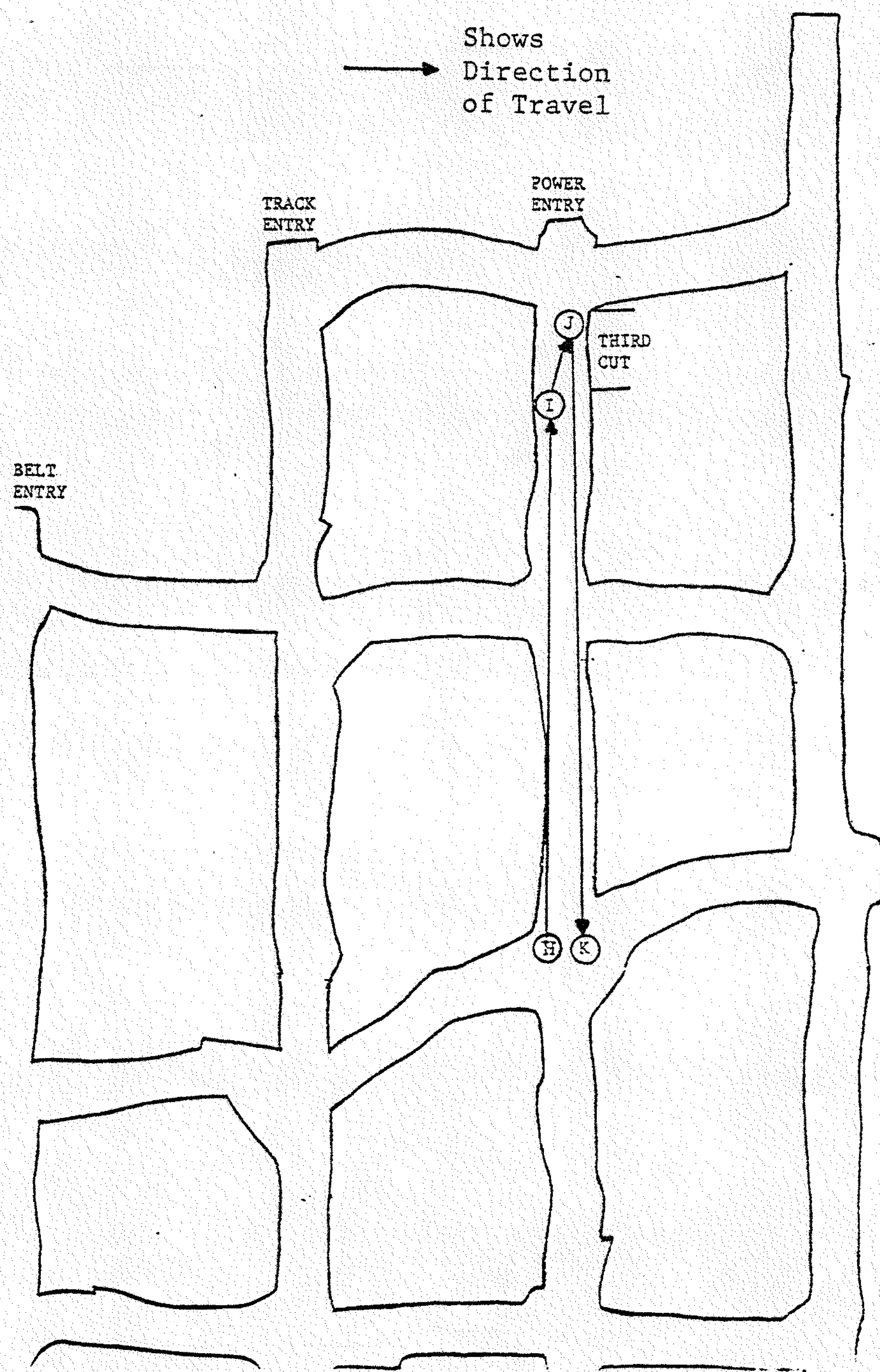


Figure 20 (Continued)
Movement of AES During Second
Production Period (Before Shutdown Period)
(April 13 - May 23, 1978)

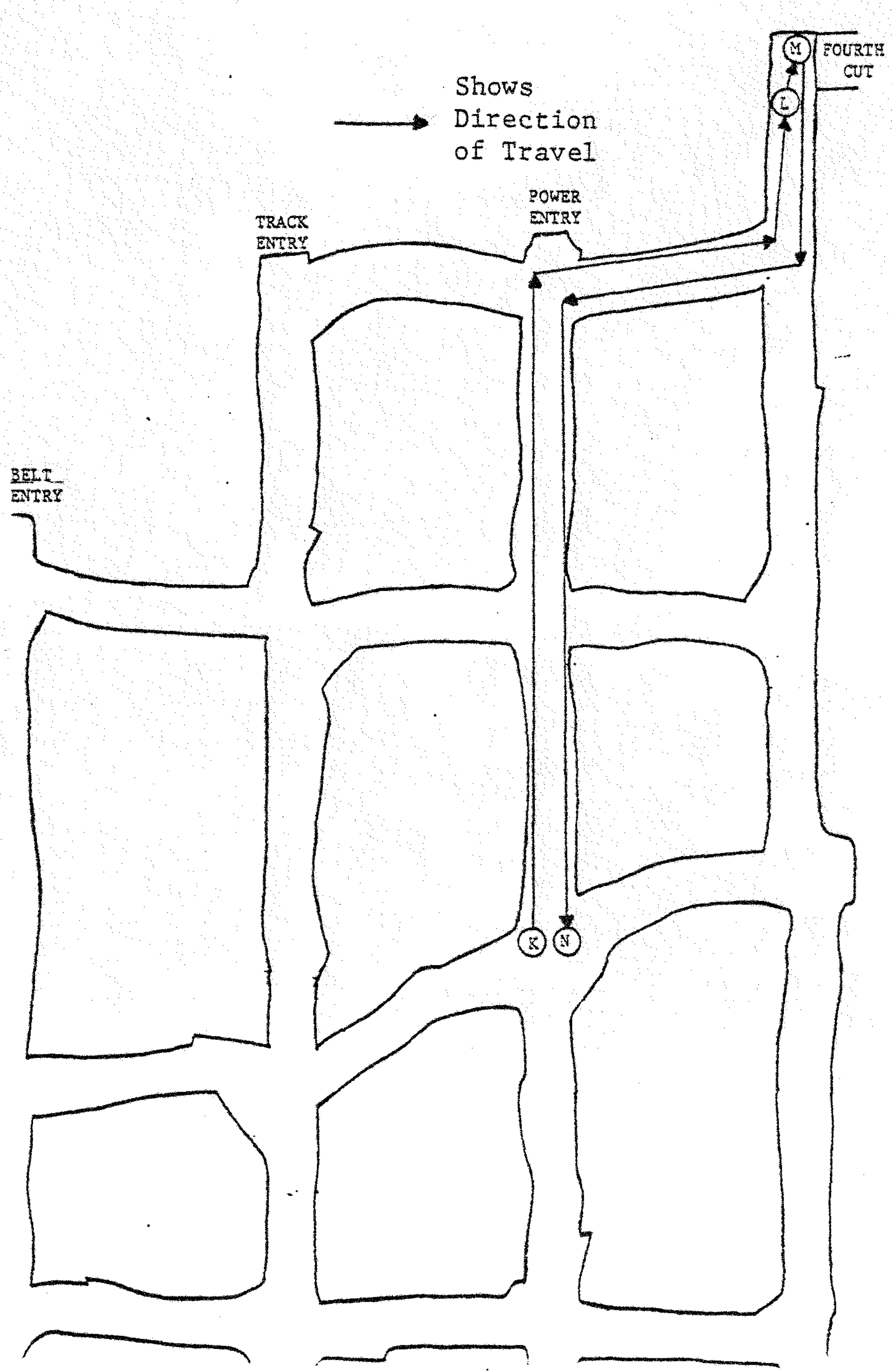


Figure 20 (Continued)
Movement of AES During Second
Production Period (for Special Production test)
(May 24 - May 25, 1978)

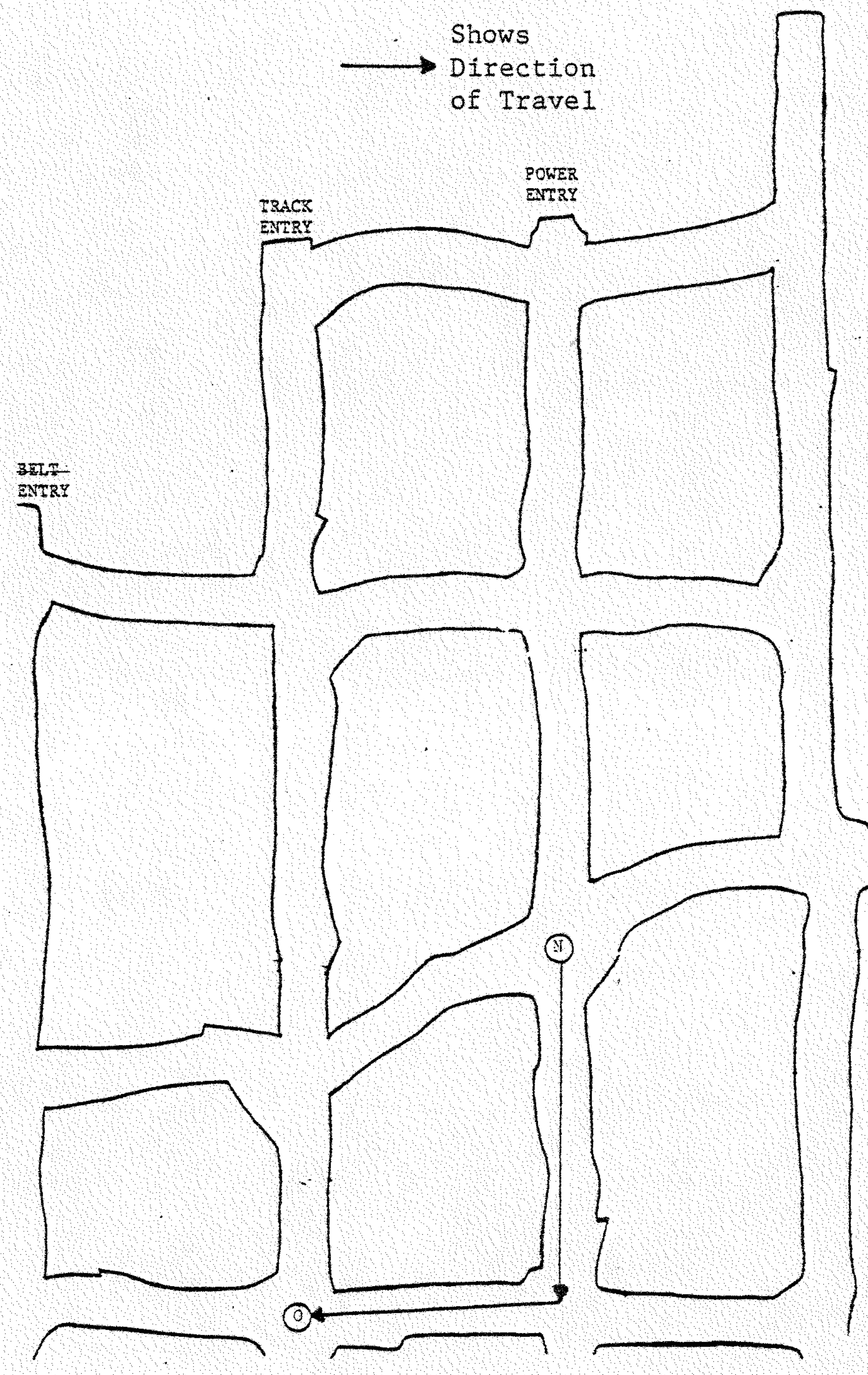


Figure 20 (Continued)
Movement of AES During Disassembly Period
(May 25 - September 2, 1978)

TABLE II

Section Equipment and Personnel Used During the In-Mine Trial

EQUIPMENT	PERSONNEL
NMSC'S AES	AES Operator (1) Bolter Operators (2)
NMSC Shuttle Cars (Torkars)	Shuttle Car Operators (2)
Joy Loading Machine	Loader Operator (1)
	Mechanic (1)
	Mechanic Helper (1)
	Foreman (1)

from NMSC, one U.S. Bureau of Mines (USBM) or DOE engineer, and one or two Penn State observers.

During the in-mine trial, the Full Time Audit Team's observers documented all events on every production shift and on 94 percent of the maintenance shifts. Data on the eight maintenance shifts that were not monitored by the Audit Team observers was supplied by either NMSC or Consol personnel.

2.2 AES PERFORMANCE OVERVIEW

2.2.1 Production

During the in-mine trial, the AES mined 1,429 raw tons of coal during 21 production shifts out of 71 planned shifts. The production rate varied from 0.69 to 2.90 tpm over a total of 1031.74 minutes (cutting-loading, and cleaning and trimming combined). (See Table III). The AES did not operate on many of the scheduled production shifts because either maintenance was required or geological conditions were adverse.

Table IV is a summary of the operating shifts, showing total and average times for each element in each production trial and in the two combined. The percent change between the two trials is also shown. Note that maintenance accounted for 2,588.18 minutes; necessary delays, 1,298.07 minutes; and unnecessary delays, 1,035.17 minutes. However, cut (load) amounted to only 877.86 minutes. Many of the necessary, unnecessary, and maintenance delays were caused by poor roof and floor conditions, and were complicated by the inability of the AES to cope with these problems.

2.2.2 Safety

During the in-mine trial, no accidents were reported. In addition a majority of the workers responded favorably when asked questions regarding the safety additions to the AES. However, there was some concern, on the part of the safety committee, regarding time when each bolter would install two bolts simultaneously. This situation did not occur, but no problems were anticipated by NMSC.

2.2.3 Special Production Test

The first production trial with the AES was stopped while modifications were made. When production resumed, additional problems were encountered with the local geology and the Automatic Control System (ACS). After several days, during which the roof fell on one occasion, the AES was again idled to allow the 6 CM to advance the face of Entry 6 into an area of better operating conditions.

The roof was bolted by the use of only plates against the roof coal, so that the AES' roof-beam assembly could be tested for an entire cut. The idle time was also used by NMSC to repair the ACS and the

TABLE III

Production Tonnage Summary

DATE	PRODUCING TIME (MIN)	TONNAGE	PRODUCTION RATE (TPM)
8/30	31.72	51	1.6
9/6	7.60	0	0
9/7	9.10	0	0
9/13	75.60	120	1.6
9/15	24.50	60	2.5
9/19	27.35	20	0.7
9/20	21.00	30	1.4
9/23	43.90	100	2.3
9/28	28.25	48	1.7
10/5	62.60	75	1.2
10/10	0	0	0
10/11	62.70	125	2.0
10/12	30.90	90	2.9
10/13	49.16	115	2.6
10/14	23.51	50	2.1
10/18	70.60	80	1.1
4/11	85.85*	100	1.2
4/12	67.00**	72	1.1
4/14	74.40**	120	1.6
4/17	82.60**	77	0.9
5/24	153.40***	96	0.6
Total	1031.74	1429	---
Average	49.13	68	1.4

Manual crawler cutting except where otherwise noted.

*Mined coal left on bottom for a distance of 120 ft.

**Crawler assisted sump-cylinder cutting using the roof-beam assembly as an anchor.

***Automatic Control System cutting.

Table IV
Production Element Summary

ELEMENT	FIRST PRODUCTION PERIOD (16 SHIFTS) (MINUTES) TOTAL	AVERAGE	SECOND PRODUCTION PERIOD (5 SHIFTS) (MINUTES) TOTAL	AVERAGE	PERCENT CHANGE	OVERALL (MINUTES) TOTAL	AVERAGE	PERCENT
1. CUT (LOAD)	535.16	33.45	342.70	68.54	+ 104.9	877.86	41.80	8.7
2. PLUGGED	95.01	5.94	—	—	- 100.0	95.01	4.52	0.9
3. ADVANCE MACHINE	5.04	0.32	7.20	1.44	+ 350.0	12.24	0.58	0.1
4. MANEUVER IN CUT	47.28	2.96	12.70	2.54	- 14.2	59.98	2.86	0.6
5. TRAM	141.15	8.82	41.70	8.34	- 5.4	182.85	8.71	1.8
6. BOLTING DELAYS	319.75	19.98	129.50	25.90	+ 29.6	449.25	21.39	4.5
7. MAINTENANCE DELAYS	2212.58	138.29	375.60	75.12	- 45.7	2588.18	123.25	25.8
8. NON-AES MAINTENANCE	140.60	8.79	16.00	3.20	- 63.6	156.60	7.46	1.6
9. SERVICE	157.90	9.87	82.50	16.50	+ 67.2	240.40	11.45	2.4
10. ADJUST ROOF SUPPORTS	—	—	120.40	24.08	—	120.40	5.73	1.2
11. CLEAN AND TRIM	33.50	2.09	120.55*	24.11	+1053.6	154.05	7.34	1.5
12. VENTILATION DELAYS	—	—	26.50	5.30	—	26.50	1.26	0.3
13. PREPARE TO START	435.72	27.23	217.90	43.58	+ 60.0	653.62	31.12	6.5
14. PREPARE TO LEAVE	259.60	16.23	23.10	4.62	- 71.5	282.70	13.46	2.8
15. LUNCH	498.64	31.17	60.00	12.00	- 61.5	558.64	26.60	5.6
16. MANTRIP IN	611.40	38.21	180.00	36.00	- 5.8	791.40	37.69	7.9
17. MANTRIP OUT	351.35	21.96	98.70	19.74	- 10.1	450.05	21.43	4.5
18. NECESSARY DELAYS	1151.32	71.96	146.75	29.35	- 59.2	1298.07	61.81	12.9
19. UNNECESSARY DELAYS	636.97	39.81	398.20	79.64	+ 100.1	1035.17	49.29	10.3
TOTAL	7632.97	477.08	2395.74	479.96	—	10028.71	477.75	100.0

*Includes one shift during which coal left on the bottom by the 6 CM was mined by the AES (85.85 minutes) pump-drive gear-case (several bearings had failed). May 24 was set as a special production day which would be used to evaluate alternatives regarding the continuation of the in-mine trial at McElroy.

Under starting conditions that were the best to date, the AES advanced 15.5 feet in ACS mode, mining 96 tons of coal in 139.6 minutes. The shift time available for mining (480 minutes minus all fixed delays) was 367.9 minutes. The top cut by the AES was smooth, the entry was straight, and the horizon controls appeared to work. However, the existing roof coal outby the newly mined area was disturbed by the action of the AES' roof beams (See Figure 21). The AES used about 165 gallons of oil, and the floor beams began to slip after the second sump. They continued to slip during the entire shift, which severely limited the production rate (0.69 tpm). The rate was already set low to observe the ACS operation.

On the following day, it was decided to remove the AES from the mine.

2.3 UNDERGROUND TESTING

Table V shows the tests that were performed during the in-mine trial, and Table VI shows the tests that were not. The failure to complete some of the special tests is attributed to several factors, including: 1) the hazardous nature of the mine roof over the AES, 2) the unknown hazards inherent with any unproven, prototype machine, 3) the attempt to keep the crowded face area as uncongested as possible (the Audit Team observers kept surveillance over the AES from a point outby the loading machine), and 4) the termination of the experiment.

Because the operation of the AES was inconsistent and limited, the Team gave the Full-Time Production Audit top priority. Also, the Team realistically acknowledged that initiating some parts of the Special Audits, which would have required very close observations of the AES and its crew, would possibly have interfered with the smooth operation of the machine. Thus, the tests were postponed until the AES could hopefully function more productively and in better roof conditions. This circumstance did not materialize.

2.4 FULL-TIME AES AUDIT

2.4.1 Synopsis of In-Mine Events

2.4.1.1 Assembly - April 26 - August 28, 1977: No major problems were encountered during actual transportation of the AES parts into the mine except the smashing of a bolter control panel. However, the reassembly of the major components within the confines of the underground section caused handling and orientation problems.

A stress crack, which developed because unusually high pressures were created when the beams were pressed against the roof over one

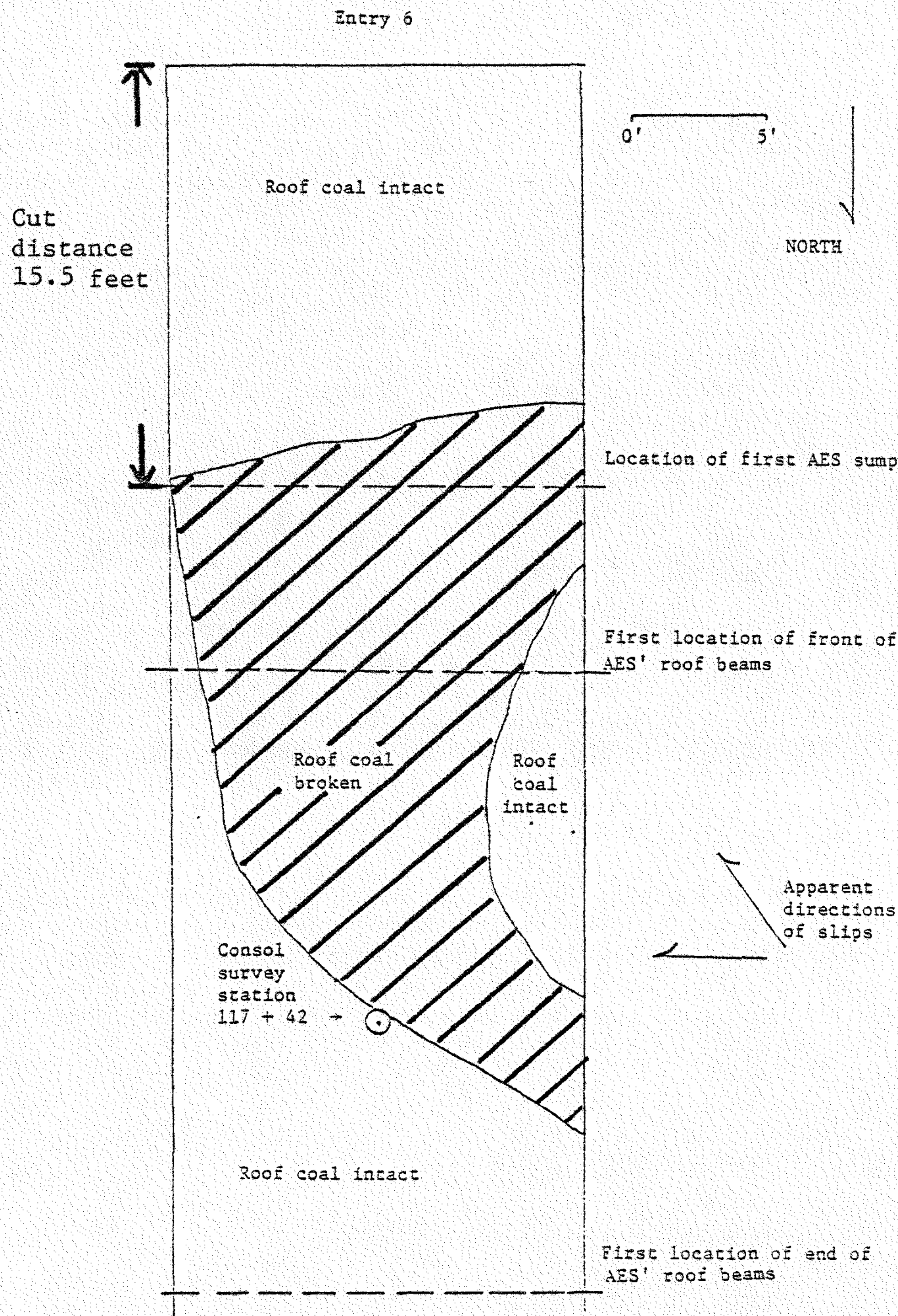


Figure 21

Roof Deterioration in Face of Entry 6 - Influenced by Action of Roof-Beam Assembly on May 24, 1978.

TABLE V

Tests Performed During The In-Mine Trial

TEST	DEGREE OF COMPLETION
1. FULL-TIME AES AUDIT	
Record of all events (Production and Maintenance	Complete
Identification of all elements	Complete
Design of time and study forms	Complete
Recording of detailed mining cycle	Complete
Recording of part, supply and labor costs	Complete
2. Worker Opinion Questionnaire Audit	Complete
3. SPECIAL AUDITS	
Record of all accidents	Complete
Roof and floor profile measurements	Complete
Record of roof falls, cracks, slickensides, joints, faults and other geologic data	Complete
Centerline and rib deviation measurements	Complete
Correlations of poor roof with centerline and rib deviation	Complete
Air velocity measurements at discharge end of tubing	Partially Complete
Methane measurements at discharge end of tubing	Partially Complete
Personal dust sampler measurements	Complete
Record of tubing condition	Complete
Record of non-AES face ventilation during non-productive time	Complete
Photometer measures outby the face	Complete
Noise dosimeter measurements	Partially Complete
Observations of bolting platform movement	Partially Complete
Observations of tramming through crosscuts	Complete
Cleanup measurements	Partially Complete
Record of clean and trim time	Complete
Out-of-seam dilution measurements	Complete

TABLE VI

Test Proposed but Not Performed By the Audit Team During the In-Mine Trial

TEST	REASON WHY NOT PERFORMED
SPECIAL AUDITS	
Extensometer measurements	Limited operating time, hazards to observers
Record of the amount of off-board guidance for machine alignment	Limited operating time, hazards to observers
Time studies of workers under canopies	Limited operating time, hazards to observers
Air velocity readings at the AES*	Limited operating time, hazards to observers
Methane readings at the AES*	Limited operating time, hazards to observers
Dust sampler readings at the operator's location, in intake air, in the various operating modes*	Limited operating time, hazards to observers
Air pattern determinations using smoke tubes	Limited operating time, hazards to observers
Photometer readings at the face	Limited operating time, hazards to observers
Noise recordings*	Limited operating time, hazards to observers
Bolting platform stability measurements drawing, or time studies	Limited operating time, hazards to observers
Crosscut turning observations, drawings, or time studies	No crosscuts mined
Cleanup measurements	Limited operating time, hazards to observers
Detailed bolting cycle time recordings	Limited operating time, hazards to observers

* In these items a very limited sampling of information was taken by others or started by Audit personnel.

weekend, was found in the underside of the roof beam behind the left ventilation-fan, and had to be repaired. Major oil leaks were found in the lower oil-reservoir and had to be corrected. Both the methane monitor (which was damaged) and the two sump-cylinders (whose rods bent during initial troubleshooting operations) had to be replaced.

The full production crew that was assigned to the AES had to be familiarized on the job with the machine's controls. Also, early attempts to initiate production were hampered by maintenance problems and by operator unfamiliarity with the AES.

<u>2.4.1.2 First Production Trial, August 29 - October 20, 1977</u>: During the first week of production, about 51 raw tons of coal were mined on one shift. Shortly after that shift, however, the cutter head tie-shaft and retaining ring failed due to a design error.

During the second production week, the tonnage mined was minimal because two gathering-head motors failed, at the time considered to be caused by incorrect overload settings.

Two production shifts did yield 180 raw tons of coal during the third week, but a fall of the roof in Entry 6 during this time hampered production. Then, a gathering-head float-control problem had to be solved.

About 50 raw tons of coal were mined on two production shifts during the fourth week. A third gathering-head motor failure and associated stalling was responsible. The failure caused was then attributed to either 1) excessive water pressure not allowing recirculation through the motor casings or 2) the sideboard extensions were being pressed against one disc, causing the motor to stall. Then, another maintenance problem arose when the motor seal failed on the left ventilation fan. A second small fall occurred in Entry 6.

During the fifth week, approximately 148 raw tons of coal were mined during two production shifts. However, poor top- and bottom-conditions necessitated moving the AES to Entry 5. Then, the left ventilation-fan motor failed again; the cause was determined to be an undersized relief-valve spring that allowed excess pressure to accumulate and cause seal failure and motor cavitation.

During the sixth week, 75 raw tons of coal were produced on one shift. The left-front inner-roof-support cylinder had to be replaced because the piston port threads were stripped. Also, the Sundstrand variable-volume pump failed because of a defective bearing on the pump-drive gear-case; the bearing bore was enlarged to correct the problem. Another sump cylinder had to be replaced because it had a bent rod. The seals on the water pump failed, and the unit had to be replaced.

During the seventh week, four production-shifts yielded 360 raw tons of coal. However, the sump-cylinder motion-control valve had to be replaced. Floor conditions began to worsen, and some water sprays were plugged to decrease the amount of water placed on the floor. And on one shift, 10 bolts were required in newly-exposed roof (the most by the AES on a single shift).

On the eighth week (the final week before the modification period), 117 raw tons of coal were produced on two shifts. But, a major maintenance problem occurred when a crack developed in the stinger frame which required over 18 manhours to repair. Also, a major geological problem was created when the roof fell at the end of the week.

Consol decided to advance the section beyond the area of poor geological conditions with a Joy 6 CM miner, and NMSC used the available time to modify the AES to alleviate some major design-problems. NMSC had already begun to modify the roof-beam assembly by retrofitting controls to make the inner roof-beam independently adjustable in both the front and back.

2.4.1.3 Modification - October 20 - December 6, 1977; and April 4 - April 10, 1978: A total of 662.24 manhours was required to modify the AES during this period. The major modifications were:

- Relocation of the drill dust-collector boxes.
- Revalving of the drill centralizers.
- Installation of deep-chuck drill pods.
- Redesign of the stinger.
- Plugging of water sprays.
- Redesign of the guide slide.
- Redesign of the sump cylinders.
- Minor changes to facilitate other modifications.

The dust-collector boxes were relocated because they became caught on the outer roof-support cylinders. The drill centralizers and drill pods were replaced with better units to make their operation safer. Because the stinger was redesigned to aid tramming and to shift the center of gravity from the front of the crawler pads to near the middle of the pads, parts of the machine-floor problem was solved. Also, additional water sprays were plugged to further alleviate some of the floor problems. The sump cylinders and guide slide were modified to prevent bending of the sump cylinder rods.

The deep-chuck drill-pod modification that was begun during the modification period had to be completed during the second production trial because not all of the drill units were ready for installation during the modification period.

2.4.1.4 Strike Interval, December 6, 1977 - April 4, 1978: During the in-mine trial, the contract for the United Mine Workers of

America expired, halting work at the mine until March 27. Work on the AES was delayed until April 5 to prevent interference during the back-to-work period.

2.4.1.5 Second Production Trial - April 10 - May 24, 1978: When AES production resumed, the inconsistent operation continued. During the first week, 172 raw tons of coal were mined on two production shifts. About 100 tons were mined on the first shift because 120 feet of coal had been left on the bottom by the 6 CM. During the next production shift, the roof began to improve, and there were no major maintenance delays. A small crack that developed in the new stinger was corrected.

During the second week, however, although two shifts produced 197 raw tons of coal, another gathering-head motor failed and delays caused by bolting (4 bolts, 75.80 minutes) became excessive because of operator inexperience with the controls, the steels becoming stuck in the holes, and the centralizer arms requiring adjustment. Also, the roof fell in Entry 5, where the AES was mining, because an 11-foot span of roof was left unsupported for three days. The area had been mined with the use of the AES' roof supports, and the pressure from the roof-beam assembly may have affected the span. This fall caused several maintenance problems on the AES which had to be corrected later.

The AES was again idled to allow the 6 CM to prepare the face of Entry 6 for the May 24 special trial. The shutdown time was also used to correct some maintenance problems and to get the Automatic Control System (ACS) into working order. During the third week, work on the AES included adding conveyor-lift-cylinder stabilizer bars, attaching one deep-chuck drill, and trouble shooting the ACS.

During the fourth week, it was discovered that the bearings in the pump-drive gear case had failed reportedly because of undersized bores of the pistons. The four main-pumps were replaced, and the gear case was replaced during the fifth week.

During the sixth week the AES was prepared for production. The ACS and a stinger malfunction were repaired. When the AES mined 96 raw tons on May 24 (see Section 2.2.3, Special Production Test), the ACS was used for the first time. However, AES-roof problems, chock slippage, and extensive oil leaks halted production. Consol, NMSC, and DOE personnel met on the following day and decided to disassemble and remove the AES from McElroy Mine.

2.4.1.6 Disassembly - May 25 - September 2, 1978: During the disassembly period, delays caused by personnel availability and AES parts transportation problems, and the intervening miner's vacation, considerably extended the disassembly period.

During the disassembly, the roof-beam assembly was dismantled, and it was found that two roof-support cylinders had been pulled out of their lower housings. The remainder of the components (cutter head, gathering head, roof beams, and conveyor did not appear to be damaged. However, several major oil leaks were repaired at the start of the period to facilitate tramming.

2.4.2 Delays and Other Factors

2.4.2.1 Roof Control Delays: Time lost because of falls and breakage of the roof totalled 8,983.8 minutes, including 18 production shifts lost to support the fall areas and to advance Entry 6 during the second production trial by the 6 CM. During designated production shifts, the total delay time attributed to roof falls amounted to 1.5 hours because in these conditions the machine was taken off production.

2.4.2.2 Oil Leak Delays: During the 21 production shifts, the total delay time due to oil leaks amounted to 513.51 minutes. One and one-half operating days were lost in looking for the correcting leaks. Approximately 1,540 gallons of oil were used by the AES during the in-mine trial. The sources and causes of the leaks are discussed in Section 2.4.4.2.

2.4.2.3 Tramming Delays: During the 21 production shifts. the AES was trammed in and out of the face 39 times. Tram delays amounted to 182.85 minutes during the in-mine trial. The AES was trammed around ten corners (see Figure 20). Table VII shows a typical move as recorded as either necessary or unnecessary delays amounted to 501.2 minutes, including delays for building bridges over muddy areas, directions, shop talk, handling cables, adjusting the roof-beam assembly for clearance, and machine complexity.

2.4.2.4 Gathering Head and Conveyor Delays: The time that was lost because of the failures of the four gathering-head motors totaled 1,254.3 minutes (excluding wait-for-parts time). Five and one-half operating shifts were lost because of the motor failures. The fact that the gathering head and conveyor stalled was documented 13 times during the Full-Time Audit. However, the event was very commonplace, and the observers did not document every time the breaker tripped, but rather, only when the stalling occurred several times. Usually, only a few seconds were lost each time the gathering head and conveyor stalled.

2.4.2.5 Ventilation Delays: Delays due to ventilation totaled 26.5 minutes. Two days were lost because of the two failures of the left ventilation-fan motor. (See Section 2.4.1.2)

2.4.2.6 Absenteeism: During 21 production shifts, neither the miner operator nor the bolter operators were unnecessarily absent. On one occasion, the miner operator attended a mine rescue class, and the AES was idled. On another occasion, the usual operators were required to work on another section. The mechanic operated the AES, one shuttle car was used behind the loader, and 100 raw tons were produced. However, no bolts were installed during this shift. On other occasions, various other crew members, especially the loader

TABLE VII

Typical Tramming Move

May 23, 1978 - Day Shift

ELEMENT	START	STOP	TIME (min)	REMARKS
Prepare to tram - adjust roof supports, etc.	2.35	2.36	1.0	
Tram	2.36	3.15	39.0	
Maintenance delay	3.15	3.30	15.0	Damaged hydraulic hose
Tram	3.37	3.47	10.0	
Handle cables	3.47	3.52	5.0	
Tram	3.52	3.55	3.0	
Handle cable	3.55	4.27	32.0	

Total time	105.0 min
Tram time (excluding delays)	52.0 min
Tram time, including handing cables	89.0 min
AES traveled from point K to point L (Figure 2-4)	
Approximate distance	300 ft
Approximate tram rate	
Excluding handle cables	5.77 ft/min
Including handle cables	3.37 ft/min

operator or one shuttle-car operator were absent and required replacement workers. Production did not decrease noticeably during these shifts.

2.4.2.7 Possible Pressure by Consol Officials for Production: During the two production trials, Consol officials visited the section on at least five days. Their primary concern during these visits appeared to be with the low production being achieved by the AES and possible causes.

2.4.2.8 Detailed Mining Cycle: Figure 13 shows a simplified version of the cycle elements for ACS operation. Table VIII lists the actual steps that occurred on the May 24 special trail; included are the achieved times.

2.4.3 Production Costs

During the in-mine trial, labor and supply costs were recorded by Consol, including the payroll for hourly and salaried employees, and expenditures for water line, brattice curtain, resin cartridges, five-foot resin bolts, six-inch resin-bolt plates, six-foot conventional bolts (used by stoper crews to support fall areas), and four-inch bolt plates. Consol did not charge some supplies on a per section basis, including ventilation tubing, crib blocks, timber, cap pieces, wedges, point anchors, planks, or rock dust. Therefore, the costs for these supplied are not included.

The cost for labor was $56,173 and the supply cost for the trial was $990. (See Table IX) The total reported operating cost was $57,163.

2.4.4 Maintenance Audit

2.4.4.1 Maintenance Elemental Data: A summary of the maintenance manhours by component is shown in Table X. Maintenance work on the gathering head and conveyor required the highest amount of effort, followed by the roof drills and dust collectors, roof-beam assembly, and the cutter head. These figures represent corrective and preventive manhours with the exception of the modification to the inner-roof-beam assembly (providing independent operation from front-to rear) which was classified as preventive maintenance when it was performed. The "other" maintenance category represents minor and miscellaneous work on electrical or hydraulic systems.

Table XI is a compilation of the maintenance manhours by category, including corrective, preventive, service, and other and non-AES maintenance. "Other" maintenance refers to small jobs, such as "turn the power on" or "get the tools."

The corrective maintenance manhours are further divided in Table XII into hydraulic, mechanical, electrical, lighting, or other

TABLE VIII

Detailed Mining Cycle Times

ELEMENT	AVERAGE TIME (MIN)
1. Raise all supports	0.95
2. Set cutting limits	4.50
3. Start cycle and lower inner supports	0.18
4. Sump	1.37
5. Raise inner supports	0.31
6. Shear	1.56
7. Lower inner supports	0.15
8. Cut cusp	0.39
9. Raise head	0.24
10. Advance head	0.61
11. Repeat 4-10	4.63
12. Repeat 4-10	4.63
13. Repeat 4-10	4.63
14. Wait for bolter consent	0.00
15. Raise inner supports	0.31
16. Lower outer supports	0.18
17. Advance AES (hydraulically)	0.40
18. Delays	1.95
Total	26.99
Estimated Advance	4 feet
Estimated tons per foot of advance	6 tons (ROM)

$$\text{Instantaneous Cutting Rate} = \frac{\text{Tonnage for One Cycle}}{\text{Sump Time + Shear Time}}$$

$$= \frac{6}{1.37 + 1.56} = 2.05 \text{ tpm}$$

$$\text{Average Cutting Rate per Cycle} = \frac{\text{Tonnage for One Cycle}}{\text{Sump Time + Shear Time + Raise Head Time + Cut Cusp Time + Advance Head Time}}$$

$$= \frac{6}{1.37 + 1.56 + 0.24 + 0.39 + 0.61} = 1.44 \text{ tpm}$$

$$\text{Average Cutting Rate per Four Foot Advance} = \frac{\text{Tonnage for Four Cycles}}{\text{Total Time for Four Cycles (including Roof-Beam Assembly Movements)}}$$

$$= \frac{24}{(4)(4.63) + 0.31 + 0.18 + 0.40} = 1.24 \text{ tpm}$$

TABLE IX

Production Costs*

LABOR	HOURS	COST($)
AES operator	256	2152
Bolter operator	480	4010
Shuttle car operator	366	2757
Loader operator	244	1768
Section foreman	244	3050
Technician	1264	8058
Mechanic	3018	24235
Maintenance foreman	704	9240
Stoperman	114	903
Total	6690	56173

SUPPLIES	AMOUNT	COST($)
1" water line	400 ft	488
Bits	14	27
Brattice curtain	3-9 ft rolls	144
Resin cartridges	5 boxes	25
5' resin bolts	25	41
6" resin plates	16	7
6' conventional roof bolts	144	229
4" plates	144	29
Total		990

*Courtesy of Consol

TABLE X

Production Period Maintenance Manhours by Component

COMPONENT	TOTAL MANHOURS	AVERAGE PER WEEK	PERCENT
Roof Drills and Dust Collectors	139.35	9.29	15.7
Cutter Head	125.12	8.34	14.1
Gathering Head and Conveyor	146.07	9.74	16.4
Ventilation and Dust Suppression	89.88	5.99	10.1
Roof Beam Assembly	129.98	8.67	14.6
Tram	9.99	0.67	1.1
Lights	34.05	2.27	3.8
ACS	28.50	1.90	3.2
Methane Monitor	1.14	0.08	0.1
Water	9.20	0.61	1.0
Stinger	42.79	2.85	4.8
Other	133.45	9.68	15.0
Total	889.52		100.0

TABLE XI

Production Period Maintenance Manhours
By Category

CATEGORY	TOTAL MANHOURS	AVERAGE PER WEEK	PERCENT
Corrective	1022.48	68.17	35.8
Preventive	77.12	5.14	2.7
Service	50.40	3.36	1.8
Modification	63.66	4.24	2.2
Other and Non-AES	1644.06	109.60	57.5
Total	2857.72	190.52	100.0

TABLE XII

Classification of Corrective Maintenance Manhours

CLASS	TOTAL MANHOURS	AVERAGE PER WEEK	PERCENT
Hydraulic	379.41	25.29	37.1
Mechanical	253.98	16.93	24.8
Electrical	225.24	15.02	22.0
Lighting	39.38	2.63	3.9
Other	124.47	8.30	12.2
Total	1022.48	68.17	100.0

maintenance. Table XIII shows a similar classification for preventive maintenance.

Finally, the hydraulic element is examined in Table XIV. The categories include hoses, fittings, cylinders, control apparatus and adjustments, pumps, and others.

2.4.4.2 Additional Maintenance Data: A total of 24.1 maintenance manhours was required as a result of falls and breakage of the roof. The fallen material damaged components on several occasions.

A total of 83 oil leaks required 204.9 maintenance manhours, including work on fittings or hoses that were either loose, damaged, had been hit, stretched, pinched, or were the wrong type.

Table XV shows those jobs which required excessive amounts of manhours to complete.

Table XVI shows the comparative complexity of the AES and the Joy 6 CM through a listing of the components for each machine.

Several components that failed were returned (to NMSC plants and to other manufacturers) for analysis purposes, including the pump-drive bearings, roof-drill pump, variable volume pump, gathering head motors, sump cylinders, ventilation-fan motors, and the pilot valves. The report of the findings is presented as Appendix 5-3.

2.4.4.3 Maintenance Costs: The cost of maintaining the AES was determined for both production trials and the modification period. The cost of the major parts supplied by NMSC includes that of motors, pumps, cylinders, etc.

Minor parts (i.e., nuts, bolts, fittings, etc.) are not charged on a per section basis by Consol, so that the cost of these parts was not included in the totals shown here. The cost of maintenance parts used during the production trials was $51,545.85, and the cost of parts used during the modification was $26,496.44, for a total maintenance-part cost of $78,042.29.

The maintenance labor costs for the in-mine trial include both Consol and NMSC manhours. The costs were determined by multiplying a cost per manhour of $8.03 (supplied by Consol) by the total number of maintenance manhours (including NMSC personnel). The labor cost for the production trials was $9,084.59, and for the modification period was $5,317.89. The total labor cost amounted to $14,402.48.

The total cost for maintenance was determined to be $92,444.77.

2.5 WORKER OPINION QUESTIONNAIRE AUDIT

During the in-mine trial, a questionnaire was used to determine the attitudes of the workers toward the AES. The survey was conducted at

TABLE XIII

Classification of Preventive Maintenance Manhours

CLASS	TOTAL MANHOURS	AVERAGE PER WEEK	PERCENT
Hydraulic	63.08	4.21	81.8
Mechanical	3.10	0.21	4.0
Electrical	8.29	0.55	10.7
Lighting	0.00	0.00	0.0
Others	2.65	0.18	3.4
	77.12	5.14	100.0

TABLE XIV

Hydraulic Maintenance Manhours*

CATEGORY	TOTAL MANHOURS	AVERAGE PER WEEK	PERCENT
Hoses	30.81	2.05	6.2
Fittings	85.15	5.68	17.3
Cylinders	69.44	4.63	14.1
Control Apparatus, Adjustments	175.80	11.72	35.6
Pumps	47.68	3.18	9.7
Others	84.58	5.64	17.1
	493.46	32.90	100.0

*Modification manhours included.

TABLE XV

List of Maintenance Jobs Requiring Abnormal Manpower

MAINTENANCE JOBS	MAINTENANCE MANHOURS	REMARKS
1. Leak on Outer Bolter - Left Side (8/29)	3.0	Get fittings from outside
2. Repair the Conveyor Take-up Cylinder (8/29)	11.7	Inexperienced personnel
3. Installing Vent Fan in R.H. Roof Beam (8/29)	10.0	An improved motor mount is needed to facilitate more rapid maintenance
4. Replacing Cutter Head Tie Shaft (8/30 to 9/2)	49.9 (46.6 hrs)	Delayed 34.3 manhours (1440 min) waiting for new shaft
5. Setting Instantaneous Electrical Circuit Breakers on AES for Gathering Head Motor (9/12)	12.1	Workmen not familiar with settings required
6. Change fittings on bottom from Left Roof Support Cylinder (9/9)	10.5	Accessibility problem
7. Change R.H. Gathering Head Motor (9/7)	62.0 (21.3 hrs)	Delayed 51.2 manhours waiting for replacement motor
8. Change R.H. Gathering Head Motor (9/20)	48.1 (30.0 hrs)	Delayed 37.3 manhours waiting for replacement motor
9. Replace Vent Fan Motor on L.H. side (9/26 & 9/27)	13.4	Same as #3
10. Remove R.H. Front Inner Roof Support Cylinder and replace fitting on bottom (9/29)	17.5	Poor design Poor accessibility for maintenance work
11. Replace L.H. Vent Fan motor (9/28 & 9/29)	9.8	Same as #3 and #9
12. Replace L.H. Front Inner Roof Support Cylinder (leaking) install new cylinder	40.0	Accessibility problems
13. Repairs to pump drive gear case - replace bearings and Sundstrand Pump (10/1 & 10/3)	48.2	Waiting for replacement parts
14. Broken Frame (stinger)	18.4	Extensive welding - inexperienced welders
15. Replace 3/4" x 10" Hyd. Hose in sump cylinder circuit (10/10)	5.2	Difficult to reach no room to make repairs
16. Repair Motion Control Valve (10/12)	16.4	Location of valve dictates difficulty to perform maintenance
17. Repairs to Stinger Torque Motor	20.3	Difficult to reach - recurrent problems

TABLE XVI

Major Components on the AES and Joy 6 CM

AES COMPONENTS *	6 CM COMPONENTS
Conveyor	Conveyor
Cutter Head	Ripper Bar
Gathering Head	Gathering Head
Crawlers	Crawlers
Motors (7)	Motors (8)
Pump (1)	Main (2)
Cutter Head (2)	Conveyor (2)
Gathering Head and Conveyor (2)	Traction (2)
Dust Collector (2)	Pump (1)
Roof Drills (4)	Gathering Head (1)
Dust Collectors (4)	Roof Drills (2)
Methane Monitor	Dust Collectors (4)
Roof Supports (3)	Methane Monitor
Roof Support Cylinders (10)	
Ventilation Fans	
Dust Scrubbers	
Water Particle-Removal System	
Automatic Control System	
Area Lights (12)	
Sump Cylinders (2)	

Note: The AES has a more extensive hydraulic system compared to the 6 CM.

*See Appendix 5-5 for additional details.

the beginning and at the end of the trial period, and consisted of informal talks with each of the eight workers individually. The answers are presented here.

2.5.1 Effect of Geologic Conditions on the AES

Half of the workers stated that the AES-roof interaction did not allow the AES to perform satisfactorily. All of the workers said that the AES affected the roof adversely. Half of the workers said they thought the AES would work if there were different roof-and floor-conditions, but also said that some changes in the machine were required to improve its operation.

2.5.2 Effect of Machine Size and Weight

All of the workers interviewed stated that the size and bulk of the AES adversely affected the operator's view of the face and cutting components. Half of the workers said that place changing was not more difficult than with other machines, but one-fourth said that it was harder. All of the workers said that the modifications of the stinger aided tramming, and 37.5 percent stated that the AES was too heavy for the 1 South section bottom.

2.5.3 Gathering Head and Conveyor

Half of the workers interviewed stated that the conveyor did not stall because of excess material (to their knowledge), but 37.5 percent said that it did and that the speed was too slow. An increase in the depth of the conveyor was recommended by 12.5 percent of the crew.

2.5.4 Ventilation

Half of the workers stated that the ventilation system was better and easier to use than other systems. The other half did not comment on ventilation.

2.5.5 Noise

Three-fourths of the workers felt that the noise produced by the AES was worse than that of other machines in that it caused more discomfort.

2.5.6 Operator Comfort and Ability to Store Materials

Three-fourths of the crew said that there was enough room for the operators to work, but 62.5 percent said that there was not sufficient room for supplies. Three-fourths of the workers said that the controls were properly placed.

2.5.7 Lighting

An improvement in lighting over other machines was indicated by 87.5 percent of the workers.

2.5.8 General Component Safety

Table XVII shows the workers' opinions of the safety of various components on the AES in comparison with that of other machine components, including the roof-beam assembly, roof bolters, ventilation, water sprays, dust scrubbers, dust collectors (bolters), lights, ACS, umbilical tram, sump cycle, gathering head, and the conveyor.

2.5.9 Training

The questionnaire was also used to determine the amount of experience and training that members of the crew received both 1) prior to working on the AES and 2) for the AES itself. The responses are shown in Table XVIII. With the exception of the maintenance and section foreman, on-the-job training (OJT) was the only type received prior to working on the AES. OJT was also the only type of training that the operator and bolters received to prepare them for working on the AES.

With the exception of the operator and the mechanic, no further training was desired by the workers. The operator stated that simulated controls would aid in familiarizing him with the AES prior to production-mode operation. The mechanic stated that a more detailed knowledge of hydraulics, electricity, electronics, and mechanics than the training he had received was necessary.

2.6 SPECIAL AUDITS

2.6.1 General Safety Audit

The object of this audit is to calculate the frequency and severity rates for the accidents occurring on the AES section. However, no accidents were reported to the foreman during the operating periods. The workers said that most of the systems on the AES were either as safe as or safer than those on other machines upon which they had worked (see Table VII).

2.6.2 Roof Support and Strata Control Audit

The object of this audit is to evaluate the effectiveness of the temporary and permanent roof-support provided by the AES.

2.6.2.1 Roof Profiles: A set of levels were run throughout the entries mined by both the AES and the 6 CM. Readings were taken on the roof at ten-foot intervals except where obvious roof deterioration required more frequent measurements. Readings were taken at five-foot intervals at those places. Figures 22, 23, and 24 show Entries 4, 5, and 6, respectively. The floor profile for each entry is also shown,

TABLE XVII

Worker Questionnaire Component Safety Response*

COMPONENT	SAFER THAN ON OTHER MACHINES	NO DIFFERENCE	LESS SAFE THAN ON OTHER MACHINES
Roof Beam Assembly	87.5	0	12.5
Roof Bolters	62.5	12.5	25.0
Ventilation	50.0	37.5	12.5
Sprays	50.0	50.0	0
Scrubbers	12.5	25.0	0
Dust Collectors	12.5	50.0	37.5
Lights	87.5	0	12.5
ACS	50.0	0	12.5
Umbilical Tram	75.0	25.0	0
Sump Cycle	0	12.5	0
Gathering Head	12.5	87.5	0
Conveyor Swing	12.5	87.5	0

*Figures shown are percent of workers responding

(In some cases, not every worker responded to a certain question, usually due to lack of knowledge about a certain component. Thus, some of the figures do not total 100 percent for some components.)

TABLE XVIII

Worker Questionnaire Training Response

POSITION	MINING EXPERIENCE (YEARS)	PRE-AES TRAINING	AES TRAINING	INDICATED NEED FOR ADDITIONAL TRAINING
Maintenance Foreman	13.0	various schools	OJT	No
Section Foreman	10.0	periodic	None	No
AES Operator	13.0	OJT*	OJT	Yes
Bolter Operator	3.5	OJT	OJT	No
Bolter Operator	3.5	OJT	OJT	No
Mechanic	7.5	OJT	OJT	Yes
Loader Operator	4.0	OJT	N/A**	N/A
Shuttle Car Operator	5.0	OJT	N/A	N/A

*OJT = On-the-job training
**N/A = Not Applicable

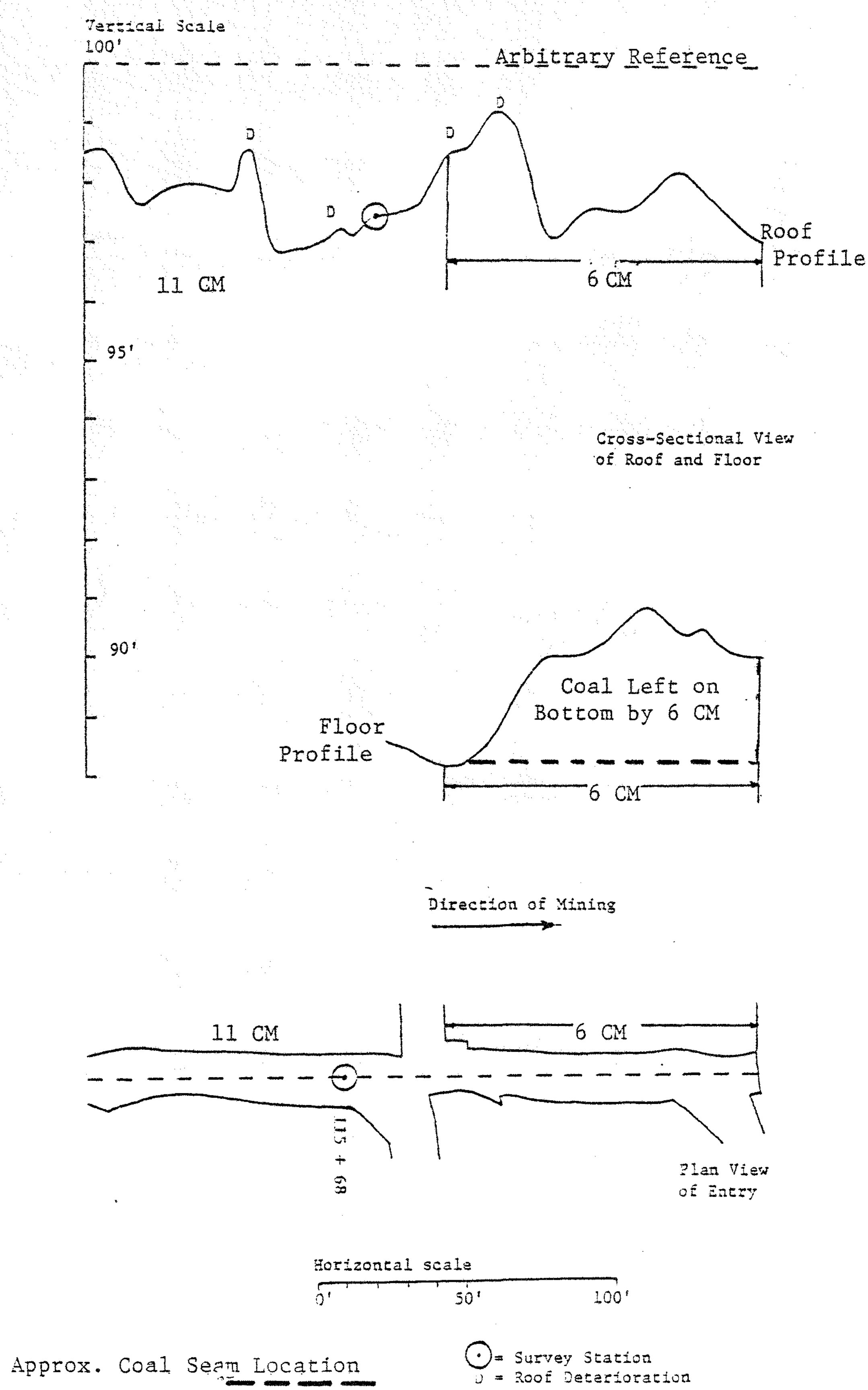


Figure 22

Roof and Floor Profile Entry 4

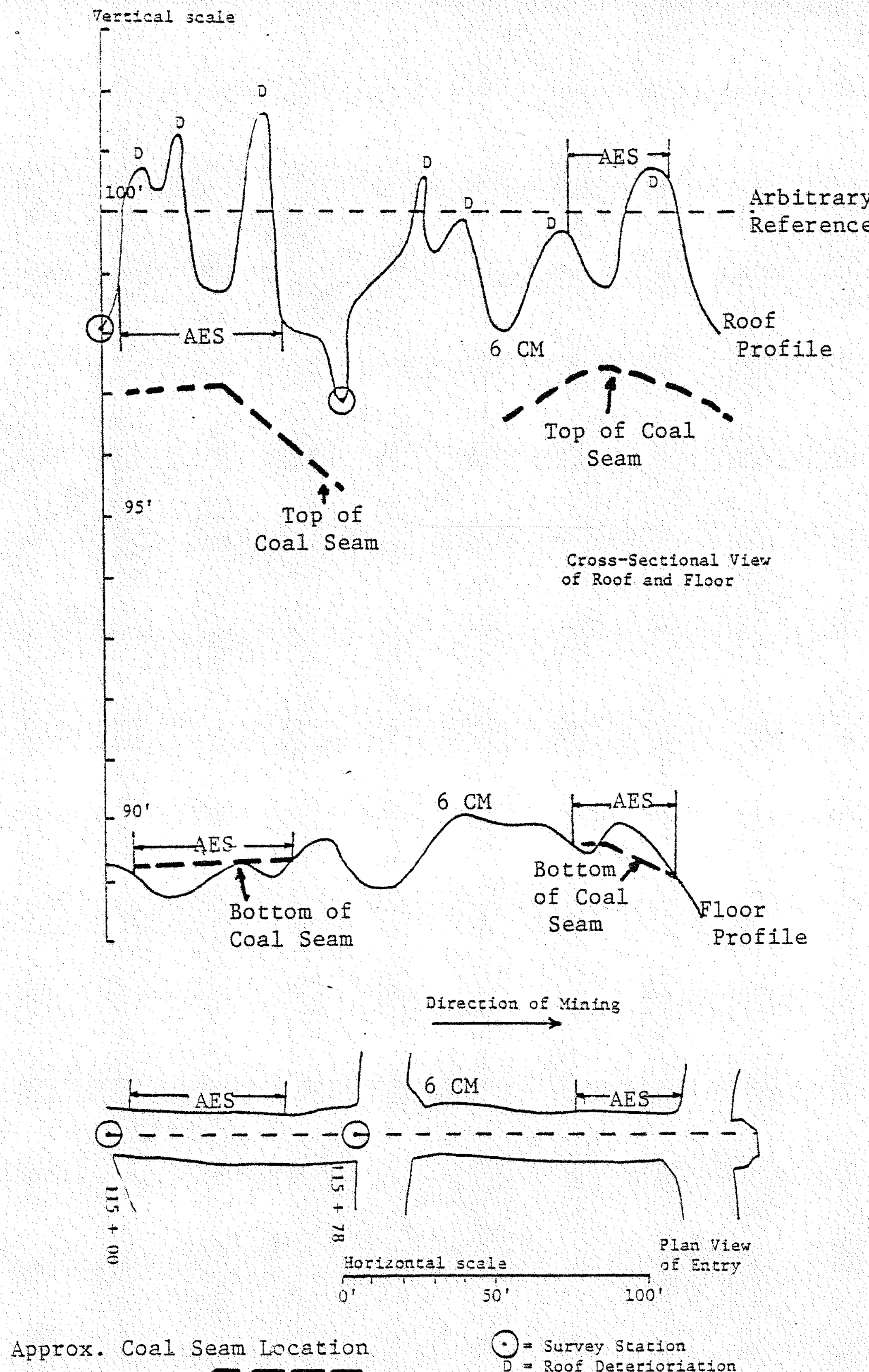


Figure 23

Roof and Floor Profile Entry 5

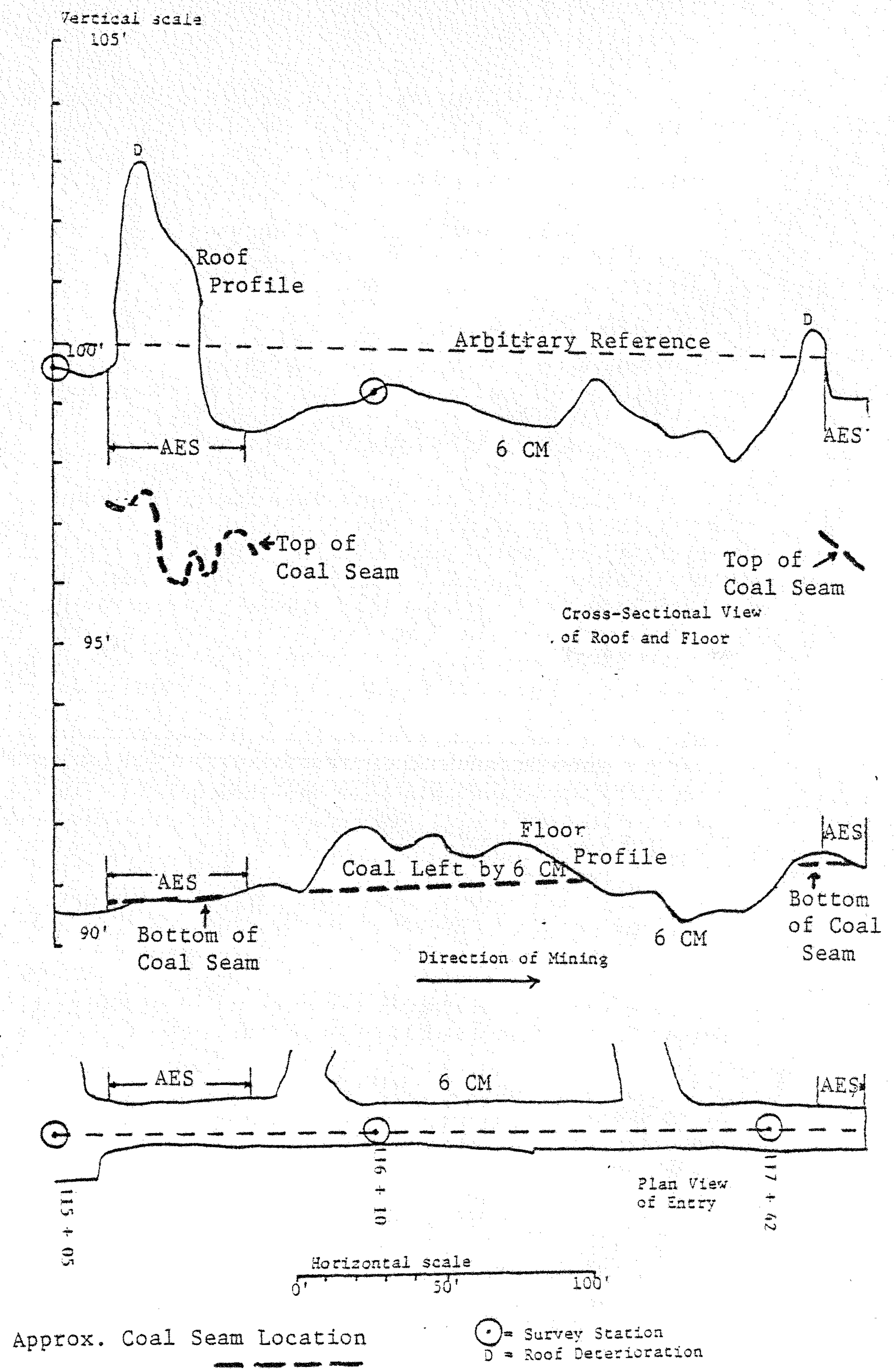


Figure 24

Roof and Floor Profile Entry 6

since difficulty in maintaining a proper floor horizon may have influenced some of the roof problems, especially when the AES was forced to mine higher and into the roof. The coal-seam-roof and -floor boundaries are shown for the cuts mined by the AES.

In Entry 4, the Joy 11 CM mined through the crosscut just inby Survey Station 115+65. The roof was deteriorated in the area of the crosscut. However, inby the area of rib deviation, the roof was relatively smooth.

Because of the limted cutting height of the 6 CM, the machine must operate on approximately two feet of coal on the bottom. This coal is then recovered at a later time. The floor profile in Figure 22 shows the intact coal left on the bottom.

In Entry 5, the area just inby Survey Station 115+00 was mined by the AES as shown in Figure 23. The roof fell in most of the area because there were planks used to support the area just outby where the AES mined. The AES' roof-beam assembly could not be used effectively because the wooden planks would be destroyed. The AES was operated manually until its beams could be used, and therefore there was an unsupported span of between 19 and 23 feet. The roof would fall out at this distance, and sometimes even before this length was reached. But, then because the roof was uneven, the AES' beams again could not be used. This situation also existed in the area just ahead mined by the AES (just outby the last open crosscut). However, the roof beams were used this time and the roof fell when the beams were tried after a span of 11 feet was left unsupported over a weekend. (See Section 2.4.1.5)

The floor horizon remained relatively constant, as is shown by the floor profile. Two floor depressions developed just inby Survey Stations 115+00 and 115+78.

The roof deterioration occurred at the intersection just inby Survey Station 115+78 which was mined by the 6 CM.

Figure 24 shows Entry 6 as mined by the AES and the 6 CM. There is an area of deterioration just inby Survey Station 115+05. The area just inby Survey Station 117+42 was mined by the AES with the use of the ACS on the May 24 special trial. No planks were used in this area.

The floor in the area mined by the AES just inby Survey Station 115+05 is noticeably lower, and the floor in the area mined by the 6 CM is higher in order to allow the 6 CM to reach the roof coal.

2.6.22 Section Plan Map: Figure 25 shows a plan map of the section showing roof falls. A detailed roof-condition report is being prepared at the time of this writing under a contract extension. That report will be submitted separately, and will show other pertinent geological data.

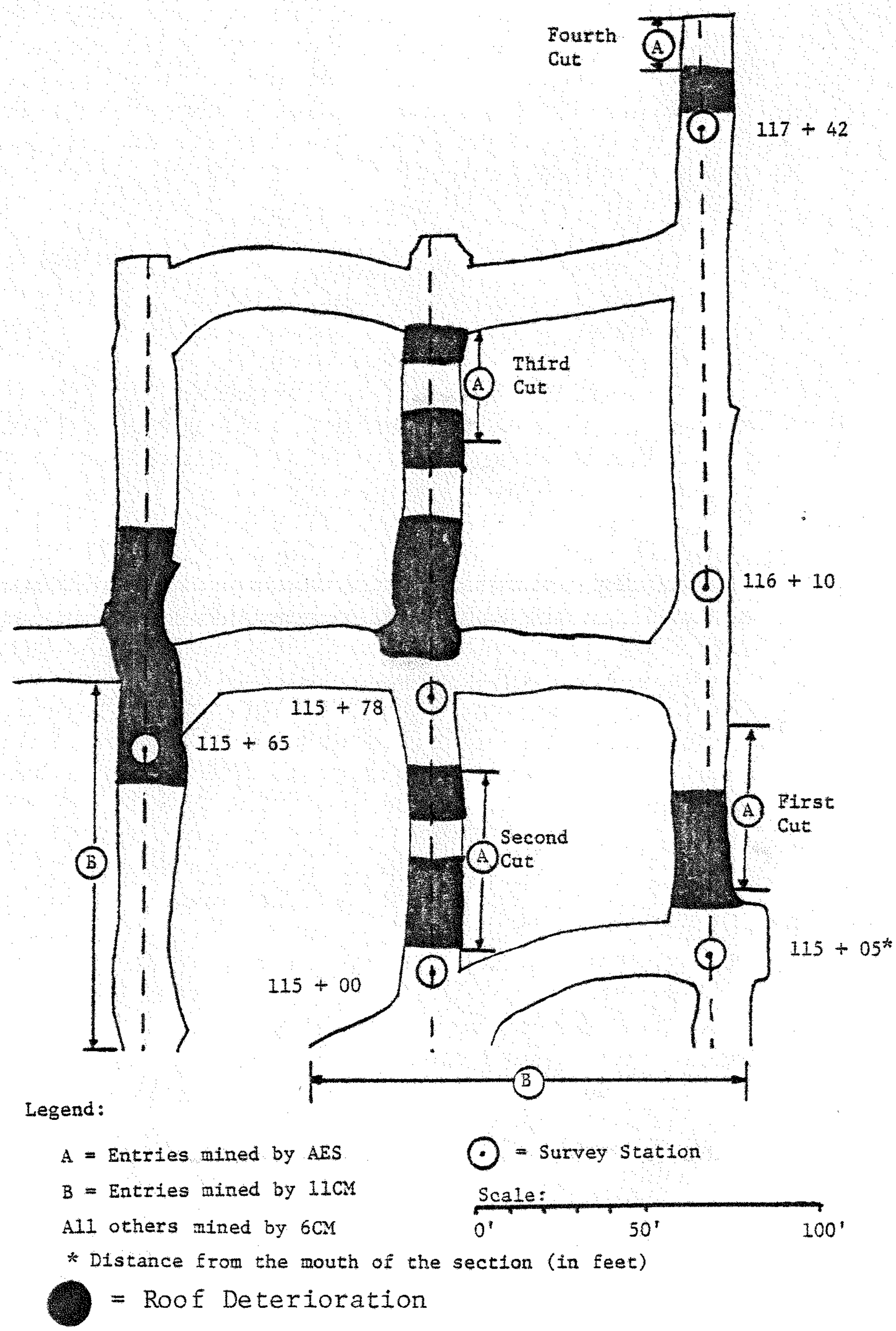


Figure 25

Areas of Roof Deterioration.

2.6.3 Centerline Orientation Audit

The object of this audit is to evaluate the ability of the AES and its operator to maintain the projected centerline and entry dimensions. With the exception of the face area of Entry 6, the AES was operated manually with the use of the remote pendant. The last 15 feet of Entry 6 were mined in automatic mode. The entry features were located by measuring the distance from the projected centerline to the rib. All intersections and obvious deviations from the cut plan were located. So that comparisons could be made, the survey included the entire section face area mined by the AES, 11 CM, and the 6 CM. Figures 15 and 19 are the products of that survey. Table XIX shows the average entry-width dimensions and centerline deviations for the AES, 11 CM, and 6 CM.

Observations made during mining, and especially during tramming, reveal that a high amount of direction (usually 2 or more extra people) was required whenever the AES was moved.

The AES' components contacted the roof and ribs often. Visibility on the left side and on the top of the roof beams was restricted, and inexperience and lack of training seemed to be evident in tramming and aligning the AES.

With the exception of the last 15 feet mined in Entry 6, all mining was done manually with the use of the remote-control pendant attached to the umbilical cable. This arrangement provided the operator with more visibility, during both mining and tramming.

When the last 15 feet in Entry 6 was mined, the AES was in automatic mode. At first, after the cutting horizons had been set, the roof beams began moving forward ahead of the bottom of the support cylinders. It was determined that the floor horizon had not been set low enough to allow the gathering head to "float" up over it and a lip of coal was left. The situation was corrected by backing up and resetting the lower limit of the shear control but two rear support-cylinders were broken from their lower housings, thus causing the floor beam to slip during mining.

In order to align the AES, the operator would tie a string through two survey spads driven into the roof. Then, by sighting over the strings, the operator would position the AES to correct any centerline deviations.

2.6.4 Canopy-Protection Audit

The object of this audit is to document and evaluate the effectiveness of the AES' roof-beam assembly as a canopy-protection device.

2.6.4.1 Observations of the Operators: It was observed that during manual operation, the operator used the control pendant and stood outside the canopy. During automatic operation, the operator remained

TABLE XIX

Mean Entry Width and Centerline Deviation Measurements

MACHINE	ENTRY WIDTH $\bar{x}$	ENTRY WIDTH s	CENTERLINE DEVIATION $\bar{x}$	CENTERLINE DEVIATION s
AES	15.95'	0.90'	-0.25'*	1.50'
11 CM	17.42	4.14	0.64'	2.91'
6 CM	17.47	1.97	-0.69'	2.45'

$\bar{x}$ = mean

s = standard deviation

*Negative sign shows deviation to left of centerline.

within the canopy, except when the operator is taking a methane check or troubleshooting any problems.

2.6.4.2 Worker-opinion Questionnaire: The workers noted that visibility was a problem, especially for the AES operator. The ease of working within the confines of the cabs did not seem to be a problem, but production time was limited, and there was no full shift during which the operators were in the cabs constantly.

2.6.5 Face-Ventilation and Dust-Control Audit

The object is to test the ability of the AES to 1) provide a quantity of at least 3,000 cfm of air at a velocity of at least 60 fpm to the face; 2) dilute, render harmless, and carry away methane; and 3) suppress and carry away respirable dust.

2.6.5.1 Air-velocity Measurements: With the use of a hand-held anemometer, air velocity readings were obtained at the discharge end of the inflatable tubing hung behind the AES. The quantity was determined by multiplying the velocity by the cross-sectional area of the duct. Table XX shows the velocity readings obtained.

2.6.5.2 Methane Measurements: With the use of a calibrated hand-held methanometer, readings of the methane content of the air at the discharged end of the tubing were obtained. These results are shown in Table XXI.

The AES was never shut down by an accumulation of methane at the face. No ignitions of methane occurred.

2.6.5.3 Personal-dust Sampler: One shift was monitored for dust concentrations by Consol with the use of a personal-dust sampler. The findings indicated that 1 mg/cm^3 of respirable dust was present in the air at the operator's location. The AES mined 120 raw tons (99 clean tons) of coal on that shift.

2.6.5.4 Worker Opinion Questionnaire: Most of the workers said that they thought the ventilation system on the AES was better than those on other machines upon which they had worked.

2.6.5.5 Tubing Conditions: The flexible tubing worked quite well during most of the in-mine trial. However, on the May 24 special-trial day, a hole was discovered in one tubing section. The hole was approximately 12 inches high and 4 inches wide.

2.6.5.6 Alternative Ventilation: During the in-mine experiment, it was necessary to ventilate the face using brattice cloth when the AES was not operating. Difficulties were encountered when the workers tried to hang the curtain beside the AES because of the AES' size. In some instances, when the curtain was hung, an additional piece was required at the bottom in order to reach from the roof to the floor because of extra height caused either by floor

TABLE XX

Air Velocity and Quantity Determinations

READING	DISCHARGE VELOCITY (fpm)	DISCHARGE AREA (ft^2)	AVERAGE FACE Area (ft^2)	FACE VELOCITY (fpm)	FACE QUANTITY (cfm)
1	2720	1.50	92.49	44.11	4090
2	3006	1.50	92.49	44.11	4509
3	2720	1.50	92.49	44.11	4080

TABLE XXI

Methane Readings

TUBING LENGTH	PERCENT METHANE
55	0.2-1.2 0.7 average
60	0.0-0.5 0.25 average

load-out or roof falls. In many cases, the crew was unable to install the curtain immediately because a large amount of time was required to tram the AES from the face.

2.6.6 Lighting Audit

The object of this audit was to determine if the AES' lighting system meets MSHA specifications of 0.06 ft Lamberts and to obtain the workers' attitudes toward the lighting system.

2.6.6.1 Photometer Measurements; Incident-light readings were obtained at various positions around the AES. The readings were primarily taken when the AES was located in a crosscut some distance from the face. Four readings, each taken at a distance of 2 feet from the AES and at an average height of 5 feet, were obtained, and an average value calculated. Table XXII lists the readings. The data shows that all locations are in compliance with the Federal Law.

2.6.6.2 Worker-opinion Questionnaire: Worker opinion of the lighting system was generally favorable. However, several workers experienced a brief moment of "blindness" when the lights suddenly went out and their eyes were not adjusted to the new level of illumination. They reported feeling uncomfortable during this time.

2.6.7 Noise Audit:

The object of this audit is to determine the amount of noise energy generated by the AES and the workers' opinion of the noise. Because of the limited production period, few measurements were taken.

2.6.7.1 Docimeter Readings: Several readings of the noise energy levels were taken at the operators' stations. The measurements are shown in Table XXIII. The highest readings were found to exist at the operator's station on the right side during cutting.

2.6.7.2 Worker-opinion Questionnaire: Generally, the workers said that the noise from the AES caused more discomfort than from other machines which they had operated.

2.6.8 Bolting Platform Stability Audit

The object of this audit is to determine if the AES is able to maintain a stable platform during simultaneous cutting, loading, and bolting. However, during the special trial on May 24, it was observed that the rear bumper moved backward about 12 inches during a four-foot cycle.

2.6.9 Crosscut-Turning-Ability Audit

The object of this audit is to evaluate the AES' ability to turn out a crosscut. However, no crosscuts were mined by the AES; therefore, no data was collected.

TABLE XXII

Incident-Light Readings and Corresponding Reflective-Light Values

MACHINE LOCATION	AVERAGE INCIDENT LIGHT (ft-candles)	AVERAGE REFLECTED LIGHT* (ft-Lamberts)
1	6.58	0.20
2	8.96	0.27
3	12.97	0.39
4	18.44	0.55
5	9.81	0.29
6	8.00	0.24
7	10.61	0.32
8	5.38	0.16
9	12.00	0.36
10	15.98	0.48
11	14.34	0.43
12	5.85	0.18
13	2.81	0.08
14**	0.13	0.00

*Average Reflected Light = Average Incident Light x Reflectivity. The average reflectivity for coal is assumed to be 3 percent (Chironis, 1976).

**MSHA does not specify that the rear conveyor boom be illuminated. This reading was taken for information purposes only and will not be used in data analysis.

TABLE XXIII

Noise Measurements

LOCATION	ACTIVITY	AVERAGE READING (dBA)
Right Operator Cab	Pump motor running, leveling roof supports	86.3
Left Operator Cab	Pump motor running, leveling roof supports	92.0
Right Operator Cab	Cutting/loading	100.0
Left Operator Cab	Cutting/loading	97.6

Tramming around previously mined corners was sometimes a problem. Ease of turning the corner seemed to improve with increasing operator experience. The stinger modifications and the 60-degree conveyor-swing angle aided tramming around corners.

2.6.10 Cleanup-Ability Audit

The object of this audit is to evaluate the cleanup ability of the AES. As shown in Table IV, clean-and-trim time amounted to 154.05 minutes. Included in this data is the time the AES was used to mine the 120 feet of floor coal left by the 6 CM in Entry 5. It should be noted that clean-and-trim time for the first production trial was much less than that recorded during the second, when it increased from 33.50 to 120.55 minutes as a result of mining that coal left by the 6 CM for 85.85 minutes.

During production periods, it was noted that at times the gathering head did not clean well. On one occasion, the pilot pressure for the gathering-head float controls was adjusted and the situation improved considerably. However, during the last production shift, as much as five inches of loose material was being left on the bottom by the AES believed to be caused by problems in the hydraulic system.

2.6.11 Out-of-Seam Dilution Audit

The object of this audit is to evaluate the ability of the AES to limit out-of-seam dilution. A tape measure was used to determine the amount of coal and rock mined. The out-of-seam dilution was 34.3 percent on a volume basis. Roof-fall material was included in the calculations.

CHAPTER 3

ANALYSIS OF THE RESULTS OF THE AES IN-MINE TRIAL

3.1 INTRODUCTION

The purpose of this chapter is to 1) analyze the data obtained during the in-mine trial, 2) determine what influenced the performance of the AES, 3) determine how the AES affected the mining operation, and 4) make observations based on the data to formulate conclusions and recommendations to improve the AES. The analysis must determine if the data is adequate to make accurate conclusions and recommendations. Because the data was so difficult to obtain, some of the conclusions and recommendations are based on partially complete information. However, with the qualifications stipulated herein, the Audit Team believes that the preliminary data is representative enough to ensure correctness in the findings. Sources of error will be discussed to determine their effect on the data that was collected.

3.2 FIFTEEN AREAS OF SPECIAL INTEREST

Consol and NMSC met to discuss the experiment shortly after the May 24 special-trial day. The decision was made to discontinue the trial on the basis of 15 problem areas that were severely limiting the amount of new data that was being obtained. The areas, not necessarily in order of importance, are:

- Falls and breakage of the mine roof.
- Oil leaks.
- Chock slippage.
- Low production.
- Repairability.
- Accessibility and complexity.
- Bulk.
- Weight.
- Gathering head and conveyor.
- Ventilation.
- Safety.
- MSHA compliance.
- No technical gain in continuing the experiment.
- Worker morale.
- Upper and middle management pressure for production.

The Audit Team was asked to supply quantitative data concerning these 15 areas, and an interim report was submitted to the DOE on September 13, 1978. The data was gathered from the results of the Full-Time Audit, the Worker Questionnaire, and the Special Audits, the Audit Team observer's analyses and opinions, and machine specifications supplied by NMSC. The data has been included in Chapter 2 of this report. The analysis of that data is a substantial part of this chapter.

3.3 ERROR ANALYSIS

Accurate data collection in any underground machine evaluation is challenging and the Comprehensive Field Documentation (CFD) of the in-mine trial was no exception. However, the Audit Team believes that the techniques used have ensured data results that are within reasonable limits of accuracy. The circumstances of limited data has been discussed earlier. The following points are considered to be the major areas where errors in observed data may be claimed by others. However, we believe any impart would be negligible.

Errors may have been introduced because the observers prudently elected to collect data from a safe vantage point outby the loading machine. However, any error introduced in this manner is believed to be negligible.

Error could enter the data because more than one observer was used for the documentation. However, all of the observers used the same, pre-determined procedure, so that this error was minimized.

The use of non-calibrated instruments for the special audits could have been a source of error. But, all the instruments used had been calibrated both before and after the observations, thus minimizing any error.

Some error may have resulted from the eight maintenance shifts not personally monitored by the Audit Team personnel. However, the Audit Teams believes that the data supplied by NMSC and Consol is reliable.

Some error is noticeable in the element summary of Table IV, since the average per shift total (477.75 minutes) does not add to 480 minutes. However, the results are within 0.5 percent, and the error may be considered negligible.

The final possible error source might have resulted because the Audit observers were not necessarily informed of every detail. The Audit observers may have been completing another task when some information was being discussed and therefore that point may not have been documented at that time. The extent of this type of error is believed to be insignificant.

3.4 ANALYSIS OF THE RESULTS OF THE FULL-TIME AES AUDIT

3.4.1 Introduction

One goal of the ARCCM program was for the AES to consistently demonstrate a capability to mine for 100 minutes per shift at a rate of 4 tpm. However, the AES only achieved an average of 49.13 loading minutes per production shift and a production rate of 1.4 tpm. Also, because of maintenance, geological, and other problems, only 21 shifts out of a planned total of 71 were actually used for production resulting in a very limited amount of data. With full awareness of this deficiency and without further qualification this section examines the production, maintenance, and geological data obtained during the Full-Time AES Audit for identification of spreads between realized and target objective goals. The costs incurred during the in-mine trial will also be discussed.

3.4.2 Successful Machine Operation

Some components are considered to have been reasonably successful within the limited production operating time. The area-lighting system posed no major problems. The disc-type gathering head limited spillage over the sides and rear of the head. The extensible conveyor worked without damaging any conveyor chains and moved the coal satisfactorily. The basic structural members, with the exception of the stinger frame, held up well during the limited in-mine trial. However, it must be noted that these observations are based on only 21 operating shifts out of the planned 71 production shifts. It is obvious that additional in-mine duty is required to verify these findings. In addition the Audit Team believes that a rigorous out-of-mine testing program should be conducted on any second generation prototype machine to spot and correct problem areas before going underground.

3.4.3 Production Rate and Tonnage Analysis

The AES mined 1429 raw tons of coal and advanced 261 feet (including 100 tons of bottom coal left by the 6 CM over a distance of about 120 feet) for an average of 68 raw tons and 7.1 feet mined per shift for 21 shifts. The average production rate was determined to be 1.4 tpm for the 21 shifts. The highest production on any one shift was 125 raw tons, and the highest producing rate during the in-mine trial was 2.9 tpm for a 30-minute period on one shift. (Table III).

The production rate might have been low because: 1) Manual operation was less productive than expected because of operator unfamiliarity with the AES and inexperience with the controls. 2) Cleanup time was also included in the production rate because the observers could not always determine when the AES was manuevering to clean the floor or to cut the cusp. 3) The ACS operation on the special-trial day was less productive than expected because both the sump and shear rates were set low and because the floor beam moved backward during sumping. The beam slipped because two rear roof-support cylinders had pulled out of their sockets and because oil and water on the bottom and poor cleanup reduced the coefficient of friction between the floor and the AES. 4) The observers recorded loading time from the moment the conveyor was started until it was stopped, except for obvious times when the operator was performing another job with the conveyor running. There were times when there appeared to be no coal being discharged from the conveyor, but often, because the pile of coal behind the AES blocked the observer's view, this conclusion could not be validated. Thus, the rate is only indicative of loading time, not cutting time.

From the detailed mining cycle times (Table VIII) obtained during the special-trial day, 1) the instantaneous cutting rate, 2) the average cutting rate per one-foot cycle, and 3) the average cutting rate per four-foot cycle were determined to be 2.05 tpm, 1.44 tpm, and 1.24 tpm, respectively. These rates do not reflect delays or time that the conveyor was operating without coal being loaded. The values are low because the sump and shear rates were set low and because slippage of the floor beams limited the rate of advance of the cutter head.

3.4.4 Production Shift Analysis

The time elements varied considerably during the in-mine trial. During the first production trial, cutting and loading time accounted for an average of 33.45 minutes per shift; necessary delays, 71.96 minutes; bolting, 19.98 minutes; maintenance delays, 138.29 minutes; and unnecessary delays, 39.81 minutes. The remaining time (176.51 minutes) was distributed among lunch, mantrip time, preparatory time, and other operating delays. (See Table IV).

After the modification cutting and loading improved to an average of 68.54 minutes per shift; necessary delays, to 29.35 minutes; bolting, to 25.90 minutes; and maintenance to 75.12 minutes. Unnecessary delays worsened to 79.12 minutes. The remaining time for fixed and other operating delays was 201.49 minutes. (Table III).

The average times for the entire trial period are as follows: cutting and loading, 41.80 minutes (8.7 percent of the shift); necessary delays, 61.81 minutes (12.9 percent); bolting, 21.39 minutes (4.5 percent); maintenance delays, 123.25 minutes (25.8 percent); unnecessary delays, 49.29 minutes (10.3 percent); and fixed and other operating delays, 182.46 minutes (38.0 percent). (Table IV).

Table XXIV compares the Unit Shift Production times versus predicted times. Table XXV shows the major items of potential production time improvement for cutting and loading. If one-third of this potential time, or 52.5 minutes, could be translated to cutting and loading, the total production time would increase to 101.7 minutes. At the achieved production rate of 1.4 tpm, 142 raw tons of coal could be produced; and at the target rate of 4.0 tpm, the production extrapolates to 407 raw tons of coal per shift. (This value compares favorably with the production achieved by the miners in the 1975 Marrus study of 409 raw tons per shift.) The remaining two-thirds of the potential time could be applied to necessary delays, bolting delays, place change time, plugged time, ventilation delays, etc. These calculations are shown with Table XXV.

If nine workers including the foreman, are required to operate the section as was the case in both the in-mine trial and the Frantz and King (1977) report, the ton per section-man-day figure at the 1.4 tpm rate would be 15.8, and at the 4.0 tpm rate, the figure would become 45.2 tons per section-man-day for the AES section.

3.4.5 Production Delay Analysis

3.4.5.1 Roof Support Delays: An average of 1.5 bolts was installed during each of the 21 production shifts, resulting in an average delay of 14.5 minutes per bolt. If two bolts are installed

TABLE XXIV

Unit Shift Production Analysis

ELEMENT	PREDICTED[1] TIMES (MIN)	ACHIEVED TIMES (MIN)	VARIATIONS FROM PREDICTED
Cutting and Loading	95	49.14(1, 11)[2]	- 45.86
Place Change	25	8.71(5)	- 16.29
Shuttle Car (Plugged)	40	4.52(2)	- 35.48
Mantrip	50	59.12(16, 17)	+ 9.12
Lunch	30	26.60(15)	- 3.40
Fireboss Inspection	15	---	- 15.00
Necessary Delays	25	70.98(3, 4, 10, 18)	+ 45.98
Bolting Delays	30	21.39(6)	- 8.61
Prepare to Start	20	31.12(13)	+ 11.12
Prepare to Leave	20	13.46(14)	- 6.54
Maintenance	75	123.25(7)	+ 48.25
Ventilation	10	1.26(12)	- 8.74
Outby Haulage	20	---	- 20.00
Other Delays (Unnecessary)	25	68.20(8, 9, 19)	_ 43.10
Total	480	477.75	
Tons per Shift	380	68.0	
Production Rate (tons per minute)	4.0	1.4	

[1]Extracted from Table 4-1, Study of the Human Factors Aspects of an Automated Continuous Mining Section, Frantz, R. L., and R. H. King, USBM Final Report, Grant No. S0144115, March, 1977, page 74.

[2]Extracted from the Full-Time AES Audit - (N) is the element number from Table III.

TABLE XXV

Elements of Potential Time Improvement
for Cutting and Loading

ITEM	VARIATIONS (MIN)
Mantrip	9.12
Necessary Delays	45.98
Prepare to Start	11.12
Maintenance	48.25
Other Delays (Unnecessary)	43.10
Total	157.57

If 1/3 of this variation (52.52 minutes) is applied toward cutting and loading,

$$49.14 + 52.52 = 101.66 \text{ minutes}$$

could be used for producing coal. At the achieved production rate of 1.40 tpm, the amount of raw coal that could be produced would be

$$(1.40)(101.66) = 142.33 \text{ raw tons.}$$

If the production rate increases to the target rate of 4.00 tpm, the tonnage becomes

$$(4.00)(101.66) = 406.64 \text{ raw tons.}$$

The remaining 2/3 of the variation (105.05 minutes) could be applied towards bolting, necessary delays, ventilation, rock dusting, etc.

Note: This hypothetical analysis shows the performance improvements required to achieve the 400 tons per shift target objective and the typical items that are involved.

simultaneously on each side, the average delay per row of bolts would also be 14.5 minutes. Assuming there was an average of 6 tons of raw coal per foot of advance and a four-foot cycle was mined (as was the case during the in-mine trial), the production rate that would let bolting and mining end simultaneously would be

$$\text{Production Rate} = \frac{\text{Tons/Total Advance}}{\text{Bolting}}$$

$$= \frac{24}{14.5} = 1.7 \text{ tpm.}$$

If the production rate were to increase to the target rate of 4.0 tpm, the bolting time would have to decrease to

$$\text{Bolting Time} = \frac{\text{Tons/Total Advance}}{\text{Producing Rate}}$$

$$= \frac{24}{40} = 6.0 \text{ minutes}$$

per four-bolt set.

During the 21 production shifts, the average delay to production because of roof falls was determined to be only 4.2 minutes or less than one percent of shift time. However, this results from the judicious selection of production shifts and does not realistically express the adverse condition of the 1 South section mine roof.

3.4.5.2 Oil Leak Delays: An average delay of 34.23 minutes per production shift was incurred from oil leak delays. (Based on 21 shifts.) A total of one and one-half operating days were lost because of oil leak repairs. Over half (55.4 percent) of the total effort (204.9 manhours during the entire trial) was required for fittings; hoses, 31.3 percent. Loose and damaged fittings each contributed 27.7 percent of the leaks; hitting components, 14.5 percent; stretched threads, 7.2 percent; pinching, 6.0 percent; wrong fitting-type, 1.2 percent; and unknown causes, 15.7 percent. An average of 2.5 manhours were required per oil leak. (Based on 83 leaks.)

The AES used 1540 gallons of oil during the in-mine trial, resulting in 1.1 gallons of oil per ton of raw coal. (Based on 1429 raw-tons mined.)

3.4.5.3 Tramming Delays: Tram delays amounted to an average of 8.7 minutes per production shift. (Based on 21 shifts.) Delays during the tramming (including non-production shifts) averaged 33.41 minutes per week. (Based on 15 weeks.) Building bridges comprised 52.5 percent of this time; directions and shop talk, 15.0 percent; handling cables, 11.9 percent; adjusting the roof-beam assembly for clearance, 11.9 percent; machine complexity, 1.7 percent; and other problems, 6.8 percent.

3.4.5.4 Gathering Head and Conveyor Delays: An average of 59.73 minutes per shift were lost because of the four gathering-head motor failures. (Based on 21 production shifts.) The average delay per occurrence of the conveyor stalling (including the motor failures) was 96.48 minutes.

3.4.5.5 Ventilation Delays: Ventilation delays on production shifts totaled 1.26 minutes per shift. (Based on 21 production shifts.) This delay was usually required to add a piece of tubing or to re-hang fallen tubing.

3.4.6 Maintenance Analysis

3.4.6.1 Logistics Performance Analysis: During schedule production shifts, maintenance delays comprised 25.8 percent of the shift time. The maintenance was primarily corrective, hydraulic work. The major components requiring the most repair work were the gathering head and conveyor, roof-beam assembly, roof drills and dust collectors, ventilation fans, and the cutter head. Oil leaks caused many of the maintenance problems including one and one-half production days that were used entirely to repair oil leaks. Loose and damaged fittings were the primary causes of the leaks, but hose replacements also required a substantial number of maintenance manhours.

A number of maintenance logistics parameters have been determined. Table XXVI lists the results. The values achieved by the AES will be subject to improvement whenever increased or persistent production, not present during the in-mine trial, is attained. Average values for corrective maintenance on other continuous miners (Marrus, et al., 1975) compare with the achieved values in the following ways:

- The mean tons between failures for the continuous miners in the Marrus study was about 1400, but for the AES, the value was 17.6 tons.
- The mean time between failures for those miners was about 6.23 hours, but for the AES it was 3.1 hours.
- The mean corrective time for those miners was 2.22 hours, but for the AES it was 1.2 hours.
- The mean corrective manhours for those miners were 4.80 manhours, but for the AES it was 3.0 manhours.
- The maintainability index for those miners was 3.53 manhours per 1000 tons. The figure for the AES was 170.1 manhours per 1000 tons.
- The corrective-maintenance manhours per hour of operation for those miners was 0.79, but for the AES it was 1.8.
- The achieved availability for those miners was about 73 percent as calculated on the basis of maintenance delays only on production shifts. For the AES it was 63.6 percent. Adjusting for unavailable production shifts (70 percent of of time) the achieved availability for the AES becomes 19.1 percent, which is a more representative comparison.
- The Marrus report also listed the average-shift production for the miners studied to be 409 raw tons per shift. The figure for the AES was 63 raw tons per shift.

TABLE XXVI

Maintenance Logistics Parameters

PARAMETER	CORRECTIVE	PREVENTIVE	SERVICE
Mean Time per Action (Hrs)	1.2	1.6	1.1
Mean Downtime (Hrs)	0.8	0.0	0.1
Mean Administrative Downtime (Hrs)	0.4	0.0	0.0
Mean Manhours (mH)	3.0	5.5	0.8
Mean Time Between Failures (Hrs)	2.1	21.0	2.0
Mean Tons Between Failures (T)	17.6	178.6	17.0
Mean Manhours per Operating Hour (mH/H)	1.8	0.3	0.4
Mean Manhours per Ton (mH/T)	0.2	0.03	0.04

Comparison of Corrective Logistics Parameters

	Marrus Study	AES
Mean Tons Between Failures	1400	17.60
Mean Time Between Failures	6.23	3.10
Mean Corrective Time	2.22	1.20
Mean Corrective Manhours	4.80	3.00
Manhours/1000 Tons	3.53	170.10
Manhours/Operating Hour	0.79	1.80
Availability	0.730	0.191*
Production per Shift	409	63

*0.636 based on only 21 production shifts but 0.191 when adjusted for unavailable production shifts 70 percent of the time.

This comparison shows the extent that the AES is less available and less reliable than the production-proven continuous miners studied in the Marrus report based on their definition of criteria. The lower values (except for the mean corrective maintenance times and manhours) are generally due to lower tonnage and operating time during the in-mine trial. However, the lower values for corrective time and manhours could be misleading because the substantially long maintenance periods between production shifts have not been factored into their analysis. However, with discretion these are the yardsticks that express the target objectives of new and old machines.

3.4.6.2 Additional Maintenance Analysis: During the in-mine trial, there was no time spent locating the mechanic. The reason was in part that the mechanic was usually near the AES. Also, a NMSC serviceman or engineer was always with the AES, and that person would usually begin the work immediately. Therefore, no loss of time resulted.

Also during the in-mine trial, a total of 36.6 hours was spent troubleshooting failed items, resulting in an average of 0.1 hours per action. The troubleshooting delay time amounted to 4.3 hours, or an average of 0.01 hours per corrective-maintenance action.

There were four times that the AES maintenance crews had to wait for spare parts to arrive, for a total of 165.0 hours, or an average of 0.44 hours per corrective-maintenance action.

During the in-mine trial, a total of 4.0 hours was spent testing components for an average of 0.01 hours per corrective-maintenance action. However, there were no production delays attributed to testing components since the work was performed on non-production shifts.

During the in-mine trial, a total delay time of 18.8 hours was found to occur after the AES was repaired on designated production shifts and before it again began to mine. Thus, there was an average of 0.05 hours used per corrective maintenance action.

Many of these parameters were low because 1) only several hours are distributed over many maintenance actions and 2) the maintenance personnel were very attentive to the AES. The values would be much higher if the time were distributed over the maintenance actions during which the delays actually occurred. However, because the values listed represent averages, further testing will be needed for verification.

3.4.6.3 Correlation of Maintenance and Operating Parameters: Student's "t" distribution was used to determine the confidence level of correlations between maintenance and operating parameters as shown in Appendix 5-2. Table XXVII shows the correlation coefficients and the significance of those coefficients.

The authors are assuming that a 95 percent degree of significance of correlation (DSC) is to be considered high, a 75 to 95 percent DSC is moderate, and a 55 to 75 percent DSC is poor. Likewise, a correlation coefficient (r) of greater than $\pm$ 0.60 is a good correlation, and r between $\pm$ 0.60 and $\pm$ 0.40, respectively, is an indicated correlation, an r between ± 0.40 and ± 0.20 is a possible correlation, and an r that is less than ± 0.20 shows no correlation.

If the DSC is low and r is high, then the correlation might be based on chance. If the DSC is high and the r is also high, there is a definite linear correlation. If the DSC is high and r is low, there is no correlation. On the other hand, if both the DSC and r are low, one could assume that 1) there is no correlation; 2) the correlation is not linear, but may be defined by a polynomial expression; or 3) the correlation is linear, but this sample does not show the relationship. Then the graphs must be more closely examined to determine any relationships.

It must be noted that for small samples, the value of r might be slightly biased, so that higher values of r, and correspondingly, for t might result.

The correlations can be used to formulate inferrences regarding the AES' components and systems. Those inferrences are:

1) The maintenance on several components and systems correlates well with operating parameters: the roof drills and dust collectors, the roof-beam assembly, the Automatic Control System, the stinger, the methane monitor, the tram system, and the hydraulic system. Because the correlations are moderately to highly significant, it may be said that these correlations are probably not the result of chance sampling, but rather indicate that as the machine operates more, the amount of maintenance required on the component or system increases. This raises the question, "How reliable are these components and systems?" It is the opinion of the Audit Team that these components and systems require further in-plant testing to improve their reliability. Subsequent individual component testing should be used to verify the improvements.

2) There was no correlation of corrective and preventive maintenance with operating parameters. Since preventive maintenance was not scheduled but was performed (in many cases) during extended AES downtime, the Audit Team agrees with this finding. Preventive maintenance should have little correlation if performed routinely. However, the low correlation for corrective maintenance might be the result of chance because it is near the poorly-moderately significant border (75.4 percent).

3) The possible correlations of corrective electrical and preventive-mechanical maintenance with operating parameters is probably due to chance.

4) There were several components and systems whose correlations were in the poorly-significant, no-correlation range. Therefore, the results of the in-mine trial will be used to explain the relationships: A) The maintenance of several components and systems was probably a result of design errors, and is not correlated with operating parameters.

These are: the gathering head and conveyor (motor failures), the ventilation and dust suppression system (motor cavitation and seal failures), the ventilation and dust suppression system (motor cavitation and seal failure), and the cutter head (tie-shaft failure and sump-cylinder rods bending). B) There was very little data base correlations of preventive-lighting or other-preventive maintenance with operating parameters. Therefore, no inferences can be made. C) The maintenance on the lighting system seems to be indicating that the "bugs" in the system were being ironed out. Therefore, the Audit Team believes that the maintenance on the lighting system is becoming less correlated with operating parameters. However, the results of the in-mine trial indicated that maintenance on the lights increased when the roof fell and (crushing some cables) when the roof-beam assembly was fully extended (pulling light cables from the lights). The Audit Team believes that if the light cables are a) protected against roof falls and b) made long enough to allow the roof-beam assembly to extend completely, then the lighting system will be more reliable. D) The maintenance on the water system was only caused by production once (on the special-trial day) when a manifold block was broken by falling roof material. The water pump seemed to fail gradually and was replaced during a week in which there was little cutting and loading time. Therefore, the correlation might actually be somewhat higher than this analysis would indicate. However, only three data points are above zero for the maintenance on the water system, so further calculations would probably be meaningless. E) The modification maintenance is not related to operating parameters because the work was performed whenever there was an extended downtime and the parts were available.

5) Service correlates very highly, probably because as operating time increased, the more hydraulic fluid was needed. If service is scheduled with preventive maintenance, and number of oil leaks is reduced, service should correlate to a much lesser degree.

As stated earlier the Audit Team recognizes that this maintenance-operating parameter correlation is based on very limited operating time and a large amount of maintenance time. Therefore, the results might be skewed; but the significance of the analytical technique is valid. The results point to those maintenance areas that should be more thoroughly investigated and validates the need for bench type component analysis, systematic systems investigations and above ground prototype machine service simulation for extended periods of time in a more favorable corrective atmosphere.

3.4.6.4 Additional Maintenance Analysis: The frequency of machine failures at the face was determined to be 0.013 occurrences per minute, or one failure for every 78.6 face minutes (excluding mantrip in and out, lunch, prepare to leave times).

From a review of Table XVI, a number of maintenance jobs required excessive amounts of repair time, which for some was caused by poor accessibility to some components (ventilation-fan motors, inner roof-support cylinders, sump-cylinder motion control value, stinger torque motor). Some of the delays were caused by the workers having to wait for parts (see Jobs 1, 4, 7, 8, and 13).

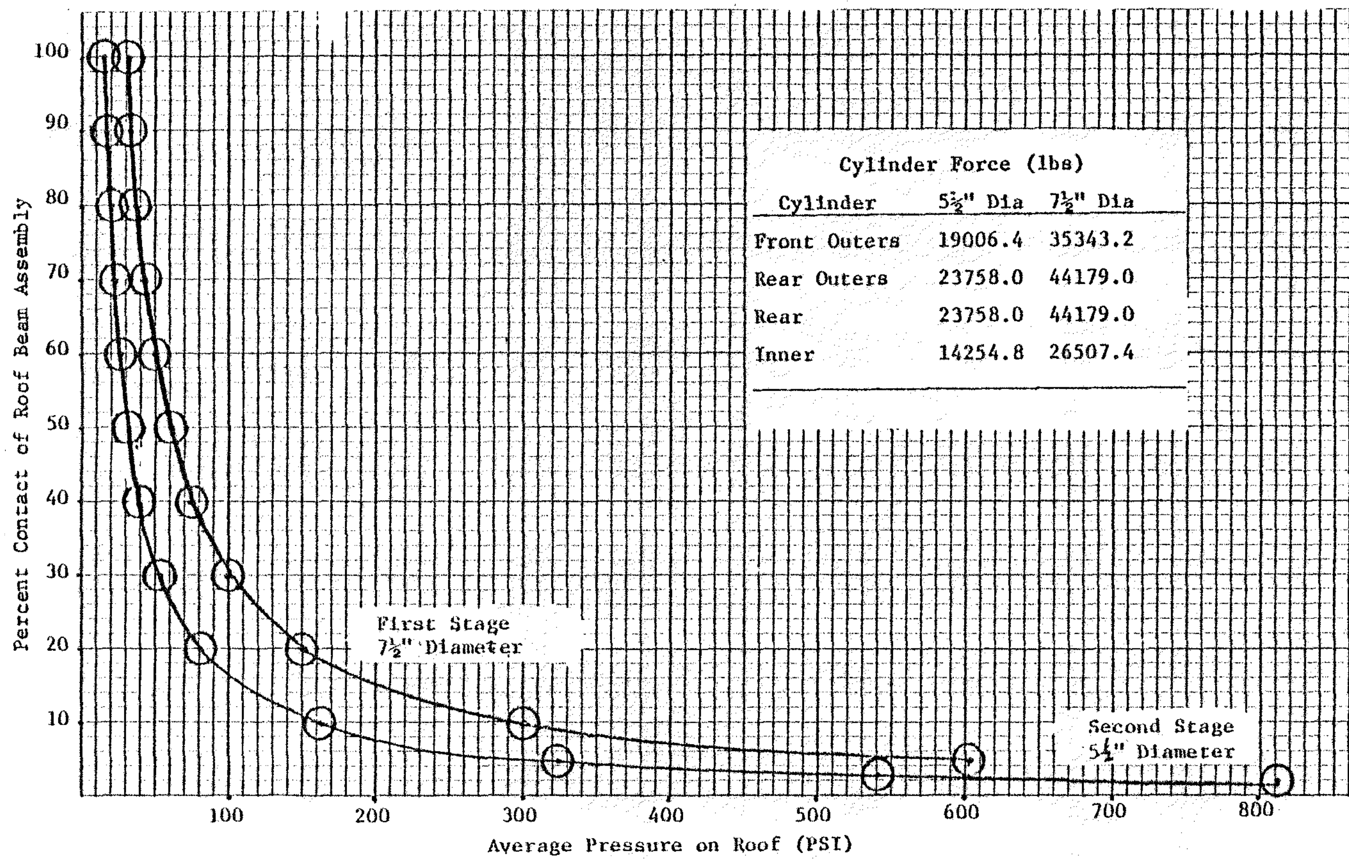
A review of Table XV and Appendix 5-4 clearly shows that the AES was significantly more complex than the Joy 6 CM, but does not fully portray the additional complexity of the AES hydraulic system and the additional number of hydraulic hoses and fittings.

3.4.7 Geologic Analysis

The operation of the AES was hampered by geological conditions during the in-mine trial, including falls and breakage of the roof and poor floor conditions. In some instances, the AES affected the roof and floor and complicated the problems.

3.4.7.1 Roof Problems: Several factors were responsible for the roof-control problems: 1) The 1 South section was idle for eight months prior to the beginning of the in-mine trial in la e August, 1977. During this time, the roof deteriorated and many minor falls (usually material from between the roof bolts) had to be cleaned up before the AES and the shuttle cars could tram to the existing faces. McElroy Mine officials stated that such an idle period was notorious for causing poor roof and floor conditions in the face area when mining resumes. 2) There were areas of slickensides and weaknesses in the immediate roof rock that caused the boundaries between rock layers to separate; thus the roof was made relatively unstable. 3) Surface streams were notorious for causing poor roof conditions in mines operating below. Natural water also accumulated in the faces of Entries 1, 2, and 3 during the in-mine trial, a fact which may indicate that the surface stream could have affected the roof strata. 4) There were several instances where the crew either failed to install temporary supports, temporary supports were left in place for long periods, or there was a long delay before the temporary supports were emplaced. The roof could have been weakened as a result of any of these situations.

Then, there were three factors attributable to operational problems: 1) The AES' roof bolters were located a minimum of 19 feet from the front of the machine. If a four-foot sump cycle was completed during concurrent mining and bolting, the bolters would then have been 23 feet from the face. At that distance, the roof might not support itself, and a fall would result. 2) The roof-beam assembly could not be used. Planks and cap pieces, placed against the roof before the in-mine trial began, caused point loads to develop between the roof-beam assembly and the wooden members. The stresses that developed on the planks and cap pieces were high enough to shatter the wood and damage the already-existing wooden-supports. A stable bolting platform was not used until bolts could be emplaced, and so the assembly was not used during manual operation at the start of a cut. Then, when the roof was allowed to sag, it lost its structural competency and would sometimes fall. This situation was further complicated because the roof-beam assembly could not be used effectively on the resulting uneven roof. The roof would sometimes fall when the AES again attempted to mine far enough (19-23 feet) to insert a row of bolts. On one occasion, two rows of bolts were inserted, but the roof fell, and covered the roof-beam assembly before a third row of bolts could be installed. Large



Cylinder Force (lbs)

Cylinder	5½" Dia	7½" Dia
Front Outers	19006.4	35343.2
Rear Outers	23758.0	44179.0
Rear	23758.0	44179.0
Inner	14254.8	26507.4

Figure 26

Percent Contact of Roof Beams vs. Roof
Pressure at Cylinder Operating Pressure

Data supplied by National Mine Service Company.

delays during bolting are believed to have been partially responsible for this fall. 3) When the roof beams were used, they caused the roof to deteriorate. Point loading situations appeared to have increased the pressure applied to the roof. Figure 26 shows the relationship between roof pressure and the amount of contact area achieved by the roof-beam assembly. At 100 percent effective contact, the roof pressure ranges from about 15 psi to about 32 psi. However, at ten percent contact, the range is from about 163 psi to about 302 psi.

It is evident that a different arrangement of machine anchorage that does not break the roof is necessary.

3.4.7.2 Floor Problems: The performance of the AES was limited by floor conditions. Some of the conditions were made worse by the AES.

The fireclay bottom was very soft, especially when mixed with water and oil introduced onto the floor by the AES. Because the weight of the AES (75 tons) and the initial location of the center of gravity of the machine, unusually high bearing pressures developed under the crawlers during the first production trial (see Figures 27 and 28). Of the 280 psi that developed, 91.9 percent was concentrated in the front eight inches of each crawler which, in digging into the floor, made mining and maneuvering difficult. Many delays resulted from building bridges to aid machine maneuverability in the muddy areas.

During the modification period, the AES' stinger was redesigned in order to shift the center of gravity to between one-third and one-half of the way from the crawler front idler (see Figures 29 and 30). Therefore, tramming and mining were made easier. Also, the number of sprays was reduced from 40 to 14, and the stream-type nozzles were replaced with fog-type nozzles. Three sprays were also plugged in each scrubber to reduce the amount of water.

3.4.8 Analysis of Other Factors

3.4.8.1 Absenteeism: Absenteeism did not greatly affect production.

3.4.8.2 Training: No formal training was given to the AES operators. Rather, they were given some hands-on experience before the first production trial began and became familiar with the controls and operation during actual production. In the judgment of the Audit Team, production and bolting were considerably less efficient than if the workers had been given some formal above-ground training.

3.4.8.3 Worker Morale: The morale of the crew was observed by the Audit Team to be quite high for the most part (with certain exceptions). The workers were operating a new machine, one that contained promises of safer, easier operation, and they were working steady day shifts. There was a challenge to make a prototype machine work during its initial in-mine trial. These factors combined to establish very high morale.

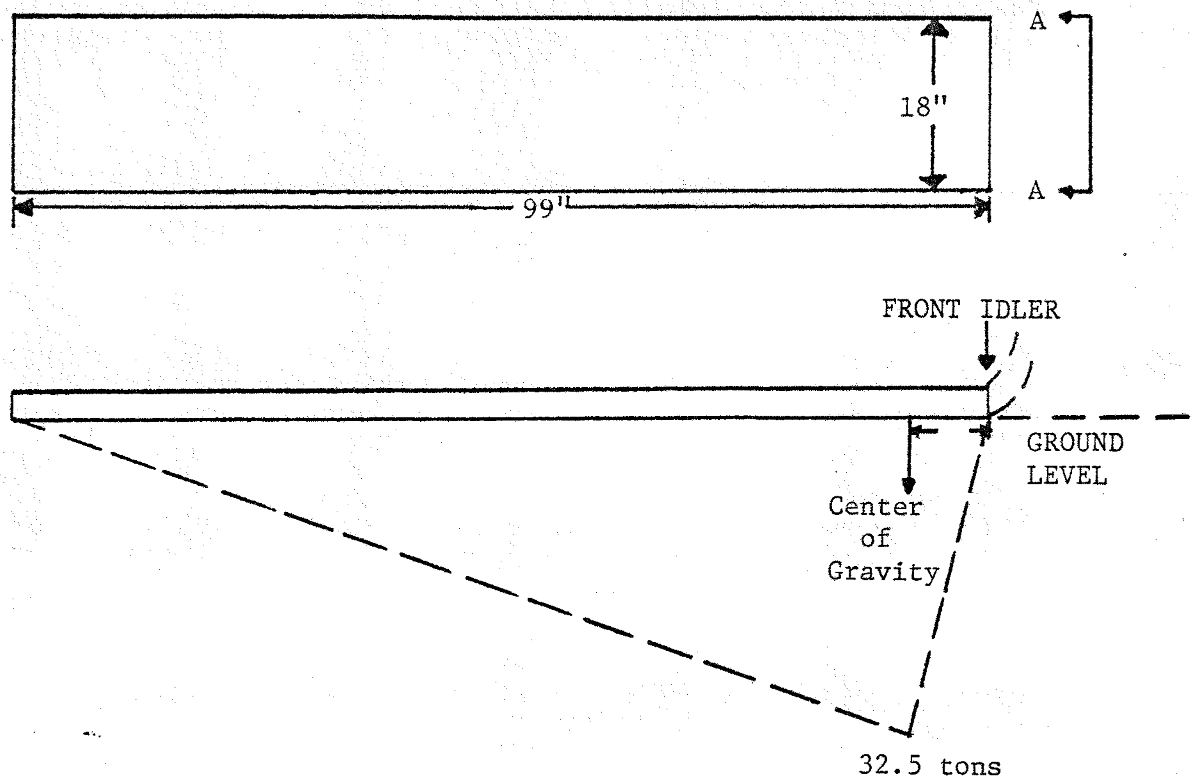


Figure 27

AES Crawler Pad
Weight Distribution in Longitudinal Direction
Before Modification

(Karafaith and Nowateki, 1978)

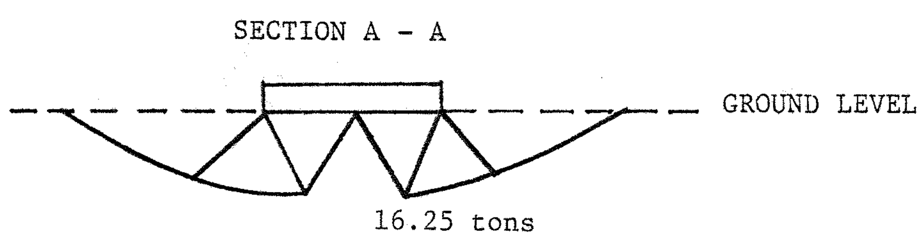


Figure 28

AES Crawler Pad
Weight Distribution in Transverse Direction
Before Modification

(Karafaith and Nowateki, 1978)

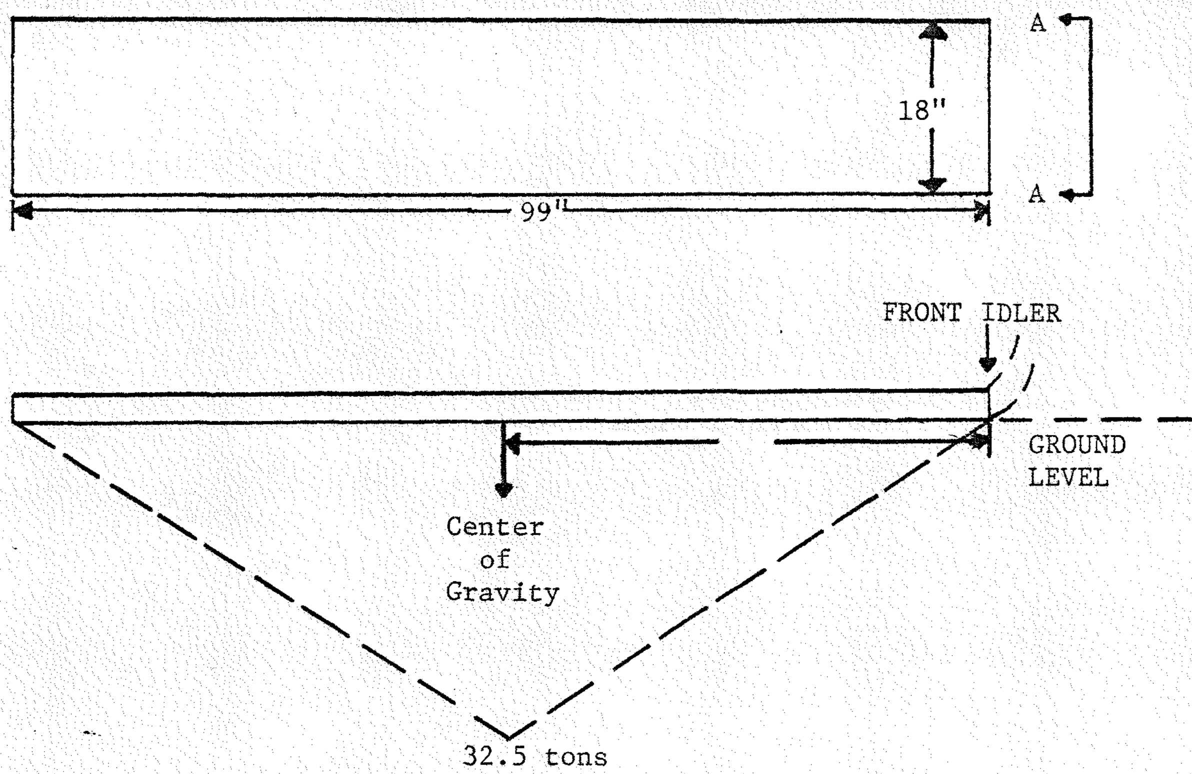


Figure 29

AES Crawler Pad
Weight Distribution in Longitudinal Direction
After Modification

(Karafaith and Nowateki, 1978)

SECTION A - A

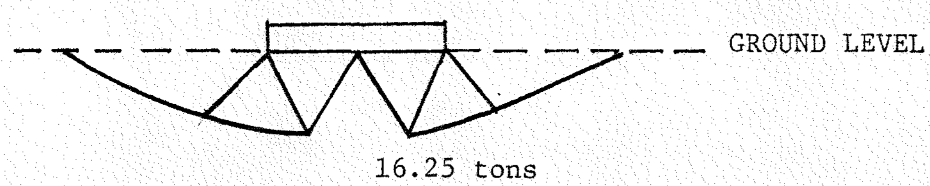


Figure 30
AES Crawler Pad
Weight Distribution in Transverse Direction
After Modification

(Karafaith and Nowateki, 1978)

However, after an extended period of maintenance difficulties at the outset of the first production trial, there were some instances of low morale, especially among the mechanics. They were originally told that they would reassemble the AES and that they could remain with the machine during the trial if they so desired. However, when several mechanics asked for their former positions, Consol management wanted to keep them on the AES which caused their morale to decline. They were eventually returned to their former positions. Another problem was encountered when labor-intensive jobs were required, such as building bridges, breaking rock, and hanging curtain, and thus, this unexpected amount of manual labor also helped to lower morale.

The crew's enthusiasm was probably greatest at the time of the May 24 special-trial day. The crew knew that their performance on this day would mean that the trial was either going to continue or end. They were even prepared to work into the afternoon shift if there was sufficient need to do so. The crew's attitude and morale on this day were quite laudable.

3.4.8.4 Pressure for Production: At various times, the section foreman seemed to be under some pressure to increase the production of the AES. This was especially evident when the Consol upper- and middle-management officials were on the section.

It must be recognized that most mines have limited locations for operating sections. Therefore each section carries its proportionate part of mine overhead capital- and operating-costs. It is very difficult for production-oriented front-line and middle management to accept, over an extended period of time, the concept of an experimental mining section or the long-development times associated with a new mining concept.

3.4.8.5 Technical Gain in Continuing the Experiment: Near the end of the in-mine trial, it became apparent, primarily, because of the maintenance and geological delays, that a limited amount of data was being collected and that a redesign of the AES was desirable to achieve ARCCM objectives. To be sure, many aspects of the AES were not evaluated, but the events of the in-mine trial indicated that the AES could not achieve desired goals at that time.

3.4.9 Cost Analysis

Since the AES was a first generation prototype machine, capital costs were not available. Furthermore, because of the high development capital that is required in such a project they were not comparable to the cost of a machine that is already proven.

3.5 ANALYSIS OF THE RESULTS OF THE WORKER OPINION QUESTIONNAIRE AUDIT

The worker questionnaire was used twice to determine attitudes toward the components and operation of the AES. The survey was conducted by interviewing each worker individually, and thus the answers seemed very candid. Peer pressure did not appear to be a factor in the answers.

There was little change in the answers from the first to the second questionnaire. The only change was that the modified stinger was more helpful in maneuvering and mining with the AES than before the modifications were made.

Table XVII shows the response of the workers to the safety additions on the AES The fact that the workers said that the components were as safe as or safer than those on other machines is an important ARCCM accomplishment. The fact that training was so limited (Table XVIII) but was recommended by some of the workers indicates the need for improved training for future in-mine trials of first generation prototype machines.

3.6 ANALYSIS OF THE RESULTS OF THE SPECIAL AUDITS

3.6.1 General Safety Audit

3.6.1.1 Safety and MSHA Compliance: There are several question about the safety and the compliance with Federal specifications of the AES, especially in the areas of roof support, ventilation, lighting, noise, and cleanup. The following observations can be made on the basis of the limited in-mine trial data: 1) The concern for the bolters by the safety committee seemed to be with one worker installing two bolts simultaneously on each side of the AES. Although this operation was never performed, NMSC anticipated no problems. 2) The roof-bolting plan was usually followed closely. 3) There may be problems with the air velocity provided by the AES at the face. (See Section 3.6.5) Also, face ventilation was disrupted when the AES was de-energized. The ventilation was then out-of-compliance with the Federal Law. 4) Measurements of lighting shows that Federal specifications were met. However, the area light wiring was arranged so that the lights would be turned off when the oil level in the upper reservoir became depleted. When this situation occurred, the operators were blinded for a moment. Equally important, the AES was then out-of-compliance with the Federal Law. 5) Although the noise readings were limited, preliminary results showed the AES was not within the specified limits of the Federal Law. 6) The AES was observed to leave as much as five inches of material on the floor during loading. To comply with the Federal Law, a secondary cleanup pass will have to be made if the problem is not solved.

3.6.2 Roof-Support and Strata-Control Audit

The roof problems appeared to be aggregated by surface streams, machine conflicts with the planks and cap pieces used during previous mining, slickensides and weaknesses in the immediate roof, the inability to use the roof-beam assembly effectively or the roof bolters immediately, and the action of the roof-beam assembly on the mine roof. (See Section 3.4.7.1) The roof profile diagrams indicate the resulting roof roughness.

This is a very serious problem which strikes at the very heart of the AES roof support-machine advance concept.

It is evident that some of these problems could be eliminated through the relocation of the bolters closer to the face, and improvement of the machine anchorage ability without subsequent roof deterioration.

3.6.3 Centerline Orientation Audit

The AES mined an average entry width of 15.95 feet (standard deviation of 0.90 feet), and varied to the left of the centerline an average of 0.25 feet (standard deviation of ± 1.50 feet). (See Table XIX). These values are significantly less than for the 11 CM and 6 CM included oversized crosscuts, and since the AES mined only 141 feet (no crosscuts were mined) during the in-mine trial, the results may be somewhat misleading. This fact is further substantiated in that the AES was only operated in the ACS mode for 15.5 feet of the total advance. Therefore, the data for the most part represents only the ability of the operator, not of the ACS, to maintain the designed entry dimensions and centerline orientation. However, the limited data did indicate that the correct centerline orientation and pillar dimensions were maintained by the AES.

Since the mean deviation from the mining plan is less than one foot for both entry width and centerline deviation in the AES-mined entries, there would appear to be no correlation with the roof deterioration that occurred. However, entry width does seem to be a factor in Entries 4 and 5 where they were mined by the 6 CM (see Figures 22 and 23) in the intersection just inby Survey Stations 115+65 and 115+78, some roof deterioriation recurred.

3.6.4 Canopy-Protection Audit

Because of the limited operating time the results of the canopy-protection audit were not totally conclusive. Preliminary results indicated the following:

- The canopies were effective, as evidenced by the fact that no roof-fall injuries occurred.
- The canopies restricted the vision of the bolter operators, though this restriction did not seem to affect performance of the bolting job.
- The canopy (roof-support) cylinders restricted the vision of the AES operator, which would appear to limit the efficiency of the system for manual, on-board operation. However, manual operation was capable from a remote location by the use of the umbilical cord, although it should be noted that the operator was then removed from the protection of the AES canopy.
- The roof-beam assembly caught cap pieces and disturbed the the roof coal during tramming. Many delays to tramming were attributed to lowering the roof-beam assembly for clearance purposes.

3.6.5 Ventilation and Dust Control Audit

3.6.5.1 Ventilation System Analysis: The AES' fans were rated to provide 6000 cfm of air to within 81 inches of the face. Measurements show that the average entry dimensions were a width of 15.95 feet and a height of 8.45 feet. These figures produce an average area of 134.78 square feet. The AES, at the intake of the ventilation duct, has an area of 4108 square feet. The area of the remaining opening, then, is 93.70 square feet. The area of the openings of the air duct is 1.21 square feet. Therefore, the maximum air velocity (V_{max}) that can be supplied by the AES is

$$V_{max} = \frac{600\ ft^3/min}{(93.70\ ft^2 - 1.21\ ft^2)} = 64.87 \sim 65\ \text{fmp},$$

which is approximately 5 fpm above requirements of 60 fpm as specified by the Federal Law. For the AES to be in compliance with the Law, it must also supply a quantity (Q) of

$$Q = (60\ \text{fpm})\ (93.70\ ft^2 - 1.21\ ft^2) = 5549.40\ \text{cfm},$$

excluding losses. Therefore, theoretically, the AES met the specified air quantity and velocity requirements of the Federal Law. However, the readings of velocity taken in the mine showed that velocity requirements may not always have been met at the face. Further study is necessary to verify the performance of the ventilation system especially because head losses could not be identified during the in-mine trial.

Methane readings were within Federally specified limits (less than two percent). However, only a limited number of readings were taken, and additional studies over an extended period are necessary in order to verify how well the AES diluted and carried away methane.

The AES seemed to limit respirable dust. However the finding obtained is based on one data point. Additional measurements of the average amount of respirable dust are necessary.

Methods of ventilating the face without the AES should be investigated in order to maintain the necessary quantity and velocity of air moving across the face at all times. A brattice system was used during AES downtime and withdrawal from the face, but the optimal system should be self-deploying and should be capable of working automatically even if the AES' ventilation is disrupted.

3.6.6 Lighting Audit

The results of the Lighting Audit indicate that Federal standards were being met. However, face readings were limited and, therefore, additional ones are needed to verify the findings.

3.6.7 Noise Audit

It is apparent that noise may be a problem, since the readings show that the energy level is above allowable specifications. Also, the workers stated that the noise produced by the AES causes more discomfort than other miners. However, the results of the Noise Audit are not completely conclusive since measurements were limited.

3.6.8 Bolting Platform Stability Audit

The following preliminary observation can be made regarding the stability of the bolting platform: The force applied to the roof and floor was not sufficient to stabilize the floor platform. The stability was influenced by the oil and water on the floor which reduced the coefficient of friction between the floor beams and the fireclay, and by the fact that two roof-cylinders were pulled out of their lower sockets.

It is necessary, therefore, to determine the amount of anchoring pressure that is necessary to stabilize the bolting platform. When this pressure is tried on the AES, any movement of the platform should be noted.

3.6.9 Cleanup Ability Audit

Preliminary indications show the following results regarding the AES' cleanup ability: 1) Clean-and-trim time for the AES was limited (7.34 minutes per shift) because a pickup loader was used behind the machine. The AES was only used for cleanup when the material on the bottom affected its performance. Furthermore, the observations show that as much as five inches of material was left on the floor. This amount of material would be excessive if shuttle cars or continuous haulage were used directly behind the AES. Therefore, it may be said that the AES did not clean the floor well. 2) The material on the floor, when mixed with water and oil, may have contributed to 1) the gathering head motor failures by clogging the teeth on the underside of the discs and 2) the instability of the bolting platform.

3.6.10 Out-of-Seam Dilution Audit

It was suggested that the AES would limit out-of-seam dilution by mining under the draw slate or by mining it separately. However, neither was done, and so dilution remained the same as for the other units in McElroy Mine. In fact, if the fallen roof material is included, the dilution was substantially higher than for other sections of the mine.

CHAPTER 4

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CHAPTER 5

APPENDICES

APPENDIX 5-1

DETAILED COMMENTARY OF THE AES IN-MINE TRIAL

The AES in-mine trial, as shown in Figure 17 , has been divided into periods as follows:

- Assembly, April 26 - August 28, 1977.
- First Production Trial, August 29 - October 20, 1977
- Modification, October 20 - December 6, 1977; April 5 - April 10, 1978.
- Strike Interval, December 7, 1977 - April 4, 1978.
- Second Production Trial, April 11 - May 24, 1978.
- Disassembly, May 25 - September 2, 1978.

This appendix is a detailed account of these periods. The assembly is discussed with regard to problems encountered and component testing. The production periods are discussed on a week-by-week basis, including tonnages and distances mined, component failures, and problems encountered. The modification periods are discussed with regard to each major component or system that was changed. The final period is discussed with regard to the discoveries made during the disassembly of the AES.

5-1.1 ASSEMBLY - APRIL 26 TO AUGUST 28, 1977

On April 26, the first shipment of AES parts arrived at McElroy Mine and were sent underground soon afterward (within 1 month). No outstanding problems were encountered during transportation of the parts underground, but the roof drill control panels were damaged during the assembly process and were replaced.

The limited amount of working room did pose problems during assembly. An oil leak (due to insufficient weld) was discovered in the lower oil reservoir tank and was corrected. The automatic control cycle (ACS) did not operate properly because of improper adjustments on the roof-support pressure switches and because of a faulty relay in the sump counter. A stress crack developed on the underside on the inner-roof-beam assembly, just behind the left ventilation fan. The cause was determined to be unusually high pressures developing when the assembly was pressed against the roof over a three-day period. The crack was filled with weld and additional support members were added around it. The snap rings that hold the conveyor-takeup roller in place were accidentally not installed. This omission was corrected. The AES had some other major oil leaks (that caused the loss of approximately 25 gallons per day) located on the lower oil reservoir, the front inner-support cylinders, and the stinger value bank. The leaks were corrected. The methane monitor was replaced because it was faulty, and the sump cylinders had to be replaced because they were bent (cause unknown at that time). Approval for the roof-drill dust-collectors was granted following underground testing of the units.

A full production crew was assigned to the AES, and NMSC personnel and the mechanics familiarized the operators with the controls. Only nine shuttle-cars of coal were loaded from Entry 6 during the week of August 22 through August 28, so this week was not considered production time. Oil leaks on the roof supports, drill hoses, and ventilation fans, and a control cable that was crushed during operation hampered production.

5-1.2 FIRST PRODUCTION TRIAL - AUGUST 29 TO OCTOBER 20, 1977

5-1.2.1 August 29 - September 1, 1977

August 29 was spent completing the final preparation for mining, including the reinsertion of a control cable between junction boxes, the repacking of one conveyor-takeup cylinder, the replacement of a fitting in one ventilation fan, and the repairing of one faulty wire controlling the vertical movement of the tail boom.

On August 30, 54 raw tons of coal were mined, with an advance of 10 feet. The advance was achieved manually, by sumping from the crawlers. Ventilation was accomplished by the use of rigid tubing behind the AES.

Also on August 30, the cutter-head tie-shaft and retaining ring failed because of a design error. The cutter-head extension-jacks, designed for a 10-inch stroke, extended 10-1/4 inches. The retaining ring was positioned for the 10-inch movement, and allowed for 1/8-inch of clearance. The additional 1/8-inch movement caused the right-side retaining ring to be sheared and the left-side grove shoulder in the shaft to be damanged. A modified shaft that corrected the error was machined and installed on the AES.

The left dust-collector boxes were removed from the rigid brackets that held them in place and were remounted on chains. It was believed that this adjustment would prevent damage to the hose fittings and blower couplers that was caused when the boxes contacted the outer roof-support cylinders.

5-1.2.2 September 2 - September 8, 1977

On September 6, no coal was produced because of failure of the right gathering-head motor. The cause of failure was unknown at this time. A second gathering-head motor was installed on the afternoon shift.

Also on September 6, inflatable tubing was tried with the AES' fans. The most successful system consisted of using a slightly smaller diameter tubing inside a large section. As the AES advanced, the smaller tubing slid forward and provided continuous ventilation for 25 feet. Ventilation was interrupted only momentarily as other sections were added and the smaller tubing would again be inserted into the larger section.

A second gathering-head motor on the right side failed after 9.1 minutes of operating time. This failure halted production for the remainder of the week. At the time, the cause was believed to be in the gathering-head motor overloads. The heater overloads (NMSC #3006-6591B) were rewired in order to enable the breakers to trip if only one head stalled. Previously, both heads were required to stall to trip the heater overloads.

A pin on the right-rear-outer support cylinder was sheared during mining and was replaced. The left sump-cylinder was replaced because of a bent rod. The cause was unknown. The replacement cylinder leaked because an O-ring had been omitted during manufacturing. Several oil leaks on the front-inner-support cylinders were repaired.

5-1.2.3 September 9 - September 15, 1977

The heater overloads (NMSC #3006-6591B) for the gathering-head motors were replaced with ones requiring less amperage in order to trip. The instantaneous breakers were found to be set too high and were reset to trip at approximately seven amps. The third gathering-head motor was installed on September 12.

On September 13, 120 raw tons of coal were mined with an advance of approximately 23 feet. However, before any bolts could be installed, a short circuit was discovered in the ventilation-fan electrical circuit. Shortly afterward, the draw slate fell from the roof and smashed 1) a cable to the area lights, 2) the sump-counter cable, and 3) an amphenol connector on the main control cable. These were replaced on the afternoon shift. A damaged hydraulic hose to the left gathering-head-extension cylinder was also discovered and replaced.

A loading machine was used behind the AES for the first time on September 13; and during the afternoon shift, it was used to remove some of the rock that had fallen.

No coal was mined on September 12 because of the short circuit in the ventilation-fan-electrical circuit. A major oil leak on a hydraulic return line was discovered and corrected during the afternoon shift.

On September 15, 60 raw tons of coal were mined with and advance of approximately 11 feet until a minor roof fall covered the cutter head and part of the roof beams. A cable to the rear area-lights was severed by the left sump-cylinder. Much time was spent placing cribbing and cap blocks under the crawlers to facilitate tramming in the muddy bottom. The mud was created by excess water and oil, and the weight of the AES aggrevated the problen.

It was discovered that the gathering head did not float but rode up and over the rock on the uncleaned bottom. As a result, the AES cut higher into the roof than was desired, and exposed a claystone that deteriorates more rapidly than the usual immediate roof.

5-1.2.4 September 16 - September 22, 1977

No coal was mined on September 16 because the operator was attending a mine-rescue class. However, a broken hose on the bottom of a left-inner bolter-cylinder was replaced, and belting was placed on the sides of the conveyor compartment in order to prevent spillage into operators' compartments.

On September 19, 20 raw tons of coal were mined with an advance of approximately 4 feet. The gathering-head extensions would not open because one was bent and contacted another part of the head. The problem was corrected. A great amount of time was spent leveling the AES in order to mine down to the proper floor horizon. Cribs were built and the AES backed on to them. When this arrangement did not work, the cribs were replaced with wooden ties. This system was successful, and the AES mined down close to the floor rock.

The newly exposed roof was stopered on the next shift in an attempt to hold the poor roof. In addition, work was begun to install controls which would provide independent operation of the inner roof-beam from front-to-rear.

On September 20, 30 raw tons of coal were mined with an advance of approximately 6 feet. The right gathering-head motor failed again. Two possible causes were postulated, and thus two actions were taken. 1) Water recirculation to the motor was poor. Investigations revealed that the back pressure from the mine water supply feed (190 psi) was too high, thus a pressure reducing valve opened. The valve was removed and recirculation improved. 2) The gathering-head-extensions wings (3/4 inches x 23-1/2 inches x 4 inches) were contacting the vanes on both small discs. The extensions were forced down upon the vanes when the sideboards were retracted, and the cutter-head stop-block rested on the gathering head. Both extensions were removed. However, the motor failure halted production until September 23.

No work was done on the AES on the September 21 day shift because there was no mechanic for the section. On the afternoon shift, a broken grease breather on the bottom of the water pump was discovered and repaired. The dust-collector boxes on the right side were suspended from the outer roof beam with chains in order to permit more flexibility and movement.

On September 22, a rebuilt gathering-head motor was partially installed. The installation was completed on September 23, and 100 raw tons of coal were mined with an advance of approximately 20 feet. The pilot pressure to the gathering-head float control was adjusted with the result that the problem of cleaning loose coal and rock from the bottom was corrected. During mining, a leak in the left ventilation-fan motor was discovered. During the afternoon shift, a minor roof-fall occurred.

5-1.2.5 September 23 - September 29, 1977

The left ventilation-fan motor was removed and replaced on September 26. The afternoon shift crew did not work because ventilation in the mine had been disrupted from a fall.

No work was done on the day shift of September 27 because no fireboss was present. The work on the left ventilation fan was completed on the afternoon shift. Also, the dust-collector blower-couplers were replaced (Figure 31); these had become damaged because the dust-collector boxes were contacting the outer-support cylinders.

On September 28, 48 raw tons of coal were mined, with an advance of approximately 9 feet. The first place change occurred when the AES was moved from Entry 6 to Entry 5. The place change occurred before the AES could mine to the next crosscut, because of 1) poor roof-and-floor conditions and 2) the inability of the shuttle cars to negotiate the turn into Entry 6 from the crosscut because of the hole dug by the action of the AES crawlers.

During the afternoon shift, a damaged connector to the methane monitor was replaced and the sump-counter cable was reinserted. The left ventilation-fan motor was again replaced. A spring (NMSC #9261 - 2845) in the relief valve was found to be undersized and thus to allow excess pressure to build up. The buildup caused seals failure and motor cavitation. The spring was replaced with one that was the correct size.

Both shifts on September 29 were used to look for hydraulic leaks. Some major ones were discovered on the front inner roof-support cylinders. The fittings were being stretched and damage when they contacted the cylinder housing. The left-front, inner-support cylinder was removed. While it was being reinserted, the key support broke and the new fitting was also damaged.

5-1.2.6 September 30 - October 6, 1977

On September 30, the left-front inner-support cylinder was replaced because the piston-port threads were stripped.

On October 1, the Sundstrand variable-volume pump (NMSC #7800-6525), and the piston shaft between the variable-volume pump and the water pump were removed. Also, the in-line flow meter downstream of the variable-volume pump failed, with the result that some pieces of metal entered the hydraulic system. These repair actions halted production until October 5.

On October 3, a defective bearing (NMSC #1101-8868) was discovered in the pump-drive-gear case. The bearing may have caused the failure of the Sunstrand pump. After the bearing bore was enlarged, a new gear was installed. Installation of a new variable-volume pump was completed.

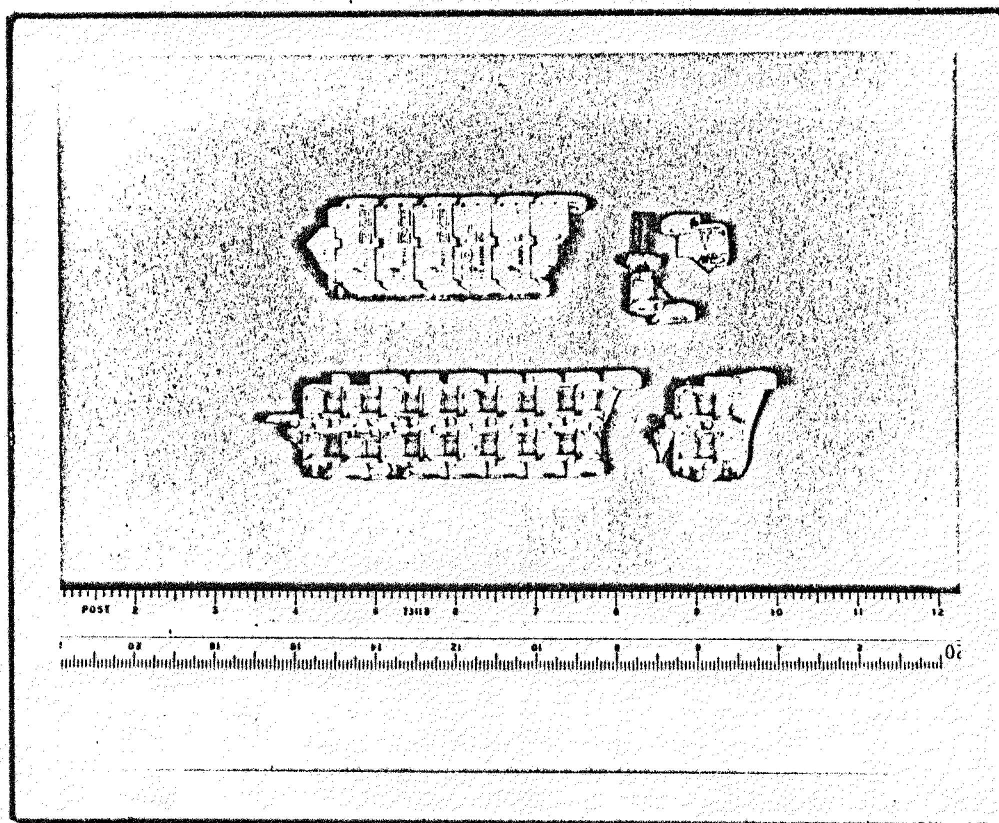


Figure 31

Broken Dust Collector Blower Coupler

On October 4, three aluminum fragments from the in-line flow meter were found in the outer-support valve bank. They jammed the valves and prevented retraction of the outer supports.

On October 5, 75 raw tons of coal were mined with an advance of approximately 15 feet. During the shift, two additional pieces of metal from the flow meter caused malfunctions in the conveyor vertical-movement valve and outer roof-support valve-bank. On the afternoon shift, work continued on the modified inner roof-support controls.

On October 6, a bent sump-cylinder rod was discovered and the cylinder was removed. The water pump was replaced because its seals had failed. The AES was serviced, and Junction Box 3 was relocated to prevent it from contacting the cutting boom.

5-1.2.7 October 7 - October 13, 1977

On October 7, the right sump-cylinder was replaced, and the installation of the new inner-support controls was continued.

On October 10, a small hydraulic hose to the sump-cylinder motion-control valve that had failed prevented the AES crew from loading coal. The hose failed when difficulty was encountered in retracting the sump cylinders. In addition, a fall occurred in the face area during the repair operations.

On October 11, 125 raw tons of coal was mined with an advance of approximately 24 feet. Considerable production time was lost because of delays such as aligning roof beams, cribbing under crawlers, and retracting the sump cylinders. It was believed that the sump cylinder motion control valve might be damaged and thus might be contributing to the sump-cylinder delays. Repairs to one of the dust-collectors' motor mounting brackets was completed.

On October 12, 120 raw tons of coal were mined with an advance of 24 feet. Delays included maintenance on the dust-collector blowers, the reducer-bushing-muffler assembly, and the sump-cylinder-motion control-valve, which was relocated for easier access. On the afternoon shift, 14 water-spray nozzles were replaced with mist-type nozzles.

On October 15, 115 raw tons of coal were mined with an advance of approximately 23 feet. Four major delays resulted from the placing of cribs (necessitated by muddy floor conditions) under the crawlers. Additional water sprays were plugged in order to reduce the amount of water introduced to the face. The roof exposed during the day shift was supported by stoper crews on the afternoon and midnight shifts.

5-1.2.8 October 14 - October 20, 1977

On October 14, 50 raw tons of coal were mined with an advance of approximately 10 feet. During mining, the main frame failed in three places where the stinger assembly is attached (see Figure 32). The

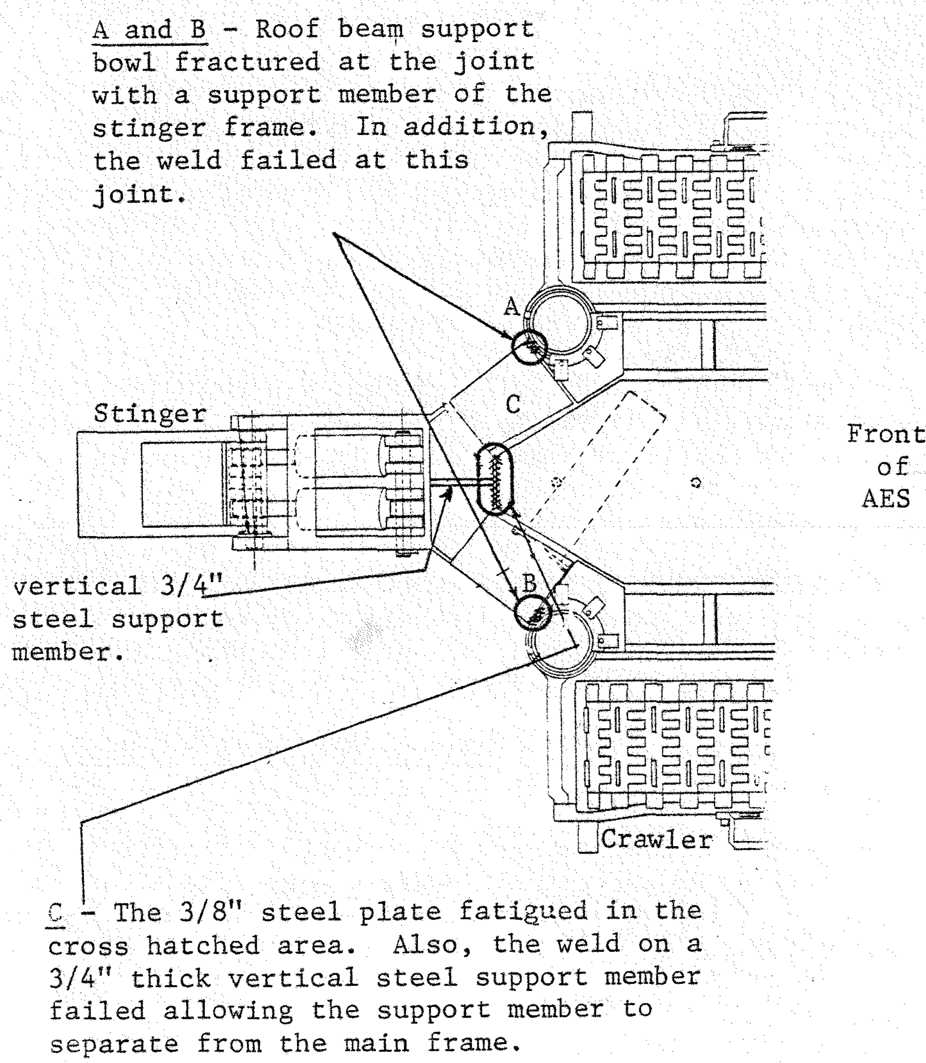


Figure 32

Structural Damage to the AES at the Stinger
(Before Modification)

Courtesy of National
Mine Service Company

failure was apparently caused by insufficient weld at the rear of the oil reservoir and on the top plate. The weld had been subjected to stresses over a long time, and to strengthen the area, five reinforcing-bars were welded across the broken area. The frame failure halted production until October 18.

On October 17, a relay in the sump distance counter was found to be defective and was replaced. Also, several hydraulic leaks were corrected, and the AES was serviced and made ready for the next production shift.

On October 18, 67 raw tons of coal were mined with an advance of approximately 13 feet. Much time was spent placing cribs and planks under the crawlers. Over an hour of production time was needed to install four roof-bolts, because of 1) improper drill rotation, 2) inexperienced bolter operators, and 3) the lack of good drill steels. At the end of the shift, the unsupported, exposed roof fell and covered the cutting head and roof beams. Stopering of the fall area was not completed until the October 20 day shift.

On October 19, a major oil leak was discovered in the solid-pipe plumbing located in the left-outer roof-beam. The pipe was replaced with a 10-foot-long hose. Several other hoses were replaced, and oil leaks on the bolter valve-banks were eliminated. The left front dust-collector blower was also reinstalled.

On October 20, Consol management decided to advance 1 South out of the area of poor geological conditions by using a Joy 6 CM miner. NMSC personnel decided to use this time to modify the AES in order to halt some of the problems hampering its performance.

5.1.3 MODIFICATION - OCTOBER 20 THROUGH DECEMBER 6, 1977; APRIL 5 THROUGH APRIL 10, 1978

During the periods October 20 through December 6, 1977 and April 5 through April 10, 1978, the AES underwent modifications in order to alleviate several design and environmental problems. The modified components include the stinger assembly, roof-drill dust-collector boxes, roof-drill centralizers, roof-drill pods, and sump cylinders. The drill-pod modifications were completed during the second production period. In addition, the inner-roof-beam assembly was modified to operate independently from front to rear. This modification took place during the first production period and the work was classified as preventive maintenance. A total of 662.24 manhours were spent performing the modifications discussed below.

5-1.3.1 Roof-Drill Pods, Dust-Collector Boxes, Centralizers, Dust Collector-Motor Couplers, and Drill Masts (116.44 manhours, 17.6 percent of the total modification manhours)

The original drill pods were replaced with deep-chuck units in order to make operation of the bolters safer.

The dust-collector boxes were relocated on the floor beams in order to prevent them from becoming caught on the roof-support cylinders. When the boxes became caught, the couplers between the motors and blowers would break. One dust box was damaged once.

The drill centralizers were changed in order to allow them to operate independently of the drill need. This modification also provided safer operation.

The dust-collector motor couplers were changed to metal ones in order to prevent breakage of the couplers.

The center section of the roof-drill masts were welded to the other sections because the additional three feet of reach was not required at McElroy Mine. This action decreased the amount of time required to raise the drill.

5-1.3.2 Stinger Modification (442.54 manhours, 66.8 percent of the total modification manhours)

The stinger was redesigned in order to 1) improve the ability to level the rear bumper with respect to the tractor frame, 2) shift the center of gravity of the tractor frame from near the front idlers of the crawler pads, 3) allow it to raise the bumper assembly above the bottom during tramming, and 4) keep the tractor frame from pitching forward during mining--a motion which had contributed to the sump-cylinder damage. In addition to the work done on the stinger itself the guide slide, "T"-bar slide, and lower oil-reservoir sight-gage were changed in order to facilitate the stinger modifications.

5-1.3.3 Sump-Cylinder (26.0 manhours, 3.9 percent of the total modification manhours)

The stroke of the sump cylinders was shortened in order to prevent the rods from becoming bent.

5-1.3.4 Scrubber and Water Spray (5.65 manhours, 0.9 percent of the total modification manhours)

In order to reduce the amount of water introduced into the face, three sprays in each scrubber and 26 dust-suppression sprays were plugged. One spray remained in each scrubber, and 14 dust-suppression sprays remained.

5-1.3.5 Other Modifications

The remainder of the modification manhours may be attributed to miscellaneous work (e.g. prepare to start, get tools, maneuver machine). (71.6 manhours, 10.8 percent).

5-1.4 STRIKE INTERVAL - DECEMBER 7, 1977 - APRIL 4, 1978

During the in-mine trial, the 1974 UMWA contract agreement expired on December 6. The new contract went into effect on March 27. During this period, no work was accomplished on the AES. In fact, work did not begin again until April 5 in order to aid Consol personnel in achieving a smooth return to work for the miners.

5-1.5.1 April 11 - April 13, 1978

On Tuesday, April 11, the AES cut floor rock and clean bottom for a distance of 120 feet. A 7-man crew (excluding one shuttle-car operator) produced 100 raw tons of coal. No roof bolts were installed. The major maintenance delay was caused by the removal of the gathering-head extensions (20 minutes). The other major section delays were "prepare to start" and "adjusting the roof supports," actions which required 105 and 38 minutes, respecitvely. The gathering-head discs stalled as a result of the gathering-head extensions being forced down on the discs because a corner of coal had been left by the 6 CM during mining. The conveyor also periodically stalled when the AES was cutting and loading bottom. However, when the cutting head and gathering discs were raised off the bottom, the conveyor operated again.

Oil was discovered in the connection box on the right-outer drill. Three persons from the day shift worked four hours overtime on the AES.

On Wednesday, April 12, a 7-man crew advanced the AES 11 feet and produced 72 raw tons of coal. Machine cutting and loading time amounted to 62 minutes, for a production rate of 1.2 tons per minute out of an available 261.00 minutes. Three roof bolts were installed by the AES. The roof condition was beginning to improve and some roof coal began to appear. The newly exposed roof (11 feet) was not supported at the end of the shift. The bottom condition, the usual fireclay, became muddy when mixed with the oil and water deposited by the AES.

The major service-delay was for oil and required 6 minutes. The other major section delays were: cribbing under the crawlers and bolting (11 and 54 minutes, respectively). A small crack was discovered on the new stinger-assembly where it was welded to the AES.

The second deep-chuck drill pod was installed on the left-inner drill mast. The "dip" relay in the sump-counter board was replaced.

The crack in the stinger was welded on April 13 as shown in Figure 33. The sump counter was repaired by replacing a defective microswitch. The right front and both left dust-collector couplers were replaced. An oil leak was discovered on the left-inner drill-mast but not repaired because of the lack of replacement parts.

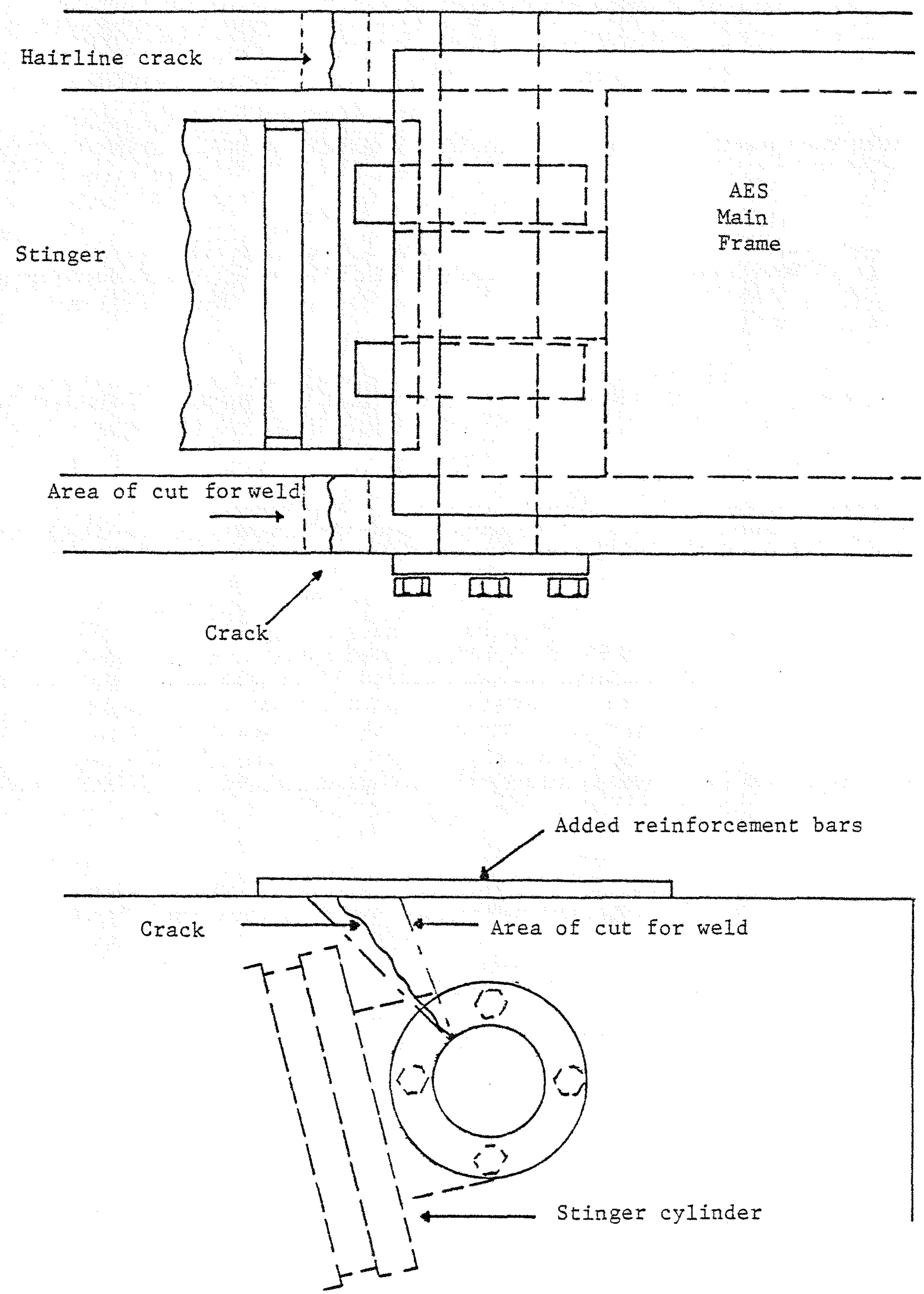


Figure 33

Structural Damage to the AES at the Stinger
(After Modification)

5-1.5.2 April 14 - April 20, 1978

On Friday, April 14, the AES advanced 20 feet in Entry 5, its 7-man crew mining 120 raw tons. Machine cutting- and loading-time amounted to 66.40 minutes (a production rate of 1.8 tons per minute) out of an available 380.50 minutes. Three roof-bolts were installed. When the AES pulled out of the face, the spot bolter was brought in immediately and part of the area was bolted. However, no temporary supports were installed and a distance of 11 feet to the face was left unsupported at the end of the shift. The bottom condition near the face was still muddy and rutted.

The major maintenance delay was to replace the right gathering-head motor that failed during operation (148.00 minutes). The other major section delay was for installing three roof-bolts (43.00 minutes). The large delay time for bolting was caused by 1) a clogged hose on the left outer-bolter dust-collection system, 2) the bolter operators experienced difficulty with the drill steels sticking in the holes and the bits coming off, and 3) the centralizer-arm linkage on the right-outer drill required adjustment.

On Monday, April 17, the new gathering-head motor installation was completed. The AES advanced 15 feet in Engry 5 and produced 77 raw tons of coal with a 6-man crew (1 shuttle car operator was absent). Machine cutting and loading time amounted to 75.40 minutes (a production rate of 1.0 tons per minute) out of an available 360.50 minutes. One roof-bolt was installed. An 11-foot span of roof had been left unsupported after the Friday shift. A 5-foot advance was made, sumping the machine with the crawlers, which were used rather than the sump cylinders because planks prevented the AES' roof beams from completely contacting the roof. Because the crawlers were slipping on the muddy bottom, it was decided to use the roof-beam assembly anyway, and both sump hydraulically and use the crawlers. After about an additional 6-foot advance, the roof fell (at about 2:00 p.m.) over the inner supports and the cutter head as the inner supports were lowered for a third sump. Time was taken to clean the debris from the AES and manuever it for clean-up. Near the end of the shift, an oil leak was discovered at the junction of a return manifold and the filter housing. Temporary supports were set at the face. The AES; trailing cable was damaged by the canopy on the loading machine, and was spliced temporarily. The major service delay was for adding gear lube oil to the cutter-head and gathering-head pots (35 minutes). Wiring the gathering-head motor at the beginning of the shift (54.50 minutes) was the major maintenance-delay.

The delay caused by the roof fall (97.70 minutes) was the major unnecessary-delay. A delay of 32.8 minutes for bolting was caused by insufficient drill-rotation speed on the right-inner bolter and difficulties with bolt placement on the left side (the resin set up before the bearing plate was tight against the roof). The threaded section of the manifold that screws into the bottom of the filter housing was found to be cracked and causing a major oil-leak. The manifold was removed, repaired, and reinstalled. The roof fall that occurred on day shift was suggested as the cause of broken manifold. The left-front area-light control cable had to be re-entered into the packing gland; it was torn loose either during the roof fall or when the outer supports were raised. So that maintenance work could be done, the AES was moved to the last open crosscut. The result of this move was the loss of a large amount of hydraulic fluid.

On Tuesday, April 18, two bolter-operators supported the fall area at the face of Entry 5. The 6 CM miner was moved into position to begin mining on April 19. Two barrels (110 gal) of hydraulic oil were pumped into the AES. The damaged power-cable was re-entered into the plug on the AES; 37 feet of cable were removed. Several oil leaks were discovered and corrected. Near the end of the shift, it was discovered that the weld on the lower left-corner of the extensible ventilation duct had failed, probably during mining. The corner was rewelded.

The control cable to the left-front area-light on the outer supports was re-entered into the packing gland. The cable had pulled out because there was not enough slack when the outer roof-supports were raised. The latter problem was corrected by routing the cable differently. The centralizer valve for the left-outer bolter was positioned so that it would not shear from position as the bolter pod was raised and lowered. The ACS was checked to see if it worked properly. It appeared that the pressure switches on the inner supports were not activating to allow continuation of the cycle.

On Wednesday, April 19, the microswitch in the sump-distance counter was replaced, the roof supports were adjusted, and the AES was trammed forward to begin testing the ACS. The outer roof-support system was checked. The advance and retract rates of all cylinders were adjusted so the cylinders move at the same rate. The right-outer centralizer arms were lubricated for smoother operation, and two oil-filters were replaced. The position of the stinger switch was reversed so that the stinger operated according to the labels on the control panel.

On Thursday, April 20, the AES was positioned for testing the ACS. It cycled through various steps, but the sump distance was incorrect. In addition, the output pressure of the Sunstrand pump was set too low to allow complete retraction of the sump cylinders; the pressure was adjusted. About 90 gallons of hydraulic oil were pumped into the AES. During the afternoon shift, the ACS was tested and slow rates to several roof-support cylinders were adjusted. The control system ran successfully except that the sump-distance counter did not work because the right crawler was slipping on the bottom. While the control system was being tested, the control cable to the left-front area-light was again pulled out of the packing gland. Plans were discussed for rerouting the cable to correct this problem.

5-1.5.3 April 21 - April 27, 1978

On Friday, April 21, four bars of steel were welded to the rear platform to restrict the movement of the conveyor-lift cylinders when the conveyor was retracted from an extended position. The original left-inner drill-pod was replaced with a deep-chuck pod. All drills, except the left-outer drill, now have deep-chuck pods. The automatic-control system (ACS) was checked, but the cycle would not reset. The wires to the pressure switch for the right-front-outer support-cylinder were found to be disconnected. When the switch was rewired, the cycle reset automatically. When the operator tried to move the AES back to an intersection, the stinger would not lift the rear platform. Adjustment to the pressure on the pilot-hydraulic circuit corrected the problem. The AES was prepared for a shutdown period; the roof supports were leveled, and the control pendant and umbilical cord were locked in the parts car.

No work was performed on the AES until after April 27 in order to allow the 6 CM to prepare the entries for another production trial for the AES.

5-1.5.4 April 28 - May 4, 1978

On Friday, April 28, the maintenance crew worked on replacing with a jet valve, the hand-filler pump for the hydraulic oil. The job was never completed.

On Monday, May 1, no work was performed on the AES because no mechanic was available, and no keys were available to the tool boxes.

On Tuesday, May 2, the crew began troubleshooting an oil leak on the left side of the AES. It was determined that a seal might have failed between a pump and the pump-drive-gear case. When the drain to the gear case was opened, hydraulic oil was found to have entered the case and the pumps, probably because the 0.003- to 0.005-inch undersizing of the bore for the bearings was responsible for the failure of the seals. The seal between the gear case and the pump motor had also been pushed out and, as a result, the oil leaked out between them. The gear case was removed and was reworked. The five hydraulic pumps were also replaced.

5-1.5.5 May 5 - May 11, 1978

There was no work on the AES from May 5 through May 11 because of the failure of the pump-drive-gear case. The Audit Team used the down-time to update the survey, and the roof and floor profiles of the 1 South entries recently mined by the AES and the 6 CM.

5-5.5.6 May 12 - May 18, 1978

On Friday, May 12, the maintenance crew bolted the gear case to the pump-drive motor and to the tractor frame.

On Monday, May 15, the following pumps were connected to the gear case: The variable volume, the conveyor, the scrubber, the right bolter, and the ventilation fan pumps. The hoses to the water pump were also connected.

On Tuesday, May 16, the left bolter pump was installed, all hoses were connected to the pumps and the gear case, and about 40 gallons of hydraulic oil were pumped into the AES. The installation of a breather line between the hydraulic fluid reservoirs was initiated.

On Wednesday, May 17, the final deep-chuck drill-pod was installed on the left-outer bolter, and the connection of the breather line between the hydraulic reservoirs was finished. The operation of several hydraulic components was checked to insure that they worked properly after the gear-case task was completed. Water hose was connected to the AES so work could be done which required the operation of the pump motor. Grease hoses which connected to the crawler-take-up cylinders were replaced because the originals were damaged during modifications. Problems were encountered with the operation of the right-inner drill, and NMSC personnel throught that a pilot valve was malfunctioning. The left-front area-light would not operate because of a damaged cable.

On Thursday, May 18, the damaged cable to the left-front area-light was replaced. The operating pressure to the feed and drill-rotation components of the right-inner bolter was found to be incorrect, and the rotation-rate potentiometer was not providing a variable signal to the Bertea control-valve. Troubleshooting efforts were not successful. However, NMSC personnel stated that the problem was probably in the hydraulic system.

5-1.5.7 May 19 - May 25, 1978

On Friday, May 19, a piece of metal was found in the relief valve of the right-inner bolter that was holding the relief valve open and not allowing the pressure to increase to its proper operating value. The roof-bolter dust-collector boxes were installed on the AES and the hoses were connected. A problem with the stinger cylinder was found. NMSC personnel thought that either the control valve was malfunctioning or the Bertea control was not receiving a signal.

On Monday, May 22, the problem with the ability to vary the drill rate of the right-inner bolter was investigated. It was determined that the potentiometer for the unit was just more sensitive than the other drill, and thus, the unit remained in service. The stinger problem was traced to the torque motor for the Bertea valve-section, where a wire was pulled from its contact. This problem was to be repaired on the next working shift.

On Tuesday, May 23, the repairs to the stinger torque-motor were completed by re-entering the wires into the motor. The stinger would not operate correctly, and it was believed that oil was getting into the motor leads. While an attempt was being made to remove the motor once more, the leads were again pulled from the motor. An additional length of wire was added, and the motor gasket was replaced to complete the repairs. The AES was trammed to the face of Entry 6 and prepared to mine on Wednesday, May 24.

Several minor maintenance-actions were necessary during tramming. The ACS was tested, and it was theorized that the pressure switches for the inner roof-supports were not making contact. However, no work was performed to correct the problem. Part of the crew worked 2.75 hours overtime to aid in preparing the AES for a good operating shift on Wednesday.

On Wednesday, May 24, the AES advanced 15.5 feet and produced 96 raw tons of coal in the automatic-control-mode utilizing a 7-man crew. Cutting and loading time amounted to 139.6 minutes, or a production rate of 0.69 tons per minute, out of an available 367.9 minutes. The roof consisted of good roof-coal that had been broken out around the bolts by the action of the roof beams (see Figure 21), which may also have been caused by areas of slickensides and weaknesses being present in the roof rock. The bottom condition was good fireclay but became muddy because of the water and oil that were introduced into the face by the AES.

The major service delay was for the addition of 55 gallons of hydraulic oil (19.00 minutes). The major maintenance delay was the repair of a broken water-spray manifold-block (34.00 minutes). The crew was prepared to work over into the afternoon shift, but the decision was made to use the shift to investigate the leaks which were depositing large amounts of oil on the bottom. Also, the floor beams were slipping during sumping, also a problem that was investigated during the afternoon shift. Four members of the crew remained to tram the AES out of the face and to bold the newly exposed roof. Much oil had been seen leaking from various places on the AES, so some time was spent troubleshooting and fixing oil leaks. It was discovered that the outer support-cylinders had broken from their lower housings because of excessive cylinder movement. It was estimated that several work shifts would be required to repair the cylinders.

On Thursday, May 25, a meeting of NMSC, Consol, and DOE personnel was held at McElroy Mine, and a tentative decision to disassemble the AES was made. It was decided to begin the disassembly on June 12. The maintenance crew did repair several oil leaks, and trammed the AES to a spot outby the last open crosscut in Entry 5 to begin disassembly. The mechanic who trammed the AES stated that the stinger modifications greatly aided tramming, and that the AES maneuvered more easily than other machines he had operated.

5-1.6 DISASSEMBLY PERIOD - MAY 25 THROUGH SEPTEMBER 2, 1978

During the disassembly period, the West Virginia Department of Mines issued an interpretation of the State Mine Law which said that no equipment should be moved while workers remained inby the move. This ruling included all equipment, so that the mine was able to move only workers, supplies, and coal. This ruling, tabled for a 45-day cooling off period before some negotiations could begin, hampered disassembly and transportation activities for the AES personnel.

The only major discovery made during this period was that the rear-left-outer, and the left-rear support-cylinders were broken from their lower housings, possibly because of excessive cylinder movement.

APPENDIX 5-2

GRAPHICAL ANALYSIS OF MAINTENANCE AND OPERATING PARAMETERS

This section contains a graphical analysis of the data that was used to compile Tables IV, X, XI, XII, XIII, and XIV. The purpose of this analysis is to correlate maintenance and operating parameters and to use a Student's "t" distribution to determine the degree of significance of each correlation. The value of the degree of significance of correlation (DSC) was used to group the parameter pairings into categories of strong, moderate, and poor correlations in order to make observations based on findings of this section.

The representative graphs presented here (at the end of this Appendix section) are explained by the use of alphabetic references, marked on each figure, which point out various events during the in-mine trial (see Appendix 5-1). The letters usually represent maintenance actions on one or more components or systems.

The following is a sample calculation to determine the degree of significance of correlation for the parameters in Figure 34. The maintenance data represents the values from the in-mine trial that were used to form part of Table X.

The lower portion of Figure 35 shows the maintenance manhours on a week-by-week basis, and the upper graph shows an operating parameter. In this case, the maintenance parameter is the manhours required to maintain the AES' roof drills and dust collectors. The upper graph portrays the number of roof bolts inserted during each week. Reference A shows maintenance on broken dust-collector couplers, motors, or blowers. Reference B shows the maintenance in broken piping or hoses that were replaced, a damaged dust collector motor that was replaced, and oil leaks that were repaired. Reference C shows when maintenance was performed on the right-inner drill-rotation potentiometer.

If these parameters were compared directly (i.e., if the graph showed the manhours required as a function of the number of bolts inserted), a scatter plot would result. Then, by regressing a straight line that fits best through the points, one may calculate the correlation coefficient, or the value r, where

$$r = \sqrt{\frac{[\Sigma(x - \bar{x})\ (y - \bar{y})]^2}{[\Sigma(x - \bar{x})^2]\ [\Sigma(y - \bar{y})^2]}}$$

At r = 0, there is no fit, and at r = ±1, the fit is perfect (minus shows a negative correlation). For this example, r - 0.73.

By use of Student's "t" distribution equation

$$t = \frac{r\sqrt{N - 2}}{\sqrt{1 - r^2}},$$

the value for t may be determined. In this case, N, the number of samples, is 15 weeks. The value for t in this case is 3.87. If one refers to a table of Student's "t" values, this value of t (at N - 2, or 13 degrees of freedom) is above the 99.5 DSC. Therefore, the significance of this correlation is extremely high, or, in other words, the correlation is not simply a "chance" situation. However, a closer examination of the references for Figure 34 shows that much of the maintenance was on the dust-collector motor-blower couplers. The damage was caused by movement of the roof-beam assembly. A check of the possible relationship between the roof drill and dust collector maintenance and the roof-beam assembly adjustment time (Figure 35 (because the couplers were damaged when the roof-beam assembly was adjusted) reveals an r of 0.56 and a DSC of 98.3 percent, which is a highly-significant, indicated correlation.

If the DSC were low and the value for r high, then the correlation was probably a chance result. The authors are assuming that a 95 percent DSC is a high degree of significance, a 75 to 95 percent DSC is a moderate one, and a 55-75 percent DSC is a poor one. In addition, the assumption is made that r values less than ±0.20 indicate no correlation, values between ±0.20 and ±0.40, respectively, are possible correlations, values between ±0.40 and ±0.60, respectively, are indicated correlations, and r values above ±0.60 are good correlations. However, if the DSC is poor (55-75 percent) and the value for r is also low (less than 0.20), one may assume that, 1) there is no correlation; 2) the correlation is not linear, but may be defined by a polynominal expression, or 3) the correlation is linear, but this sample does not show the relationship. In such cases, a closer look at the data is usually beneficial in arriving at a conclusion.

It must be noted that for small samples, the value of r might be slightly positively biased, so that higher values for r, and correspondingly for t, might result. However, this difference is usually small and may be neglected.

Figures 36, 37, and 38 represent examples of the analyses performed on the remainder of the values in Table X. The references on each graph and results of the correlations are discussed below.

In Figure 36 (Cutter Head vs. Cutting and Loading Time), reference A shows maintenance on the damaged cutter-head tie-shaft. Reference B shows work performed on the damaged motion-control valve. The DSC was determined to be 55.9 percent for an r of 0.04.

In Figure 37 (Gathering Head and Conveyor vs. Cutting and Loading Time), reference A shows work performed on the repacking of the conveyor take-up cylinder. Reference B shows gathering-head motor-failures. The DSC was determined to be 63.9 percent for an r of 0.10.

In Figure 38 (Ventilation and Dust Suppression vs. Cutting and Loading Time), reference A shows manhours used to plug water sprays in order to reduce the amount of water introduced into the face. Reference B shows repair time on a loose ventilation-fan wire in a junction box. Reference C represents a replacement of two ventilation-fan motors due to an undersized spring in a relief valve. Reference D represents welding on a crack in the extensible ventilation duct. The DSC was determined to be 56.3 percent for an r of 0.04.

Table XXVII summarizes the DSC's and the r's for the maintenance parameters investigated. The listing is made in the decreasing order of significance with divisions made to show highly, moderate, and poor correlations.

The interpretation of this analysis is discussed in Chapter 3.

TABLE XXVII
Maintenance and Operating Parameter Correlations

CORRELATION DESCRIPTION	MAINTENANCE PARAMETER	OPERATING PARAMETER	CORRELATION COEFFICIENT	SIGNIFICANCE OF CORRELATION (PERCENT)
Highly-Significant, Good Correlation	Roof Drills and Dust Collectors	RBI[1]	0.73	>99.5
	Service	CLT[2]	0.61	99.2
Highly-Significant, Indicated Correlation	Roof-Beam Assembly	RBAAT[3]	0.59	98.9
	Automatic Control System	OT[4]	0.59	98.4
	Stinger	CLT	0.45	95.2
	Roof Drills and Dust Collectors	RBAAT	0.56	98.6
Highly-Significant, Possible Correlation	Other and Non-AES Maintenance	OT	0.24	97.8
	Other Components	OT	0.39	95.2
Moderately-Significant, Possible Correlation	Preventive Electrical	CLT	0.47	93.7
	Other Hydraulic	OT	0.31	89.4
	Methane Monitor	OT	0.34	88.7
	Hydraulic Hoses	OT	0.39	88.3
	Preventive Hydraulic	CLT	0.32	87.5
	Roof-Beam Assembly	CLT	0.31	86.7
	Control Apparatus, Adjustments	CLT	0.35	86.3
	Hydraulic Cylinders	OT	0.30	85.9
Moderately-Significant, Possible Correlation	Corrective Mechanical	CLT[2]	-0.35[6]	82.4
	Other Corrective	OT[4]	0.26	82.2
	Corrective Hydraulic	OT	0.22	82.1
	Hydraulic Fittings	OT	0.25	81.1
	Tram	TT[5]	0.21	78.3
	Hydraulic Pumps	CLT	0.20	75.8

(Continued)

TABLE XXVII (Continued)

CORRELATION DESCRIPTION	MAINTENANCE PARAMETER	OPERATING PARAMETER	CORRELATION COEFFICIENT	SIGNIFICANCE OF CORRELATION (PERCENT)
Moderately-Significant, No Correlation	Preventive	CLT	0.14	88.5
	Corrective	OT	0.16	75.4
Poorly-Significant, Possible Correlation	Corrective Electrical	CLT	0.35	67.6
	Preventive Mechanical	OT	0.28	67.0
Poorly-Significant, No Correlation	Modification	OT	0.16	71.5
	Lighting	OT	-0.11	64.9
	Gathering Head and Conveyor	CLT	0.10	63.9
	Water	CLT	0.10	63.2
	Corrective Lighting	CLT	0.09	62.1
	Ventilation and Dust Suppression	CLT	0.04	56.3
Poorly-Significant, No Correlation	Cutter Head	CLT[2]	0.04	55.9
	Preventive Lighting	OT[4]	0.00	>55.0
	Other Preventive	OT	0.00	>55.0

1 - RBI = Number of roof bolts installed.
2 - CLT = Cutting and loading time.
3 - RBAAT = Roof-beam assembly adjustment time.
4 - OT = Operating time.
5 - TT = Tram time.

$$\text{Coefficient of Correlation} = r = \sqrt{\frac{[\Sigma(x - \bar{x})(y - \bar{y})]^2}{[\Sigma(x - \bar{x})^2][\Sigma(y - \bar{y})^2]}}$$

High significance = >95%
Moderate significance = 75-95%
Poor significance = 55-75%

Good correlation = $r > \pm 0.60$
Indicated correlation = $\pm 0.40 < r < \pm 0.60$, respectively.
Possible correlation = $\pm 0.20 < r < \pm 0.40$, respectively.
No Correlation = $r < \pm 0.20$

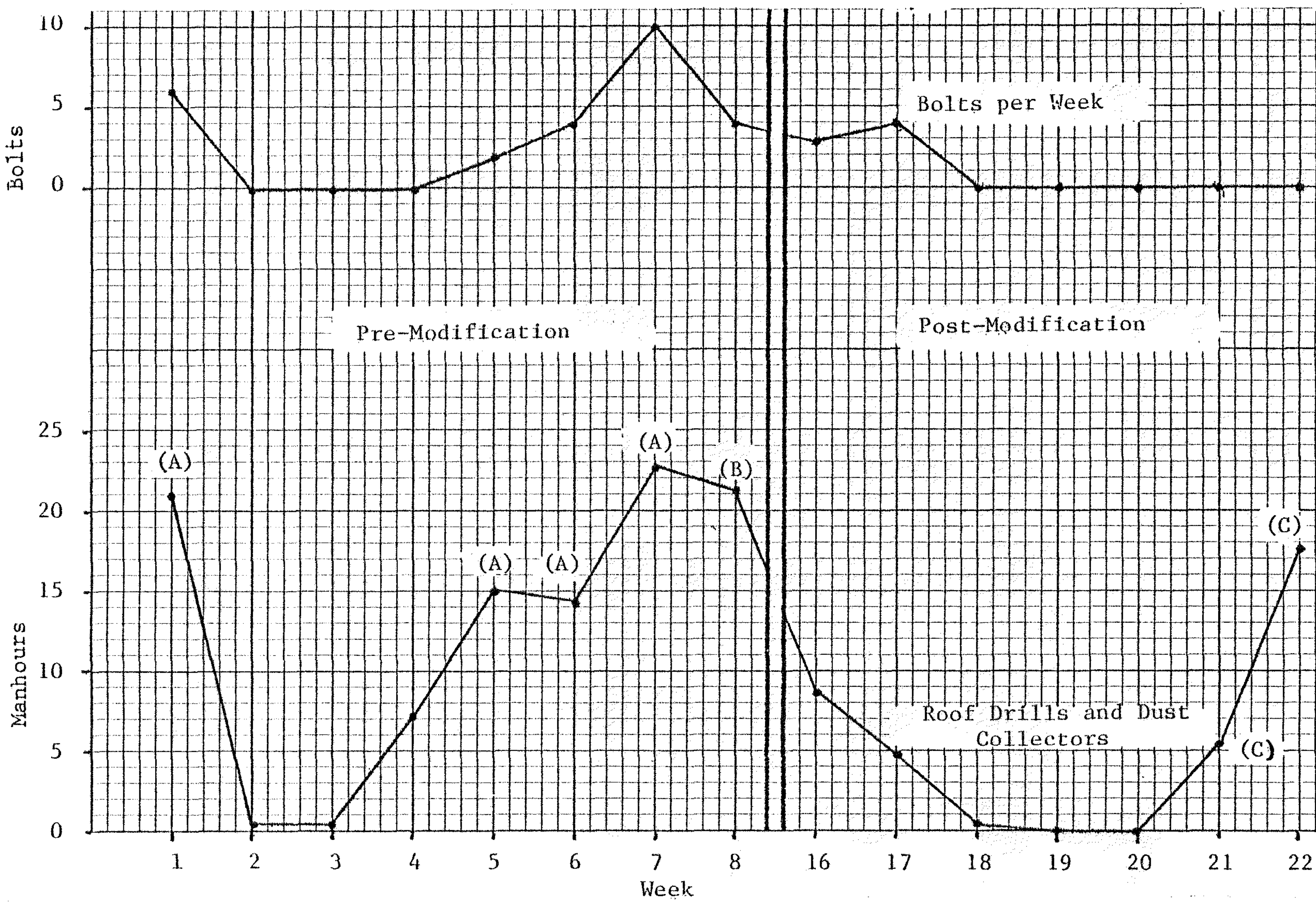


Figure 34
Roof Drills and Dust Collectors Maintenance Manhours vs. Time
Bolts per Week vs. Time

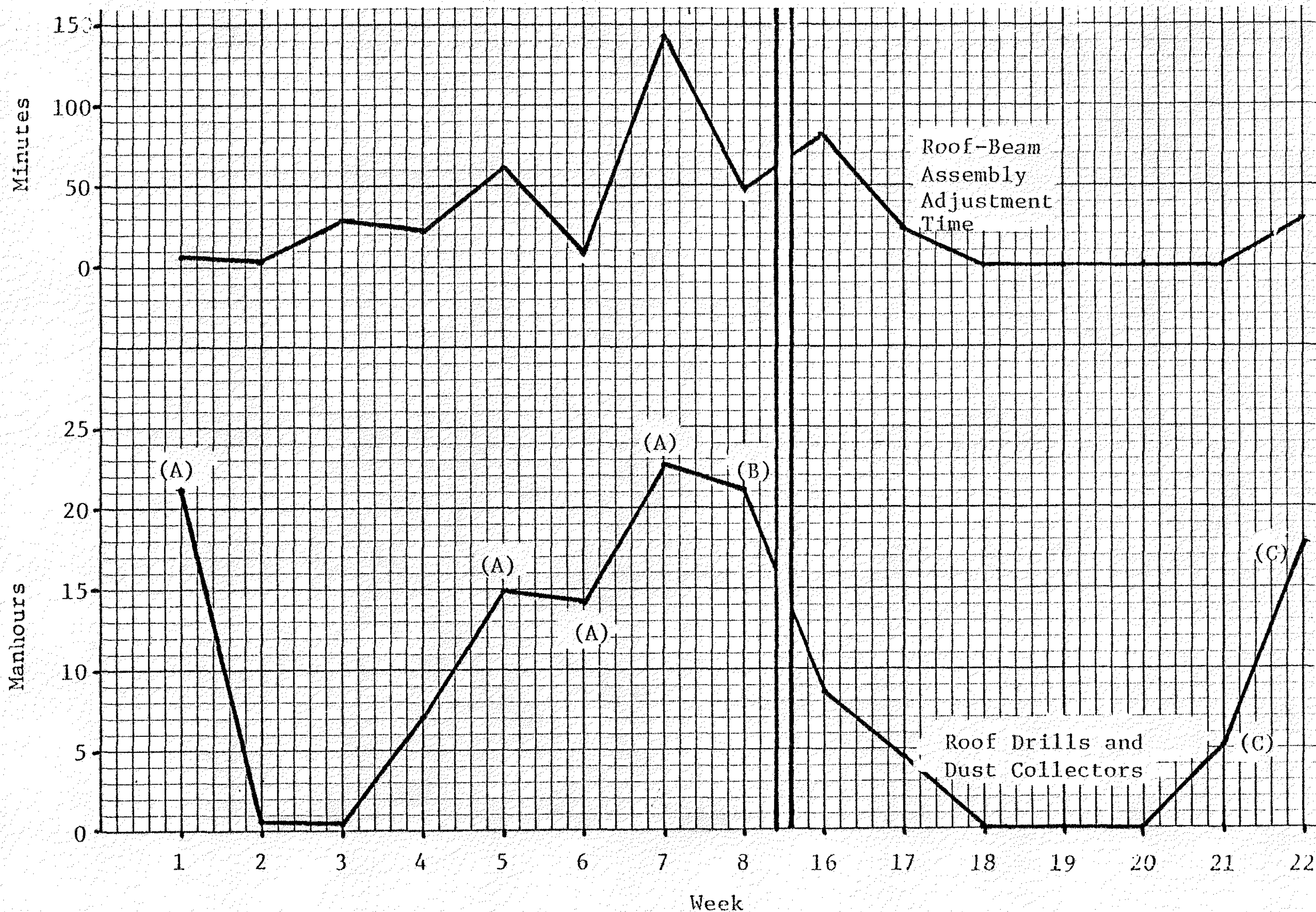


Figure 35
Roof Drills and Dust Collector Maintenance Manhours vs. Time
Roof-Beam Assembly Adjustment Time vs. Time

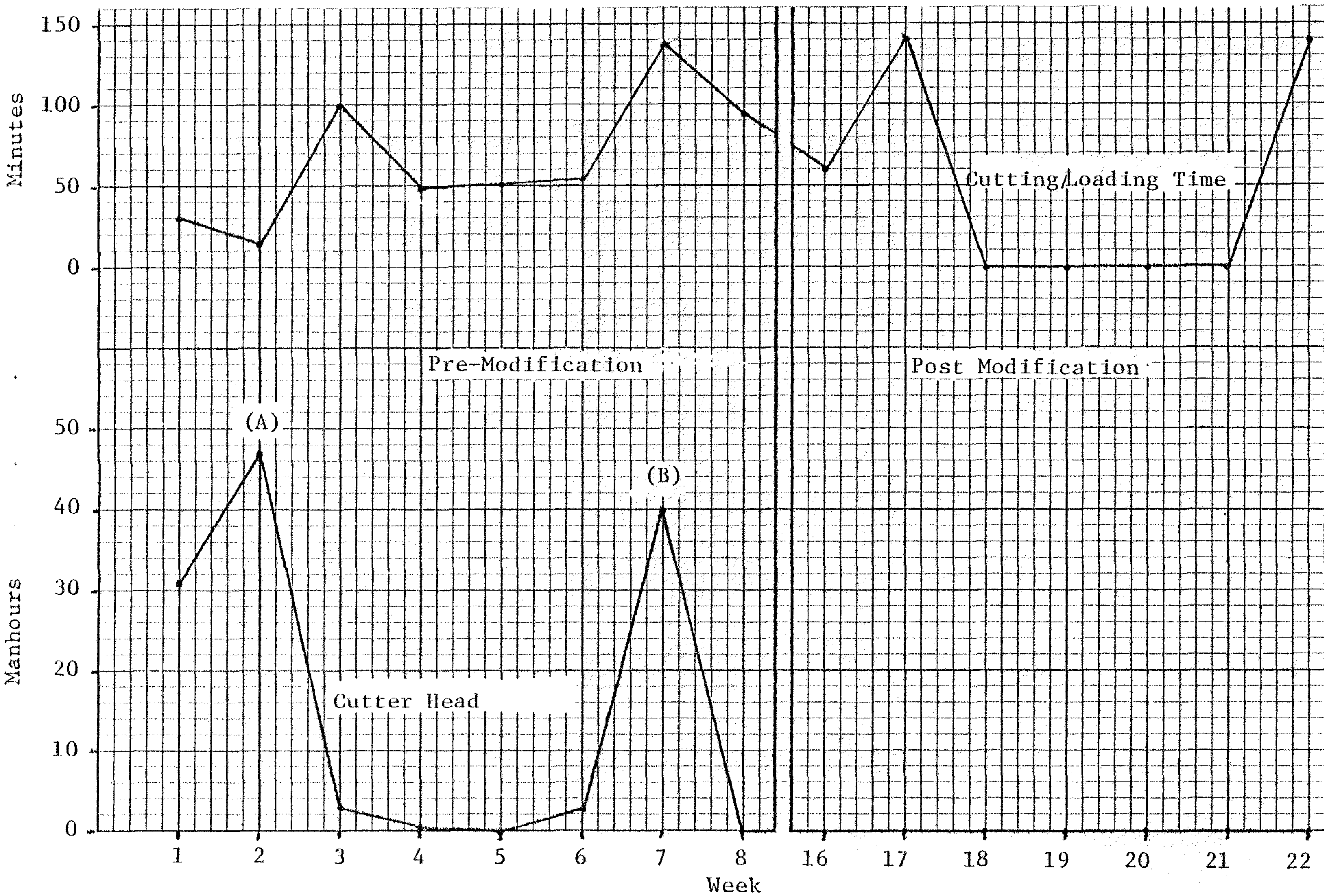


Figure 36
Cutter Head Maintenance Manhours vs. Time
Cutting/Loading Time vs. Time

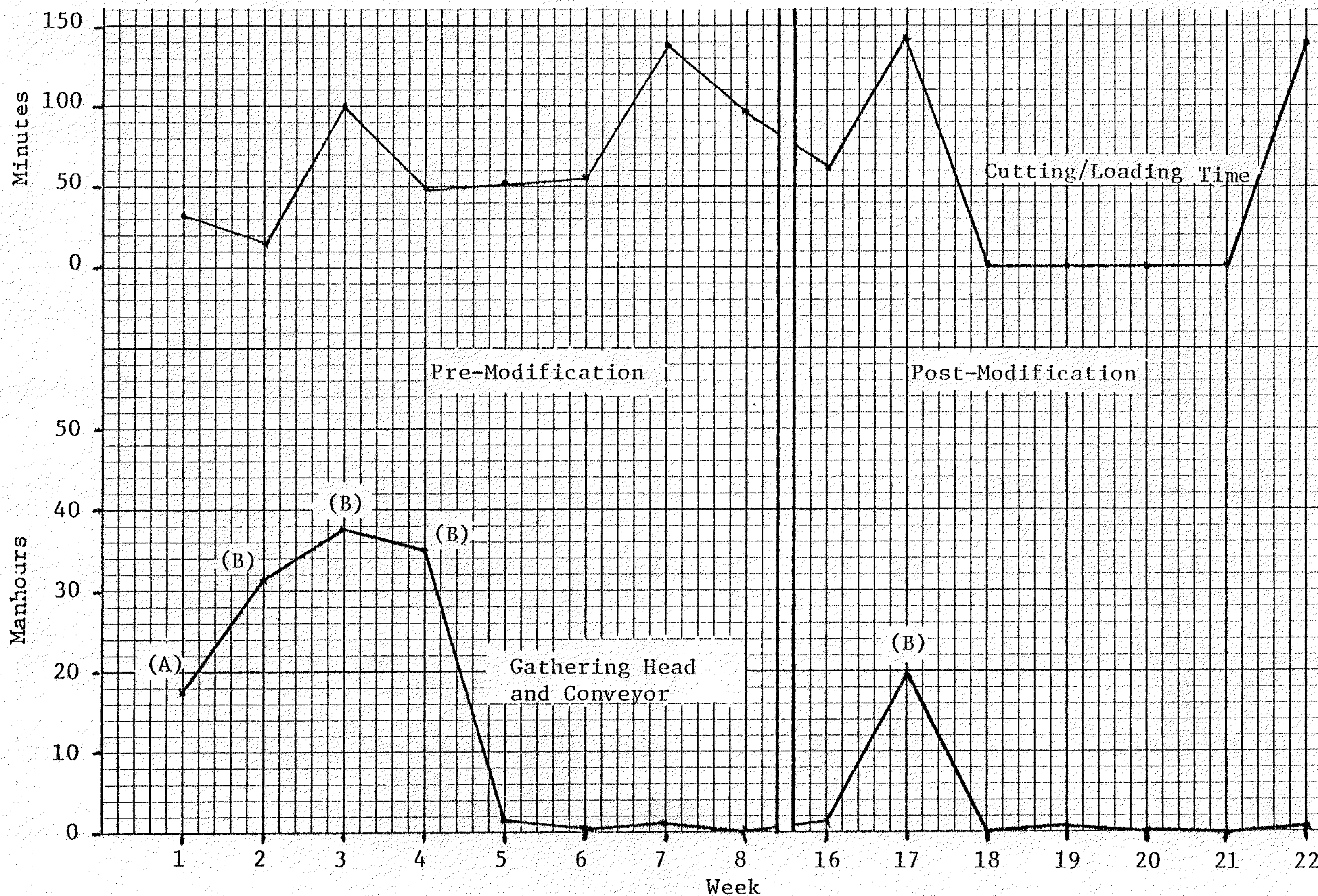


Figure 37
Gathering Head and Conveyor Maintenance Manhours vs. Time
Cutting/Loading Time vs. Time

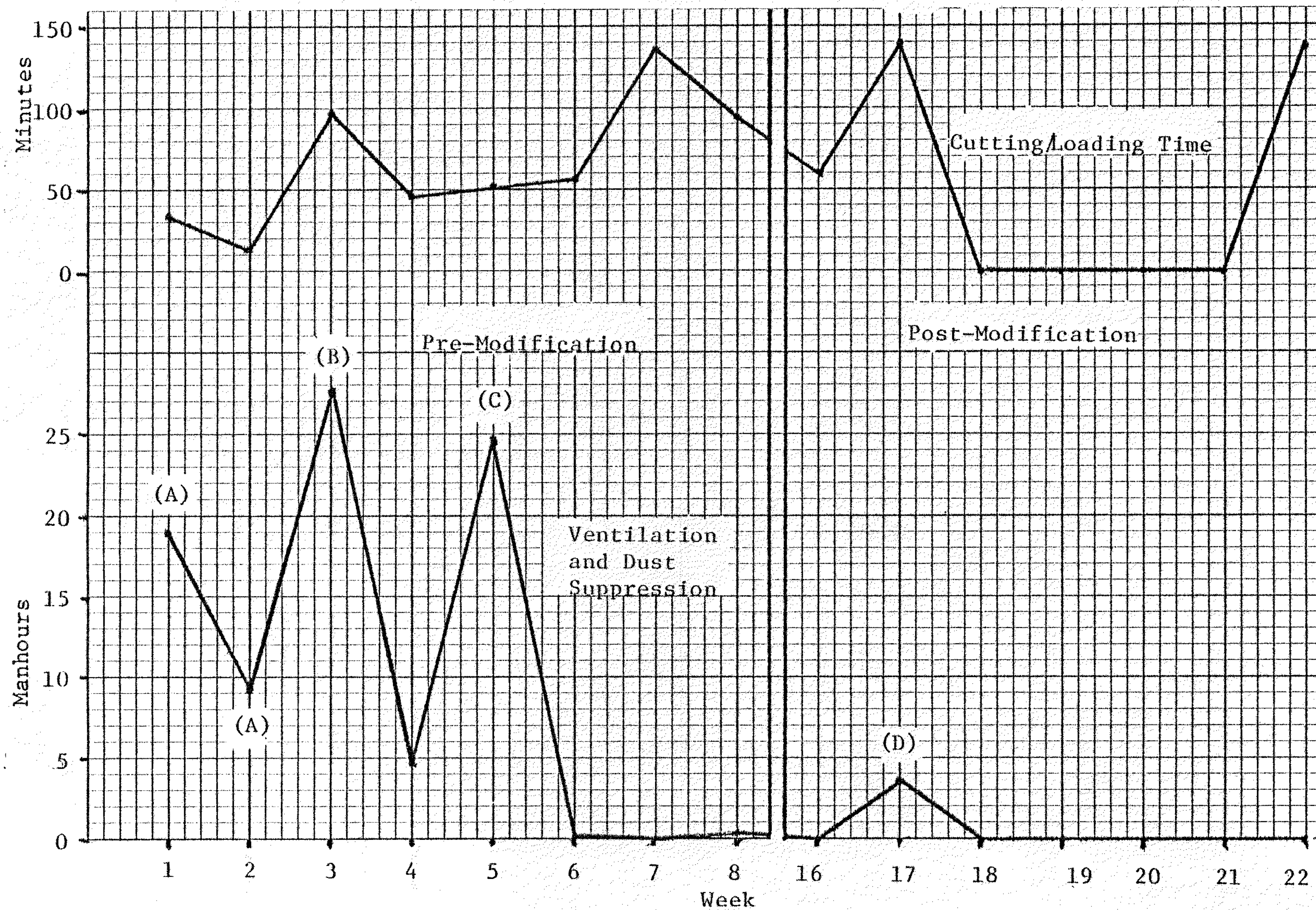


Figure 38
Ventilation and Dust Suppression Maintenance Manhours vs. Time
Cutting/Loading Time vs. Time

APPENDIX 5-3

FAILED COMPONENTS THAT WERE RETURNED TO MANUFACTURER FOR ANALYSIS

NMSC has made available information on the status of some of the AES components that failed during the in-mine evaluation period. This part of the report will present that information.

5-3.1 PUMP-DRIVE BEARING

Two NMSC #1101-8868 M.R.C. roller bearings failed during the initial production trial. The first failure was attributed to a misfit between the bearing and the shaft and was corrected in the field. The second failure was attributed to improper assembly procedure.

5-3.2 ROOF-DRILL GEAR PUMP

The NMSC #7800-8554 Hydreco Tandem-Gear-Pump Assembly and the NMSC #7800-8539 Hydreco Rear Gear Pump Section failed due to cavitation that resulted from operating the AES at a low oil level. These items were replaced with more reliable units.

5-3.3 CYLINDER FUNCTION PUMPS

The NMSC #7800-6525 Sundstrand pressure-compensated, variable-volume pump was replaced. The pump was returned to Sundstrand for analysis which revealed:

- The piston slippers had rolled, one almost completely off the piston.
- Excessive head had discolored the brass parts.
- The bearing plate had been smeared and the thrust plate was also discolored.

Given the AES operating conditions, the cause of failure was determined (by Sundstrand) to be a high differential pressure (about 15 psi) between the pump case and the inlet. Also, a less viscous oil than the S.A.E. 30 oil was required. Therefore, NMSC planned to relocate the pump-case drain and consider the use of a thinner oil.

5-3.4 GATHERING HEAD MOTOR

Four failures of the NMSC #3100-1290, 15-hp gathering-head motors occurred, caused by one or more of the following:

- The thermal overloads were of the wrong value.
- The instantaneous overloads were set too high.
- The sideboard extensions contacted the ribs on the small gathering-head discs when the extensions were completely collapsed.

- Material on the bottom clogged components on the underside of the head.

The windings in each motor were found to be grounded.

5-3.5 SUMP CYLINDERS

The rods to both sump cylinders, NMSC #4920-3631, were bent by unrestricted angular movement of the cylinders between the tractor frame and the operators' platform. While the machine rested on uneven bottom, the extension of the sump cylinders would allow the platform to fall until the cylinder bodies contacted the tractor frame "T" slide. Modifications to the tractor frame "T" slide, guide slide, and sump cylinders were completed to alleviate this problem.

5-3.6 VENTILATION-FAN MOTORS

Several failures of the ventilation-fan motors (NMSC #7800-1146) occurred. The analysis of the one motor sent to Volvo for repairs revealed that cavitation had caused the port plates to separate. This allowed a high pressure-buildup in the case, which caused the shaft seal to fail. The cavitation was caused by an undersized spring in the motor-braking valve. The spring was replaced and the problem was solved.

5-3.7 ROOF-SUPPORT CYLINDER

A NMSC #4900-0706 Hyco Cylinder developed a substantial leak on the first stage of packing because of the omission of some Chevron packing seals. The cylinder was repacked.

5-3.8 PILOT VALVES

Contamination of the pilot valves, NMSC #9230-5978, caused several hydraulic functions to operate erratically. They were replaced. The contamination was caused by:

- The buildup of silt in the valves.
- The buildup of rubber flakes from the inside of the hydraulic hoses. The flakes resulted from excessive oil temperature.
- The failure of filter screens upstream of the valves--a design error on the part of Bertea, the manufacturer of the valves.

APPENDIX 5-4

AES SPECIFICATION SUMMARY

Source - National Mine Service Co.

MACHINE ENVELOPE DIMENSIONS		
Overall Machine Length		35'5"
Cutting Head Width	Mining	15'0"
	Retracted	13'4"
Machine Width Outside Roof and Floor Beams		12'4"
Weight		150,000 lb.
Face Area		75 to 135 sq. ft.
Coal/ft. Advance		4.7 to 8.5 tons
TRACTOR FRAME		
Ground Clearance		6"
Tracks		
Type		Piano Hinge
Width		18"
Ground Contact Length		99" Nominal
Area		3564 sq. in.
Ground Pressure (Max. Tramming)		34 PSI
Tram Drive		
Type		Spur Primary Two Stage Planetary Final
Maximum Belt Pull		50,000 lb./side
Speed		13.3 ft./min. - 40 ft./min
Motor (Two Water Cooled)		7.5 HP @ 600 RPM 30 HP @ 1800 RPM
CUTTER DRUM		
Drum Outside Diameter		36"
Base Diameter		22"
Cutter Boom Height	Drum on Grade	42"
Cutter Drum Reach	Above Grade	10'0"
	Below Grade (Max.)	10"
	With Gathering Head on Grade	5"
CUTTER DRUM DRIVE		
Motors (Two Water Cooled)		200 HP ea. @ 1200 RPM
Drum Speed		57 RPM (63 and 70 optional)
Drive Type (2)		Bevel and Planetary
Bit Speed		535 RPM (595, 660 optional)
Protection - Clutches (2)		Multi Disc, 600 HP Slip

GATHERING HEAD

Type		Disc
Gathering Head Width	Mining	15'0" to 16'0"
	Retracted	14'0"
Disc Sizes	Large	4'5"
	Small	2'6"
Disc RPM	Large	70
	Small	117
Drive Motors (Two Water Cooled)		15 HP ea. @ 1750 RPM

CONVEYOR

Width	30"
Depth (steel Sideboards)	5"
Sideboards (FLexible)	4"
Chain	2 5/8-inch Pitch Redbird
Drive	Rear Hydraulic
Speed	400 ft./min.
Loading Rate (Max. with 5" Depth)	11.6 T/min.
Tensioning	Automatic Load Sensitive Hydraulic
Swing Angle	60° Left or Right

ROOF SUPPORT/OPERATOR PROTECTION SYSTEM

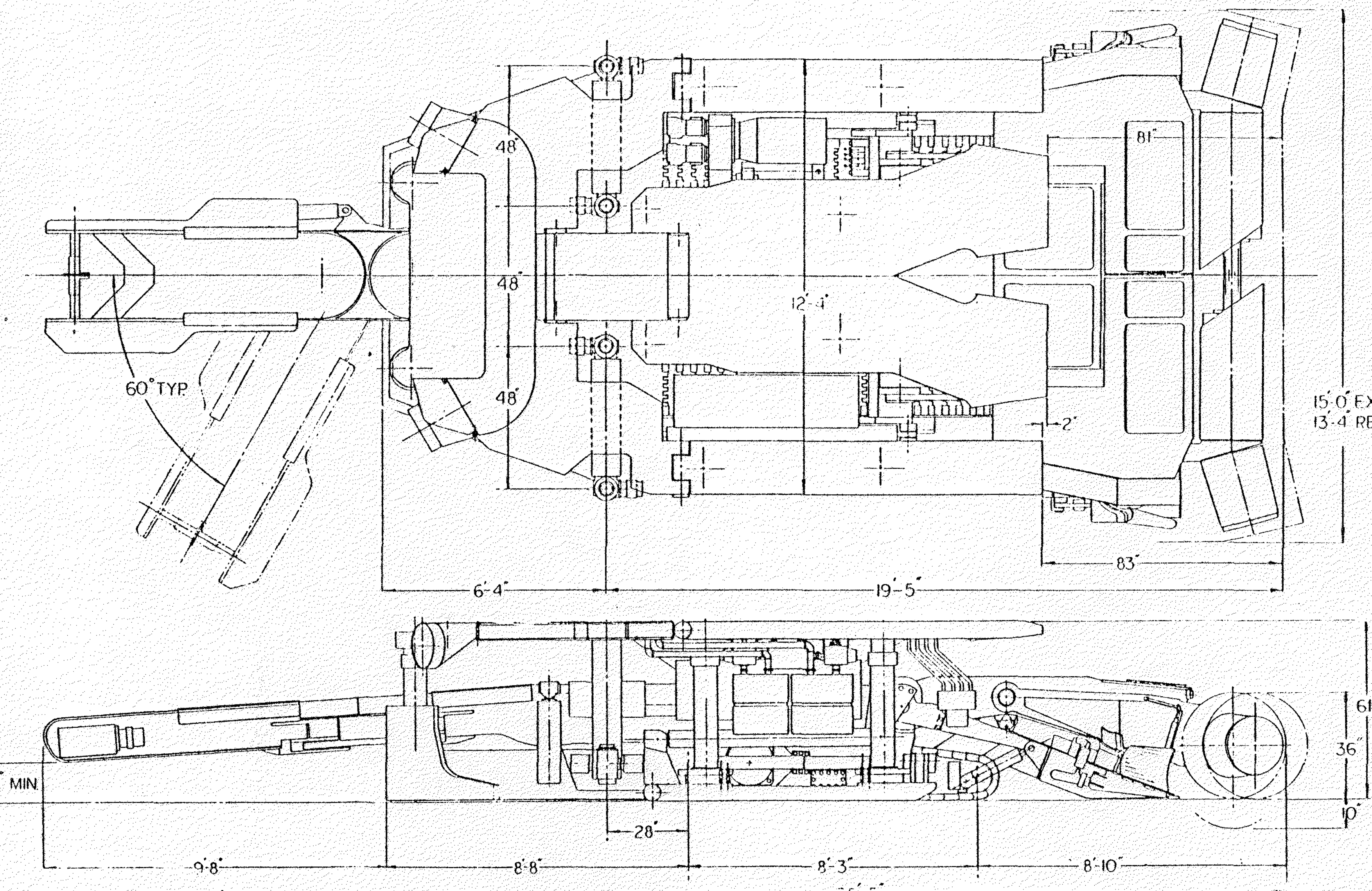
Type	Longitudinal Roof and Floor Beam
Support Cylinder	
Quantity	10
Bore	7"
Rated Pressure	3000 PSI
Capacity at Rated Pressure	35.6 Tons Each
Roof Beams - Outer	
Quantity	2
Width	18" Each
Length	10'4" Each
Average Roof Pressure at Cylinder Capacity	63.8 PSI
Roof Beams - Inner	
Quantity	1
Width (At Narrowest Portion)	5'0"
Length	11'9"
Average Roof Pressure at Cylinder Capacity	33.64 PSI
Floor Beams - Outer	
Quantity	2
Width	18" Each
Length	97-1/4" Each
Average Floor Pressure at Cylinder Capacity	81.3 PSI

Outer Rear Support	
Quantity	1
Approximate Roof Area	4,050 sq. in.
Average Roof Pressure at Cylinder Capacity	35.2 PSI
Outer Rear Platform	
Quantity	1
Approximate Floor Area	4,080 sq. in.
Average Floor Pressure at Cylinder Capacity	34.9 PSI
LOW RANGE SUPPORT SYSTEM	
Minimum Tram Height	51"
Outer Floor Beam Ground Clearance	4-1/4"
MINING RANGE (With 12" Tram Top Clearance)	5'0" to 7'4"
HIGH RANGE SUPPORT SYSTEM	
Minimum Tram Height	5'1"
Outer Floor Beam Ground Clearance	6"
MINING RANGE (With 12" Tram Top Clearance)	6'1" to 10'0"
PUMP DRIVE	
Motor (One Water Cooled)	200 HP @ 1200 RPM
Pumps	
Drills (4)	30 GPM Gear
Scrubbers (1)	30 GPM Gear
Cylinders (1)	0-37.6 GPM VV Piston
Vent Fans (1)	0-23 TPM VV Piston
Conveyor (1)	30 GPM Gear
Water (1)	30 GPM Gear
Gear Box	Spur Gear Water Cooled
Reservoir Capacity	150 gal.
ROOF DRILLS	
Quantity	4
Feed Rate - Maximum	26 ft./min.
Torque Maximum	300 lb./ft.
Bolt Torque	250 lb./ft.
Thrust - Maximum	8000 lb.
Thrust - Torquing	200 lb.
Location	2'0" and 6' either side of center line
ROOF DRILL DUST COLLECTION	
Blowers	70 CFM @ 12" Hg.
Quantity	4
Type	Positive Displacement - 2 Rotor Lobe Type

COLLECTORS	
Quantity	4--1 Per Drill
Type	3 Stage--2 Centrifugal, 1 Media Type Filter
Manufacturer	Donaldson
VENTILATION FANS	
Quantity	2
Type	Axial Flow
Capacity Each	3000 CFM @ 12" H_2O
AIR SCRUBBERS	
Quantity	2
Type	Centrifugal, Wet Impingement
Manufacturer	T.J. Gundlach
Capacity	2000 CFM Each
Water Requirement	3 GPM Each

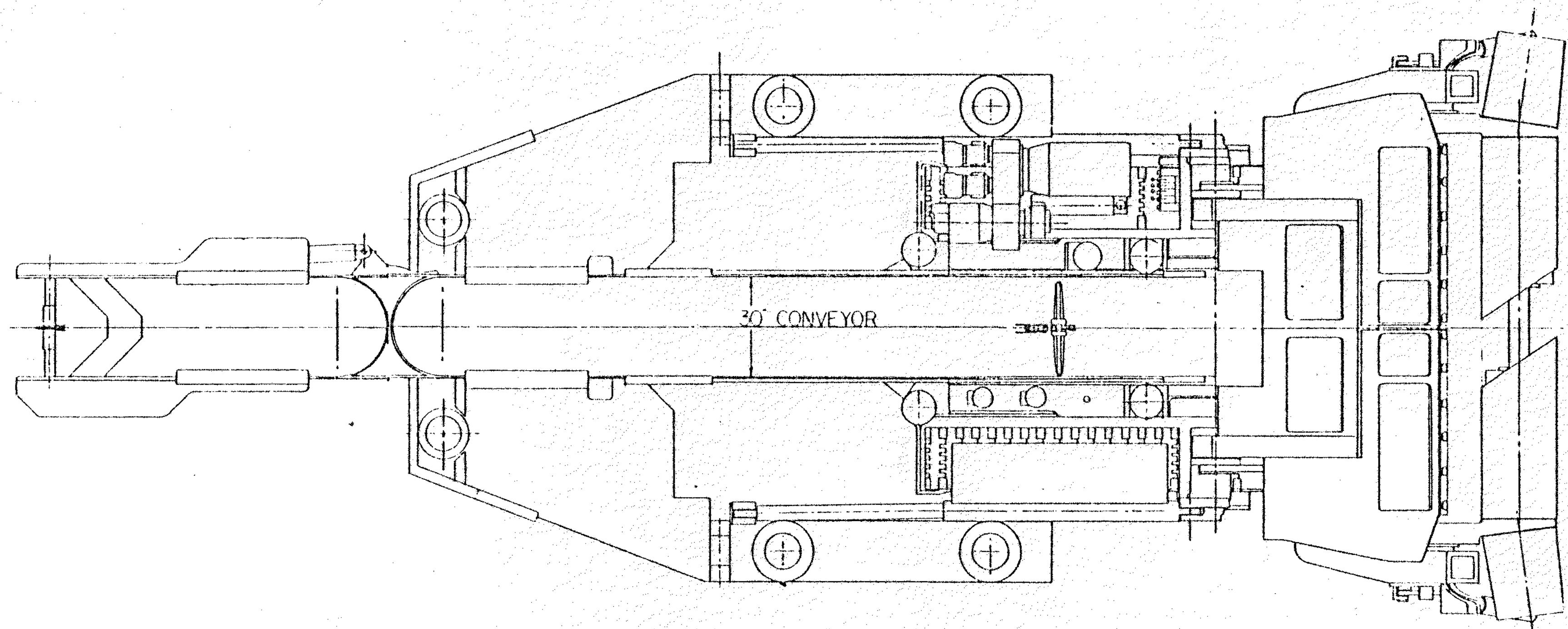
ADDITIONAL FEATURES

Water Deluge Fire Suppression System
Approved Lighting, Face and Area
High Visibility Paint
Compatible with invert emulsion fire resistant hydraulic fluids
High efficiency filtration is provided on return flow before entering tank. All replacement oil is filtered before entering tank. Pilot circuit utilizes high pressure filtration.
Improved Face Water Spray System



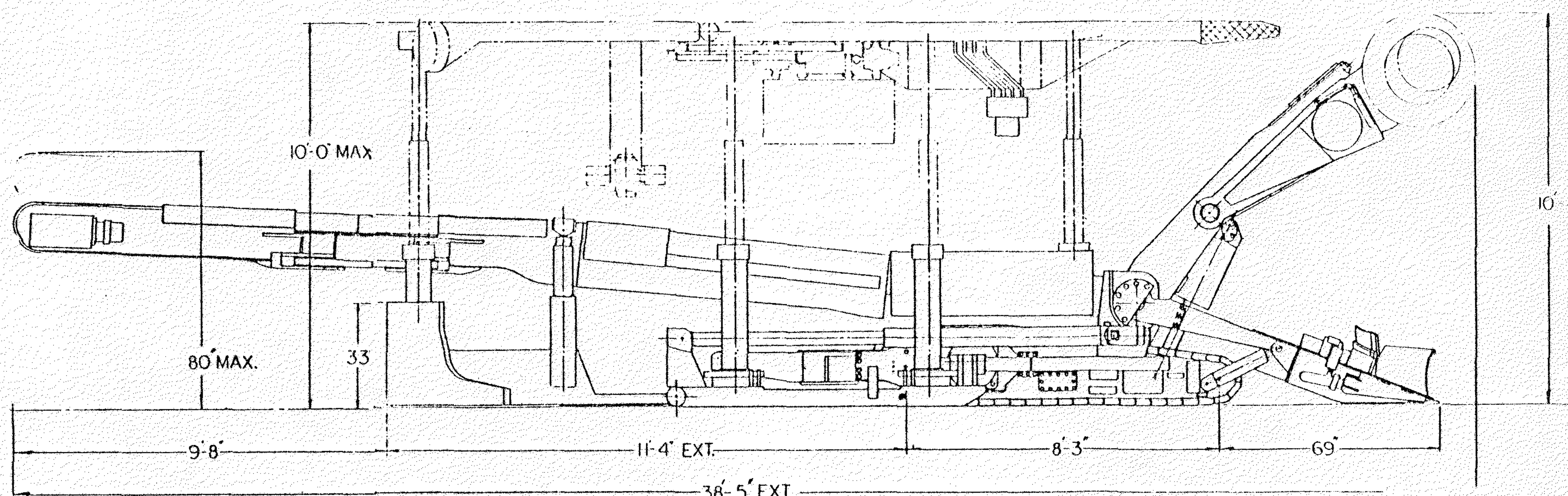
Courtesy of National
Mine Service Co.

Figure 39
Top and Side Views of AES
Machine Retracted



Courtesy of National
Mine Service Co.

Figure 40
Top View of AES
Machine Extended and Roof Beam
Assembly Omitted



Courtesy of National
Mine Service Co.

Figure 41
Side View of AES
Machine Extended and Roof
Beam Assembly Omitted

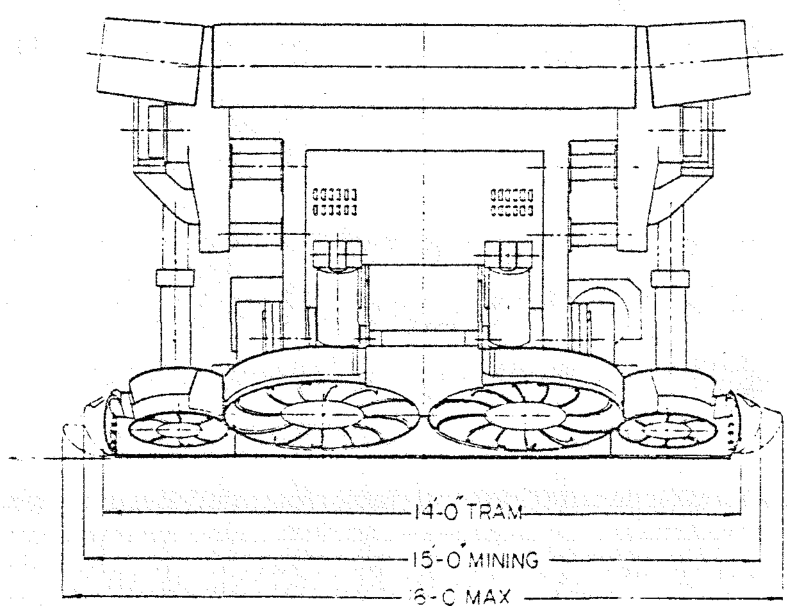


Figure 42
Front View of AES

Courtesy of National
Mine Service Co.

APPENDIX 5-5

STATE OF THE ART OF MINING EQUIPMENT EVALUATION

In developing an evaluation procedure for the AES' in-mine trial, it was necessary to determine the state of the art for equipment testing in mining. For the most part, machines are audited by three groups: equipment manufacturers, operating mining companies, and researchers. Some of the same tests and techniques are used by each group, but unfortunately, many machine evaluations are very limited in scope. This investigation of the state of the art shows how past studies were limited. They are grouped to show that the studies did not include one or more of the five major criteria that should be used to evaluate mining machine parameters - safety, production, performance, economics, and human factors.

The first section of this chapter examines techniques used by equipment manufacturers. The second part discusses operating mining companies, and the third part discusses researchers.

5-5.1 EQUIPMENT MANUFACTURERS

The most intensive machine testing that is performed by equipment manufacturers seems to occur at the plant. Most manufacturers verify specifications of all parts, including mechanical, electrical, and hydraulic components. Mechanically, components are tested to determine 1) if tolerances are met during machining, 2) if clearances between components are maintained, 3) if MSHA compliance is maintained, 4) what the actual (not design) specifications are, and 5) what the machine weight and center of gravity are. Electrically, 1) short circuits are located and eliminated, 2) the actual voltage, amperage, power, power factor, and load factors are determined, and 3) MSHA compliance is assured. Hydraulically, pumps, motors, and valves are checked for improper operation, hoses and fittings for clearance with other components, and reservoirs for meeting pressure tests.

One company performs a "heat-run" where a machine is operated for two hours. Every 15 minutes, several components are operated simultaneously to simulate peak load conditions. After 15 minutes of operation, the components are de-energized and allowed to "rest" for 15 minutes. The temperature is measured at known "hot-spots"; if it exceeds certain limits during the two hour heat-run, then that area of the machine must be dismantled and reassembled after corrections have been made. The heat-run is repeated until no hot-spot temperature exceeds the limitations. If a hot-spot is near the limit, then the test may be extended for up to two additional hours. Also, prototype machines are all heat-run tested (by one company) for four hours. In some instances, there is a lower limit that the temperature at a hot-spot must exceed in order to pass the test.

When a machine is taken underground, some equipment manufacturers feel that it is the responsibility of the operating company to verify machine operation and perform in-mine tests. The manufacturers will provide service-persons to aid in reassembly and function verification. In the case of a prototype machine, and for some machines that may be new to the mine personnel, a representative of the manufacturer will accompany a machine for a period of time to determine if it is performing as it was designed to perform and to instruct the operators in the proper techniques for running and maneuvering it. The representative will recommend modifications needed for use in a particular mine or section. Occasionally, time-and-motion studies are used by the representative, but usually observations and operator response are used in this type of underground evaluation.

Occasionally, a service-person or someone from the manufacturer's engineering office is asked to determine a specific problem in the operation of a machine. One manufacturer has an industrial engineering department that handles such assignments. Time-and-motion studies and observations are the usual techniques used.

For some machines, especially prototypes developed under research contracts, the equipment manufacturer has representatives accompany a machine during its prototype stage. Observations and evaluations are made to improve or modify the machine with regard to safety, production, and performance.

5-5.2 OPERATING COMPANIES

The goals of most mining companies are to 1) increase production and thereby increase profits, 2) better utilize manpower, and 3) improve safety. Many companies that mine steam coal have long-term contracts with electric utility companies, and must justify production decreases by proving that the cost of operating delays can be attributed to either changes in the UMWA labor contract agreement or to the Federal Law, and as such are reimbursable. In such cases, machine and mining-method evaluations are required to identify the amount of delay time that is reimbursable. Mining machines must be selected that fit into the mining scheme and deliver specified outputs.

Most in-mine evaluations of mining machines use time-and-motion studies as the most common means of determining operating parameters. Usually elemental times are identified, including mining time, necessary delays, and unnecessary delays.

For machines that are evaluated as candidate mining systems, the mine operator may use manufacturer specifications as one criterion. However, in-mine experience with a machine or machine type is a major factor when the decision is made on which machines to buy.

Commonly, mine operators develop some information concerning the following parameters in order to evaluate mining machines:

• Production (or output) capabilities.
• Cost (capital and operating).
• Maintainability and availability.
• Labor utilization.
• Equipment life.
• Safety features.

Production, cost, and maintainability are probably considered to be the most important yardsticks in a machine evaluation by an operating company. After data is collected in an area, several machines may be compared. The operator then chooses the best machine. If the evaluation is an in-mine test, then the operator knows which machines need to be improved for a particular application, and knows which machines can be used to obtain maximum output, lowest cost, and maximum reliability and safety. However, there are instances when mining machines cannot be specifically compared because mining and geological conditions are different (i.e., the controlled variables are different).

In many cases, worker opinion is used, but the results are not formally documented or reported. The operators and support personnel may have excellent judgement and, therefore, may be able to qualitatively evaluate a machine they have operated. Indeed, the operators, support personnel, and section foreman may be the only source of data available at an outset of an in-mine machine evaluation. Their knowledge and experience could be invaluable in identifying problem areas and in working out solutions.

Often, there seems to be minimal concern with identifying and documenting operator errors (delays that could be eliminated if a job were less labor-intensive, a machine were modified, or an individual were better trained). Consider, for example, the case of an operator who consistently spins the bits of the continuous miner but does not cut coal. This type of operating error can be identified through carefuly observations of the operators performing their work.

In general, the following observations may be made concerning equipment evaluation and documentation by operating companies:

• Companies evaluate machines to eliminate or minimize delays, increase production, better utilize manpower,
• Companies mainly use time and motion studies as a basis of testing.
• Several machines are sometimes compared, but the controlled variables (machine condition, geology, etc.) must be equated in order to make the comparison valid.
• The opinions of workers are often used but not formally documented. These opinions are valuable information that can be used to determine the abilities of a machine.
• Safety features are being evaluated in greater depth because of increased emphasis on preventing injuries.

- Operator errors are often not included in an in-mine test, but they may represent a significant amount of the non-productive time.

The two studies that are reviewed next illustrate the kinds of machine evaluations performed by mining companies:

One mining company evaluated a new system in a mine (Cochran, 1974), using time-and-motion studies to identify necessary and unnecessary delays according to causes. This 10-week, 68-shift study was performed on a semi-remote-controlled continuous miner with continuous haulage. In addition to delay identification, a record of production was kept, and certain components were performance tested, including the remote-control system, bridge conveyor, double-bridge carrier, and pan line. The evaluation revealed that the operator could get a "feel" of the continuous miner with the remote control after some experience was gained. For one thing, he had a better view of the area because the remote-control unit allowed him to stand to the side of the entry. The sliding sump frame required little attention. It was also discovered that the practical limit for the umbilical cord was 30 to 35 feet. The noise and dust levels were in compliance with Federal standards.

The study concluded that remote control (in this case) is a definite asset to mining, that the pendant gave excellent service, and that the continuous haulage gave steady, dependable loading and eliminated shuttle car traffic hazards. The study investigated safety, production, performance, and human factors but did not discuss economics.

Another study (Britton, 1978) was an evaluation of a water-spray fan that was retrofitted on a continuous miner on the side opposite of the operator. The fan forced air into the corners of the face and aided in methane and dust dispersal. The company used smoke tubes to determine the path of the air flow and anemometers to determine the velocity of the air. The results showed that air velocity increased 120 percent and that methane and dust were dispersed into the return more easily. The author did not discuss instrumentation with regard to dust, but said that observations of coal dust at the operator's location indicated improvement when the fan was used. One disadvantage cited was the introduction of extra water into the face area. However, the problem was controlled somewhat after the operator became familiar with the modification. The study only examined safety and performance parameters; economics and human factors were neglected.

5-5.3 RESEARCHERS

Machine evaluations have also been performed by researchers who worked under government contracts or with private funding. These evaluations have been documented, but in many cases the scope of the projects were limited and the machines were not fully tested. Typically, the projects each covered only several operating parameters. This is not to say that such studies were inadequate; however, perhaps better evaluations could have been made of a comprehensive documentation of machine capabilities had been carried out.

This section discusses the objects and techniques, and summarizes the results of some past studies of machines in underground coal mines. Most of the projects were time-and-motion studies, and most evaluated production capabilities. Many also assessed performance. The studies discussed here represent a cross section of the machine evaluations performed by researchers; they are not a total listing of all past projects.

5-5.1 Studies that Investigated Safety Only

In one study (Barry et al., 1971), face machinery was studied with regard to fatal accidents. A strong relationship was discovered between fatal accident occurrences and limited experience. An industrial-engineering analysis was used to estimate the annual cost to industry and the potential lives saved through the use of protection devices or modifications, such as panic bars or repositioning of the operator. Finally, multiflow-process charts were used to determine interactions between the job elements, and standard-time data was used to represent typical cycle times for all activities associated with the working face. The times when there was exposure to hazards were noted, and time-study forms were developed.

In a study of the safety of equipment operation (Barry et al., 1972), the opinions of the USBM and the equipment manufacturers, and the observations of mining machines were used to determine the machine operations that could be made safer. Interviews were conducted with the USBM and the equipment manufacturers to obtain their ideas regarding operator-acceptance of new components, difficulties in complying with USBM regulations, other major design problems, approaches used to satisfy requirements and to overcome design problems, safety devices included as standard but not required, sources of design or modification ideas, and operating and maintenance training programs. The underground examination of machines included interviews with machine operators regarding training received, experience on the machine, fatigue, controls, safety features, and position and seating. The study revealed that safer equipment can be realized in six major areas: control standardization, shuttle cars, tractors (scoops), auger miners, continuous miners and loaders, and roof drills.

One study developed health and safety indices to evaluate underground coal-mining systems (Pfleider, et al., 1973). The study had four objectives: to 1) determine the relationships in underground coal-mining systems which affect the health and saf ety of mine personnel, including mining methods, unit operations, coal-seam characteristics, roof conditions, etc. 2) develop the relationships into quantitative health and safety indices, 3) develop a computer-based model, and 4) establish the groundwork for further study:

The following relationships between mining factors and safety (severity and frequency rates) were derived in this study:

- Geology
  Mines with disturbed roofs were safer.
- Seam Thickness
  Non-captive mines in thick seams were safer than non-captive mines in thin seams. There was no difference in captive mines.
- Mine-Opening Time
  Safety versus mine-opening time - results inconclusive.
- Coal Use
  Non-captive, non-coking mines were safer than non-captive coking mines. The results were inconclusive for captive mines.
- State Rating
  The injury rate was higher in West Virginia than in Kentucky, Pennsylvania, and Illinois; but the result may be due to better reporting of accidents.
- Mine Ownership
  Captive mines were safer than non-captive mines.
- Mining Method
  Non-captive, continuous mines were safer than non-captive ones using conventional methods. Captive continous mines were no safer than captive conventional mines.
- Primary-Secondary Haulage Method
  Mines using all-track haulage were safer.
- Number of shifts worked per day
  Conventional mines using three shifts per day were safer than less-productive mines. There was no difference in captive versus non-captive mines.
- Number of Workers
  Mines employing more than 250 workers were safer.
- Number of days worked per year
  Number of days worker per year - results inconclusive.
- Number of man-days (m-d) worker per year
  Continuous mines with more than 60,000 m-d per year were safer.

A health and safety model was developed for coal mines. Three indices were defined. Index A was a subjective rating that used relative comparisons of alternate designs and was intended to take advantage of the experience and judgement of the mining staff. Index A was useful in the early stages of design and was easily applied. Index B was an objective rating that used actual data representative of the mine's working conditions for each occupation. Index C was a subjective modification of Index B, used when objective data was not available, for example when a new system, method, or machine was evaluated. Three model types were also defined. Model 1 used hazard ratings expressed as shift averages; Model 2 used ratings expressed as averages of activity zones; Model 3 used time spent on a task in a zone. A chi-squared statistical analysis was used to determine the relationships discussed above. Four types of injury rates were used as a measure of mine safety: total disabling injuries per million man-hours, total disabling injuries per million raw tons, fatalities per million man-hours, and fatalities per million raw tons. The mines were also grouped by frequency rate, geology, and mining-system characteristics.

5-5.3.2 Studies that Investigated Production Only

In one shuttle car study (Feng, 1947), continuous time-and-motion studies were used to determine available time, delay times, shuttle-car payload, and loader capacity. The study identified delay causes and recommended system improvements.

Another research effort (Marrus et al., 1975) was designed to collect, analyze, and verify maintenance data on continuous miners and to determine the current availability of continuous miners being used in underground coal mining. Time-and-motion studies were the primary means of obtaining data in the ARCCM project. Several other parameters were established, including the elements and amount of downtime associated with corrective and preventive maintenance servicing, and the cost, size, and weight of each part replaced. A total of 46 continuous miners operating in medium-to-high-seam coal was sampled. The results were as follows:

- Mean tons between failures = 1400 tons.
- Mean time between failures = 6.23 hours.
- Shifts between failures = 3.73 shifts
- Mean corrective time = 2.22 hours.
- Mean corrective man-hours = 4.80 man-hours.
- Maintainability index = 3.53 man-hours/1000 tons.
- Corrective maintenance per hour of operation = 0.79 man-hours/hr.
- Achieved availability = 0.73 or 73 percent.
- Average shift production = 409 tons per shift.

A number of conclusions were reached on the basis of this data, including:

- There is no demonstrated relationship between machine model and level of performance. Rather, performance seems to be based on nonuniformities of manufacture of individual machines, machine-maintenance practices, operator skills and attitudes, and seam characteristics and mine environment.
- The mean time between failures increases as the production loading increases until a loading of near 500 tons per shift is reached. After this level is attained, reliability decreases.
- Availability increases as production increases.

A similar analysis was used in the AES audit project.

A study of shuttle car capabilities (Smith and Blohm, 1978) showed that production increased 50 percent by reducing haulage downtime by 20 minutes and shuttle car wait-time by 30 minutes. Using a computer simulation model and the results of underground time-studies, the team divided the shift elements into categories: start-up, shut-down, lunch place-change, ventilation-moving, haulage-vehicle downtime, shuttle car wait, and mining time. The object of the study was to compare cable-reeled with battery-operated shuttle cars. The battery-operated units out-produced the cable-reeled cars by up to 50 percent.

One research effort (Suboleski, 1978) developed a mathematical model to predict productivity in underground mines that use continuous miners as a function of eight primary physical variables: seam height, roof quality, methane liberation, bottom quality, water, grades, seam depth, and seam hardness. Questionnaires were used to collect data from 400 operating sections, and the analysis included computer simulation and empirical methods. Some 50 controlled cases were analyzed and the productivity was linked to the primary physical variables.

The results between the 400 sections and the 50 controlled cases compared favorably. The field analysis explains about 50 percent of the underlying bases for production as being related to the physical variables. The remaining 50 percent is attributed to worker attitudes, management techniques, local laws or customs, or equipment variations. In conclusion, Suboleski said that the mining industry cannot make productivity estimates based on experience or other subjective means, because some of the eight variables are ignored. There is a need for rapid field techniques using simple mathematical relationships to describe the influence of physical factors on mine production. The models developed in this study illustrate a quick economical solution.

5-5.3.3 Studies that Investigated Performance Only

Performance and production parameters of shuttle cars were examined in one study (DuBreuil, 1947), in which battery shuttle-cars were again compared with cabled units. The comparison was based on manufacturer specifications and achieved production. Advantages and disadvantages were cited, and improvements were recommended, including cable size, discharge time, and the amount of concentration of the working places.

Several research efforts were developed towards evaluating electrical system performance. The first (Kahlon, 1960) used self-recording voltmeters and wattmeters to compare the power required per ton of coal for ripper- and boring-type continuous miners. The study also discusses the effect of worn bits, coal height, the accuracy of the results, and the effect of the type of mining (production-development, and retreat) on power consumption.

A respirable-dust study (Tomb et al., 1973) was conducted to assess the performance of machine mounted dust scrubbers. Both laboratory and in-mine evaluations were conducted by the use of personal-dust samplers. The samplers were located in the main entries and the immediate intake air, at the face, and in the main and immediate return entries. Lab tests were conducted in a to-scale chamber. The results showed that the scrubbers limited nearly 60 percent of the dust in the mine and nearly 70 percent of the dust in the chamber. Also 20 percent of the dust did not pass through the scrubber.

Another dust study (Mundell et al., 1972) examined a similar dust-collection unit that was used in conjunction with an exhausting auxiliary ventilation fan. However, in this system the scrubber did not significantly reduce the amount of respirable dust because the ventilation tubing captured the dust before it entered the scrubber. The concentrations at the operators' locations remained below 2 mg/m^3 with or without the scrubber operating, so that the operator was always in compliance with the Federal Law. It was also shown that the dust in the exhaust tubing was reduced by approximately 40 percent with the scrubber in operation.

The second study of mining electrical machine parameters (Jeng, 1974) involved the use of a power analyzer for both alternating and direct current to record voltage, power factors, current, and power. The power analyzer was attached to the trailing cables of continuous miners, loaders, shuttle cars, and bolters. An analysis of the operating cycle was made, and total duration, total operating time, idle time, percent of shift that the machine was idle, tram time, percent of shift that the machine was tramming, cutting time (as shown on a tape record of power), percent of shift that the machine was cutting, production, actual cutting time, cutting ability (tpm), and power consumption (kWH, kWH per ton, kWH per minute) were identified. Jeng also derived several graphical relationships, including the actual power versus load factor, and operating factor versus voltage regulation. In addition, some tests to determine the amount of insulation deterioration indicated the following:

- Mining machines are grossly overpowered because of low operating load factors.
- Poor load factors cause the machines to operate at poor power factors causing a large number of peak currents which rapidly degrade systems, including trailing cables.

In another dust study (Jayaraman and Grigal, 1977) a longwall shearer was equipped with a dust collector that included a 500-cfm fan, water sprays, and a mini-cyclone panel at the tailgate end of the shearer. Personal-gravimetric samplers were located at five locations in the intake, the return, and along the face. Three samplers were used at each location, and the results were averaged together. Full shifts were sampled. Also, the air velocity and quantity across the air decreases face were measured. The results show that the velocity decreases from 400 to 200 fpm from the headgate to the tailgate, and the quantity of air decreases from 30,000 cfm to 6,000 cfm because of air loss to the gob. The results of the dust sampling indicate no reduction in the intake, a 23 percent reduction 1/4 of the way across the face, 62 percent 1/2 of the way, 42 percent at the tailgate, and 66 percent in the return. The study concludes that dust is reduced significantly, but the levels were still above Federal standards at the tailgate and in the return partly because of secondary dust sources (e.g., pneumatic picks).

An underground test of diesel exhaust emissions was used to validate turbulence dispersion models (Kenzy, 1977). The tests were designed to verify the mathematical models used to predict the emission, dilution, and dispersion of gaseous pollutants under various conditions. The concentration of carbon monoxide was determined by gas chromatographic analysis of exhaust-gas samples collected in evacuated gas containers. The volumetric rate of exhaust emission was determined by the measurement of exhaust-flow velocity and exhaust-pipe diameter and solving the equation

$$\text{Quantity} = \text{Area} \times \text{Velocity}.$$

The study determined 1) the concentration of gaseous pollutant species in the exhaust, 2) rate of exhaust emission, 3) places where the samples could be collected, and 4) design procedures for monitoring mine-atmosphere contaminant concentrations and the quantities of model input parameters. The study verified the models, and indicated that they were somewhat positively biased.

A third study of electrical components (Kohler, 1977), involved the use of the power analyzer again, and discussed component failure in a power system, the effect of the power system on production and safety, and the economic aspects of this effect. Kohler developed a new method of predicting incipient failures in power-system components, using both field instrumentation and laboratory investigations. He verified the existence of pattern attributes in mine-power systems when a deteriorating component was present in the system. Finally, a correlation was revealed between harmonic currents and the integrity of a component.

The effect of water-spray location on respirable dust produced by a continuous miner was investigated (Courtney, et al., 1978). By the use of SRI dust samplers in the return air, the effectiveness of locating water sprays above and below the bits and over the conveyor was determined. Sumping was determined to be the dustiest operation, in which 60 percent of the respirable dust was produced. During this operation, the top and bottom sprays each reduced 13 percent of the respirable dust. During shearing and loading, bottom sprays reduce the dust by 60 percent, and were 50 percent more efficient than top sprays alone. During a complete mining cycle, bottom sprays reduced 33 percent of the dust versus 25 percent for the top. A combination of top and bottom sprays reduced 41 percent, probably because of increased water and area covered. Throat (conveyor) sprays decreased the overall reduction to 24 percent because they used some of the water that could have been used for top and bottom sprays.

5-5.3.4 Studies that Investigated Human Factors Only

An important research effort (Bailey and Bailey, 1975) used a questionnaire to determine worker acceptance of modified mine equipment, the FMC Corporation's Inherently Safe Mining System (ISMS), at various intervals during a 12-month period. Four interviews were conducted in order to obtain changing attitudes. The question categories included familiarities with modifications, safety features, comfort features, staff-replacement concerns, overall miner attitude, manufacturer receptivity to operator suggestions, program-management follow-through on operator suggestions, capability of technical representatives, and training and orientation programs.

The study was successful in obtaining what seemed to be candid attitudes toward modified machines. The data, though qualitative only, might be helpful in pointing other researchers towards safer, more acceptable mining systems and devices. The questionnaire procedure documented in this study was used as a guide for the AES audit project.

In a study conducted by the USBM (Adkins, 1976), the current training programs found in various mining environments were reviewed and evaluated. A survey of mining training programs was conducted to produce a general description of the variation of training program characteristics. The programs were evaluated to identify variations in organization, course content, course objectives, instructional methods, etc., and to determine relationships between injury rates and training levels and rates.

This study is important to the Penn State Audit Team because one of the objectives of the AES evaluation is the development of a training plan for the machine and section personnel.

The study included over 40 on-site visits to mining companies, schools, agencies, union representatives, and MESA (now MSHA) training facilities. Accident data was supplied by the MESA Health and Safety Analysis Center, and training information was supplied by the MESA Office of Education and Training Qualification and Certification Unit. The main effort of this study was to use the various definitions of effectiveness as the criteria for ranking mines according to the apparent effectiveness of their training programs.

The study indicated that the effectiveness of the training depends on the skill with which the available resources are applied to the perception of the need for training by the workers. The strongest relationships between training and injury rates were determined for roof and rib falls, haulage, electricity, machinery, and general accident prevention courses--probably because the training dealt more with specific cases. The study also showed that half of the reported training effort and more than half of the training administration were going into training that had recognized value but did not relate strongly to the fundamental issue of reducing injuries in mines. However, at the time the report was written, pending changes in mandatory training requirements in the UMWA were to add emphasis to more effective training. Finally, the study indicated that better reporting systems for training and accidents were needed.

5-5.3.5 Studies that Investigated Production and Peformance

A comparison of the performance of battery shuttle-cars and cabled units was also made (Piper, 1948). However, the methods of data collection were not consistent, nor were they extensive (some sections were sampled only two or three times, and some only once). The study examined performance of the haulage units under many conditions, and the results were supported by yet another study (Lindstrom, 1950). Lindstrom went further, and recommended the best shuttle car-loader combination on the basis of loading rates, shuttle-car speeds, discharge rates, car-change times, loader availability, crew size, productivity, and standard deviation production. He showed that two shuttle cars behind the loader were the best system and that the battery-cars discharged and traveled faster than cabled-cars.

One study (Zeller, 1950) discussed the effect of mining methods and seam conditions on mining. Some 70 time studies conducted in 42 mines were utilized. Zeller compared predicted values for shuttle-car parameters with achieved values.

The following conclusions were drawn: 1) As the shuttle-car capacity, discharge rate, and travel speed increased, production increased. 2) As the volume of the cut taken increased, production increased. 3) As the seam height increased, production increased. 4) As the entry width increased, production increased to a maximum and then remained nearly constant. 5) As the depth of cut increased, production increased to a maximum, then decreased. Zeller used time-and-motion studies to verify predicted values.

A study of continuous-miner, shuttle-car combinations (Skovran, 1952) used time-and-motion studies to compare the following:

- Continuous miner, 1 shuttle car.
- Continuous miner, 1 surge car, 1 shuttle car.
- Continuous miner, 1 loader, 1 shuttle car.
- Continuous miner, 1 loader, 2 shuttle cars.
- Continuous miner, 1 surge car, 2 shuttle cars.

His study determined that on the basis of production and performance, the system combining the continuous miner, loader, and two shuttle cars also worked best.

Another study, based on manufacturer specifications, compared the duty cycles and load factors of diesel-powered vehicles and electric shuttle cars, load-haul-dump units (LHD's), and personnel and supply vehicles (Alcock, 1978). The study revealed that diesel-powered vehicles out-produced shuttle cars because the former performed better over long hauls (greater than 1000 feet). The diesel supply vehicles and LHD's also out-produced conventional models and had better load factors.

Ketron & J. J. Davis have also performed extensive production studies on conventional mining and continuous mining respectively.

5-5.3.6 Studies that Investigated Production and Human Factors

Production and an incentive plan were the subject of another research effort (Frantz, 1950). Loader and shuttle car combinations at two mines were examined with respect to production and the results were compared against industry averages. Recommendations to eliminate delays were made in order to increase the output of coal. The advantages and considerations of an incentive plan, based on the time-and-motion study results, were discussed.

5-5.3.7 Studies that Investigated Production, Performance, and Economics

Another study (Oguz, 1971) evaluated the feasibility of a monorail mine-haulage system with regard to production, maintenance, cost, and power consumption. Time-and-motion studies were used to evaluate installation, loading and dumping. Power requirements were first calculated, and then verified with the field instrumentation. Manpower and cost savings were recommended, and the monorail's advantages were compared with those of track haulage.

5-5.3.8 Studies that Investigated Production, Economics, and Human Factors

Capital costs and production were the subject of a study of longwall and room-and-pillar mining in the USA and Great Britain (Purcell, 1952). Purcell conducted full-shift, time-and-motion studies and obtained values for travel time, mining, shuttle-car delays, place-change delays, loader delays, unnecessary delays, and available time. The operating summary covered: tonnage, average tonnage for the past 50 shifts, number of shuttle cars loaded, shuttle-car payload (average), coal height, entry width, average haul distance, average distance mined per shift, and the average loading rate of the continuous miner. A comparison was made of the actual performance and predicted values. The observation was made that continuous miners worked well below their rated capacities. This fact was at least partially attributed to the attitude of the crew, who were suspected of setting a ceiling on production.

5-5.3.9 Studies that Investigated Production and Economics

The production capabilities and economics of shortwall and room-and-pillar mining were compared to determine if shortwall is a viable alternative to the other method (Green and Palowitch, 1977). The analysis covered capital, operating, and labor costs. Tonnage was determined from an existing operation so that a cost-per-ton could be calculated. Straight-line depreciation was also used in the analysis. Assumptions included a 22 percent rejection rate, $40 per clean ton selling-price, 10 percent depreciation, 7 percent investment tax-credit, 48 percent tax rate, 15 percent rate of return on investment, and 2 operating shifts per day over 220 days per year. The calculations indicated that room-and-pillar mining would gross $6.08 per ton of clean coal, and that shortwall would gross $7.72 per ton of clean coal. The study concluded the following: 1) an average of 70 more raw tons of coal could be produced per day using a shortwall; 2) the increased costs of depreciation of the support system (shortwall) were only partially offset by reductions in the cost of supplies and materials; 3) the profitability of the shortwall system was a function of the direct operating costs, fixed and variable indirect costs, quality of the coal, and the selling price; 4) shortwall was a proven viable alternative to room-and-pillar mining in this case.

In another study (Mabruk, 1977) a computerized materials handling simulator was used to compare shortwall mining with normal pillar-recovery operations. Production and rate of return on investment were the bases of comparison. The results indicated that, for low coal, standard pillaring was the more profitable retreat method. This result was primarily influenced by the shuttle-car payload, which, in shortwall mining, decreased as the seam height decreased. The chock-support also decreased the allowable height of the shuttle cars, and as a result, the payload also decreased. The study showed that for medium and high coal, shortwall was the better method of retreat mining.

5-5.3.10 Studies that Investigated Performance and Economics

Determination of in-situ bearing capacities of roof and floor rock was the subject of another study (Nair, 1970). Bearing pressures were determined by loading the roof and floor with a hydraulic jack, and measuring the amount of indentation into the rock member. The tests were conducted for various plate areas, and were developed to aid a problem with chock-supports in longwalls. The tests were reliable, convenient, effective, and relatively low in cost.

One study compared the performance of five types of low-energy dust scrubbers (Divers and Janosik, 1978). The units were tested in a laboratory air chamber to determine their effectiveness with a water-droplet eliminator upstream and downstream, and without an eliminator. An eight-stage Andersen Cascade-Impactor-Ambient-Sampler was used to determine the dust concentration.

The results showed that a flooded fibrous-bed-type scrubber was the best, with a 95 percent efficiency when the eliminator was downstream of the unit. However, the scrubber might be susceptible to clogging, a condition that could be changed through modification. The cost was determined to be the highest at $2.50 per cfm volumetric capacity.

The Mertex small-diameter cyclone-scrubber demonstrated a 92 percent efficiency with the eliminator upstream of the unit, and the Donaldson unit was rated next at 89 percent efficiency with the eliminator in the same position. The advantages of the scrubbers were that the units acted as both eliminators and scrubbers, were fairly compact, and had a low water output to the floor. The units showed clogging tendencies, however, when clay was present, and the units operated at low water flow rates. The cost was estimated to be about $2.00 per cfm volumetric capacity.

The wetted fan scrubbers showed an efficiency of 87 percent with the eliminator downstream of the unit, and were the smallest and simplest to use. The efficiency increased with increasing water flow rate, spray water pressure, and increased water differential across the fan. Disadvantages included fan-blade erosion, mechanical damage from oversized particles, and fan stalling because of excessive amounts of water. The unit was lowest in cost at $1.50 to $2.00 per cfm volumetric capacity.

Wetted brush scrubbers showed an efficiency of 88 percent with the eliminator downstream. Advantages included resistance to clogging and the low cost of $1.50 to $2.00 per cfm volumetric capacity.

5-5.3.11 Studies that Investigated Safety and Performance

In an evaluation of noise produced by continuous miners, 30 tests were made of 26 machines; three cutter-head styles and four manufacturers were represented (Patterson and Rubin, 1976). Noise sources for continuous miners were located, and solutions to abate the noise discussed. Patterson and Rubin used time-and-motion studies and personal-noise docimeters to determine the noise sources, which included the cutter head, conveyor, drive train, and hydraulic system. The docimeters were placed on the operator's person during full-shift operation. The study also indicated modifications needed to the cutter head (pick holders), conveyor system, and drive-train for one miner.

In an ARCCM research project, longer-than-seam-height drills were evaluated in order to recommend continuing development efforts of the most promising units (Derby and Bevan, 1978). Six units were tested in the factory and underground. The Bendix Corporations' Flexible Drill was compared with conventional drills in one mine. Time studies indicated that the conventional drill was faster (1.5 minutes for the conventional drill, 1.9 minutes for the Flexible Drill). However, the temperature of the Bendix bit was lower and the bit life was longer. Also, dust and noise were less in the use of this drill. One disadvantage was that occasionally the operator was required to assist the bolt-bending and hole-aligning units in order to insert the bolt into the hole.

The Ingersoll Rand Research Rod-Changer Drill required an average of only four minutes to drill a ten-foot long hole. However, the unit needs improvements in order to be a productive, reliable machine. The cycle time is unusually long, but the drill is able to impart good torque and thrust to the cutting bit.

The Eimco Flexible Drill underwent minimal underground testing because of problems with the flexible-drill shaft; the shaft developed a helix shape because some of the layers of wire rope slipped with respect to other layers. At first, the average life span of the shaft was about ten feet of drilling. Modifications to the shaft increased the life to 53 feet, but to date the unit is not considered successful.

The Foster-Miller Association developed two flexible drills - one using short, sectioned tubes, the other using two concentric springs wound in opposite directions. The latter was further developed because of its simplicity, ability to operate continuously, lower cost, and lower maintenance requirements. A penetration rate of 3.84 fpm was achieved, but problems with vacuum system clogging developed.

The Washington State University Drilling System has not undergone an underground test, but a limited number of surface tests incicated that the unit showed promise of meeting specifications of thrust, torque, feed rate, retraction rate, and drilling within 18 inches of the rib. However, it did not meet the coal seam height requirement of 30 inches by six inches, and it bolted within 32 inches of the face (the requirement was 30 inches).

5-5.3.12 Studies that Investigated Safety and Production

A study was made of the operation of a truck using a remote-control system for simultaneous commands (Lefevre, 1977). The truck was steered by the use of a remote pendant attached to a coaxial cable. The study team counted the number of skips loaded per shift with and without remote control. A total of 222 shifts were monitored, and results showed a 20 percent increase in recovery rate at one mine, and district productivity increased 18 percent. Also, costs were reportedly low. Although the operators could be trained quickly and easily, some modifications of the controls and the shape of the on-board unit had to be made for greater safety, and the weight of the control pendant had to be reduced.

5-5.3.13 Studies that Investigated Safety, Performance, and Economics

In another study, four remote-control radio systems were compared on the basis of manufacturers' specifications (Lessöllmann and Hemann, 1977). Safety, efficiency, cost, and the maximum distance of transmission were compared for each unit. The Siemens Radio Control Unit transmitted well over long distances, but special cables and amplifiers were required. The Remotus system also transmitted well over long distances. The Montanforschung System had excellent safety and efficiency characteristics, and the Gunder and Hotten system was very safe and efficient. The advantages of the latter system outweighed the high cost of the unit.

5-5-.3.14 Studies that Investigated Safety, Production, Performance, and Economics

A study of scraper loaders (Wu, 1945) examined conditions, economics, and safety features common to scraper mining of coal. On the basis of manufacturer's specifications and in-mine observations, Wu found that scraper loading was very versatile, low in cost, safe (because moving parts were guarded), and had various other advantages, including 1) mobility; 2) ability to move both coal and supplies; 3) ability to timber close to the face; and 4) in the event of a heavy fall at the face, no expensive equipment would be lost.

5-5.3.15 Studies that Investigated Safety, Production, Economics, and Human Factors

As far as the AES Audit project is concerned, the most important earlier study was the Frantz and King (1977) investigation. This study, which was conducted as part of the ARCCM research program, examined the production, health and safety, economic, and human factor aspects in the automating of a continuous mining section. This study is important because it provides predicted production and economic values for the AES.

Using the AES machine as a model, Frantz and King investigated the operation of the present machine, and then theoretically advanced through four stages of mechanization, automation, and remote control in a selected mining case-study. They first identified 20 section mining functions and defined the present status of each (e.g., cutting and loading). The present operating state was represented by a continuous miner, pickup loader, shuttle cars, bolters mounted on the miner and a section spot bolter, an auxiliary face fan, a power center, a section belt with a feeder breaker, and a rock dusting machine and trickle dusters. In Stage I operations, the AES replaced the miner, no loader was used, the fans on the AES replaced the face fan, and a scoop and rubber tired rail cars were used for cleanup and supply handling. In Stage II, the shuttle cars were replaced by mobile bridge-carriers, and the rock duster was replaced by piped rock-dust. Also, a mine-environment-monitoring system, stopping emplacer, belt-moving machine, supply vehicle, section-parts vehicle, and a vacuum system for belt cleanup were added. In Stage III, a face-equipment-integration system, roof-condition-monitoring system, electrical-monitoring system, float-dust sampler, water-monitoring system, equipment-status-monitoring system, section monitoring console and a mine-monitoring console were added on the production section. In Stage IV, with fully-automated and remotely-controlled operation, all personnel were removed from the section except during short servicing periods.

Production was expected to triple (from 250 to 750 clean tons per unit shift) from the present to Stage IV, and the crews were expected to decrease from nine to six workers. Table XXVIII shows a tabulation of the shift times). However, the workers who remained were classified as technicians who were highly skilled and trained in actual mining, haulage, roof control, ventilation, utility work, and monitoring.

Health and safety were expected to improve because the exposure time to hazards would have been limited. The major types of accidents were analyzed through the use of fault-tree analysis. The major causes of accidents were to be eliminated through the use of the AES and advanced stages of automation.

The economic analysis included estimates of capital, operating, and labor costs. The profitability index method of determining the rate of return on investment showed that a 53 percent increase in the rate of return was possible between the present operating stage and Stage IV automation.

TABLE XXVIII

Advantages of Various States of Remote Control and Automation on a Coal Mine Section (Frantz and King, 1977)

INCREASING AUTOMATION AND REMOTE CONTROL →

	PRESENT	STAGE I	STAGE II	STAGE III	STAGE IV
Raw tonnage per shift	310	380	560	800	920
Number of section workers	9	8	9	8	6
Accident frequency per million manhours	42.8%	37.5%	33.7%	29.2%	24.0%
Rate of return on investment	15.0%	17.0%	20.0%	27.5%	23.0%

The human-factor aspects of automation were discussed, including the effect on union workers, management, and training. The study spoke to the importance of human engineering in equipment and system design.

The production results from that study were compared with those from the AES in-mine trial.

5-5 SUMMARY OF THE STATE OF THE ART

After a careful review of past equipment evaluations has been made, it may be observed that the mining industry lacks 1) a set procedure for testing machines, 2) comprehensive engineering documentation of any sort, and 3) a firm data base from which future automated mining machines may be developed. Manufacturers seem to be mainly concerned with evaluating machines with respect to specifications and reliability. The emphasis of mining companies is more on production, safety, cost, and manpower usage. Most researcher's work either is restricted by limited funds (or possibly by poor planning) or is far from being comprehensive in nature. However, as the Penn State Audit Team's work here clearly indicates, mining studies should be as complete as possible. The data base achieved by a comprehesive study is exactly what is needed for the most effective machine design in the future.

Additional references included in the Bibliography (Chapter 4).