A mining research contract report
JANUARY 1987

DEMONSTRATION OF LONGWALL MINING IN A STEEPLY DIPPING COAL SEAM

Contract J0233923
Snowmass Coal Company

BUREAU OF MINES
UNITED STATES DEPARTMENT OF THE INTERIOR

NOTICE

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50272-101

REPORT DOCUMENTATION PAGE

1. REPORT NO.

2.

3. Recipient's Accession No.

4. Title and Subtitle
Demonstration of Longwall Mining in a Steeply Dipping Coal Seam

5. Report Date
January, 1987

6.

7. Author(s)
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8. Performing Organization Rept. No.

9. Performing Organization Name and Address
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10. Project/Task/Work Unit No.

11. Contract(C) or Grant(G) No.
(C) J0233923
(G)

12. Sponsoring Organization Name and Address
Bureau of Mines - Denver Research Center
P.O. Box 25086 Building 20
Denver Federal Center
Denver, Colorado 80225

13. Type of Report & Period Covered
Final Sept. 77-May 86

14.

15. Supplementary Notes

16. Abstract (Limit: 200 words)

This report describes the demonstration of longwall mining in a seam dipping 27 to 34°. Equipment was selected on the basis of capability, compatibility, and cost. Snowmass, Bureau of Mines, consultants and manufacturers were involved in the selection process. The equipment arrived in early 1981, prior to completion of development for the first panel; therefore, the equipment was operated on the surface for training. The surface training paid off with a fast, efficient installation and start-up of the face. The longwall equipment operated very well on the steep pitch with an average equipment downtime of 21.9%. Coal handling outby the face was a problem which caused an additional downtime of 19.5%. Development for the panels proved to be the major problem in the economics of longwall mining on a steeply dipping seam. During the first 9 months of operation the longwall averaged 71.9 TPMS and development averaged 0.6 TPMS for an overall average of 14.1 TPMS. Had mining continued, timing of development would have been another major problem. Monitoring of subsidence, pillar stability and shield loading was done in connection with mining, but little useful information was gained from the monitoring.

17. Document Analysis a. Descriptors

b. Identifiers/Open-Ended Terms

c. COSATI Field/Group

18. Availability Statement

19. Security Class (This Report)

20. Security Class (This Page)

21. No. of Pages

22. Price

(See ANSI-Z39.18)

See Instructions on Reverse

OPTIONAL FORM 272 (4-77)
(Formerly NTIS-35)
Department of Commerce

FORWARD

This report was prepared by Snowmass Coal Company, Palisade, Colorado, under United States Bureau of Mines Contract number J0233923. The contract was initiated under the Conservation and Development Program. It was administered under the technical direction of the Denver Research Center with David Wisecarver, Gregory Morlock, and Tim Hackett acting as Technical Project Officers. David J. Askins was the contract administrator for the Bureau of Mines.

This report is a summary of the work recently completed as part of this contract during the period of September 30, 1977 to May 7, 1986.

The report contains no patentable material.

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INTRODUCTION

Coal Mining has been done in the North Thompson Creek area of Pitkin County, Colorado, off and on since before the turn of the century. These coal seams dipping approximately 30°, were of interest because of their high quality medium volatile coking coal. As noted in the "Description of Geology and Pre-mining Conditions" section of this report, Albert M. Keenan introduced longwall mining to the Thompson Creek Mines in 1958 and 1966. Economic conditions prevented continued operation of the face in 1966.

On December 27, 1976, Mr. Keenan as Chairman of the Board for Anschutz Coal Corporation, of Denver, Colorado, submitted an unsolicited proposal to the Bureau of Mines for a cooperative project. The proposal title was "Application of Mechanical Longwall Mining Systems to the Recovery of Coal Reserves in Seams of Medium Inclinations 20 - 50°". As a result, the Bureau of Mines issued a request for proposal No. H-272033, "Demonstration of Longwall Mining in a Steeply Dipping Coal Seam", on April 14, 1977. Anschutz responded with a proposal on May 16, 1977. The Bureau of Mines contract No. H0272033 was awarded to Anschutz effective September 30, 1977. The contract was transferred to the U.S. Department of Energy October 1, 1977 and given the Department of Energy No. ET-77-C-01-8903.

Anschutz Coal Company sold the mine to Snowmass Coal Company April 13, 1979. The contract was transferred to Snowmass on September 30, 1979.

The Department of Energy contract number was changed to DE-AC-01-77ET12&51 on May 7, 1980. The contract was transferred back to the Bureau of Mines on May 10, 1983, contract No. J0233923.

Mining started in August, 1981 and operated at full capacity until April, 1982. Due to the unsettled coal market, mining continued, off and on through May, 1984. Approximately half of the first of three demonstration panels was mined when mining ceased. A contract modification approved May 7, 1986 terminated mining requirements and initiated the submission of the final report.

I. DESCRIPTION OF GEOLOGIC AND PRE-MINING CONDITIONS

A. Coal Age and Description of Deposition

Snowmass Coal Company controls some 10,000 acres of coal lands located in Northwestern Pitkin County, Colorado, see Figure I.A.-1. There are nine recognized seams on the property, seven of which are of +4 foot mineable thickness.

These coal seams are located in the Bowie Member of the Mesaverde Formation[1] of upper Cretaceous Age. The Bowie member consists of late cretaceous sandstones, shales and coals which were deposited in a fresh to brackish water deltaic environment.

Deltaic depositional environments are characterized by lenticular coal bodies with channels, producing areas of thick and thin coal. Sandstone lenses and overbank splay deposits of clay produce partings and clastic channels within the coal. These have been identified in the worked out areas of the mines in the immediate vicinity.

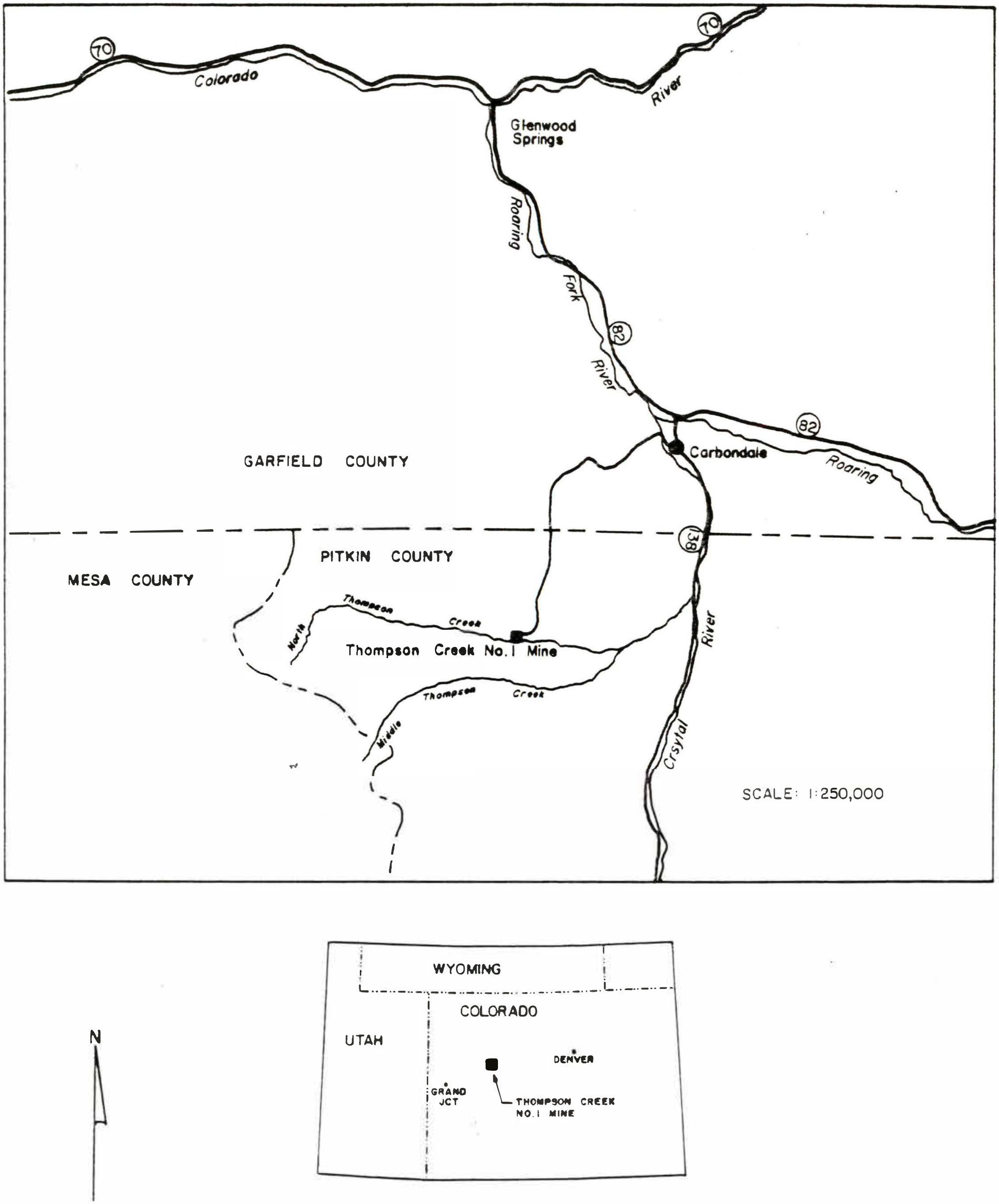
The regional strike of the Mesaverde Formation ranges from North to N 10° W, and dips 27° to 34° westerly. This represents the eastern margin of a synclinal trough whose axis lies three miles to the west and trends north westerly. The lower most rock unit of the Bowie member, known as the Rollins Sandstone, is a massive, medium grained, white to grey beach sand. The average thickness of the Rollins Sandstone is 105 feet.

The "A" Seam coal in Thompson Creek #1 Mine, Figure I.A.-2, lies directly above the Rollins Sandstone. However, in the area of Panel 1-1 in Thompson Creek #1 Mine, a carbonaceous shale, 1 - 1.5 feet thick occurred between the Rollins Sandstone and the "A" Seam. Measured thickness of the "A" Seam ranged from 7.0 to 8.5 feet. These measurements were taken in the area of Panel 1-1.

The roof above the #1 Mine "A" Seam consisted of alternating units of sandstone and shale. Typically, the immediate mine roof consisted of 1-3 feet of shale followed by 8-12 feet of sandstone. Testing of the roof and floor strata was done during the early stages of the rock mechanics program for the longwall project. A total of 86 samples of roof and floor rock were tested by a subcontractor, during the physical rock properties testing program. Average results were as follows:

Unconfined Compressive Strength

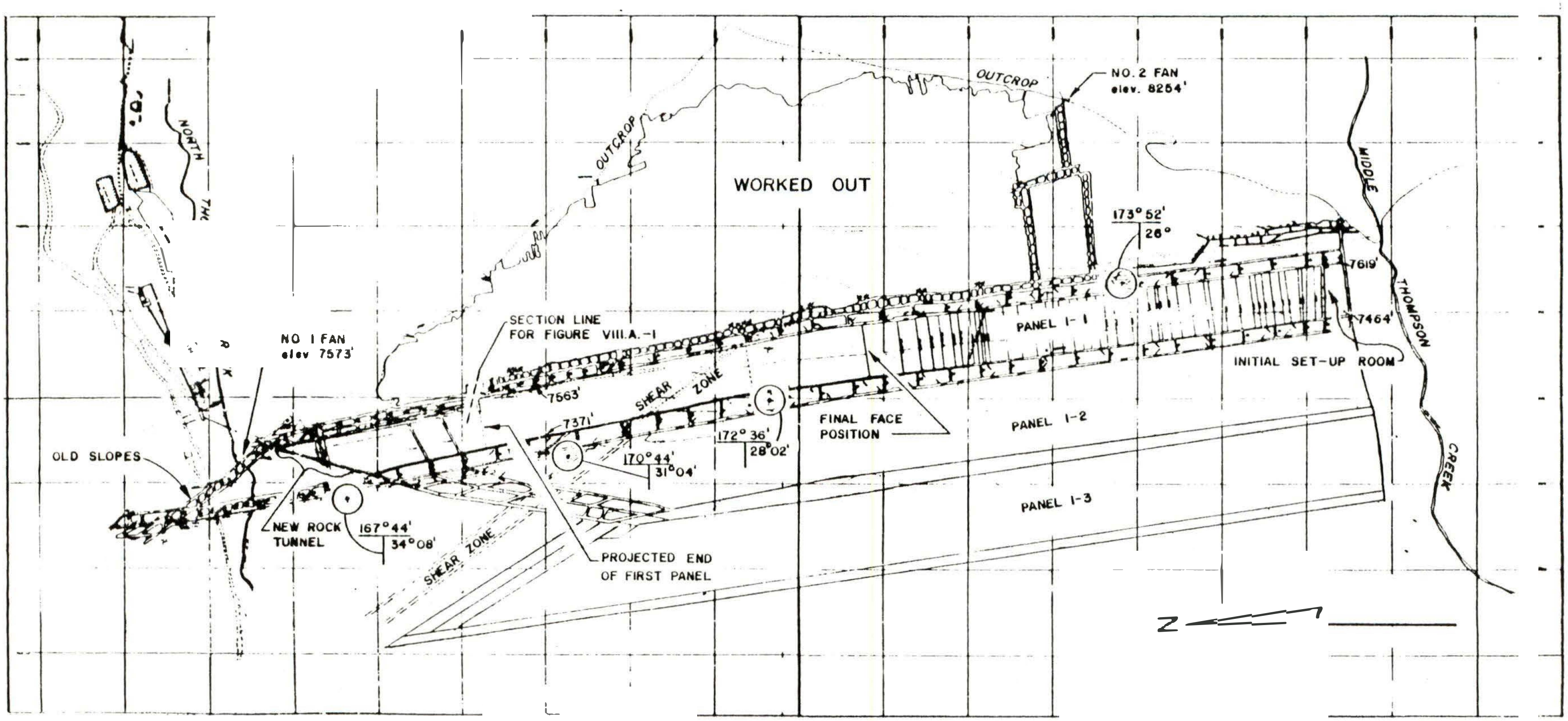
Floor Sandstone	14,092 mean (psi)
Roof Shale	13,455 mean (psi)
Roof Sandstone	17,730 mean (psi)



LOCATION MAP

FIGURE I.A.—1

THOMPSON CREEK NO. I MINE MAP



SCALE: 1" = 1000'

FIGURE IA.-2

B. Shear Zone and Other Discontinuities

There were a number of minor discontinuities in the roof of the "A" Seam. The majority of these were 1 to 3 foot displacement faults. Other common occurrences included channels or slips. These normally produced localized poor roof conditions and water discharging from the roof for a short period subsequent to mining.

One major fault, often referred to as the shear zone, occurred in the #1 Mine "A" Seam. This fault trended N 37° W through Panel 1-1, see Figure I.A.-2. It consists of soft floor and extremely poor roof conditions. The width of this fault ranged from 85 to 120 feet with the area of heavy roof loading observed to be as wide as 180 feet in some areas. The roof in this area ranged from hard sandstone and shale slabs to soft clay and "pea gravel". The coal seam through the center of the fault ranges in thickness from 6-inches to 7-feet and was observed to be nearly vertical at times through the fault zone. The displacement of this fault area ranged from 8 to 18-feet. In some areas the displacement was gradual and in other areas it was observed to be quite severe.

No longwall mining occurred in this major fault zone. However, several sets of development entries were driven through the fault. The unique problems associated with mining in this fault area, is discussed in the "Unusual Problems" section of this report.

C. Local Mining History

The main portal of #1 Mine started at the outcrop of the "A" Seam and followed the strike of the seam. This entry was the tailgate for Panel 1-1. In the 1950's and 1960's all of the coal above this main entry was mined out. The principle mining method was room and pillar, drilling and shooting from the bottom up. Sheet metal was placed on the floor before the coal was shot to facilitate the coal sliding down dip to the haulage entry. The main haulage to the surface was by locomotives and coal cars.

During this time period, a "favorable"[2] overall recovery was achieved with the room and pillar method. Consequently, most of the up dip coal above Panel 1-1 has been mined out and the roof has subsequently caved. This created a rather large cave area directly up dip from the first demonstration panel.

In 1958, Albert M. Keenan first introduced longwall mining to the Thompson Creek #1 Mine.[2] His experimental panel had a 100 foot face down the dip and advanced 227 feet along the strike. The face was advanced three feet at a time, hand mining, drilling and blasting on the solid. For support on the face 200 yielding steel mine props were purchased in Germany. The test proved that longwall mining could be accomplished successfully but at no greater increase in productivity per man.

In 1966 another experimental longwall panel was set up in the Thompson Creek #2 Mine.[2] The support system for this face was a Klockner-Ferromatic self advancing powered roof support. The support consisted of two integrating supporting chocks which alternately advanced and pulled forward. These chocks were set up on a face 320 foot down the dip. The face was set up as an advancing longwall with a two entry headgate and tailgate developed approximately 200 feet ahead of the face along the strike of the coal seam. Initially coal was loaded into the face conveyor by shooting on the solid, but a shearer was added after two months. This experiment proved that a mechanized longwall could be established and operated successfully on a seam pitching 30°.

II. METHOD OF SELECTING FACE EQUIPMENT AND MINING METHOD

A. Specifications

Equipment selection for the Longwall Demonstration Project commenced in mid 1977. It was decided by the operator, that bids would be solicited from manufacturers who were prominent suppliers in the United States market; and who, in the operators opinion, were well acquainted with applications, technology and requirements of longwall mining on an inclined seam. The bidding process commenced with five manufacturers.

A set of specifications were developed by the operator. These specifications included a basic description of the mining conditions, as well as, the method of mining proposed to the United States Bureau of Mines for the Longwall Demonstration Project. Specifications for each unit in the longwall are shown in Figure II.A.-1.

The mining conditions outlined included the seam thickness, properties of the coal seam, properties of the roof and floor, pitch of the seam, faulting, gas and water. The mining method would be longwall retreat mining, with a face length of 400 feet conducted on the full pitch of the coal seam. A brief description of the headgate and tailgate entries was also included.

FIGURE II.A.-1

LONGWALL EQUIPMENT SPECIFICATION SUBMITTED TO MANUFACTURES

ROOF SUPPORT

a. Shield-type supports (2 or 4 leg).
b. 350 ton minimum capacity or heavier if recommended by the manufacturer for the mining conditions.
c. Anti-topple provisions, anti-slew provisions, anti-creep provisions.
d. Adjacent controls (for operator safety).
e. Contact advance (conveyor advance not connected to conveyor).
f. Hydraulic canopy extensions.
g. Hydraulic anti-spalling plates.

FACE CONVEYOR

a. Capacity, 700 tons/hour, minimum
b. Manufacturer to specify drive horsepower and location, panline width, height and equipment included in the quoted price.
c. Twin outboard chain, manufacturer to specify chain and flights.
d. Chain tensioner.
e. Include electric motors and controls.

STAGE LOADER

a. Traveling swivel-mount to tailpiece of the pan conveyor. Complete unit to be furnished by manufacturer.
b. Lump breaker.
c. Capacity at least equal to that of face conveyor quoted.
d. Length, width, horsepower, chain speed, length of travel, etc., specified by manufacturer.
e. Twin outboard chain and flights.
f. Not attached to face conveyor.
g. Include electric motors and controls.
h. 100 single props.

SHEARER

a. Double ended, ranging arm with chainless haulage and safety winch.
b. Face conveyor accessories for chainless haulage to be quoted with shearer.
c. Lump breaker included.
d. Manufacturers to specify horsepower, drum size, bit lacing, rpm, speed, control, etc.
e. Radio remote control.
f. Dust control and hoses.
g. Included electric motors and controls.

FIGURE II.A.-1

CONTINUED

HYDRAULIC POWER PACK

a. Designed for mounting on monorail over panel belt conveyor tail piece.
b. Number of pumps, pressure, capacity, etc., to be specified by manufacturer.
c. Oil in water ratio to be specified by manufacturer.
d. Include electric motors and controls.

ELECTRICAL EQUIPMENT

a. Transformer of suitable capacity and voltage for face operation. Primary voltage 4160 V AC.
b. All electric controls for complete face operation.
c. Complete set of electric cables for face equipment.
d. Complete set of face lighting equipment.
e. Complete set of face communication equipment.
f. Methane control.
g. Complete set of face signalling equipment.
h. All equipment must satisfy U.S. Government regulations.

One requirement outlined to all manufacturers was that all equipment quoted must be designed to operate in normal fashion and in compliance with all existing Federal and State regulations. This specification was critical to the success of the project and was of major concern due to the fact that some of the specialized equipment had not been previously used in this country.

Selection of the roof support capacity requirement was based on testing of the roof and floor strata during the early phase of the rock mechanics program for the project. The results of this testing is given in Section I.A.

B. Selection

Quotations from the five manufacturers were received by the end of November 1977. Over the next few months, the quotations were analyzed and discussed in an effort to decide what specifications were most desirable for the Longwall Demonstration Project. Also, an effort was made to narrow the list of prospective manufacturers from five to three.

Several meetings were held during this time period between the operator and the U.S. Bureau of Mines personnel. All equipment quoted was discussed in detail including the advantages, disadvantages and compatibility of each. Naturally, price was a consideration, however the success of the longwall project was of primary importance. One of the major considerations was the ease of operation considering the steep pitch and lack of experienced operators.

Each manufacturer had opportunities to meet with the people involved in equipment selection to discuss why they quoted specific equipment. Advantages and disadvantages were discussed with each manufacturer.

During this time period, accurate records were kept regarding the most desirable equipment specifications to insure success of the Longwall Demonstration Project. Ease of operation, safety, tons per hour, pan size, etc., were all considered based on the equipment quoted by the manufacturers.

By March 1978, the operator and Bureau of Mines personnel had developed an accurate set of specifications for the project equipment. The list of manufacturers was reduced to three and each was asked to re-bid based on a much more stringent set of specifications.

The equipment for use on the longwall demonstration project was selected and ordered by the end of 1978. One manufacturer would supply the shearing machine, one manufacturer would supply the roof support, and another would supply the remaining equipment.

III. DESCRIPTION OF EQUIPMENT

A. Roof Support

The operator chose the Hemscheidt* shield type, self advancing Troika System (Type G320-14.5/33.5) for the roof support, see Figure III.A.-1. The Hemscheidt Troika System was a two leg, 350 ton support designed for a seam height of 57 to 132-inches. The shields were equipped with a 30 inch hydraulic canopy extension and 43 inch hydraulic anti-spalling plates. The support centers were 59 inches, and there was a 6 inch gap between the canopies of each shield.

The Troika System was the most modern two leg shield support system for pitching coal seams available. Its proven reliability in pitching coal seams was an important consideration.

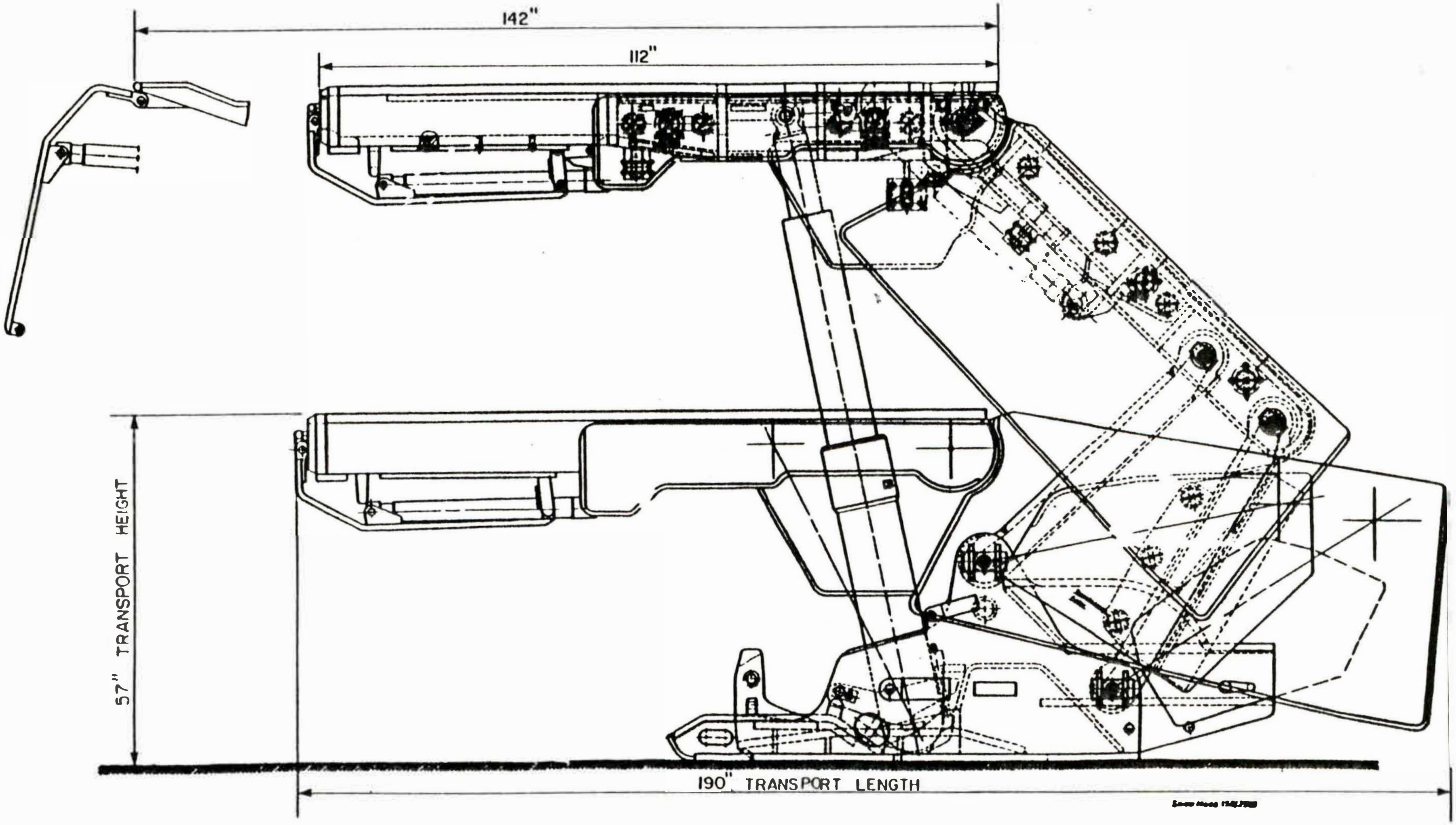
A Troika consists of a set of three (3) shields connected by a beam or Troika bar. The center shield is rigidly attached to the Troika bar and advanced by the dual acting (D.A.) rams of the two outer shields. The two outer shields are then advanced to the position of the bar on the center shield. A steering ram allows the center shield to be moved uphill or downhill if necessary. Figure III.A.-2 shows a typical advance sequence. Figure III.A.-3 is a table of the final specifications for the shields and shows how the Hemscheidt shields complied.

The face conveyor was designed to operate independent (unattached) from the roof support. The conveyor advance is accomplished by a dual acting ram attached to the uppermost shield in each Troika set. The conveyor is not attached to this ram and therefore, cannot be retracted. On every other Troika bar there is an anchor station, which is a hydraulic cylinder attached to the conveyor by a chain. This enables the face conveyor to be moved up or down the face.

B. Face Conveyor

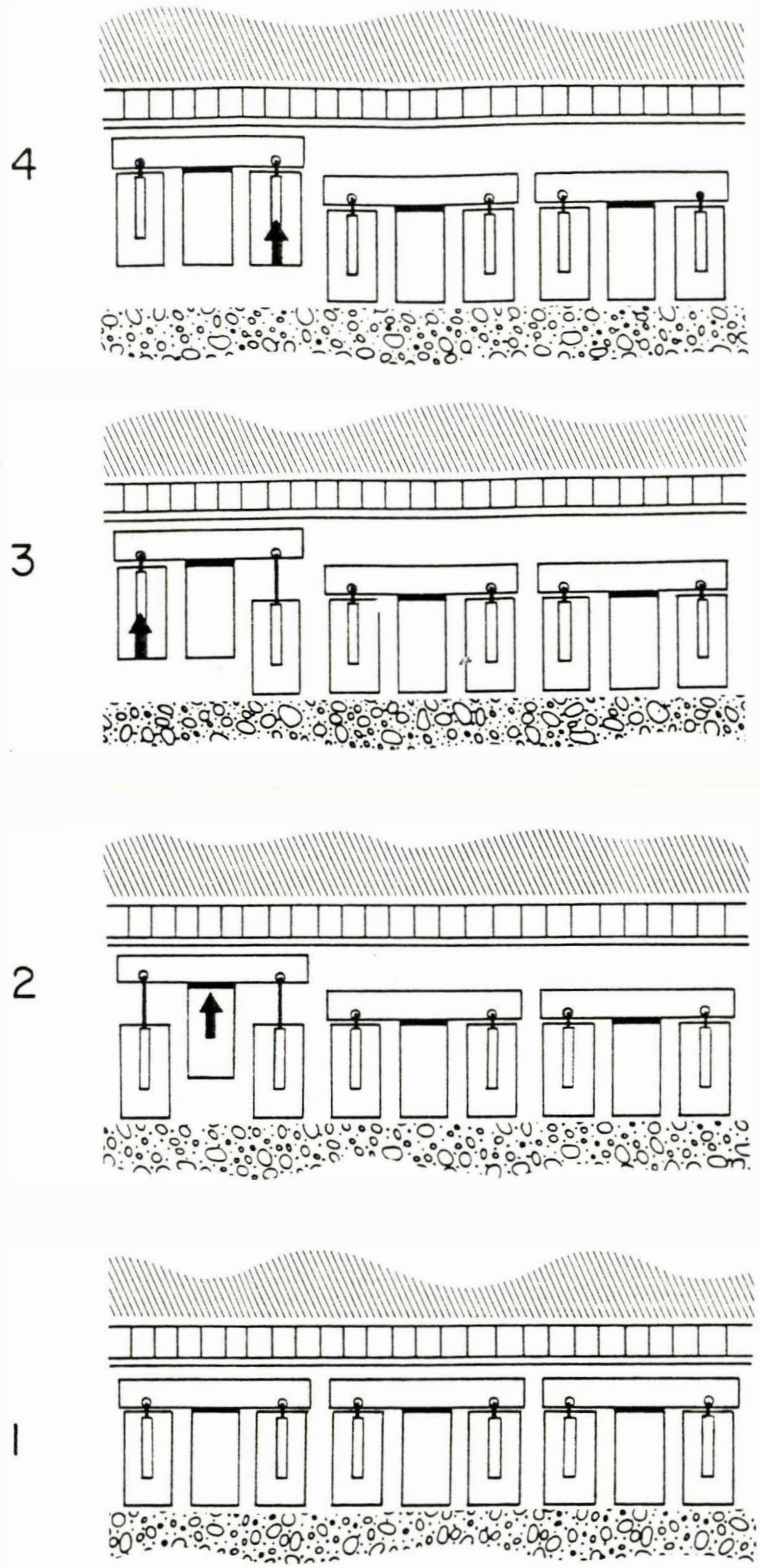
A Dowty Meco*, 800 t.p.h. armored face conveyor was selected for the longwall project. A 22 mm twin outboard flight chain was selected to give maximum retention of coal on the conveyor due to the additional flight rigidity. The conveyor pans were 59 inches long, 30 inches wide and 8 3/4 inches high. The pans had quick release type connectors, and a horizontal and vertical articulation of 3°. Attachments for the anchor stations were installed on each pan to allow connection anywhere along the face. Thirty two inch spill plates were selected to minimize the chance of injury from coal sloughage as well as coal and rock sliding down the pitch.

* Reference to specific products does not imply endorsement by the Bureau of Mines



Hemsheidt Troika Shield
Type G320-14.5/33.5

FIGURE III. A-1



Typical Advance Sequence Of Troika Shield Support

FIGURE III. A.-2

FIGURE III.A.3

ROOF SUPPORT - FINAL SPECIFICATIONS

SPECIFICATIONS	HEMSCHEIDT
1. Is all equipment included to operate the longwall face in normal fashion and and in compliance with existing Federal and State legislation?	Yes
2. What provision is made for anti-toppling?	Contact advance Auto side shields Bottom supp. is anchor
3. What provision is made for anti-slew?	Adj. dir. of adv. 5° per cut with/outside shields
4. What provision is made for anti-creep?	Same as Spec. 3
5. No. of support legs	2
6. Yield load per leg	175 Tons
7. Total support load at yield	350 Tons
8. Setting load of support	280 Tons
9. Support load-density (30° cut) - Before Cut	7.2 T/sq. ft.
- After Cut	5.7 T/sq. ft.
10. Does support have contact advance?	Yes
11. Adjustable side shields? One side or two sides?	2 sides
12. Area of roof covered at face - Before Cut	90%
- After Cut	79%
13. Area of base plate floor contact.	19 sq. ft.
14. Base plate pressure at yield load	247 psi.

FIGURE III.A.-3

CONTINUED

15. Support centers	59 in.
16. Canopy	Rigid
17. Base	Rigid
18. Advancing ram	Dbl Acting
19. Support control	Adj Bi Direct, Push Button
20. Distance between canopies	6 inches
21. Min. closed height	59 inches
22. Max. open height	132 inches
23. Operating fluid	5:95 oil/H_2O
24. System pressure	4740 psi
25. Leg setting press	4750 psi
26. Max. yield press.	5800 psi
27. Pump output	23 gpm
28. Support weight	14 tons
29. Base bearing area	19 sq. ft.
30. Canopy bearing area	43 sq. ft.
31. Yield tip load	114 tons
32. Setting tip load	94 tons
33. Legs	Dbl acting Dbl telescope
34. Yield/leg	175 tons
35. Setting load/leg	140 tons
36. Advancing ram conveyor advance	Ajs., 7-8 tons
37. Advancing ram shield advance	35 tons

FIGURE III.A.-3

CONTINUED

38. Canopy side shield rams	5 inches
39. Stroke side shield rams	7.9 inches
40. Force out	8 tons
41. Force in	5-6 tons
42. Cycle time to move shield	8-10 Sec/shield
43. No. of shields/face	37 Troika shields full pitch face
44. Tail gate anchor support, location and function	Hyd. prop. in tailgate 7 anchors along face
45. Cantilever canopy extension	Yes 30 inches
46. Face sprags	Yes about 43 inches

The original quotation consisted of one (1) 150 h.p. motor located at the delivery end of the face conveyor. Enough doubt was raised regarding start up power under fully loaded conditions, that a second 150 h.p. motor was added at the return end of the conveyor.

The ramp plate on the face side of the conveyor had a square edge to prevent the conveyor from climbing on loose coal. The hard floor prevents the ramp plates from digging in.

C. Stage Loader

The Dowty Meco* stage loader had a twin outboard, 18 mm flight chain with a pan width of 31 inches and a flight spacing of 21 inches. It was powered by a 75 h.p. motor and has a capacity of 800 t.p.h. The total length of the stage loader was 130 feet and it was capable of 27 feet of travel before it required repositioning. The chain speed was designed for 256 feet/minute. The stage loader was a self advancing type and was not connected to the face conveyor.

A lump breaker was designed and purchased to be mounted on the face conveyor where coal dumped onto the stage loader. Before installation it became apparent that height restrictions in the headgate would cause significant problems with the use of the lump breaker as designed. It was modified to mount on the stage loader. In addition to modifying the mount, heavier plate steel was added to the pan below the new lump breaker location.

D. Shearer

The shearing machine chosen, Anderson Mavor 500, was manufactured by Anderson Mavor* and purchased through the Dowty Corporation*. It is a 400 h.p. double ended ranging drum shearer, chainless haul, equipped with three, hydraulic motor driven roll rack haulage units. Each haulage drive unit is equipped with dual disc brakes. A safety winch with a constant pull of 15 tons was purchased to hold the weight of the machine allowing the haulage drive units to be used for coal cutting. The safety winch was not needed because the shearer had fifty percent more haulage power than most shearing machines and nearly twice the power needed to overcome the 30° pitch of the face.

The shearer was equipped with two, 60 inch diameter cutting drums with a design web of 30 inches and was capable of speeds from 0 - 30 feet per minute on the level. The shearer had

* Reference to specific products does not imply endorsement by the Bureau of Mines

a crusher mounted on the uphill side of the machine above the face conveyor. The lubrication of the machine was designed for effective operation on a 30° inclined face.

The shearer was also equipped with radio control for a single operator, if needed, for additional operator safety. This feature was added due to the possibility of face sloughage and dust problems, which could occur on a steep face.

E. Electrical

A 1,000 KVA power center, 4160/950V AC, was ordered along with all necessary electric cable to operate the complete longwall face. A complete control console SIVAD, manufactured by Davis of Derby*, was purchased for installation at the stage loader. From the SIVAD an operator could control (start or stop) all face functions, monitor equipment and cable conditions, and communicate with men on the face or throughout the mine. Face lockouts were provided every 25 feet of face length for safety during maintenance or downtime.

Communication was provided every 50 feet along the face to enable workers to communicate with the operator at the SIVAD. Methane was also monitored with a digital readout at the SIVAD.

One Victor* fluorescent light was installed on each shield with an additional light every other shield. These lights were located under the canopy of the supports.

F. Hydraulic Pumps

The hydraulic system consisted of two Hauhinco* pumps, type EHP 3K60/40, equipped with 100 h.p. motors. The reservoir capacity of each pump was 240 gallons and each was rated at 32 gallons per minute.

The hydraulic pumps were originally designed to be attached to a monorail at the stage loader. Before face installation, it was decided to place the hydraulic pumps and the power centers in the entry below the headgate. This reduced dust build-up as well as allowing additional clearance in the area of the stage loader.

The entry below the headgate had track installed previously. The placement of hydraulic pumps and the power center on flat cars in this entry enabled the equipment to be easily moved as the face was retreated.

* Reference to specific products does not imply endorsement by the Bureau of Mines

Also on the flat car was a water pump to pressurize water for the face. This pump operated at a pressure of 400 psi, 60 gallons per minute and supplied 125 psi to the spray nozzles on the shearer.

G. Haulage From Longwall Face

For haulage from the longwall face, 42 inch conveyor belts were installed. At 500 feet per minute these belts had an effective capacity of 800 tons per hour. The conveyors were equipped with a belt storage unit capable of storing one 500 foot roll of belt before removal was necessary. Panel belts consisted of rigid frame, roof hung structure with 35° troughing idlers. The rigid structure facilitated installation and, more importantly, removal as the longwall face was retreated.

IV. MINE DESIGN

A. Slopes

When Thompson Creek #1 Mine was re-opened in 1975, every effort was made to take advantage of old works abandoned in the 1960's. See Figure I.A.-2 for a mine map. A set of slopes mined by drill and shoot methods in the 1960's was rehabilitated. These two slopes had a dip of 18° in a N 35° W direction. Track was laid down the haulage slope and a 250 H.P. hoist was installed. A 36 inch beltline was installed in the second slope.

The new long range mine plan developed in 1980 called for a rock tunnel to be driven from the surface to intersect the coal seam. From there, a new slope would be driven in a S 10° W direction at an apparent dip of 15° in the coal seam. These new slopes are shown on the mine map on Figure I.A.-2. The rock tunnel and slope was installed as planned. The new slope intersected the lower gate entries of the first longwall panel. The rock tunnel and slope contained the main slope belts and served as the main supply haulage slope. The mine supply system was converted to rubber tire diesel supply and the rail battery locomotives were abandoned.

From the lower gate entries of the first longwall panel, two additional slopes were started parallel to the belt/haulage slope. Mining of all three slopes was conducted two shifts per day in an effort to develop a second longwall panel as soon as possible. The newest generation of heavy duty roadheader miner, high power shuttle cars and belt haulage was then used for development of the slopes.

B. Panel Entries

A two entry panel had been started from the bottom of the old slopes. An Alpine F-6A* roadheader miner was installed in each entry and mining commenced. The uppermost entry of the two would be the longwall headgate when the panel was completed. Face fans were used to ventilate the faces. The upper entry served as the return.

The haulage behind the Alpine roadheaders, was roof hung monorail bridge conveyors. The coal from the bridge conveyors was dumped into 6-ton bottom dump coal cars. The coal cars were pulled by 8 and 12 ton battery locomotives which were unloaded onto the slope conveyor.

This two entry panel was mined with the entries rising 2° to the strike of the seam to facilitate water drainage from the face. Coal from the upper entry was transported down the crosscut into the coal cars using sheet metal chutes and the force of gravity. Crosscuts were mined from the lower entry to the upper entry by drill and shoot methods.

This mining method was slow and labor intensive. The Alpine roadheaders were adequate for cutting the coal, but it was also necessary to cut the floor to insure a flat roadway. The Alpine roadheaders were not designed to cut the hard sandstone floor, therefore, development was slow and maintenance costs were high. Figure IV.B.-1 shows a typical entry cross-section.

At the same time, rehabilitation work was done on entries near the level of the portal and a new entry was mined part of the panel length for the tailgate. The mining method was the same as described for the lower set of development entries.

The battery locomotive system of coal haulage was used because the existing mining law prohibited the use of intake escapeways for conveyor belts or trolley locomotives. Coal haulage by battery locomotives was continually hampered by local rolls in the coal seam, water on the track, battery trouble and the time element involved in hauling the coal cars to a dumping point as far as 5,000 feet outby the face.

C. Longwall Panels

The equipment purchased for the longwall demonstration project was designed for a face length of 400 feet. The configuration of the old mine works resulted in a face length of 340 feet and a panel length of 5,000 feet. The Longwall Demonstration Project consisted of the mining of three consecutive panels down dip. The first panel was proposed as a 400 foot face

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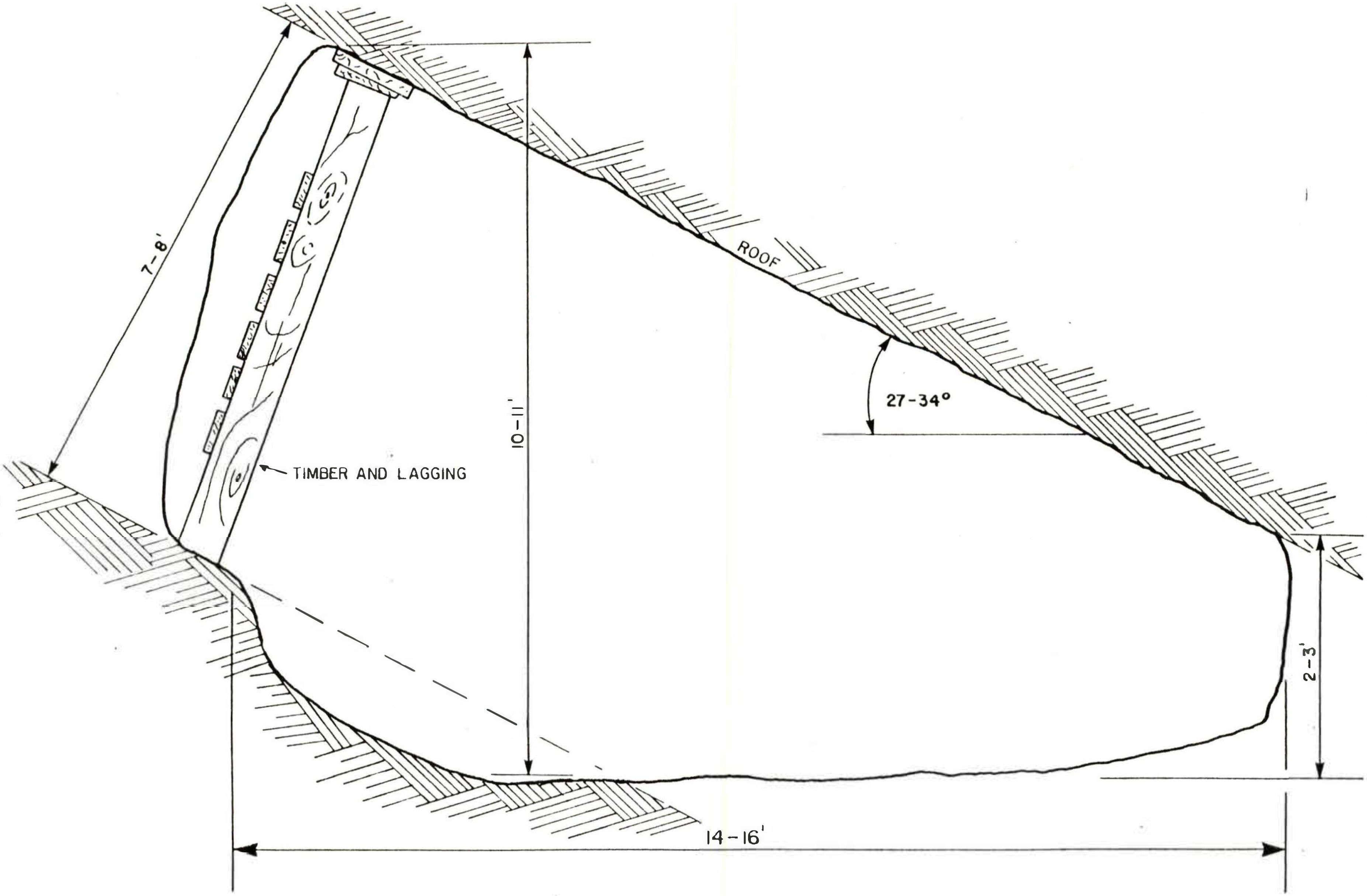


Typical Entry Cross Section

FIGURE IV.B.-1

length, the second was to have a 500 foot face length, and the third was to have a 600 foot face length. The project was altered slightly in 1980 when a new long range mine plan was formulated. The new plan called for a 500 foot face length in both the second and the third panels.

V. LONGWALL CREW TRAINING

A. Basic Background

By early 1981, all of the longwall equipment had arrived at the mine site, but the panel and slope development had not yet been completed. Very few experienced longwall miners were employed at Snowmass, therefore, a complete training program was instituted. Classroom lectures and hands-on training programs were instituted using available expertise from a local college and factory representatives. Classes were taught in hydraulic, electrical, mechanical and safety aspects of longwall mining.

Three foreman were sent to Europe to observe pitching seam mines under similar seam conditions. These foreman were able to work on these European faces and become familiar with the day to day problems and practices of pitching seam longwalls. These foreman spent six weeks on the European faces and gained valuable longwall experience.

Also, a maintenance foreman was sent to Scotland to the Anderson Mavor* factory for hydraulics and electrical training on the AM 500 Shearer. This provided excellent, in depth, hands-on training on the same machine that would be used on the Snowmass face.

Since virtually all of the longwall equipment arrived unassembled, a potential training program existed. All 84 roof supports were "hosed up" thus providing an excellent opportunity for the mechanics and the surface shop employees to become totally familiar with the hydraulics.

A 150 foot "training face" was assembled on the surface by the underground mechanics. This provided the electrical and mechanical training on the longwall face. All of the assembly work was supervised by representatives of the manufacturers. In addition, many of the compatibility problems were identified and dealt with at that time. Although no major problems occurred, there were several minor problems that could have delayed installation in the face.

After the "training face" was operational, most of the underground workforce was given two day training on the face. This gave each man a chance to see what longwall mining was all about.

* Reference to specific products does not imply endorsement by the Bureau of Mines

After the training, longwall job bids for all three shifts were posted and awarded. The longwall manufacturers assisted with this process by providing job descriptions.

B. Training on the Surface

After the longwall crews were assembled, each crew was given intensive, two week, hands-on training in the operation of the longwall. Potential problems and their solutions were discussed. Each man was given ample opportunity to become familiar with his job, as well as, other jobs on the face that he might be called upon to perform. Additional classroom training was provided as well. When the panel development was completed, the longwall crews dismantled the "training face", transported the equipment into the mine, and installed the full face underground.

Although everyone involved had estimated an 8 - 10 week surface to mine set-up time, the installation actually took 5 weeks. The intensive training and in-depth planning made the difference.

C. Training Underground

As part of the training program, the two production crews would work together underground on day shift under the supervision of the manufacturers. The representatives from each manufacturer were responsible for production training on their particular equipment. The plan was to start production very slowly, one or two shears per day, for a thirty day period with both crews. This would provide thorough training on shearer operation and horizon control, advancement of face conveyor, roof support, and movement of the stage loader. Proper maintenance would also be taught under actual production conditions.

The longwall started production on August 11, 1981. Two shears were cut on each of the first two days and three shears were cut on the third day. By the end of the second week it was time to split the crews and begin two shift per day production. The thirty day production training period was cut in half. The manufacturers continued to provide supervision on all three shifts for the first two months of operation. Production was scheduled on the first two shifts with maintenance scheduled on the third shift. The third shift was also trained to cut coal if needed.

VI. TRANSPORTATION AND INSTALLATION OF FACE EQUIPMENT

A. Roof Support

In the months proceeding the longwall installation, #1 Mine utility crews were very busy getting things ready. Maintenance work was performed on the track from the portal in the tailgate entry to the longwall set-up room. Rail was re-spiked and additional ties were added as needed. It was also necessary to move the track toward the high rib in some areas to provide additional clearance for shield transportation. The roof was drilled, shot and loaded out in areas where proper clearance could not be achieved any other way.

Two drop deck shield cars were purchased. These cars were designed for transporting the 14 ton shields and were the lowest profile cars practical. The shields were loaded onto the shield carriers by a crane on the surface and pulled to the tailgate side of the longwall by 8 and 12 ton battery locomotives.

The longwall set-up room was driven from the headgate up the pitch to the tailgate by drill and shoot methods. The coal was slushed to the entry below the headgate with electric slushers. It was then loaded into 6 ton coal cars and transported to the conveyor belt on the old slope. This set-up room for the initial installation of the long wall was driven 24 feet wide to provide enough room to maneuver the shields and face conveyor into place.

The tailgate entry was widened from 14 feet to 20 feet in the area 35 feet on both sides of the set-up room. This involved drilling and shooting the sandstone floor. It was necessary to keep this floor as level as possible so as not to cause problems in skidding the shields along the rock floor. A 15 ton hydraulic winch was installed in the tailgate entry, 30 feet inby the set-up room. Heavy duty anchor stations for the sheave blocks were also installed in the tailgate at the top of the set-up room. A ramp was fabricated out of 1 inch steel plate and anchored to the rock floor along the track. This ramp was used to skid the shields off of the drop deck carriers and onto the floor without tipping them over.

An old 40 h.p. army electric winch was rebuilt in the shop and equipped with 400 feet of 7/8 inch steel cable. This winch was anchored in the headgate entry at the bottom of the longwall set-up room. The long wall control console and the hydraulic power packs were then transported in the entry below the headgate on track and winched up the crosscut to the headgate entry. This equipment provided the electric power for the two winches and the hydraulic power to operate the shields once they were in place. The electric power cable for the 15 ton winch was run up the set-up room suspended from the roof. The hydraulic high pressure line was installed to the bottom of the set-up room (in the headgate) for connection to the first shield when installed.

The shields were installed on day shift and swing shift. The third shift installed the face conveyor and stage loader.

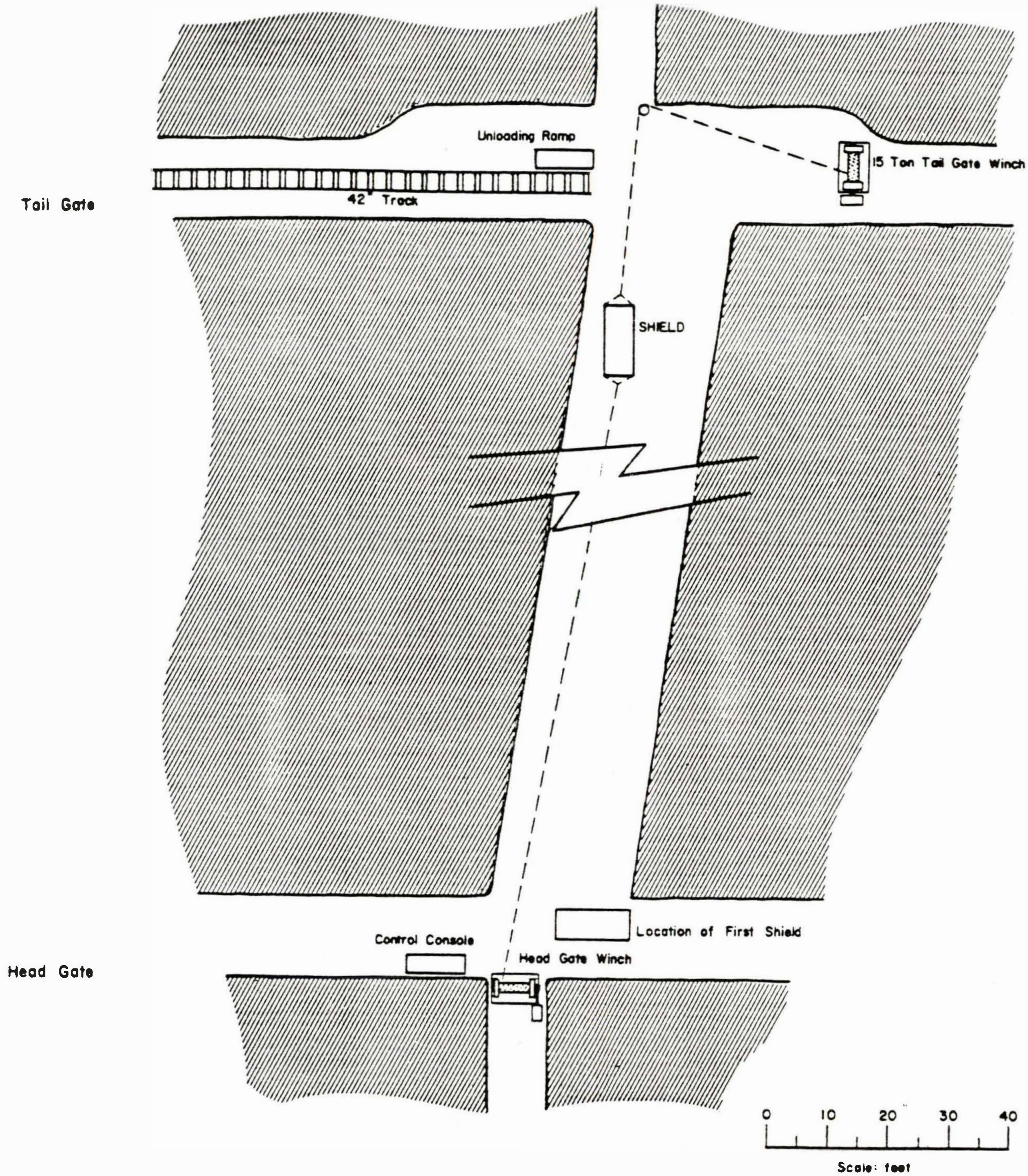
The shields were brought into the tailgate side of the set-up room and skidded off of the car with 6 ton air powered hoists. They were then skidded approximately 20 feet inby to the top of the set-up room using the tailgate winch. The headgate winch was connected to the back of the shield, and the tailgate winch was connected to the front of the shield. The shield was then pulled down the 30° pitch by the headgate winch, keeping the canopy of the shield facing up the pitch. The tailgate winch was used to hold tension on the shield to minimize the danger of a runaway. See Figure VI.A.-1.

The first shield was pulled down into the headgate entry. A stoper and jackleg were kept close by during positioning of shields for installation of roof bolts to hang sheave blocks. Six ton air hoists and sheave blocks were used to move the shields precisely into place. A paint line on the roof was used to aid in the positioning of the shields. The first shield would act by itself, not in a Troika set of three. The purpose of this shield was to aid in roof control of the headgate and it also contained two push rams for the advancing of the stage loader. This shield was placed slightly toward the up-pitch side of the headgate entry. The hydraulic lines were connected and the shield was pressurized tight to the roof.

While the next shield was being unloaded, additional roof bolts were installed in the area where the next two or three shields would be placed. This made the sheave placement much faster when positioning the shields.

The second shield was installed in the same manner as the first, using the headgate winch to pull and the tailgate winch to hold tension. The shield was placed in its proper position with the 6 ton air hoists. The hydraulic connections were then made from the first shield to the second shield. The second shield was pressurized tight to the roof.

The third and fourth shields were then moved into position on the face. Each was connected and pressurized as soon as it was positioned. At this time the first Troika bar was installed. This Troika bar was rigidly attached to the number three shield with the number two and four shields attached to the Troika bar by the connection of the D.A. rams. Thus, the number two, three and four shields formed the first Troika set installed



INITIAL SET-UP ROOM LAYOUT

FIGURE VI. A.-1

on the face. For Troika bar illustration see Figure III.A.-2.

The next three shields were lowered down the pitch, the Troika bar attached and so on. Normally, seven to nine shields per day were installed on this two shift per day operation. The delays usually occurred with positioning of the shields. This was a very difficult operation with the limited equipment available for use in a 30° pitching seam.

B. Face Conveyor and Stage Loader

On the third shift the face conveyor was installed. This installation was done only as far as the shields were installed. The headgate drive was first lowered down the face and positioned. The face conveyor installation was accomplished by lowering the pans in 15 foot long, sets of three. Each set of three pans had been previously bolted together on the surface. This helped the installation of the face conveyor proceed quickly. It was necessary to position the face conveyor by the use of 6 ton air hoists.

When the third shift finished the face conveyor installation up to the point that the shields were installed, they proceeded to the headgate and worked on the installation of the stage loader for the remainder of the shift.

The stage loader parts were lowered down the track slope on flat cars. From there they were pulled by the battery locomotives, in the entry below the headgate, into the longwall set-up area. Since the track was located in the entry below the headgate, it was necessary to pull the stage loader parts up the crosscut to the headgate entry. This was accomplished by the use of air hoists and hand winches. The stage loader was installed beginning at the bottom of the longwall face and working outby.

C. Panel and Slope Conveyor Belts

Since the method of longwall panel development did not include the use of conveyor belts, it was necessary to install the entire 42 inch belt haulage system at the same time as the longwall was being installed. This was accomplished as a two shift per day operation, beginning at the stage loader tailpiece and working outby. Due to angles in the headgate entry, three conveyor drives were necessary in the 5,800 foot long panel belts. The two inby panel drives included belt storage units capable of storing one 500 foot roll of belt before removal. All belt equipment was pulled up a diagonal crosscut into the headgate entry using cables and sheaves pulled with a diesel LHD. The belt structure was loaded onto a 5 foot by 12 foot metal skid and pulled into the work area. In addition to the panel belt 700 feet of slope belt and one conveyor drive had to be installed in the new belt/haulage slope before the longwall could begin production.

D. Shearer

The shearing machine was the last part of the longwall face to be installed. Due to the tremendous weight (38 tons) of the shearer as well as the large size, it was necessary to dismantle the shearer before transporting it to the face. Both of the cutting drums were removed, hydraulic hoses were labeled and removed, and the top half of the shearer was removed from the undercarriage.

The undercarriage was the first part installed on the face. It was transported on two lowboy railcars in the tailgate entry to the top of the setup room. Using both the headgate and tailgate winches along with several air hoists it was moved into position at the top of the face where the front and rear trapping shoes were connected. It was necessary to lift the top half of the shearer using the wenches and sheaves to position it over the undercarriage. These two parts were bolted together and the hydraulic hoses were connected in their proper locations. The cutting drums were then transported to the face and installed.

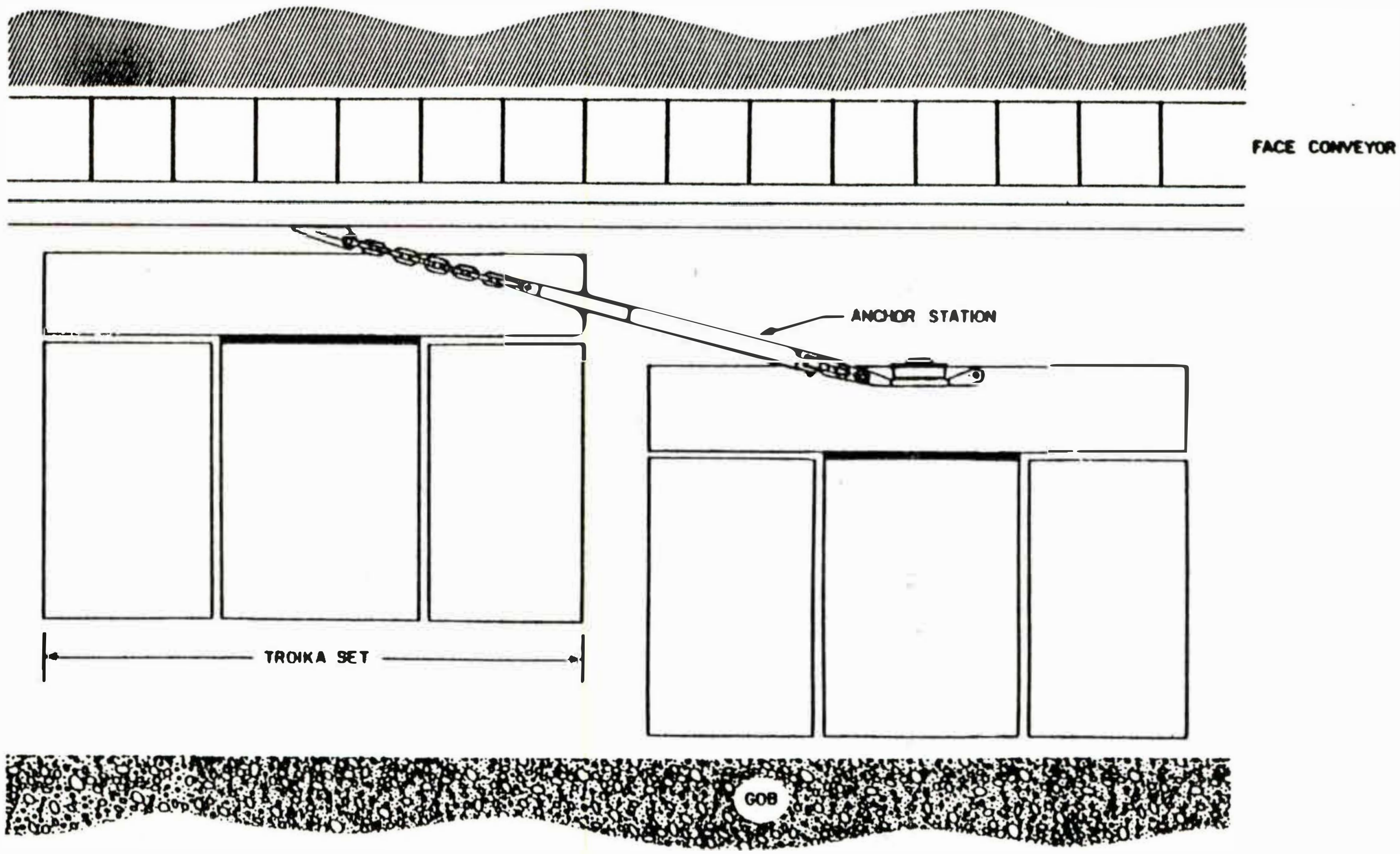
VII. OPERATIONS

A. Startup and Related Problems

As previously mentioned, the first two weeks of longwall operation were used for training with both production crews working on the day shift. The last week of August 1981, the crews were separated and mining was conducted on two shifts.

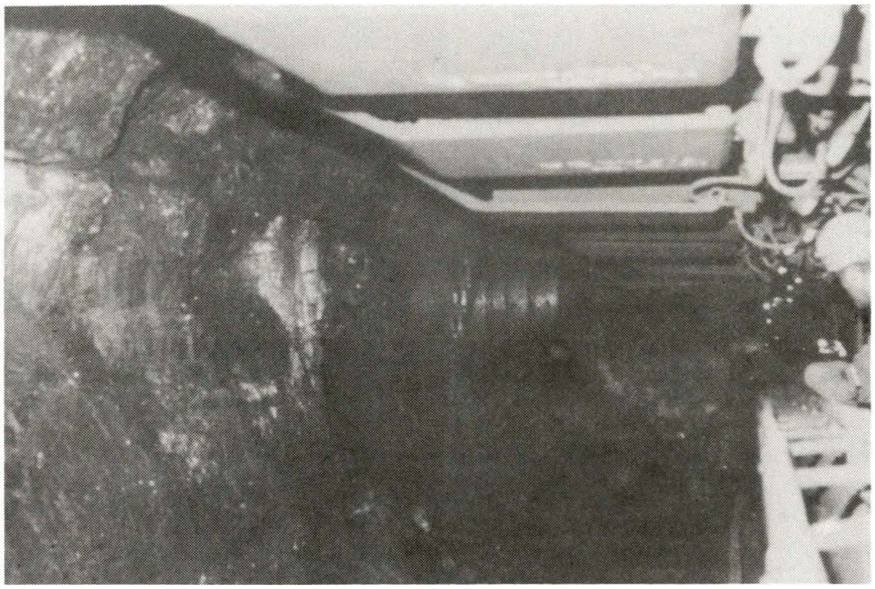
As with any new operations, there were problems to iron out. Conveyor creep was one problem that surfaced almost immediately. Although the face was planned with anchor stations on every other Troika set, only seven of the stations were initially installed. Theoretically, seven anchor stations were adequate. However, within the first two weeks of operation, it became obvious that the five additional anchor stations would have to be installed. Much of the conveyor creep problem was due to the inexperience of the operators. As the operators gained experience the conveyor creep problem was solved. See Figures VII.A.-1 and 2 for typical anchor station installation.

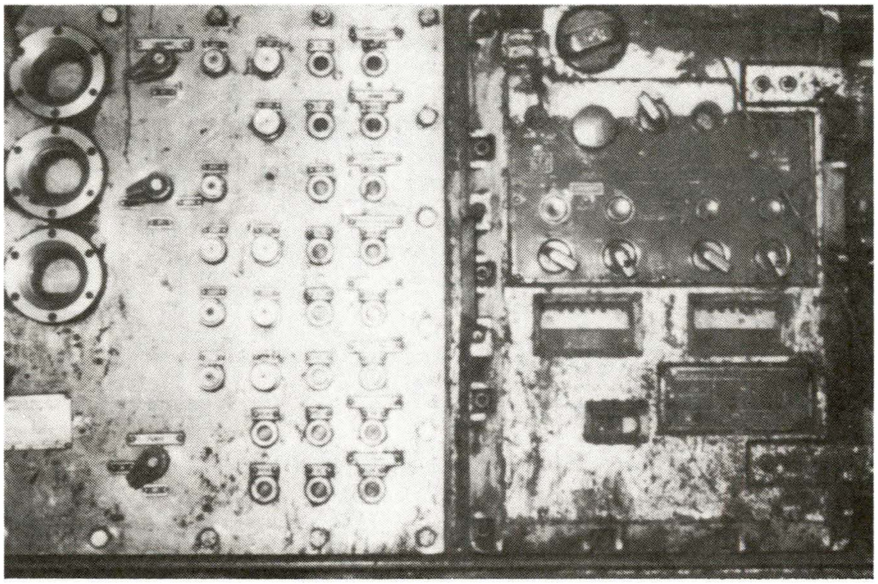
There were several minor problems with the shearer during the start-up phase. One of the problems was broken hydraulic hoses.

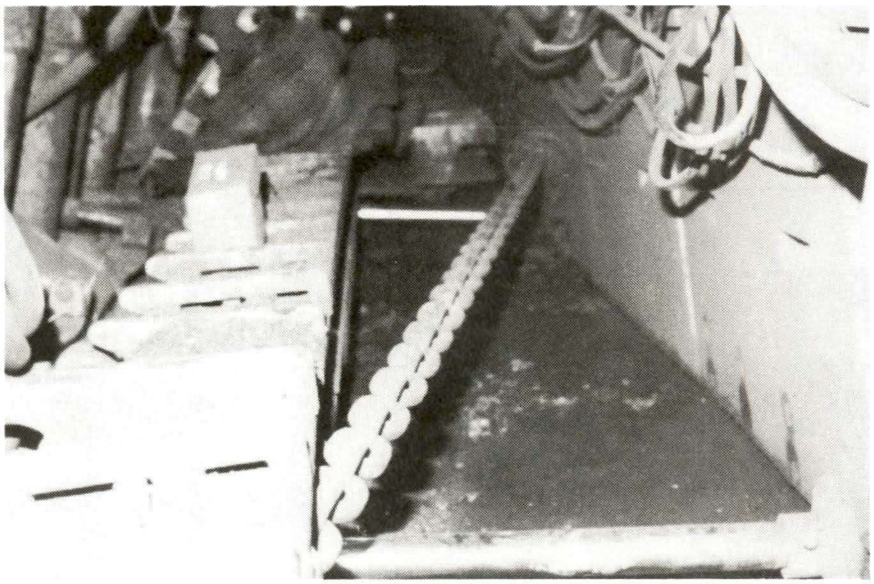


Anchor Station Installation

FIGURE VII.A.-1







Top left. Shearer in operation (looking east-update).

Center. SIVAD control console in the headgate entry.

Bottom. Anchor station attached to the face conveyor (looking-west-downdip). Note Troika bar with steps on the left and D.A. rams for advancing the face conveyor in the center.

Top right. Redesigned lump breaker mounted on the stage loader.

Figure VII.A.-2

This problem was traced to the hoses rubbing on various parts of the shearer during operation. It was necessary to reroute many of the hydraulic hoses to correct this problem.

The face side of the shearer rode on the face side of the face conveyor and allowed minor horizontal movement. The gob side of the shearer was attached to the gob side of the face conveyor with two sets of trapping plates. These plates partially encircled a bar on the conveyor preventing vertical and horizontal movement. The trapping plates supplied with the shearer were not heavy enough and were bent during operation. This made alignment of the shearer difficult. Eventually the trapping plates were replaced with a heavier duty plate to correct this problem.

It was necessary to rebuild the brakes on the traction unit. This was one problem that proved to be a part of ongoing maintenance on the shearer. Due to the pitch of the coal seam, the disk pads would wear and frequent replacement was necessary. On a flat seam the brakes would rarely need replacement.

Problems were also experienced with the radio control on the shearer. The functions of the radio control would not operate in the way they were designed. This problem was worked on for three months by the representatives of the shearer manufacturer before it was fixed. The radio control with one operator could be used by the face crews after it became operational. However, one operator could not properly see both the uphill and downhill cutting drums on the shearer. This caused the floor to be cut unevenly which caused problems with advancement of the face conveyor. It was decided to abandon the use of the radio control and proceed with two shearer operators manually controlling the operation of the shearer.

When the longwall operation began the face conveyor lump breaker did not operate properly, therefore it could not be used. Although the shearing machine was equipped with a lump breaker, it was located on the up pitch side of the machine. When cutting coal up the pitch the lump breaker on the shearer was able to handle most of the lumps. However, on the downhill cuts, most of the lumps went to the stage loader. An effort was made to break the biggest lumps with a sledge hammer, but many large chunks of coal and rock ended up on the beltline. These large lumps had a tendency to plug the chutes at the transfer points on the main beltline causing unnecessary delays in production. The face conveyor lump breaker was modified and put into operation on the stage loader within two months of the initial longwall start up, thereby minimizing the lump problem. A photo of the lump breaker mounted on the stage loader is shown in Figure VIII.A.-2

During the first three months of operation longwall availability averaged 48%. Availability was calculated on 6½ hours available face time out of the 8 hour shift. This includes both surface and underground delays that interrupted the production of the longwall. With the ability of the longwall system to mine coal quickly and continuously, much downtime occurred due to surface problems. Such large tonnages had never been mined in such a short period of time at Snowmass; therefore, the longwall start up was as much of a learning process for the surface employees as it was for the longwall crews. By the end of the second month, longwall crews were averaging 850 raw tons per shift. Within three months it was not uncommon for the two longwall production shifts to mine 2,500 total raw tons per day. The capacity of the preparation plant and the raw coal storage silo often caused more delays than did equipment downtime.

Although the capacity of the surface facilities was a definite factor in limiting longwall production, the upgrading of these facilities would have taken major capital expenditures. Without a long term coal contract in place, justification of these expenditures was not possible. Therefore, only the preliminary plans were made to upgrade the capacity of the surface facilities. Modifications to the preparation plant were made to improve production quality and product yield.

Another problem that was identified in the early stages of longwall production was the amount of coal that could actually be cut per shear. Although the design web of the cutting drums was 30 inches, it was not possible to take a full 30 inch cut on each shear. The reason was that the anchor station hydraulic rams and chains prohibited the shields from being advanced a full 30 inches on each cut. It was determined that the easiest and safest method was to take an average web of only 24 - 26 inches. This enabled the face conveyor to be pushed and the shields advanced to their proper position without causing unnecessary delays for "special positioning" of the shields. Although the smaller cut had an effect on production, it was less than the time required to position the shields of a full 30 inch cut.

B. Crew Set-Up and Mining Method

The day shift crew consisted of one foreman, two shearer operators, one face conveyor man, two shield movers, one control console operator, and two maintenance men. One of the maintenance men was specially trained in the hydraulic operation of the face and one was specially trained in the electrical operation. Both maintenance men were trained to perform other face jobs as needed.

The second shift crew was the same as the first shift except that only one maintenance man was used. General face equipment maintenance and supply haulage was conducted on the third shift by one maintenance foreman and four service people. Their work on third shift consisted of everything from routine maintenance to building cribs in the gate entries. Although the main function of the third shift was to assure that the face was ready to resume production the next day, they were also trained to cut coal if necessary.

The shearing machine was normally left at the uphill, tailgate side of the longwall face at the end of second shift. This enabled third shift crews to perform shearer maintenance at this location. Three sections of the armored face conveyor at this location were hinged and could be laid flat to allow access to the underside of the shearer. The top Troika set had to be left a full cut back to allow room for the face conveyor sections to be dropped. The canopy extensions were fully extended at this time to provide maximum protection to the service people. Maintenance supplies for the shearer were stored outby the face in the intake section of the tailgate entry. Due to the steep pitch, movement of supplies along the face was difficult, therefore, the planning of when and where to perform maintenance work was very important. Parts and supplies were brought into both the headgate and tailgate ends of the face depending on where work was to be performed.

The experience of most operators has been that three shifts of production without scheduled downtime for maintenance causes to rapid deterioration of equipment. Consequently, Snowmass performed routine longwall maintenance on schedule without exception.

The longwall face was operated on the full 30° pitch at a 90° angle to the gate entries. Bi-directional cutting was performed to avoid walking the entire length of the pitching face without coal production. Bi-directional cutting was accomplished by sumping-in over a distance of 50 feet from either gate entry. Once the shearer was sumped-in 24 inches, the face conveyor was advanced behind the shearer and a clean up pass was made back to the gate entry before the full face length was cut. By advancing the face conveyor immediately behind the shearer cutting up hill, the potential for a possible shearer runaway was minimized.

The design maximum tramming speed of the shearing machine was 30 feet per minute which was a design mining rate of 1,150 tons per hour. The actual uphill tramming speed averaged 12 feet per minute (500 tons per hour). The downhill mining speed was frequently restricted by lumps encountered ahead of the shearer. Average mining rate down the pitch was 22 feet per minute (850 tons per hour).

The shields were generally kept tight to the face. The gate end Troikas were operated one web back to allow room for the face conveyor drive units at each end. The Troika sets were advanced in pairs with the Troika containing the conveyor anchor station advanced first. These anti-creep devices had to be relaxed to advance the Troika bar and prevent damage to the hydraulic rams and the chains that are attached to the face conveyor. Normally, the face conveyor man and one shield mover worked together to advance and align the conveyor and the Troika set with the anti-creep devices attached. The second shield mover follows behind advancing the second Troika set. Figure VII.B.-1 shows a typical Troika advance sequence.

The longwall crews walked along the face on top of the Troika bar which had ladder rungs welded to it. This enabled crews to move up and down the pitching face without slipping and falling as would occur when trying to walk the pitch on a rock floor.

The face conveyor could be walked up the pitch by pressurizing all of the anti-creep devices, or it could be walked down the pitch by relaxing up to half of the devices. The force of the shearer on the face conveyor caused the conveyor to move, thereby requiring occasional adjustment up hill or down hill.

The initial set-up of the face was angled slightly up the pitch to assist in offsetting the tendency for downhill creep of the face conveyor and roof support. In other words, the headgate was ahead of the tailgate. It was discovered very early during start-up that this problem could be easily controlled by the steerability of the roof supports and the anti-creep devices. Consequently, the face was squared up to a position ninety degrees to the gate entries and that practice was continued.

Recommended spacing between Troika sets was twelve inches. Initially mining crews had difficulty maintaining the proper spacing of each Troika set. Proper spacing assisted in advancing and steering supports when operating the face. The problem was solved by attaching twelve inch long wire rope spacers to the uphill end of each Troika bar. This enabled miners to tell at a glance whether proper spacing was being maintained.

Another modification made during the initial start-up phase concerned the conveyor push rams. The Troika push rams were originally fitted with a domed or rounded push head which was the point of contact with the face conveyor as the conveyor was advanced. This domed design had a tendency to slip sideways along the conveyor as the conveyor was pushed. This caused misalignment problems and in some cases caused the rams to be bent. The problem was solved by fitting four miner bits into the domed head. The points of these miner bits eliminated the tendency for the rounded push head to slip against the conveyor. When necessary the bits could be replaced by new bits in much the same manner as bits are changed on a cutting head.

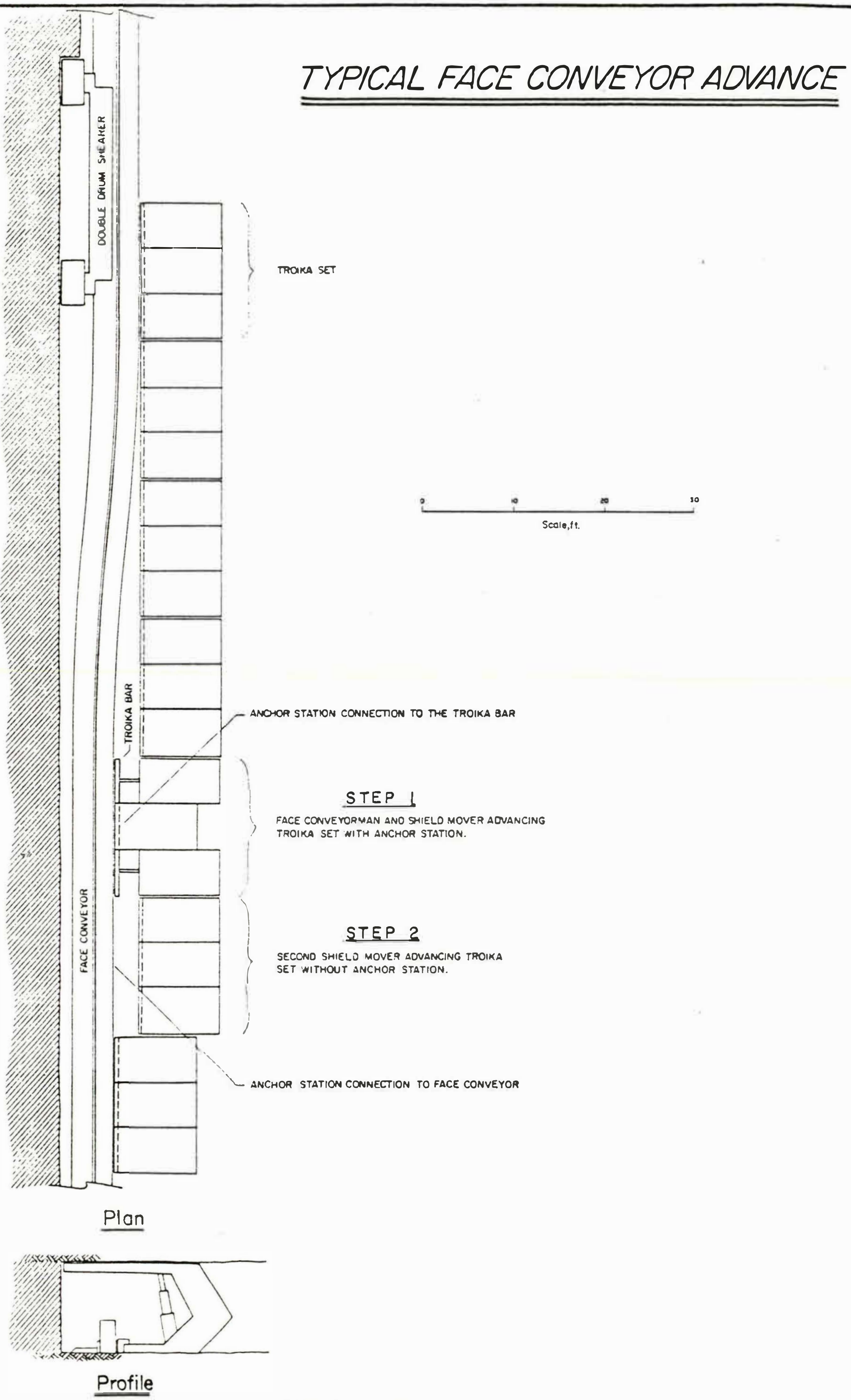


FIGURE VII.B.-1

VIII. VENTILATION

A. Mine Fans

Thompson Creek #1 Mine was ventilated by two mine fans located at opposite ends of the mine workings. The #1 fan was a six foot diameter, 100 h.p. exhausting fan and was located near the main "A" seam portals at the north end of the mine workings. The #1 fan was located at the portal of an old "B" seam entry of Thompson Creek #1 Mine. The "B" seam lies an average of 35 feet above the "A" seam workings of #1 Mine. This "B" seam entry was rehabilitated in 1975 when the #1 Mine was reopened and was used for 1,400 feet of the main return aircourse. Return air traveled from the "A" seam workings to the "B" seam return by way of two rock tunnels. See Figure VIII.A.-1 for a cross section at the second rock tunnel. The #1 fan ventilated the north end of the mine and the main development slopes down-dip to the west.

The #2 Mine fan was a six foot diameter, 200 h.p. exhausting fan located on the "A" seam outcrop at the southeastern end of the mine workings. This portion of the mine was part of the old workings in the "A" seam and was also rehabilitated when the mine was reopened. The #2 fan ventilated the south end of the mine which included the longwall face and the longwall bleeder entry.

A new fan entry was driven to the surface 550 feet southeast of the #1 fan in 1983. Plans included installation of a fan at this new location. The #1 and #2 fan installations would then be abandoned. As additional panels were mined down the dip, a small fan could be installed at the extreme south end of the mine workings in order to effectively ventilate the longwall bleeder entries.

B. Two Entry Ventilation

Both the headgate and tailgate development sections were mined with a two entry development system. The lower entry in each section served as the intake and the upper entry was used as a return aircourse. Fifty horsepower auxiliary fans with 24 inch fiberglass tubing were used for face ventilation.

Since the current mine law prohibited the use of an intake escapeway for conveyor belts or trolley locomotives, coal haulage was accomplished by the use of battery locomotives and 6 ton coal cars operated in the intake aircourse. The intake and return entries are separated by cinder block stoppings with

CROSS-SECTION THROUGH SECOND ROCK TUNNEL

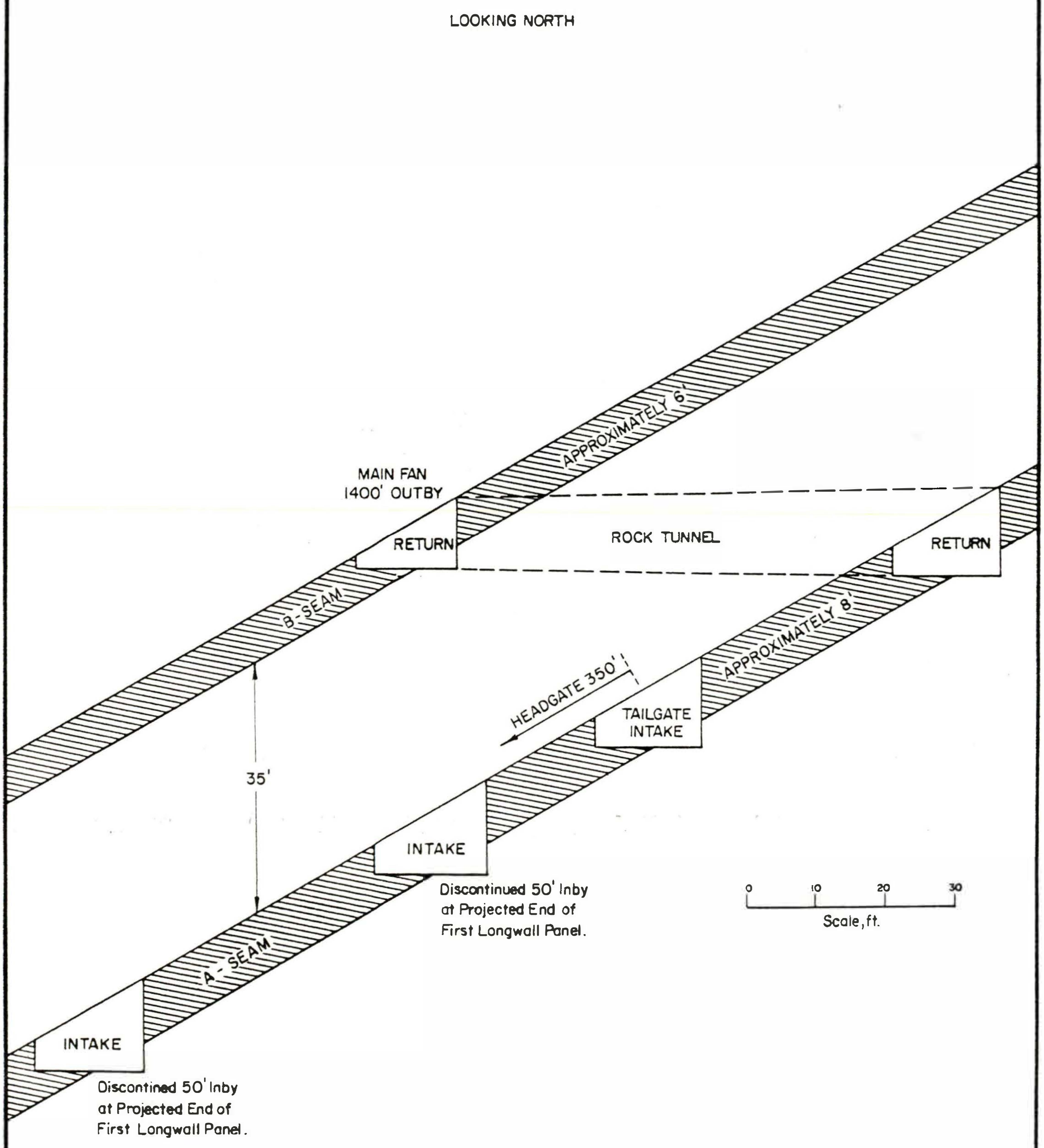


FIGURE VIII.A.-1

man-doors located in every other crosscut. There was no haulage system in the return airway of the two entry system. All coal mined from the face of the return entry was transferred to the intake entry by coal chutes in the crosscuts.

As technology improved so did the possibilities for improved and safer ventilation of two entry systems. In June 1982, Snowmass purchased a computerized mine monitoring system. This system provided continuous low level monitoring of carbon monoxide and methane gas. This mine monitoring system would aid in the safe mining of two entry panel development with a conveyor belt located in the return aircourse. Mining laws required extensive monitoring for this type of mine plan.

C. Longwall Ventilation

At the completion of the two entry headgate and tailgate development sections, a 24 foot wide longwall set-up room was driven. This room was driven from the bottom entry (headgate), through the longwall panel to the tailgate entry. This set-up "raise" was mined by the drill and shoot method. At the completion of the set-up raise, mining crews moved 150 feet inby to the farthest point of panel development and began drilling and shooting a bleeder entry or raise. This bleeder raise was mined from the last open crosscut in the headgate, through the panel to the main return aircourse above the tailgate entry. It was decided that it would be wise to continue mining the bleeder up the pitch to an old abandoned mine portal at the outcrop. As mentioned previously a small fan could be installed on this opening to ventilate future panels. The bleeder raise and the raise to the outcrop were driven sixteen feet wide.

The entry below the headgate was the main intake during development with the headgate entry serving as the return. When all development was completed and the installation of the headgate beltline and the longwall face began, the headgate entry was also changed to an intake aircourse. Although the entry below the headgate remained the primary intake, 8,000 to 12,000 cfm normally flowed in the panel belt entry. Low level carbon monoxide sensors were installed at each panel belt drive and at 2,000 foot intervals.

The main intake for the longwall face (entry below the headgate) usually provided 50,000 to 55,000 cfm. The stopping was removed and the air was split at the last crosscut outby the longwall face. The face split was routed up the open crosscut through the headgate along the stage loader and up the longwall face. The stoppings below the headgate were rebuilt after the longwall face has passed and the next stopping outby was opened up.

Normally 45,000 to 55,000 cfm was maintained across the operating longwall face and 12,000 to 15,000 cfm continued to the bleeder entry. This volume of air was more than adequate to control methane and dust along the face. The stoppings in the tailgate section were removed as the face approached allowing the air easy access to the main return aircourse. The stoppings in the tailgate section were not rebuilt after the face had passed.

A temporary curtain was installed in the tailgate outby the crosscut being used as a return. This maintained the tailgate on the outby side of the curtain as an intake. The tailgate entry had to be maintained as an intake in order to use the battery locomotives for supply haulage. See Figure VIII.B.-1. for the typical face ventilation.

This method of ventilation left two separate intake escapeways and one return escapeway. Since the stoppings were left intact in the headgate, in the event of a belt fire miners could drop down to the entry below the beltline and escape in fresh air. Miners in the tailgate area also had easy access to an intake escapeway.

Emergency self contained, self rescuer units were stored in the tailgate just outby the temporary curtain along with emergency barricade material. S.C.S.R. units were also stored near the stage loader in the entry below the headgate along with emergency barricade material. Miners working on the face carried filter type self rescuers at all times. Any miner who desired to, could carried his own S.C.S.R. while working on the longwall face.

D. Dust Control

Specially designed longwall helmets were purchased and made available to longwall crews. These helmets had a face shield which came down even with the miners chin. In the back of the helmet was a small pump that pulled air through a filter and directed it down the inside of the face shield. These helmets had to be serviced prior to each shift. Although these special helmets served their purpose, most miners preferred not to use them due to their bulky size weight.

The primary dust control measure on the longwall face consisted of water sprays. Two mist type spray bars operating at 60 psi were installed at the stage loader. One of the spray bars was directed at the transfer point between the face conveyor and the stage loader. This spray bar operated continuously during the mining process. The other spray bar was directed at the stage loader lump crusher and was only used on an as needed basis.

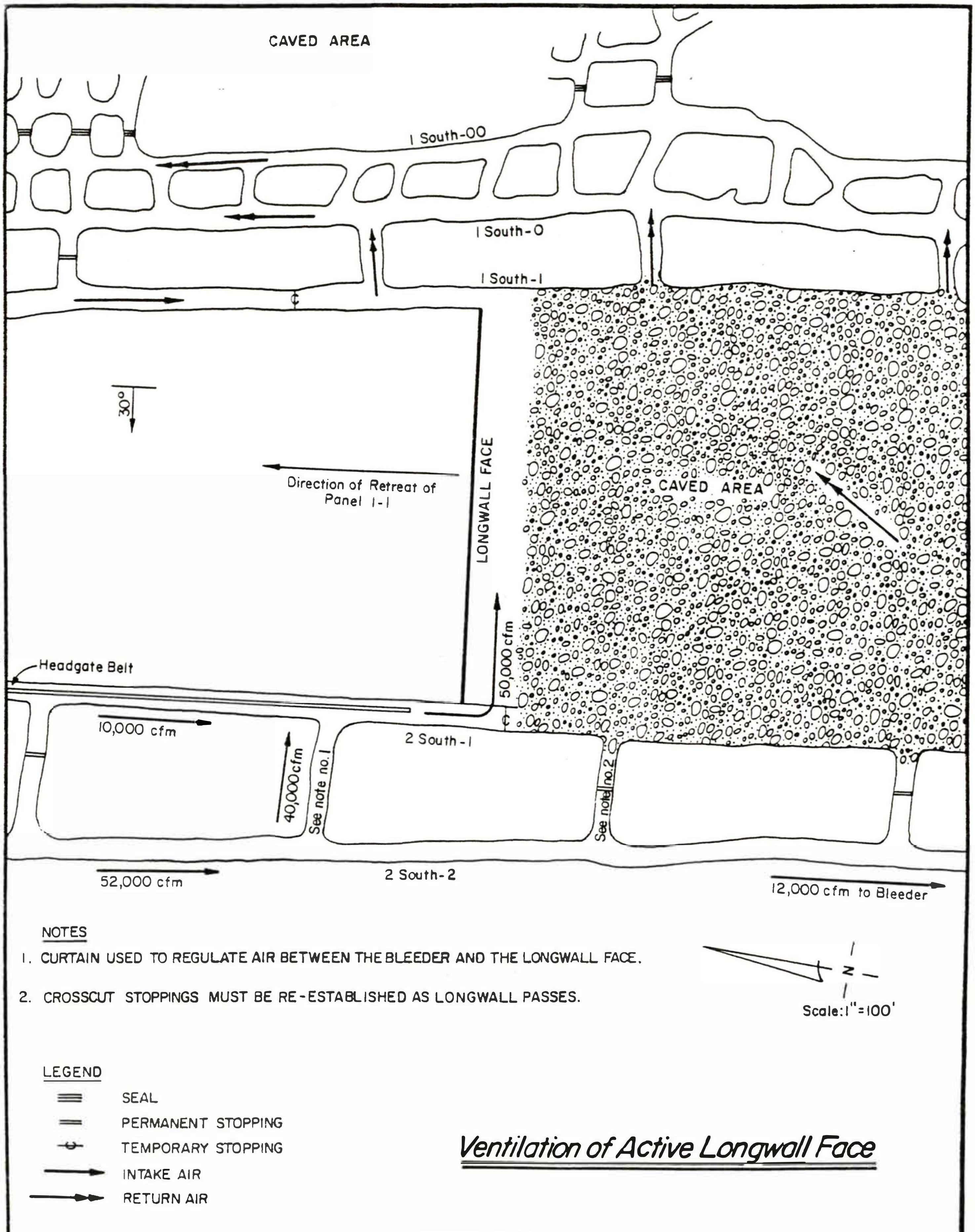


FIGURE VIII.B.-1

Dust control on the shearer consisted of 104 water sprays located on the cutting drums. There were 52 on each drum. These sprays operated at a minimum pressure of 125 psi. The normal operating pressure usually averaged 175 psi. The water that was used in the dust suppression water sprays was first routed through the shearing machine to cool the hydraulics and the motors.

Initially during the coal cutting process, dust would roll back over the top of the shearing machine due to the spinning action of the cutting drums. Snowmass personnel fabricated an additional spray bar for the shearer to direct this dust away from the operators. A four foot long, one half inch diameter spray bar was added to the headgate end of the shearer. This spray bar came straight out from the end of the machine facing down the pitch of the face. Three spray nozzles were attached to the spray bar. These nozzles were directed up the pitch of the face and at a 45° angle to the face. Not only did the water assist in dust suppression, but the action of the high pressure water sprays pulled the intake air straight across the cutting drums keeping dust from flowing back over the machine. When cutting near the headgate this spray bar was turned off to avoid spraying the workers in the stage loader area. Water sprays on the cutting drums were used at all times while cutting coal.

Adequate ventilation and water sprays proved to be an effective dust control method. The Snowmass face was continually in compliance with the 2.0 mg/m^3 respirable dust standard. In addition, the friable nature of the "A" seam coal gave Snowmass and a definite advantage in the control of respirable dust on the face.

E. CONTROL OF SPONTANEOUS COMBUSTION

Spontaneous combustion is typically a concern of an underground coal mine and this mine was no exception. The primary method used to prevent spontaneous combustion was an effective coal clean-up program. The design of the face conveyor and the fact that it was kept tight against the face reduced coal spillage. Care was taken by the shearer operators to cut a smooth clean floor. Clean-up along outby belts was another factor in the effective clean-up program.

To monitor for fires whether from spontaneous, electrical or mechanical sources, a computerized mine monitoring system was installed. As noted previously, the system was installed to monitor a two entry mining system with the belt in the return. The system was initially set up to monitor carbon monoxide, methane and flow from mine dewatering pumps but the system was capable of monitoring many other phases of operation such as production, ventilation, electrical and mechanical functions.

IX. ROOF CONTROL

A. Slope and Panel Development

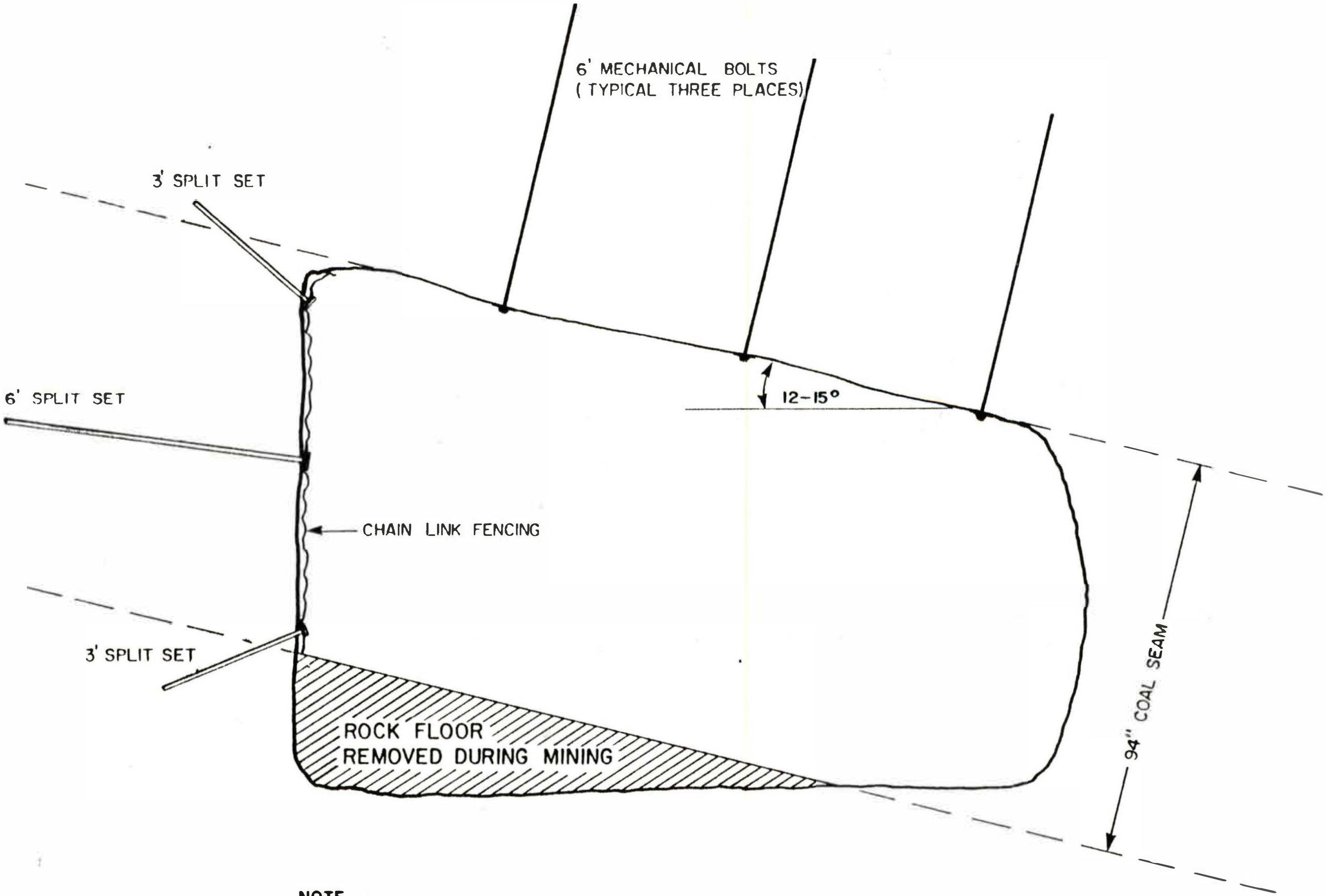
Roof control practices for the development of the panels and the slopes were carried out in accordance with a Mine Safety and Health Administration (MSHA) approved roof control plan. Many provisions of this plan were standard MSHA requirements, but other provisions applied specifically to a steep pitch operation. The approved bolting pattern was 5 foot by 5 foot with a minimum length of 4 foot. Resin, resin point anchor, conventional and truss bolts were approved. Conventional bolts were generally used in areas of good top. Figure IX.A.-1 shows a typical cross section of a slope entry and the typical bolting pattern. The bolts were installed perpendicular to the roof. Roof mats were used in conjunction with the bolts in areas where the immediate roof was slabbing off.

The installation of bolts was done using pneumatic stopers and jack legs. The entries generally ranged in height from 2.5 feet on the low side to 10.5 feet on the high side. The cross cuts were mined perpendicular to the entry and up dip. The Alpine miners had bolter heads but were not capable of bolting perpendicular to the dipping roof. Figure IX.A.-2 shows photos taken in panel entries and a cross-cut.

During the normal development of entries and slopes the high rib was susceptible to sloughage. This rib in the panel entries was timbered and lagged. When the slope was started below the first panel, attempts were made to use chainlink fence bolted to the rib to control sloughage. Conventional bolts would not hold in the coal seam, therefore split set bolts were tried and found suitable. Figure IX.A.-1 shows a typical bolting pattern on the rib. The split set bolts were installed on five foot centers down the entry. Figure IX.A.-3 shows photos of timber and lagging rib control and chainlink fence rib control.

Secondary support was used in areas of poor roof conditions. Forms of secondary support included wood and steel cross bars, yieldable arches, rigid arches, timbers, wooden cribs and steel fiber reinforced concrete cribs. In extremely poor roof a polyurethane binder was injected into the roof ahead of mining. This binder would fill cracks and fractures then expand and setup creating a tight bond in the roof material. This binder was used extensively through the major fault zone discussed in the "Description of Geologic and Premining Conditions" section

FIGURE IX.A.-1

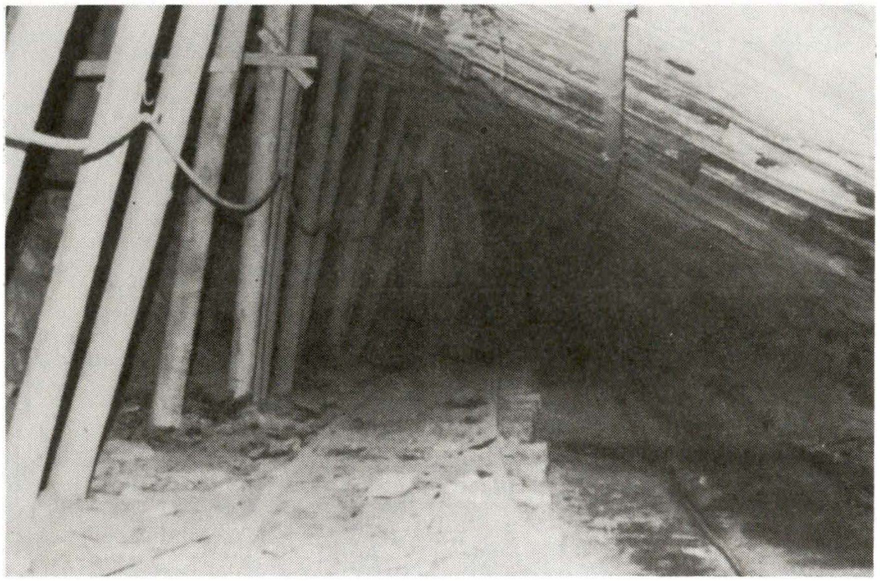


NOTE

THIS IS THE TYPICAL BOLTING PATTERN INSTALLED EVERY FIVE (5) LINEAR FEET OF ADVANCE IN EACH SLOPE ENTRY.

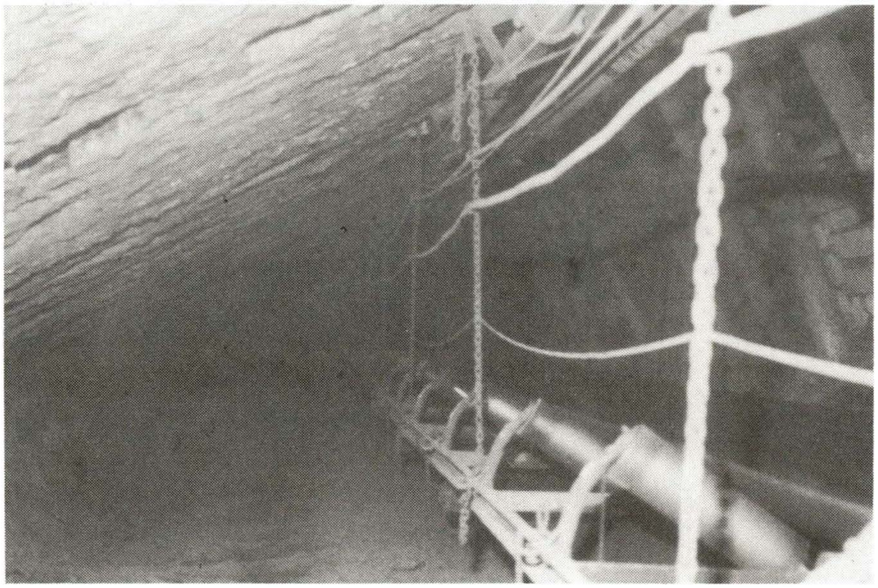
TYPICAL DESIGN OF ROOF AND RIB STABILIZATION OF SLOPE ENTRY

SCALE: 1" = 3'



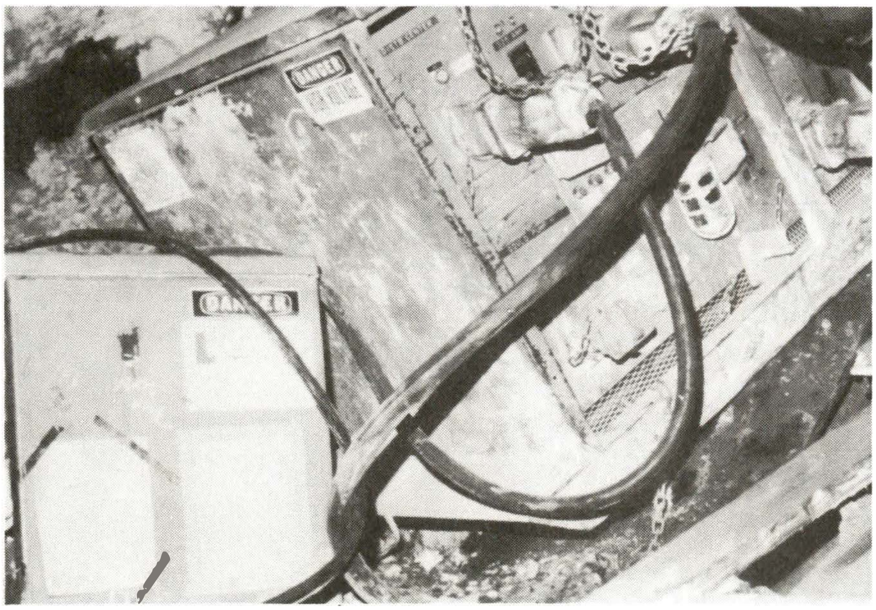
Track Entry

Track parting in the tailgate entry (looking south). Note the higher track level on the left to reduce the volume of rock that had to be cut.



Belt Entry

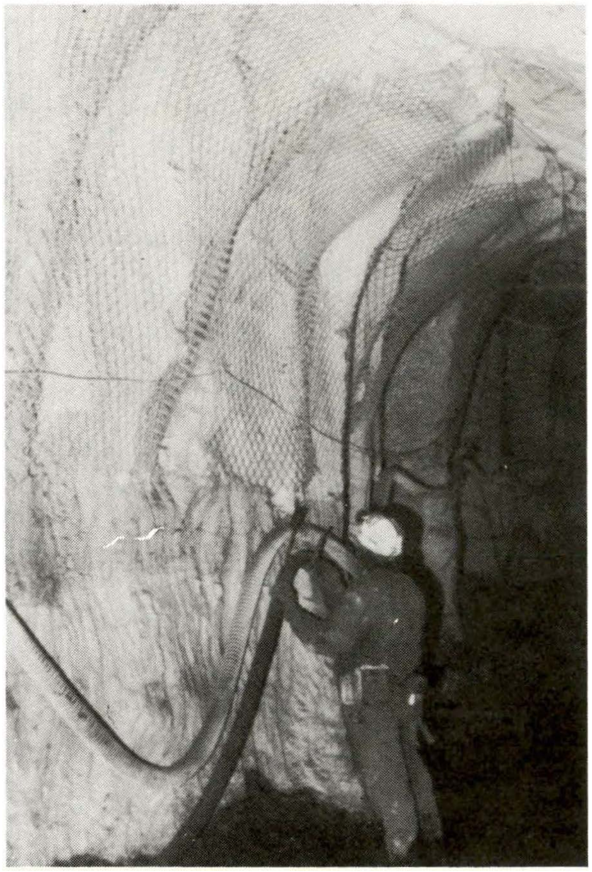
Panel belt in the headgate entry (looking north).

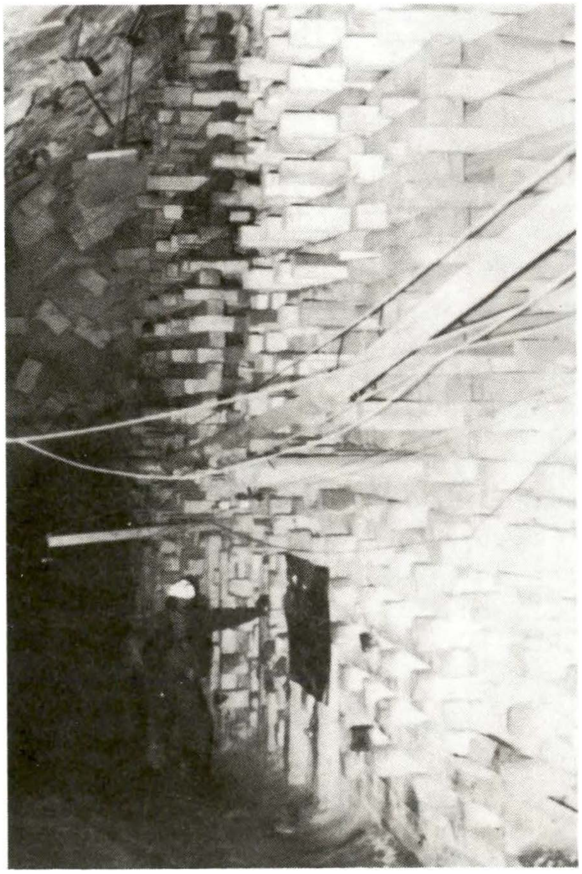


Cross-Cut

Power center installed in a cross-cut during the headgate development. Note, the boards beside the power center are part of the ladder up the cross-cut.

Figure IX.A.-2





Top left. Rib control in the tailgate entry. Timber and lagging installed when mining. Chainlink fence and roof mats installed as secondary support when rehabilitating the entry.

Top right. Rib control in the slope belt entry. Chainlink fence bolted to the rib with split-set bolts.

Bottom. Secondary roof control installed at the intersection of the hoist slope and the headgate entry during the rehabilitation of the mine.

Figure IX.A.-3

of this report. Figure IX.A.-3 shows an example of secondary roof control needed during rehabilitation of the mine.

B. Headgate and Tailgate Support

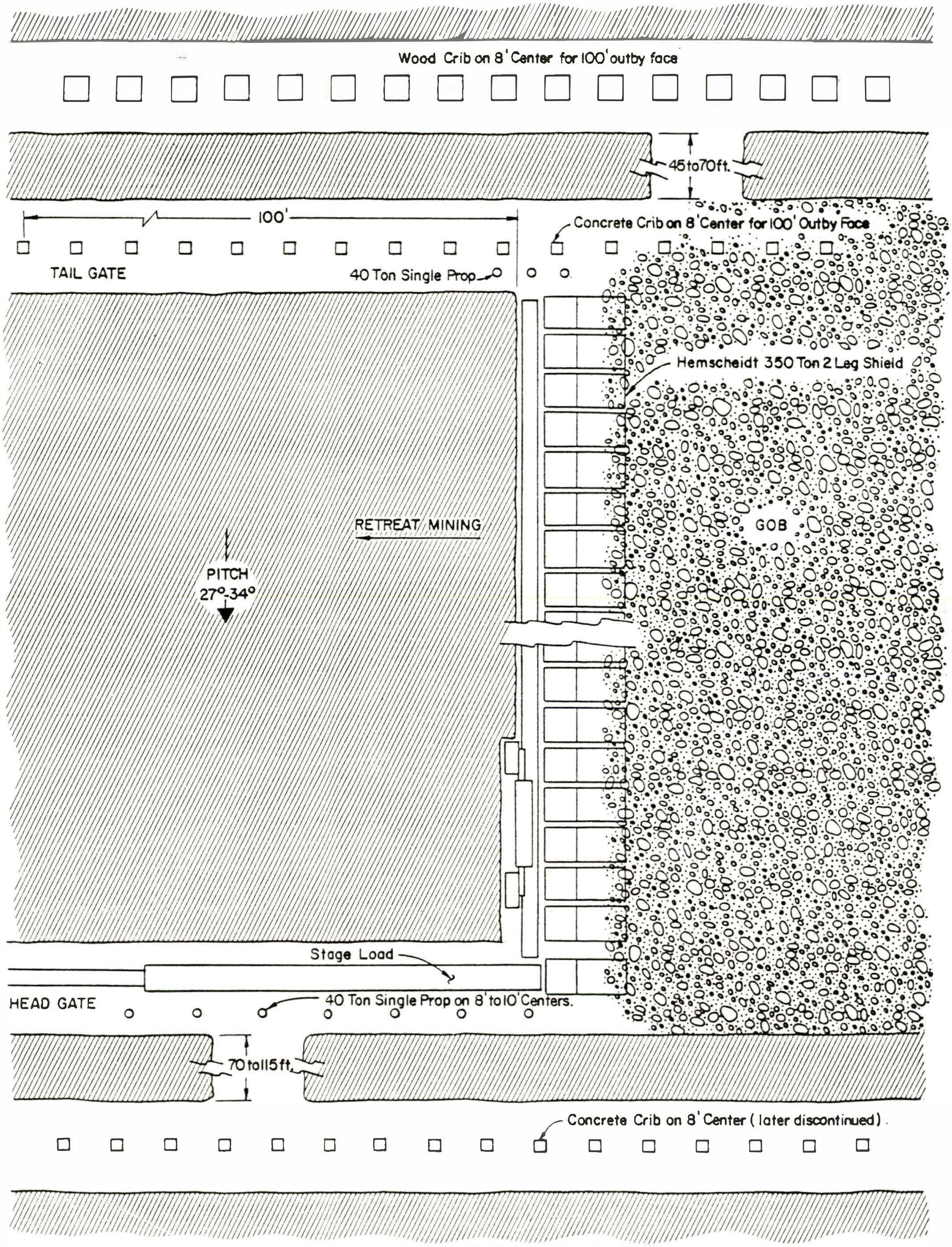
Additional secondary support was installed in the gate entries during the operation of the longwall. This secondary support primarily consisted of wood or fiber reinforced concrete cribs. Figure IX.B.-1 shows the typical secondary support arrangement for the gate entries. Figures IX.B.-2 and IX.B.-3 shows typical installation of the two types of cribs.

The entry above the tailgate was supported with wooden cribs on 8 foot centers. These cribs were constructed with 6 inch by 6 inch by 4 foot locally supplied pine or spruce. These cribs were installed 100 feet ahead of the longwall face. Maintenance of this entry was important because it was the main return for longwall. No major roof control problems were encountered in this entry but rib sloughage increased as the longwall passed.

The tailgate was supported with fiber reinforced concrete cribs on 8 foot centers and 100 feet ahead of the longwall. Hand operated single props were installed at the end of longwall to further protect the tailgate drive of the face conveyor. No shields were operated in the tailgate entry. Maintenance of this entry ahead of the longwall was necessary to protect the intake escapeway. No major roof control problems were encountered in this entry.

The headgate was supported with a single shield (not attached to a Troika set) and with single props. The shield was located above the center lines of the entry and in addition to roof control provided a means of advancing the stage loader. Single props were installed to 8-10 foot centers along the length of the stage loader. No major roof control problems were encountered in this entry. Rib sloughage was anticipated to be a problem in this entry. Steel plates were installed on the high rib side of the stage loader to protect the equipment and work area from sloughage.

The entry below the headgate was initially supported with fiber reinforced concrete installed on eight foot centers as the face retreated. Monitoring of these cribs revealed no major loading, therefore regular installation of these cribs was discontinued. Monitoring for loading was done with "Flat Jack" pressure cells, see the "Subsidence and Rock Mechanics" section of this report for further information.



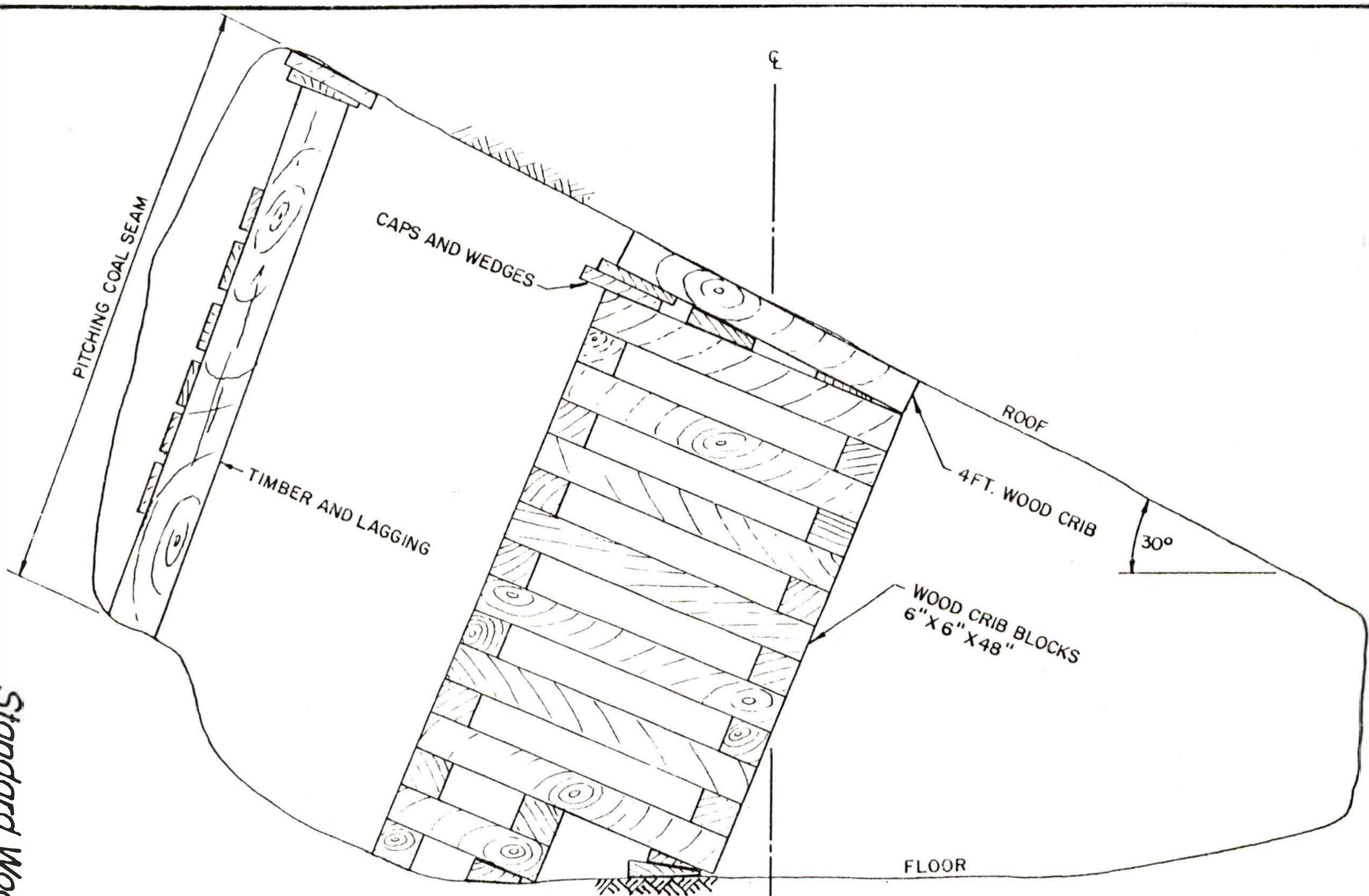
LONGWALL ROOF CONTROL PLAN

FIGURE IX.B.-1

Standard Wood Crib Installation in Longwall Entries

FIGURE IX.B-2

Scale: 2" = 1'

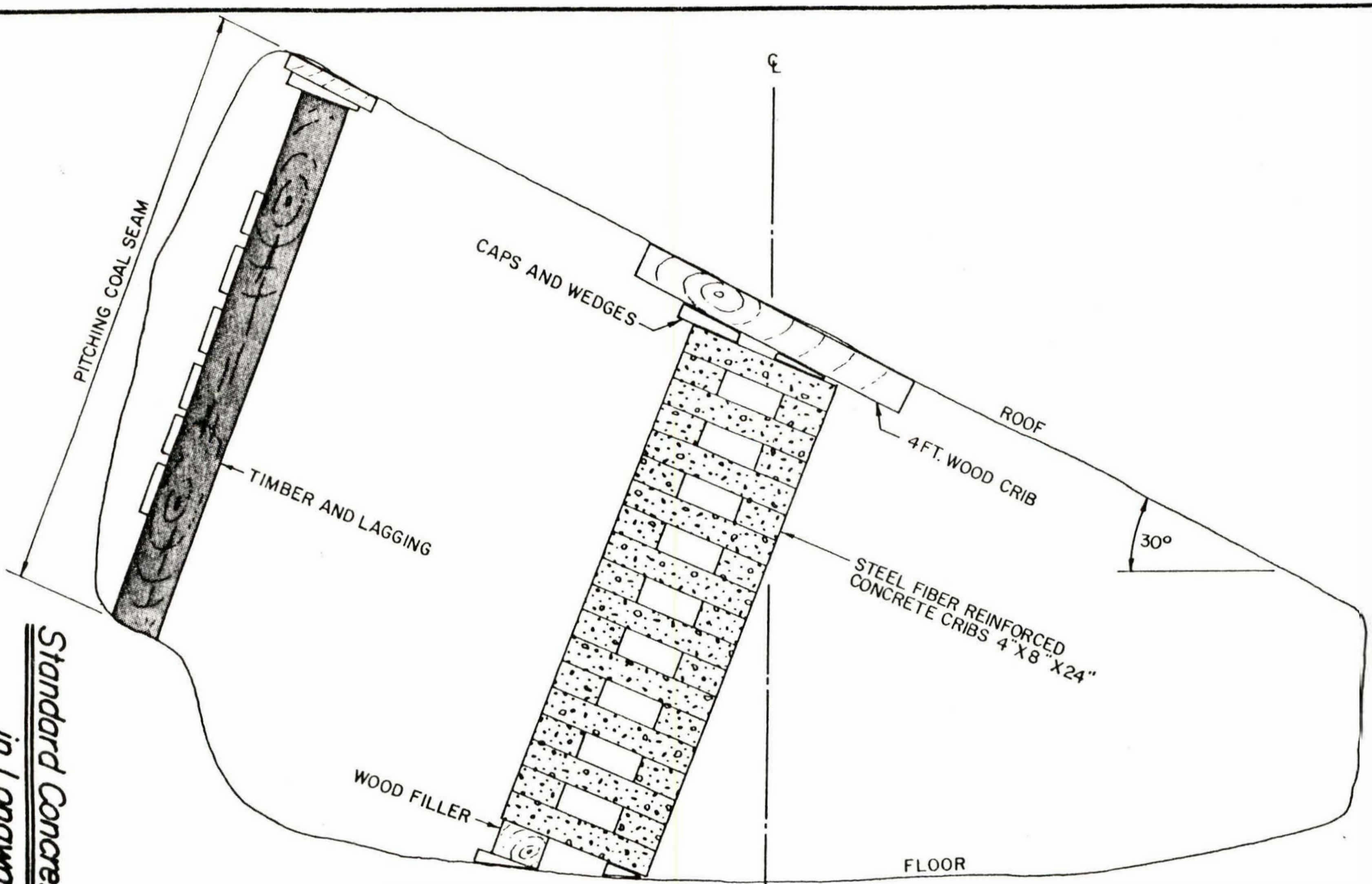


NOTE

1. ALL CRIBS ARE SET APPROXIMATLY PERPENDICULAR TO THE ROOF AND INSTALLED ON THE CENTERLINE OF THE ENTRY ON 8 FT. CENTERS.

METHOD OF INSTALLATION

A HAND OPERATED SINGLE PROP IS SET ON THE LOW RIB SITE. THE CRIB IS BUILT AGAINST THE PROP. THE PROP IS THEN REMOVED AFTER THE CRIB IS WEDGED TIGHT TO THE ROOF.



Standard Concrete Crib Installation in Longwall Entries

FIGURE IX.B-3

Scale: 2" = 1'

NOTES

1. ALL CRIBS ARE SET APPROXIMATLY PERPENDICULAR TO THE ROOF AND INSTALLED ON THE CENTERLINE OF THE ENTRY ON 8 FT. CENTERS.

METHOD OF INSTALLATION

A HAND OPERATED SINGLE PROP IS SET ON THE LOW RIB SIDE. THE CRIB IS BUILT AGAINST THE PROP. THE PROP IS THEN REMOVED AFTER THE CRIB IS WEDGED TIGHT TO THE ROOF.

C. Longwall Face

The roof support along the longwall face consisted of Hemsheidt Troika* system (type G320-14.5133.5) shields, see Figure III.A.-1. Each of the three shields in the Troika configuration had two legs and were capable of supporting 350 tons. Each shield had an optional hydraulic canopy extension that could be extended to the face after the shearer passed and before the shield could be advanced to the face. Attached to these shield extensions were hydraulic anti-spalling plates to assist in rib control. These anti-spalling plates could be folded down against the face to prevent slabs of coal from falling across the face conveyor and into the legs of the shields. The canopy extension extended 30 inches and the anti-spalling plates extend down 42 inches. The canopy extensions were not used much because the shields were kept tight against the face and because of good roof conditions. The anti-spalling plates were not used much due to better than expected face conditions.

Secondary support on the longwall face consisted of cribbing over the shields and injecting polyurethane binder into the roof where necessary. Where irregularities or poor roof conditions prevent good contact between the shield and the roof, crib blocks were added. The only cave on the face was the result of a 20 foot wide fault zone. Subsequent to the cave, the roof in the fault zone was injected with a polyurethane binder ahead of the face. This binder effectively held the roof as the longwall mined through the fault zone.

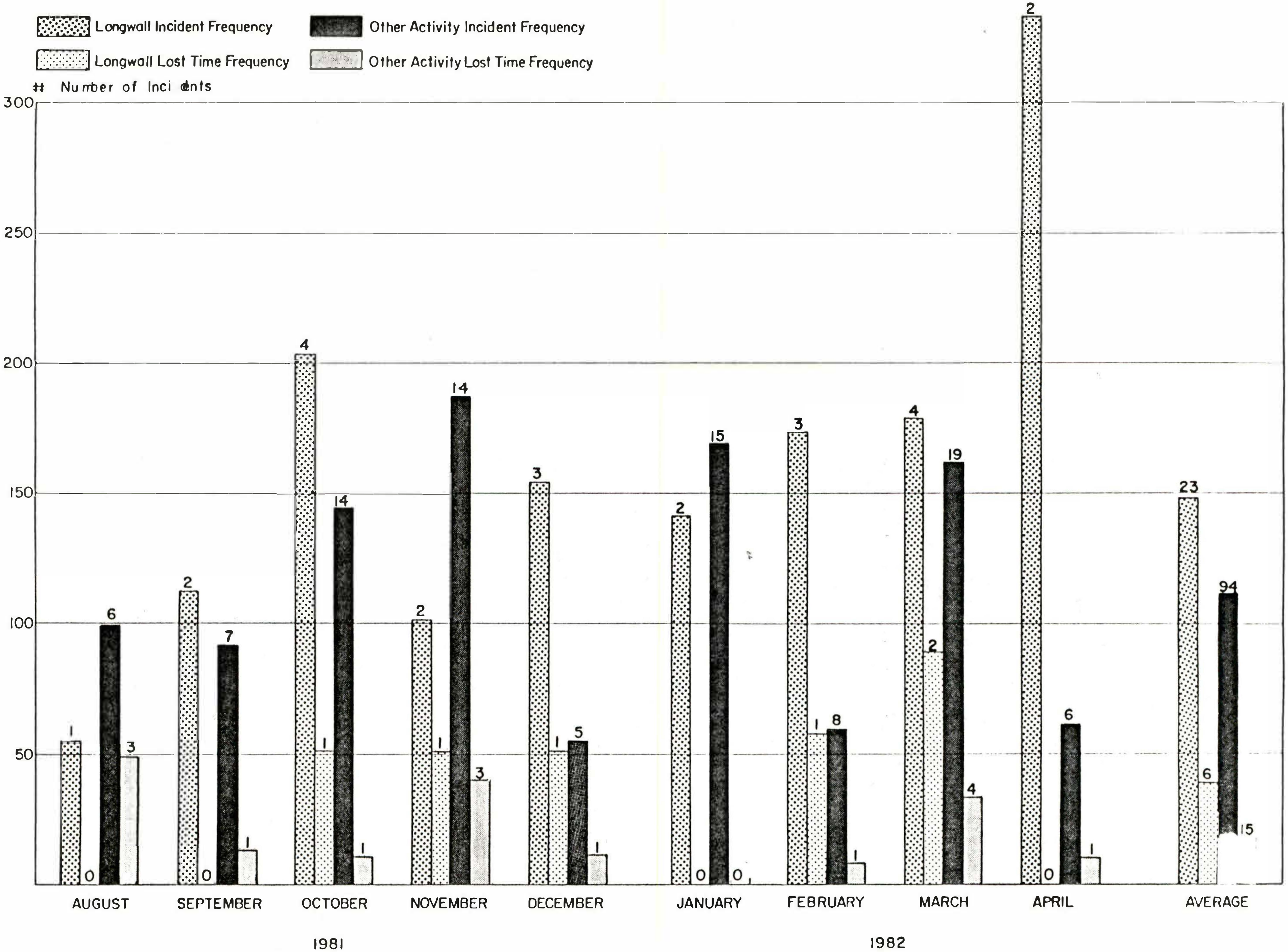
X. SAFETY RECORD

A. Accident Frequency as Related to Other Phases of the Mining Operations

The accident frequency for this mine for both development and longwall mining was high. The major contributing factor to the elevated accident rate was the pitch of the coal seam. The low rib contributed to strains due to low clearance. The high rib contributed to lacerations and contusions from rib sloughage. The pitch of the crosscut, slopes and longwall face contributed to strains, lacerations and contusions associated with twisting, slipping and tripping. The extra manhandling of equipment and supplies contributed to strains.

Figure X.A.-1 is a graph showing injury numbers and an incident frequency comparison between the longwall and all other mining activities underground. The frequency of incidents was calculated as the number of incidents per 200,000 man hours.

* Reference to specific products does not imply endorsement by the Bureau of Mines



Injury Frequency For Longwall Mining Verses Other Mining Activity At The Thompson Creek Mines

Figure X.A.-1

$$\text{Frequency} = \frac{\text{Number of Incidents x 200,000}}{\text{Total Number of Man Hours}}$$

These figures for injury incidents were compiled for August 1981 through April 1982, a period of time in which 80% of the longwall mining was done. Figures for incident frequency beyond April 1982 were not complied due to the irregular schedule of mining on the longwall.

The longwall showed a higher incident frequency than other activities in the mine for both incidents and lost time injuries. The average incident frequency for the nine month period was 148.8 for the longwall compared to 111.9 for the rest of the mine. The lost time accidents for the longwall was 38.8 compared to 17.9 for the rest of the mine. The national average for lost time frequency for 1983 was 28.84. The average lost time accident frequency for the entire underground operation was 21.1

The high incident frequency in April 1982 represents two head contusions. These injuries were attributed to surface employees being sent underground and not being familiar with the low clearance.

B. Types of Injury and Frequency

The four tables in Figures X.B.-1 and X.B.-2 show the types of injuries for the longwall and the rest of the mine for August 1981 through April 1982. The injuries are listed in order of frequency. The longwall showed high percentages of the total accidents being contusions and lacerations. The percentage of these accidents are typical of a highly mechanized operation but the frequency is greatly exaggerated. The greater frequency may be partially due to inexperience but was primarily due to difficulties associated with the steep pitch. The rest of the mine showed higher than normal percentages of the total accidents being strains. The high strain rate was primarily due to the lack of mechanization and other difficulties associated with the steep pitch.

XI. PRODUCTION RECORD AND PRODUCTIVITY

A. As Related to Manpower

The majority of production from the longwall occurred between August 1981 and March 1982. The lack of a market for the coal

FIGURE X.B.-1

INCIDENTS

TYPES OF INJURIES AND FREQUENCY

Longwall	**NUMBER**	**FREQUENCY**
Hand/finger contusion and/or laceration	10	64.7
Foot/ankle/leg contusion and/or laceration	5	32.3
Head/face contusion and/or laceration	3	19.4
Foreign object in eye	2	12.9
Back strain	1	6.5
Neck strain	1	6.5
Wrist strain	1	6.5
Total	**23**	**148.8**

Total Man Hours - 30,920

Other Mining Activity		
Hand/finger contusion and/or laceration	21	25.0
Back strain	14	16.7
Knee strain	14	16.7
Foot/ankle/leg contusion and/or laceration	11	13.1
Foreign object in eye	9	10.7
Head/face contusion and/or laceration	9	10.7
Ankle sprain	5	5.9
Arm/shoulder contusion and/or laceration	4	4.8
Neck sprain and/or contusion	4	3.6
Back contusion	2	2.4
Chest contusion	1	1.2
Finger fracture	1	1.2
Total	**94**	**1.2**

Total Man Hours - 167,992

FIGURE X.B.-2

LOST TIME INCIDENTS

TYPES OF INJURIES AND FREQUENCY

	NUMBER	FREQUENCY
Longwall		
Hand/finger contusion and/or laceration	3	19.4
Foot/leg	2	12.9
Head/face	1	6.5
Total	**6**	**38.8**
Total Man Hours - 30, 920		
Other Mining Activity		
Hand/finger contusion and/or laceration	4	4.8
Back strain	4	4.8
Foot/leg contusion and/or laceration	3	3.6
Knee strain	1	1.2
Ankle sprain	1	1.2
Arm/shoulder contusion and/or laceration	1	1.2
Neck sprain and/or contusion	1	1.2
Total	**15**	**17.9**
Total Man Hours - 167,992		

prevented mining on a continuous basis from April 1982 on and prevented completion of this steep pitch mining demonstration as proposed. Figure XI.A.-1 shows monthly production from August 1981 through May 1984, the period of time in which production was done. This figure also shows the average monthly tons per man shift.

The extensive training discussed in the "Crew Training" sections of this report paid off. The average monthly production for the longwall for September 1981 through March 1982 was 33,730 tons per month and the average productivity for this seven months was 72.4 tons per manshift. The production and productivity

for the first full month, September 1981, were 33,671 tons and 75.5 tons per man shift respectively. The higher productivity in late 1982 through 1984 was a function of only the supervisory staff and the most experienced longwall operators surviving a layoff in April 1982. Other factors in later productivity were flexibility in scheduling runs and a switch to 10 hour shifts in 1983.

The maximum production for one shift was 2380 tons. Maximum production was severely limited by the coal handling facilities outby the face. The total coal mined on the longwall was 327,513 tons.

The following is a summary of production during the first nine months of longwall mining.

	Tons	**Man Shifts**	**Tons/Man Shift**
Longwall	259,686	3,865	67.2
Development & General Mine	7,848	16,572	0.5
Total Mine (Including Surface	**267,534**	**30,489**	**8.8**

These numbers like other numbers in the partially completed Demonstration Project were not necessarily representative. Development during this time included slope development and development for a new slope unrelated to this demonstration project. The total mine man shifts includes office, shop, and wash plant employees.

B. Downtime

Figure XI.B.-1 was prepared to show the down time on major components of the longwall, coal handling outby and other factors. Down time was calculated as a percentage of the 6½ hours per

MONTHLY PRODUCTION AND PRODUCTIVITY

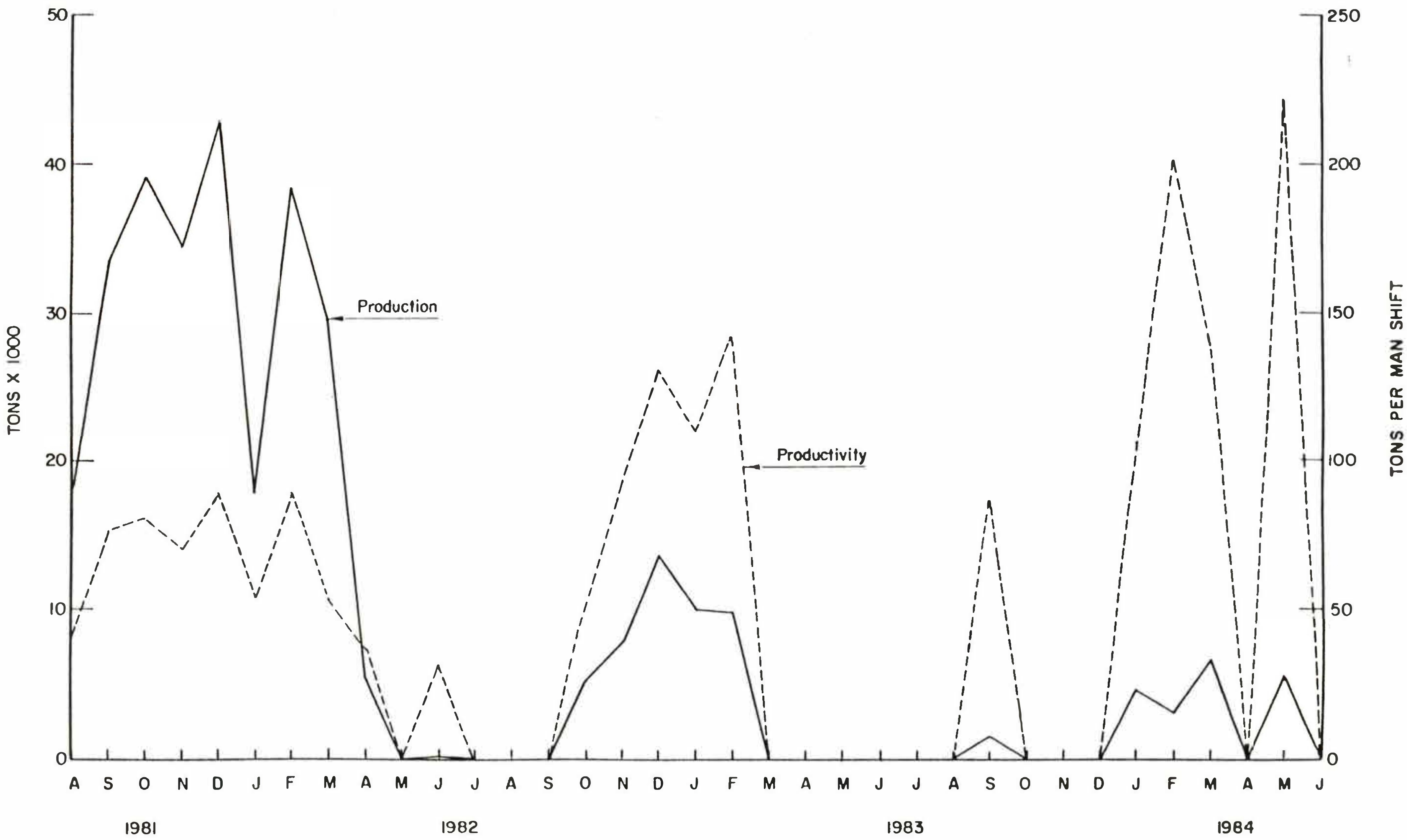


Figure XI.A.-1

MONTHLY DOWN TIME

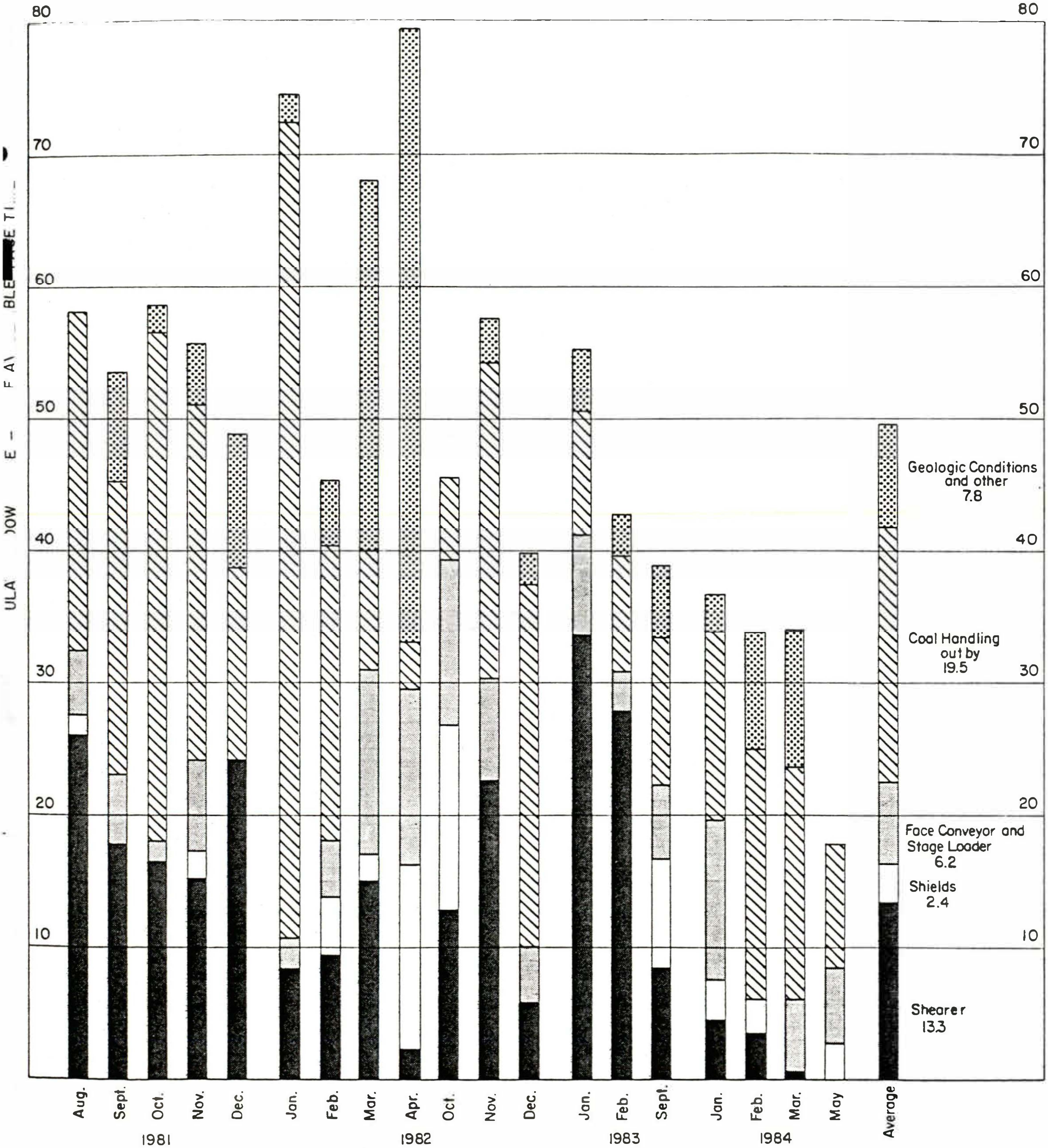


Figure XI.B.-1

hift available face time that mining was prevented. The overall average down time was 49.7% of the available time. Less than 20% of the projected steep pitch longwall demonstration was completed, therefore, the down times shown are not expected to be typical.

In the development phase of the longwall operation, low production was more of a function of slow mining procedures than down time. The miner was cutting and/or loading coal 7.8% of the available time according to a series of time studies done on the development entries and crosscuts in the summer of 1981. According to the time studies, down times were as follows:

Miner	8.1%
Shuttle Car	7.0%
Drills	1.3%
Coal Handling Outby	18.3%
Other	6.2%
Total	40.9%

All of the coal handling delay was from the monorail conveyor and the track haulage to the slope belt.

As discussed in the "Crew Set Up and Mining Methods" section of this report maintenance was performed on all three shifts. The majority of the maintenance program was done on the graveyard shift. In general the manufacturer's maintenance schedules were followed. Preventive maintenance schedules were accelerated for items that required more frequent maintenance, such as the trapping plates and the brakes on the traction unit.

XII. UNUSUAL PROBLEMS

A. Shear Zone

The shear zone discussed in "Description of Geologic and Premining Conditions" proved to be a major problem. Its presence was known from mining done in the 50's and 60's, but its extent and severity were unknown. When mining across the shear zone in the tailgate and headgate for the first panel of the Longwall Demonstration, its extent and severity were better defined. This new information caused revisions to the mine plan to limit the number of times the zone would be crossed.

The extremely poor roof and floor conditions of the shear zone caused dangerous, slow and expensive development. Secondary support in the form of resin bolts, cross bars, cribbing, arches and polyurethane binders were used. As a result of secondary support, access was restricted. Transporting the longwall equipment into the first panel was difficult but transporting it past the shear zone into the second panel promised to be even more difficult.

The possibility of mining through the shear zone was analyzed. It was determined that for a manpower and material cost of over one million dollars the longwall could mine through the 750 feet of shear zone influence. The analysis assumed the extensive use of polyurethane binder to stabilize the shear zone and a time period of over a year.

B. Other Faults

A small fault was encountered in March 1982. The longwall hit the fault at an angle of 30 degrees and required 170 feet of advance to mine past it. The width of the fault was 20 feet. One cave occurred on the face requiring several shifts to clear. The remainder of the fault was injected with polyurethane binders in advance of mining and presented no further roof control problems. The face was left idle in the fault for six months with no roof control problems upon restarting it.

Small faults and rock spars caused occasional delays. Cribbing was placed on top of shields to get under rock extending down from the roof. Rock extending up from the floor required drilling and shooting to move the face conveyor and shields over it.

C. Two Entry System

As discussed in "Mine Design", the development of the two entry system presented many unusual problems. Most of the key elements of mining presented special problems: mining, haulage, supplying ventilation and roof control.

Mining in the face was difficult due to the necessity to mine rock in part of the floor to maintain a level roadway. The roadheaders used were not adequate to cut the rock. Future plans included using a larger machine. Mining the crosscuts was difficult because mechanized equipment was not available. The coal in the crosscuts was drilled with pneumatic drills and shot on the solid.

Haulage of coal out of the face presented difficulties again due to the equipment being used. The problem started with getting coal from the solid boom of the roadheader onto the 2 foot wide bridge conveyor or into a shuttle car. The problems continued with loading coal cars with the bridge conveyor or the chutes in the crosscuts. The problems continued with electrical and mechanical trouble with the locomotives. Future plans included a belt in the return with a chain conveyor from the bottom entry to the top entry.

Supplying the two entry system was difficult due to the manhandling required to get supplies up the crosscuts and into the faces. Future plans included the use of diesel supply haulage in each entry.

Ventilation presented special difficulties due to the isolation generally required for belts or trolly haulage. Battery locomotives were used for the first panel, but a belt in the return with adequate computer monitoring was scheduled for the next panel.

Roof control presented difficulties unusual to a non pitch mine. The flexibility of bolting perpendicular to the steep pitching roof, of bolting on the low side, of bolting on the high side and of bolting in the steep crosscuts could only be met with a pneumatic drill.

XIII. SUBSIDENCE AND ROCK MECHANICS

A. Chain Pillar Design Rationale

The chain pillar widthsfor the first panel of this Demonstration Project was dictated by prior mining on the site. The previous mining in the mine was done using room-and-pillar methods. Entries had been driven from the portal updip to the strike 2° to allow drainage. Mining had been completed above these entries where no hoisting had been required. Just prior to closing this mine slopes had been driven below the portals and entries were started across the strike. These two sets of entries were the basis for projections on the first longwall panel. The entries at the portal level were not straight due to the desire to establish a drainage grade, therefore, a new entry was driven for the longwall tailgate. Pillar width was primarily a function of an attempt to straighten the tailgate. The location of the headgate was dictated by keeping a uniform panel width as measured from the tailgate. The pillar size between the headgate and the entry below the headgate was primarily a function of an attempt to straighten the next longwall panel. The tailgate solid pillar width varied from 45 to 70 feet and the headgate pillar varied from 70 to 115 feet.

Design of chain pillars was one of the objectives of the Demonstration Project. Colorado School of Mines (CSM) was subcontracted to determine, among other things, the behavior of pillars to longwall mining, including stress distributions, and lateral and vertical deformation. CSM's research started with core drilling in the roof, coal and floor. These cores were used to determine physical properties of the rock. Tests run on the core included Brazilian Disc tension strength, unconfined compression strength, peak intact cohesion, peak intact angle of internal friction, elastic modulus and Poisson Ratio. Testing was done on the MTS 810 testing machine at the Bureau of Mines, Denver Mining Research Center. Research continued with the installation of Irad vibrating wire stress meters and extensometers at varying distances through the chain pillar and the installation of convergence points in the adjacent entries. Due to the premature completion of this Demonstration Project no conclusive results were compiled by CSM.

A preliminary pillar size design was done for Snowmass by Jim Walter Resources. Their study utilized the theory of transferred load from the waste area to the barrier pillar using formulas derived by Dr. Authur Heriot Wilson, in his The Stability of Underground Workings in the Soft Rocks of the Coal Measure.[4] Information for this analysis was taken from a paper by David W. Wisecarver, B.O.M., and James K. Greenlee, Western Associated Coal Corp., entitled "Steep Seam Longwall," and from a mine map. The geometry assumed was a 400 foot panel width, a 14 foot entry width, a depth range of 500 to 2,250 feet and a cross cut center of 214 feet. Many assumptions were made to arrive at the following recommended pillar sizes:

Up to 1,000' depth	214 x 55' (centers)
1,000-1,700' depth	214 x 60' (centers)
1,700 - 2,250' depth	214 x 65' (centers)

Additional mine site testing would have been required to verify these preliminary results.

Additional pillar design work has been done by Nicholas P. Kripakov[5] of the B.O.M., Denver Research Center Ground Control Division. He determined the magnitude and orientation of the principal stresses in the roof and floor using deformation measurements obtained from stress relief overcores and average elastic properties obtained from core samples. Measurements and cores were taken in the lower headgate entry in the summer of 1981. Further testing was proposed for the three south entries, headgate entries for the second panel, but was denied due to the idle status of the mine.

B. Longwall Panel Layout Rationale

The initial proposal for panel widths was three panels at 400 feet each. The first panel width like the pillar design was dictated by previous mining. This first panel ended up with a face width of approximately 340 feet. The initial panel length was dictated on the north by a projection of the existing slopes and on the south by Middle Thompson Creek.

The projections were changed to panel widths of 400, 500 and 600 feet to allow assessment of panel width as part of the rock mechanics and subsidence investigation. Later, long range mine planning dictated 500 feet widths for the second and third panel.

The projected panel lengths was also changed. Considering the difficulty of mining through the shear zone in first panel, the projections were changed to have new slopes driven from the portal to the south west on a grade of 12-15° through the shear zone. On the south side of shear zone the slopes would zig-zag down dip along the shear zone. This slope configuration eliminated crossing the shear zone with each panel and reduced the panel length to approximately 4,000 feet.

Another objective in this Demonstration Project was to determine the rock mass response to various width of extraction. Instrumentation was installed by CSM to monitor subsidence, shield loading, panel loading and pillar loading. The evaluation of these factors for various panel width was expected to assist in panel design for the future.

C. Surface Subsidence

This Demonstration Project represented an unique opportunity to collect data on mining induced surface subsidence resulting from longwall mining a steep pitching coal seam in mountainous terrain. CSM was subcontracted to design, install and monitor a subsidence net over the first three panels. Three subsidence monitoring lines were installed approximately on the dip or perpendicular to the panel lengths. These lines were placed near 25, 50 and 75% of the first panel length. See Figure XIII.C.-1 for lines SP-1, SP-2 and SP-3. The survey net included six major turning points, five minor turning points, 75 subsidence monuments on the three monitoring lines and 20 subsidence monuments along the centerline of the first longwall panel. Major turning points were 2-inch schedule 80 steel pipe mounted in a 12-inch concrete post that was poured to bedrock. Attached to the pipe was a trivet plate to allow direct installation of the theodlite. Minor turning points had the pipe in an eight-inch concrete post and the subsidence monuments had the pipe in concrete poured to ground level. The monuments were installed on 80-foot centers. In between monuments leveling points were installed on 20-foot centers. These leveling points were rebar with an aluminum cap. Mining in the first panel progressed past the SP-1 in December 1981 and had progressed approximately 1,400-feet past SP-1 when last surveyed in August 1983. No subsidence was detected on any of the points through August 1983.

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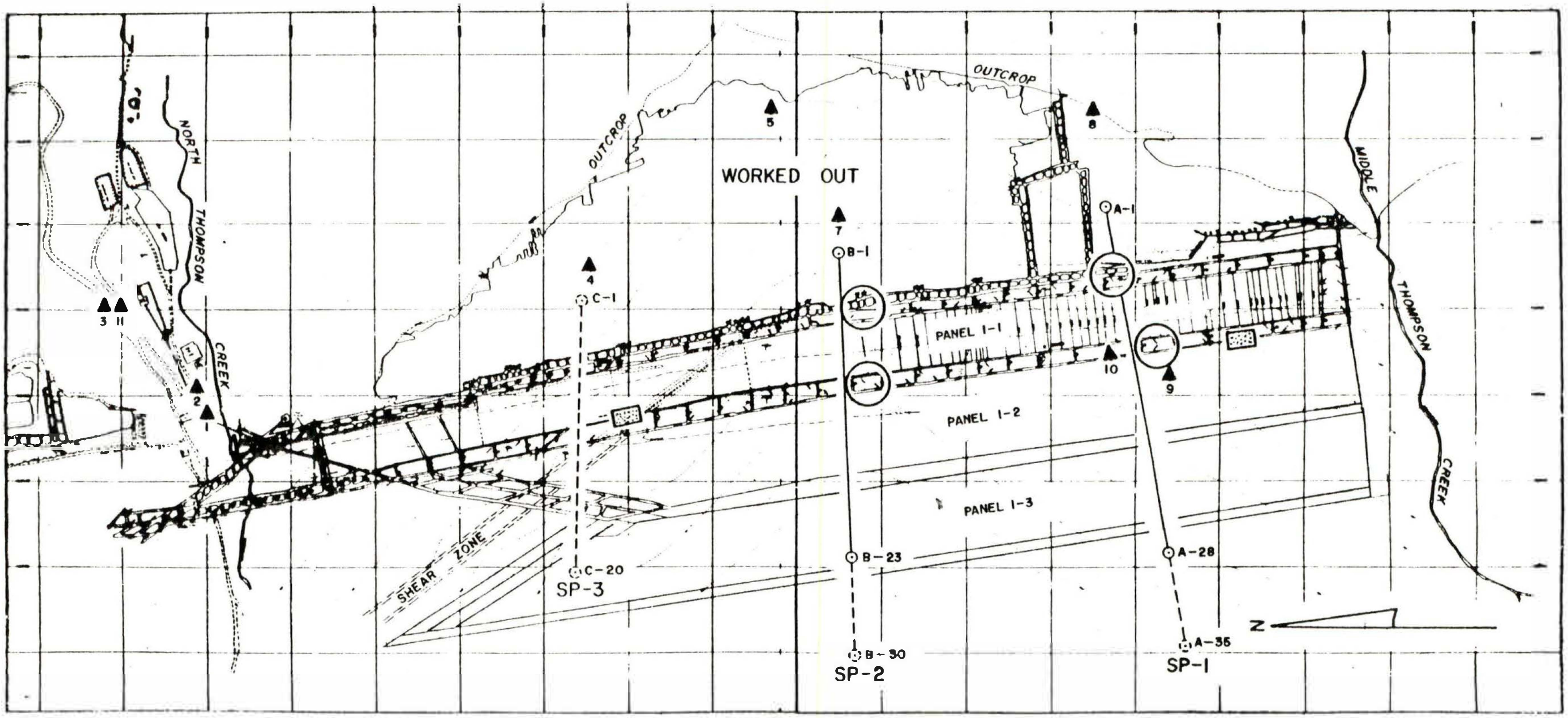


FIGURE XIII.C.-1

Subsidence Monitoring Network and Rock Mechanics Instrument Areas

Subsidence Monument Line Surveyed

Subsidence Monument Line Not Surveyed

Turning Points — Subsidence Survey

Underground Pillar Monitoring Stations

Flat Jack Monitoring Locations

Scale: 1" = 1000'

headgate entry near the end of the longwall panel. As development of the headgate for the first panel mined through the shear zone, monitoring was done to determine the effectiveness of conventional wood and steel support compared to polyurethane binder injection. As a follow up, monitoring continued with flatjacks, flat pressure cells, installed between timber and cross bars through grouted zone. All five pairs of flatjack registered pressure increases but three pairs also recorded pressure decreases. The extensometers and convergence points at SA11 and SA21 were destroyed by face preparation activity prior to the face advance. The B.O.M., Spokane Research Center, conducted studies of fiber reinforced concrete crib blocks in the lower headgate entry. Flat jacks were installed on all four corners of cribs and convergence points were installed around the cribs. The first loading was shown 133 feet behind the face and only on the high side of the entry. The pressure on these flat jacks increased slightly until the face was 1,000 feet past then stabilized. The findings from these flat jacks indicated that routine installation of cribs in the lower headgate was unnecessary.

Instrumentation to monitor the hydraulic pressures developed in the shield leg ram was installed on select shields along the face. The recorded pressures could be used to determine the effect of panel width to depth ratios on support loading and the effect of previous panels on support loading. The 22 chart records purchased to monitor shield pressures could not withstand the mining conditions, therefore little useful information was collected.

E. Potential Problems in Second and Third Panel

All of the monitoring done on the first panel and projected for the second and third panel were designed to help project potential problems with successive panels. The information gained on the first panel, subsidence and rock mechanics monitoring, revealed little to change the design of the entries, pillar or panels.

Entry development is difficult due to the lack of adequate mechanized equipment. Mining conditions in these entries will get worse under higher cover and will make development even more difficult.

Due to the unknowns of pillar design on a steep pitch with a wide range of cover, pillar failure is a potential problem. The extraction of the previous panel or panels may cause unsuspected lateral movement in the tailgate chain pillars or may cause higher than expected loading. No pillar failure potential was indicated.

The loading on panels presents potential problems with successive panels. Excessive loading could cause poor roof and face condition and could lead to shield failure. Instrumentation in the second panel, below SA21, showed only slight additional loading as the longwall face passed. No unexpected high loading was indicated.

Another potential problem is the unknown geologic conditions. The unknown location and extent of the shear zone and other minor faults could adversely affect mining on future panels. Due to the rugged terrain and high cover, extensive exploration was not possible.

XIV. PLANNED METHOD OF FACE RECOVERY AND TRANSPORTATION TO THE NEXT PORTAL

A. Recovery

Due to the premature completion of this Demonstration Project, the longwall face was never removed or installed in another face. Like many other aspects of mining on a steep pitch the removal of the face promised to be difficult.

The plans were to remove all but the headgate equipment through the tailgate. To get equipment out the headgate would have required a dual winch set up similar to that use for installation. Maintaining clearance for tailgate cable would have been difficult. By winching the components up the face, a second cable would not be needed and a cave on the face where shields had been removed would not stop removal.

To prepare for removal the face would have to be sheared a minimum of two additional cuts. After each cut the face conveyor would be advanced but the shields would be left in place. To accomplish this blocking would be required between the D.A. rams and the conveyor, and extensions would be required on the anchor stations. Bolts would be installed on 4-foot centers in the exposed roof. Anchors would be installed in the face and roof to attach the hoists and sheive blocks used to position shields for removal. The first equipment removed would be the shearer. It would be positioned at the tailgate end of the face and dismantled into four or more pieces for removal. The face conveyor would be disconnected and removed in three pan sections starting at the tailgate.

The equipment in the headgate would be removed prior to removal of the shields. The remaining belt in the headgate would be removed to allow access to the stage loader, the control console, the hydraulic pumps, the face conveyor drive, the power center and other face support systems.

The shields would be removed out the tailgate starting with the shield in the headgate. This arrangement provides support for shields as they are being removed. As a shield is removed cribs would be installed in its place to provide support for the next shield removal. Figure XIV.A.-1 shows a typical shield removal position. For operation of the shields during removal, a temporary hydraulic pump would be installed in the tailgate.

B. Transportation

The plans for moving the longwall from the first panel to the second panel envolved the use of existing track. The mine was changing over to rubber tire diesel for transportation, therefore, successive panels would require special crawler or rubber tired shield movers to transport the equipment.

To establish access to the face in the tailgate secondary support other than cribs may be required. The drop deck shield cars would be backed in as close as possible to the face and necessary ramps would be installed to allow loading of the shearer, the conveyor and the shields. The equipment would be transported out to the old track slope and lowered down to the next tailgate entry with the hoists. Temporary track would be restabilized across the new belt slope to allow transportation to the top of the set up room on the second panel. Secondary roof support work and clearance work would be required to get the equipment in the new tailgate.

The equipment removed from the headgate would be transported using a rubber tire diesel scoop. The equipment would be carried or skidded out the old headgate entry, down the belt slope and into the next headgate entry.

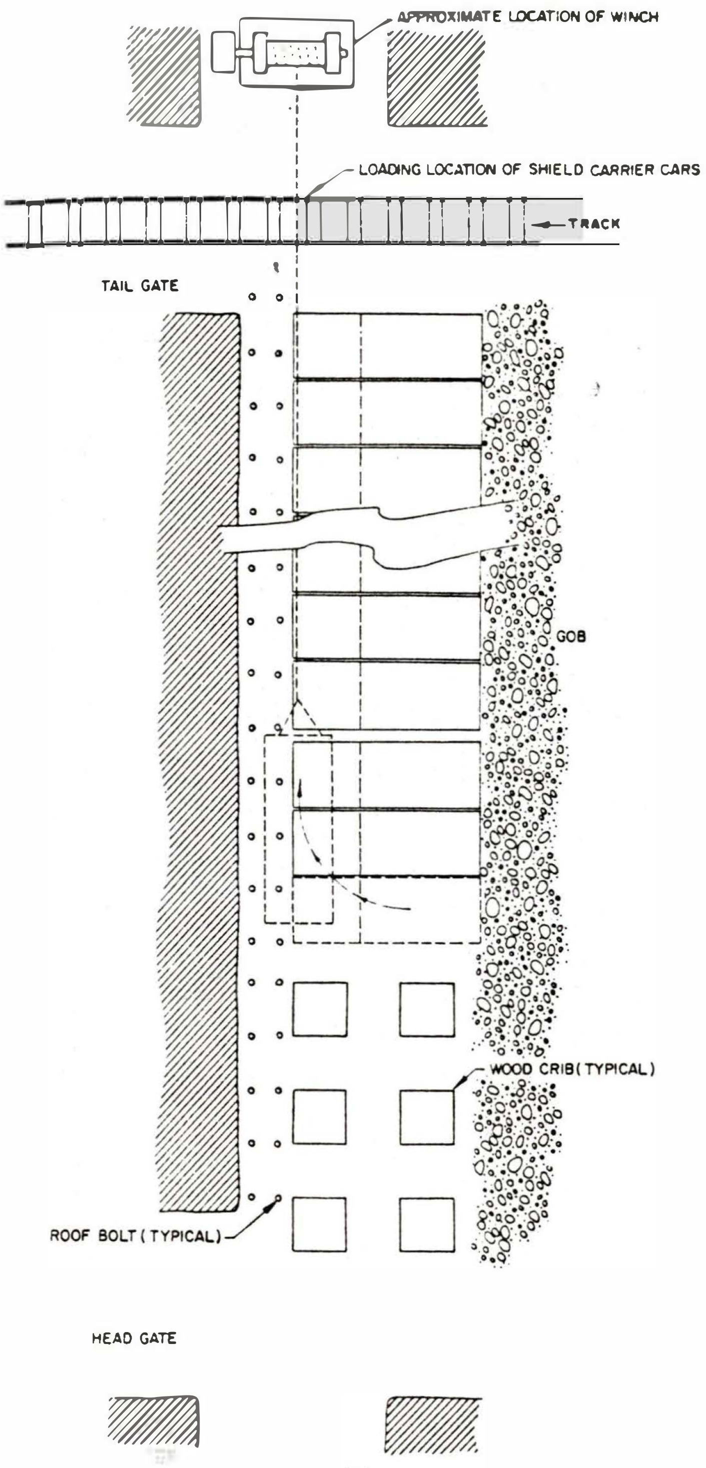
The installation of the longwall in the next set up room would be very similar to the method used for the initial installation.

XV. EVALUATION OF SYSTEM PERFORMANCE

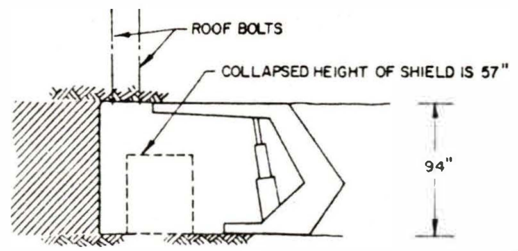
A. Equipment Operation

Snowmass Coal Company considers the performance of the equipment on the longwall as very good. As with any piece of new equipment there were minor modifications necessary. By the time mining was completed most of the bugs were out of the system. Had the Demonstration Project been completed as planned, major equipment problems may have surfaced. Durability could not be properly evaluated due to the small amount of mining done.

Typical Shield Removal System



Plan



Profile

FIGURE XIV.A.-1

As shown on Figure XI.B.-1 the shearer caused the majority of down time among longwall equipment. On the steep slope the training and braking systems did require extra maintenance but that could be expected. The radio control, once operational, did not prove useful because it lacked the human factor of being able to watch the floor and roof while cutting.

The shields worked exceptionally well on the pitch. The combination of the Troika system, the face conveyor advance system and the roof control system proved to be a good choice. The canopy extensions and the anti spalling plates were seldom used.

The face conveyor and stage loader generally worked well. Moving coal from the face to the belt worked much better than moving coal from the stage loader to the clean coal stockpile.

Equipment performance in the panel development phases of the Project was considered to be poor. The lack of adequate equipment for panel development seriously hindered success of the Demonstration Project.

B. Production

Snowmass considered the longwall production to be average when compared to other longwalls and excellent when compared to alternate means of mining a steep pitched seam. Considering the overall production for panel development plus longwall as discussed in section XI, "Production Record and Productivity," the system did poorly.

C. Ventilation and Roof Control

The ventilation system for the mine worked fine for two working sections with little methane. Maintenance of the system was difficult due to the pitch. Daily inspection of the #2 fan was difficult from underground because of the slope and difficult from the surface because of the secondary mountain access road. The methane levels were expected to increase with each successive panel. Increased levels of methane could have easily taxed the system as it was.

Roof control was generally good. Roof control on the longwall face and in the gate entries was excellent. The major roof control problems were in the shear zone during development. The polyurethane binders used in the shear zone worked well but were very expensive. The roof was very competent in most of the mine but roof control during development was difficult due to the lack of adequate mechanized equipment.

D. Safety

Steep pitch mining proved to be less safe than mining in seams with little or no pitch. The steep pitch contributed substantially to the normally high mining accident frequency.

E. Costs

Costs for the demonstration project were not broken down in detail between the longwall, development and other underground development and support. During the 9 months from August 1981 through April 1982, $4.26 million was spent on labor for the #1 Mine and $2.21 million was spent on materials. The labor and material costs were $24.18/ton for the 267,534 tons of raw coal or $35.46/ton of clean coal. These costs do not include depreciation and other site specific overhead costs which, in this case, were approximately 40% more during the same period of time. The average selling price for clean coal during this 9 month period was approximately $28/ton, FOB mine site. Although costs were not broken down in detail, the productivity on the longwall indicates that production could have been profitable if the combination of development, cleaning and overhead costs had been more favorable.

XVI. RECOMMENDATIONS AND CONCLUSIONS

The recommendations include items to continue and items to change. The major recommendations concern the development of panels. Other recommendations concern roof control, safety and longwall installation.

Development was the largest bottleneck in the Demonstration Project. Development ended up very slow and expensive. Had the first longwall panel been completed there would have been a several month delay before the next panel development was completed. Snowmass Coal Company was in the process of making changes but was not to a satisfactory production level when mining was discontinued. A larger roadheader miner and a heavy duty 4-wheel drive shuttle car were being used in the slope development. The process was still being slowed by a short section of the 2 foot wide bridge conveyor and by manual bolting of the roof and high rib with pneumatic drills. The majority of development with the larger road header and shuttle car was in a crossing of the shear zone which was not a representative trial. Another major change planned for the panel development was to install the longwall panel belt in the higher return entry while developing. Coal from the lower entry would be conveyed up crosscuts using a chain conveyor or high angle belt. Longer crosscut centers was also planned for the next panel entries. The change to rubber tire, diesel tractors did prove to be a more efficient means of transporting men and materials than the hoist and battery locomotives. A long range plan being considered was to develop a two seam longwall mining system with the same pair of development entries. The plan was to develop an entry in both seams at the same elevation and drive rock tunnels between then for crosscuts. The following is a list of other possibilities that might be investigated:

- Use a more modern high volume bridge conveyor behind the miner to transfer coal to a panel belt.

  Use a heavy duty continuous miner and cut rock in the roof rather than the floor. The parallel roof and floor would allow more clearance, eliminate the high rib, allow bolting with a roof bolting machine, Etc.

- Use hydraulic drills powered by face equipment for bolting to reduce the labor and expense of air lines.

- Use an auger type miner with jack posts and cables to mine rock to rock in the entries and an angled crosscut. Rock in the floor could be shot and leveled in the entries at some point behind the miner. The bridge conveyor and builtin roof bolters common to this machine could be used in the mining cycle.

- Develop a continuous miner with a head that rotates parallel to the roof.

- Develop a suitable boring machine to drill the crosscuts.

- Develop a suitable bolting machine to bolt perpendicular to the roof without switching positions with the miner.

- Use an advancing longwall with packwalls.

- Angle the panels more to the strike to reduce the floor or roof mining requirements.

Roof control results were very favorable during this project. The installation of bolts during development and installation of support through the shear zone did prove to be very expensive and could use more research. Positive approaches to roof control that were used by Snowmass and that would be recommended for similar projects are:

- Use of the Troika system of shield.

- Use of polyurethane binders in extremely poor roof conditions.

- Use of chain link fencing installed with split bolts for rib control.

- Use of fiber cribs in the tailgate entry.

The outside hands-on longwall training is believed to have had a large part in keeping the accident rate as low as it was. An effective safety and training program will always be necessary in an operation like this. Safety bonus programs might also be a serious consideration. A major safety improvement recommendation for use throughout the mine is to develop better materials handling systems. With the mobility of rubber tire diesel equipment, one improvement in materials handling would be mobile "cherry pickers" or hydraulic lifting devices.

The positioning of shields during the set up of the longwall proved to be difficult. A possible improvement in the installation technique might be to develop a hoisting system which would allow the shield to be dropped down the slope sideways directly into position. Another possibility might be to lower a shield mover down with the shield to do the final positioning.

The objective of the Demonstration of Longwall Mining in a Steeply Dipping Coal Seam according to Section 1.1 of the Contract was

> "to stimulate the development of underground mining technology by demonstrating a system and mining steeply dipping coal beds (25° or more) by longwall methods and techniques capable of increasing coal production and resource recovery."

The amount of development that was stimulated by this longwall demonstration is unknown. There were no known patentable inventions made associated with this demonstration. We do believe that several manufacturers were "stimulated" into evaluating their equipment more thoroughly and would be better prepared to discuss steep pitch mining.

The demonstration did prove that longwall mining on a steep pitch is possible with current technology and equipment. Unfortunately, the demonstration also proved that the most difficult part of any longwall operation, development, was even more difficult on a steep pitch.

The demonstration did increase resource recovery and is believed to have increased productivity on a steep pitch. Exact figures for extraction ratio are not available for previous room and pillar mining in the Thompson Creek #1 Mine but were estimated to be no more than 60%. With the longwall and a two entry development system, 80 - 85% extraction was expected. Increased resource recovery due to higher cover limits was not demonstrated but may be possible. Productivity from previous mining in the Thompson Creek #1 Mine is unknown but could not have been very good if Snowmass' productivity during development was any indication of the productivity in the labor intensive operation.

REFERENCES

(1) Collins, B. A. Geology of the Coal Bearing Mesaverde Formation (Cretaceous), Coal Basin Area, Pitkin County, Colorado. M.S. Thesis, CO School of Mines, Golden, CO, 1970.; pp 38 - 55.

(2) Keenan, A. M. Longwall Mining of Coal in Seams Pitching 30 Degrees. Pres. at SME Fall Meeting, Seattle, WA, September 22-24, 1971. Soc. Min. Eng. AIME preprint 71-F-336

(3) MSHA, Denver Health and Safety Analysis Center. Phone Communications, 1986.

(4) Jim Walters Resources, Inc., Mining Division. JWR STRATA Control Report 1982, Pillar Size Study, Thompson Creek #1 Mine, Pitkin County, CO. December 14, 1981. Available from F. Carr, Jim Walters Resources, Birmingham, AL.

(5) Kripakov, N. P. Analysis of Pillar Stability on Steeply Pitching Seam Using the Finite Element Method. Bureau of Mines RI 8579, 1981.