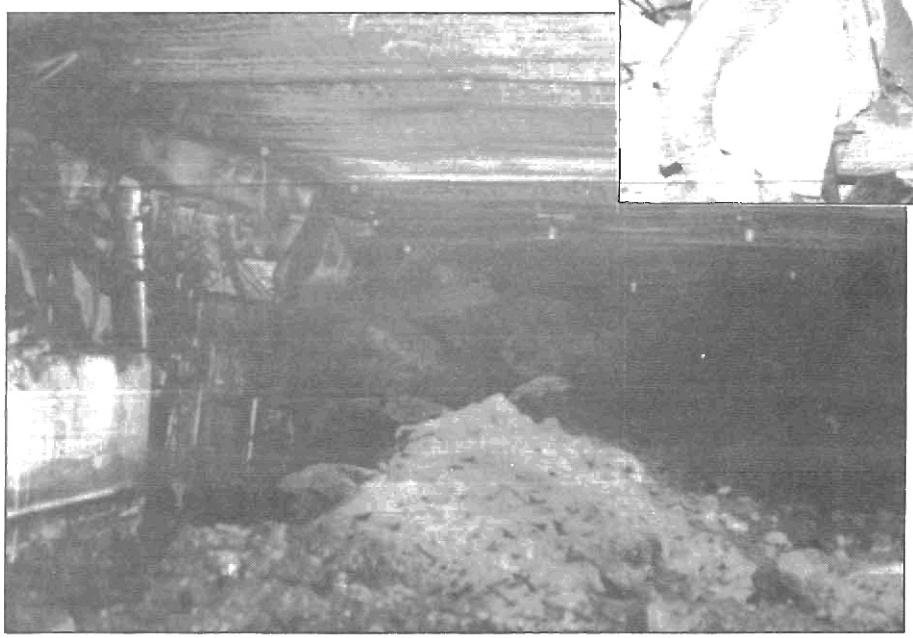
RI 9533

REPORT OF INVESTIGATIONS/1995

# Field Evaluation of Cable Bolts for Coal Mine Roof Support



By John P. McDonnell, Stephen C. Tadolini, and Paul E. DiGrado



UNITED STATES DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

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***Cover:** U.S. Bureau of Mines researchers in fully cribbed tailgate entry of a longwall (upper right) and tailgate entry utilizing only cable bolts (no wood cribs) for secondary support (lower left).*

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UNITED STATES DEPARTMENT OF THE INTERIOR
Bruce Babbitt, Secretary

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Rhea L. Graham, Director

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## UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

Metric Units

| | | | |
|---|---|---|---|
| cm | centimeter | kN | kilonewton |
| cm/h | centimeter per hour | kN/cm | kilonewton per centimeter |
| GPa | gigapascal | kPa | kilopascal |
| h | hour | m | meter |
| ha | hectare | MPa | megapascal |
| kg/m | kilogram per meter | $t/m^2$ | metric ton per square meter |
| $kg/m^3$ | kilogram per cubic meter | | |

U.S. Customary Units

| | | | |
|---|---|---|---|
| ft | foot | $lb/ft^3$ | pound per cubic foot |
| in | inch | lb/in | pound per inch |
| in/h | inch per hour | lbf | pound (force) |
| lb | pound | $lbf/in^2$ | pound (force) per square inch |
| lb/ft | pound per foot | $st/ft^2$ | short ton per square foot |

Reference to specific products does not imply endorsement by the U.S. Bureau of Mines.

# FIELD EVALUATION OF CABLE BOLTS FOR COAL MINE ROOF SUPPORT

By John P. McDonnell,[1] Stephen C. Tadolini,[1] and Paul E. DiGrado[2]

## ABSTRACT

Cable supports offer several advantages over traditional secondary support methods by enhancing stress redistribution to pillars and gob areas, minimizing or eliminating timbers and cribs that reduce ventilation, eradicating material-handling injuries related to placement of crib supports, and providing a cost-effective alternative to secondary support. The U.S. Bureau of Mines, in researching alternatives to traditional roof support methods, designed and installed high-strength cable supports to improve the stability of longwall gate road and bleeder entries in a Western U.S. coal mine. With the cooperation of industry, methods were developed to install cable supports in a tailgate and bleeder entry test area using traditional resin cartridges. Resin-grouted cable bolts were also installed and evaluated in additional longwall gate road and bleeder entry systems at the study mine. The cable-bolted areas successfully maintained roof support throughout the tailgate and bleeder entries. Cable supports replaced wood cribbing as secondary support in the bleeder entry system and minimized the use of cribbing in the longwall tailgate entries.

This report describes the theory, application, and advantages of cable supports and presents mine measurements made to assess the cable performance during the retreat process of longwall mining.

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# INTRODUCTION

Gate road entries remain the most hazardous areas in longwall mines, and keeping them stable is a challenging task. Gate roads provide access and escapeways for the miners, coal transportation, and ventilation (intake and return). Headgates are supported primarily with roof bolts during development mining and, during longwall face retreat, require secondary reinforcement for a nominal distance outby the face, around 152 m (500 ft). Generally, wooden posts, hydraulic jacks, or spot roof bolts are used to cope with the front and side abutment pressures caused by mining. Headgate entries must stay unobstructed and completely open, with minimum convergence, for access of personnel and equipment and for coal transport. The tailgates, however, are used for return air, sometimes intake air passages, and travelways. Much higher ground pressures exist around the tailgate entries, making them difficult to support. Moderate entry closure or partial blockage of the tailgate entry is tolerated. Generally, wood or concrete cribbing materials are used for secondary supports in the tailgate entries, where cribbing patterns and densities are varied to support difficult ground pressures.

A recent analysis on the number of reported accidents related to cribbing, timbering, and blocking revealed that in U.S. underground mines in 1991 and 1992, a total of 734 accidents pertinent to these activities were reported. The majority of the serious accidents occurred in Western U.S. underground longwall mines. These accidents directly relate to the height of the coal seam, which requires the use of ladders to build the cribs with heavy and cumbersome materials (*1-2*).[3] Use of cable bolts in lieu of cribbing in tailgates reduces the exposure of the worker to the hazards of cribbing.

Using less crib material will also help minimize the ventilation resistance in air passages, which results in savings in air pressure and increases airflow at the working face, while removing gas and dust more efficiently (*3*). This creates a safer and cleaner mine environment for underground workers, while saving on mine ventilation costs to the mine operator. Additionally, subject to the availability of a reliable source of quality timber, the amount required to support a single 2,530-m (8,300-ft) gate road with 1.8-m by 20.3-cm by 20.3-cm (6-ft by 8-in by 8-in) crib blocks in a 2.9-m (9.5-ft) high opening would be 613 ha (248 acres) of select-cut prime timber.[4] Consequently, eliminating or minimizing crib supports can reduce accidents related to support installation, improve ventilation, and reduce wood consumption.

Cable bolts were introduced to the U.S. mining industry in the early 1970's as a method to reinforce ground prior to mining; discarded wire rope was the preferred choice by most ground control engineers. Today, the basic cable bolt support consists of a high-strength steel cable installed and grouted with cement in a 4.1- to 6.4-cm (1.625- to 2.5-in) borehole. Recent advancements in resin and cable technology permit the use of resin as an anchorage material in a 2.5- and 3.5-cm (1- and 1-3/8-in) diam hole.

Traditional cables have an ultimate strength of 244.7 to 266.9 kN (55,000 to 60,000 lbf) and a modulus of elasticity of about 203.4 GPa (29.5 million lbf/in$^2$). Cables are 1.52 to 1.59 cm (0.6 to 0.625 in) in diameter and consist of seven individual wires. Driven by the demand for high capacity and large deformations in coal mine gate roads, high-strength cables measuring 1.78 cm (0.70 in) diam with a yield strength of 244.7 kN (55,000 lbf) and an ultimate strength of 378.1 kN (85,000 lbf) are being introduced. These cable material characteristics provide large amounts of deformation at a high degree of loading and ultimate strength. These steel cables are flexible and can be coiled to about 1.2 m (4 ft) diam for handling. This flexibility is one of the primary advantages of cable supports since the support length is not limited or restricted by the opening height.

For a cable support system to be effective, it is necessary for the loads to be successfully transferred from the rock to the cable through the grouting material. Laboratory and field investigations have determined that to achieve a load of 258.0 kN (58,000 lbf) when resin products are used, 1.2 m (4 ft) of grout is required to develop the ultimate capacity of the cable. Of course, adequate anchorage should be evaluated on a site-specific basis by performing standard pull tests. Laboratory and field results indicate that at least 6% elongation for the ungrouted portion of the cable can be expected (*4-5*). This permits the designer to vary the amount of grouted and ungrouted portion of cable to obtain various degrees of stiffness, depending on the specific strata to be supported.

From 1992 to 1994, the U.S. Bureau of Mines (USBM) investigated the use of cable supports at Mountain Coal Co.'s West Elk Mine, Somerset, CO, to evaluate the potential to supplement or replace conventional headgate and tailgate secondary supports. The results indicated that given the pillar layout and site-specific roof conditions and strengths at the test mine, the use of cable bolt systems was successful in maintaining the longwall tailgate entries (*6-7*). The effective application of this technology indicated the potential for several mining benefits. A direct benefit of cable systems is the installation of a support system capable of strengthening and reinforcing roof members, transferring high pressures into the main and

[3]Italic numbers in parentheses refer to items in the list of references at the end of this report.

[4]Data provided by the U.S. Forest Service.

immediate roofs and onto supporting structures away from the periphery of the entries. The major disadvantages of the cement-grouted cable systems are the amount of time required to pack and grout the holes, the additional step required to install the bearing plates, and the system component costs.

The USBM, with the cooperation of industry, has developed methods to install cable supports using traditional roof bolt resin cartridges. The first resin-grouted cable bolts were employed to support coal mine bleeder entries directly behind a longwall panel at the West Elk Mine. Resin-grouted cable-supported bleeder entries have proven to be an effective alternative for secondary support when compared with wood timbers and cribs in bleeder entry applications (*8*). The success of this investigation prompted the mine operator to investigate the application of resin-grouted cable supports in a longwall gate road and assess support performance during the extraction of the adjacent longwall panels.

The ultimate goal of this investigation is to provide engineered and economically feasible support systems and designs that provide safe work areas under diverse and hazardous ground conditions while enhancing production in U.S. coal mines.

This report presents results from the initial cement-grouted cable test area and subsequent cable support sites in the bleeder and gate road entries.

## CABLE SUPPORT DESIGN

Although cables have been used in U.S. hard-rock mines, the procedure is new for underground coal mines, and applications are significantly different from hard-rock employment. Wire rope and cable supports have been used in Australian coal mines for about 8 years. However, Australian mining methods differ from U.S. methods in that large barrier pillars, approximately 61 to 122 m (200 to 400 ft), are left between two entries and cables are frequently supplemented with steel arches and beams (*9-10*). The analytical and theoretical approach, ongoing during the investigation, is initially simplistic in nature, but is being constantly modified based on field measurements.

The first step of the design procedure requires a detailed data gathering to determine the ground conditions in the underground mining environment. This includes a general estimation of the rock mass quality, the geological structure, and the strengths of the immediate and main roof members. This information can be obtained from roof core samples and supplemented with a borescope or camera used in the borehole. Secondly, an estimate of the induced stresses calculated using empirical methods or modeling should be included in the stability analysis. The weight of the rock mass should be expressed per lineal meter (foot) of roof for mine entries to determine the required cable spacing. The zone of rock material that must be supported by the cable system can be determined several ways, and design principles are constantly being updated.

The simplest approach is to identify a parting plane where separation above the roof bolted zone is likely to occur. For the worst case scenario, it can be assumed that the bolted strata will shear at the pillar boundaries of the opening and that the entire block must be supported by cables, as shown in figure 1. The weight of the material can be determined by using the equation

$$F_w = W_e \times H_p \times \gamma, \qquad (1)$$

where $F_w$ = weight of rock per lineal meter (foot), kg/m (lb/ft),

$W_e$ = effective width of opening, m (ft),

$H_p$ = distance from coal mine roof to parting plane, m (ft),

and $\gamma$ = rock density, kg/m$^3$ (lb/ft$^3$).

This method is conservative since it is more likely that the coal pillars will provide additional support to the detached roof structure, forcing the formation of a pressure arch of the failed material. As shown in figure 2, the cables must be capable of supporting only the material within the boundaries of the pressure arch. The height of the arch can be correlated to the vertical and horizontal stresses acting in the immediate roof and is believed to increase as the in situ horizontal stress increases. A generally used criterion for determining the height of the pressure arch is that the failure height is 0.5 to 2.0 times the mined seam height, varying with the direction and magnitude of the stress field (*11*). The weight of the material within the arch can be estimated with respect to the opening width and the height of the arch by the following equation:

$$F_a = 1/2 \times \pi \times W_e/2 \times H_a \times \gamma, \qquad (2)$$

where $F_a$ = weight of rock within pressure arch per lineal meter (foot), kg/m (lb/ft),

and $H_a$ = height of pressure arch, m (ft).

The behavior of the pillar under different loading conditions determines $W_e$, shown in figure 3. Wilson defines the depth of the yield zone as the depth at which the coal strength in the entire pillar is exceeded by the loads imposed; therefore, Wilson's equations can be used to estimate the depth of this yield zone for rigid roof-floor conditions (equation 3) and yielding roof-floor conditions (equation 4) (*12*).

Wilson's equations are

$$w = 2\frac{m}{F}\ln\left[\frac{q}{p+p'}\right] \tag{3}$$

and

$$w = m\left[\left[\frac{q}{p+p'}\right]^{1/k-1} - 1\right], \tag{4}$$

where $w$ = pillar width, m (ft),

$F = \frac{k-1}{\sqrt{k}} + \frac{(k-1)^2}{k}\tan^{-1}\sqrt{k}$, where $\tan^{-1}\sqrt{k}$ is expressed in radians,

$m$ = seam height, m (ft),

$q$ = overburden load, $t/m^2$ ($st/ft^2$),

$p$ = artificial edge restraint, 0 $t/m^2$ ($st/ft^2$)

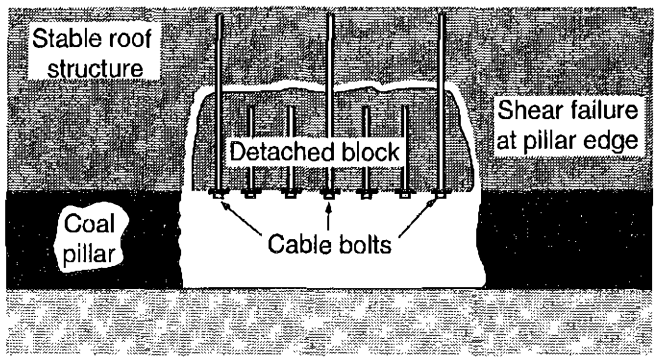
$p'$ = uniaxial strength of fractured coal, 1 $t/m^2$ ($st/ft^2$),

and $k$ = triaxial factor = $\frac{1+\sin\phi}{1-\sin\phi}$, where $\phi$ = angle of internal friction, deg. (5)

The depth of this yield zone can be calculated or approximated using the charts shown in figure 4. The charts were created using a value of 35° for the angle of internal friction. $W_e$ can then be calculated by using the following equation:

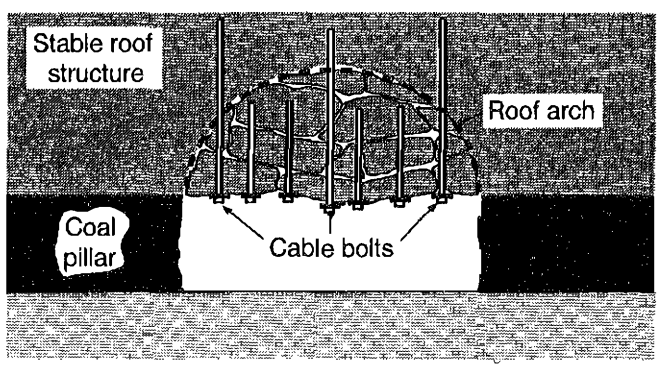
$$W_e = W + Y_{p1} + Y_{p2}, \tag{6}$$

*Figure 1*



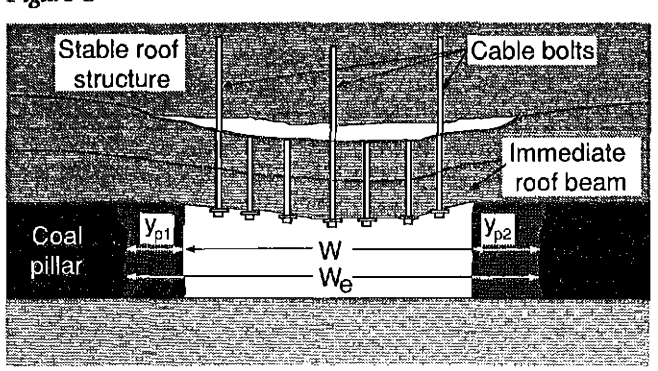

***Detached block failure supported by cables.***

*Figure 2*



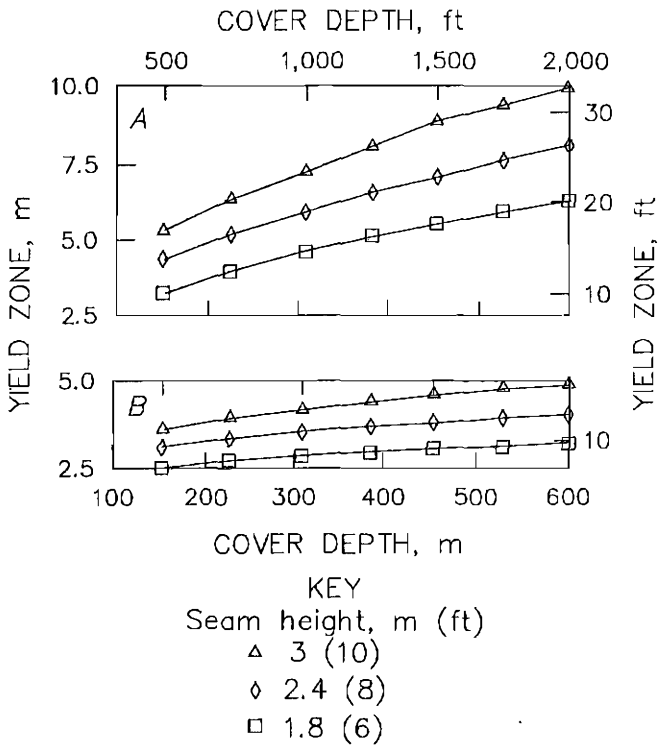

***Formation of pressure arch of failed mine roof material.***

*Figure 3*




***Formation of yield zone in coal pillars. ($W_e$ = effective width of opening; $y_{p1}$ = yield zone for pillar 1; $y_{p2}$ = yield zone for pillar 2; W = mined width of opening.)***

*Figure 4*




***Design chart to determine yield zone width in coal pillars. A, Yielding roof; B, rigid roof.***

where $W$ = mined width of opening, m (ft),

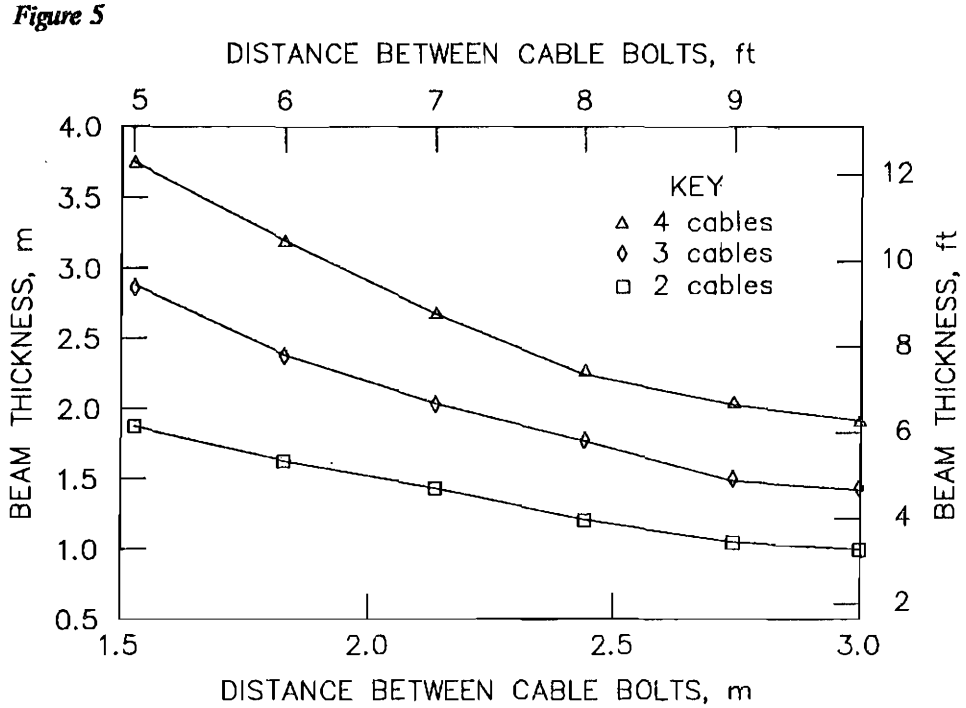
$Y_{p1}$ = yield zone for pillar 1, m (ft),

and $Y_{p2}$ = yield zone for pillar 2, m (ft).

Once the volume and weight of the material to be supported with cables has been determined, it is then possible to determine the number and spacing of the required cables to support the gate road entry. Using a cable capacity of 258.0 kN per cable (58,000 lb per cable) and varying the number of cables across the opening, the best design can be determined for a specific application and operation.

Figure 5 shows a design chart calculated for two, three, and four cables installed for $W_e$ of 7.6 m (25 ft) and a material weighing 2,403 kg/m³ (150 lb/ft³). For example, drawing a line up from the x-axis, spacing distance along the entry to the cable number line, and going left from that point to the y-axis indicates the thickness of a failed beam member that can be entirely supported with the installed cables. In the example, two, three, and four cables spaced across the width, at 2.1-m (7-ft) spacing along the entry, would have the capacity to support 1.4, 2.0, and 2.7 m (4.5, 6.7, and 8.9 ft) of separated material, respectively.

*Figure 5*




***Cable support design chart, where effective width of opening ($W_e$) = 7.6 m (25 ft) and rock density ($\gamma$) = 2,403 kg/m³ (150 lb/ft³). For example, a 2.1-m (70-ft) cable bolt spacing along entry could support 1.4, 2.0, and 2.7 m (4.5, 6.7, and 8.9 ft) of separated material using two, three, and four cables, respectively.***

## INSTALLATION METHODS

Cable supports have been successfully installed using both the "traditional" cement grout and resin grout. Both systems have advantages and disadvantages, but a close examination of both systems indicates that resin-grouted cable bolting is superior for most coal mine applications from a productivity and cost standpoint. However, there may be circumstances when a cement cable system would be preferred. For example, if large voids or washout of the roof rock was present, the concrete would completely fill the voids and develop the required anchorage, whereas the resin would be lost in the voids and may not develop enough anchorage to develop the strength of the cables. Another example of when cement grout would be preferred is when a full column of grout is used to increase the system stiffness. The volume of resin is currently constrained by the diameter of the tubes, and obtaining a full column of resin is extremely difficult.

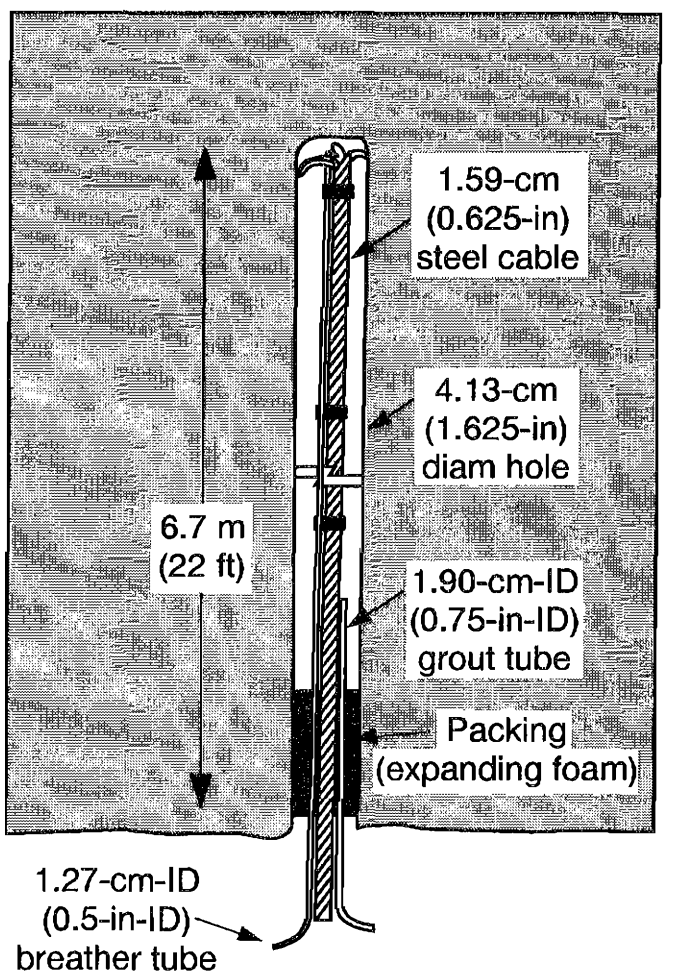
### CEMENT GROUT INSTALLATION PROCEDURES

Cement-grouted cable bolts can be installed at any angle in the rock. To install the cables with concrete grout, the following steps are taken:

1. A hole is drilled, with a diameter of 4.1 cm (1-5/8 in), to a depth of at least 5.1 cm (2 in) deeper than the desired cable length.
2. The cable, with the appropriate retaining anchor, plastic breather tube, and grout tube, is inserted into the hole. The breather tube is almost as long as the cable and allows the air being displaced by the grout to escape. Also, grout running out of the tube indicates to the cable bolt crew that the hole is filled.
3. Water is sent through the breather tube to flush the hole and clear debris from the breather tube.
4. A plastic grout tube is pushed approximately 45.7 cm (18 in) into the hole.
5. The bottom 20.3 cm (8 in) of the hole is plugged, sealing the area around the cable, grout, and breather tubes. This can be accomplished by stuffing shredded cotton waste material around the tubes or using an expansive foam. The combination of both provides an excellent seal.
6. The hole is then filled with a cement-based grout through the grout tube. The grout commonly used consists of a water and cement mixture at a ratio of 0.35 part of water to 1 part of cement by weight. Several laboratory studies have been completed that investigated the effects of water-to-cement ratios. The strength selected is site specific and related to the available pumping equipment. The greater the cement-to-water ratio, the higher the final viscosity. This can be partially overcome by adding a chemical plasticizer, which makes the grout slicker and easier to pump without adversely impacting the final strength.
7. After the hole is filled, the ends of the two tubes are folded over and tied off to prevent the grout from draining.
8. The next day, or after approximately 24 h, the tubes can be cut off to allow the installation of a bearing plate and cable grips. A hydraulic cable jack can be used to tension the cable to the desired preload condition.

A completed cement installation and the required components, without the bearing plate and the cable grips, are shown in figure 6.

*Figure 6*




*Components of concrete-grouted cable support. (ID = inside diameter.)*

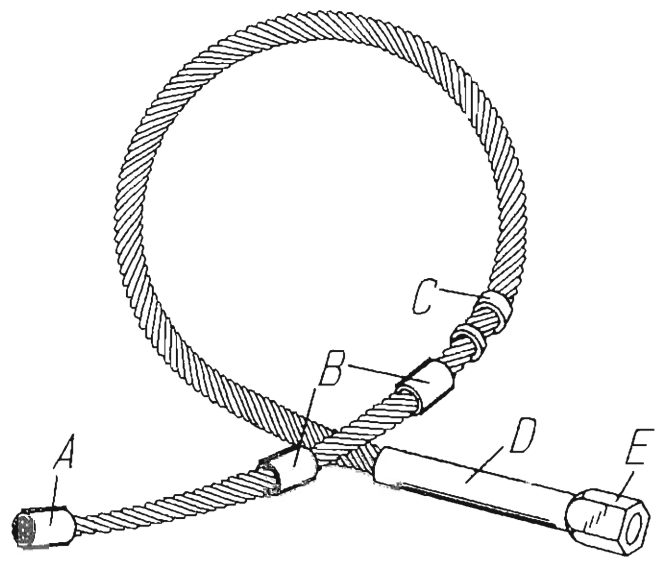
## RESIN GROUT INSTALLATION PROCEDURES

Resin-grouted cable bolting was initiated in the United States in 1992. Several required installation parameters were identified to make cable bolting with a resin anchorage system as routine as headed rebar. Numerous design evolutions were investigated before resin-grouted cable bolts were fabricated on a production level. An example of the cable bolt used in several USBM investigations is shown in figure 7. Each component serves a specific function that contributes to the overall success of the cable bolt system.

The end of the cables are clamped together using a swedged-on fitting, as shown in figure 7*A*. This ties all the cable strands together, including the king wire, which is the center cable strand. The next swedged-on buttons serve two functions (figure 7*B*). First, they provide additional anchorage and resistance to pullout forces. It is extremely difficult, if not impossible, to pull the cable out of the resin when these buttons are imbedded in the borehole. Secondly, the buttons help mix the resin during placement by forcing the resin around the tight opening; the turbulence enhances the mixing and helps to detach the cartridge cover. The short button (figure 7*C*) generally pserves the same functions as the large button and also holds a plastic seal (dam) to restrict the flow of resin down the hole. Laboratory and field results indicate that keeping the resin at the top of the hole can be the difference between adequate and inadequate anchorage for resin-grouted cables. Any resin loss for a critical length of anchorage may allow the cable to pull out of the hole before developing ultimate strength. Because the cables are flexible, they may bend when the back pressure from the resin becomes too high, causing the cable to bend or kink. The cable stiffener (figure 7*D*) provides stiffness to the bottom portion of the cable to ease the installation of the last 1.2 to 1.5 m (4 to 5 ft) of cable. The length of the stiffener can be adjusted to the mining height and the effective resin column length desired. It eases installation to have the stiffener in the hole before any back pressure causes the cable to bend or kink. Additionally, during the process of mixing the resin, the bearing plates, placed on the cable before installation, tend to spin rapidly. Field observations indicate that this spinning can cause the bearing plate to nick the cable, which may lead to premature failure. The cable stiffener eliminates cable nicking. The final component of the resin-grouted cable bolt system is the installation head or cable nut (figure 7*E*). This makes the cable easy to rotate and install with conventional roof bolting machinery. The nut is capable of handling loads in excess of the ultimate bearing capacity of the cable.

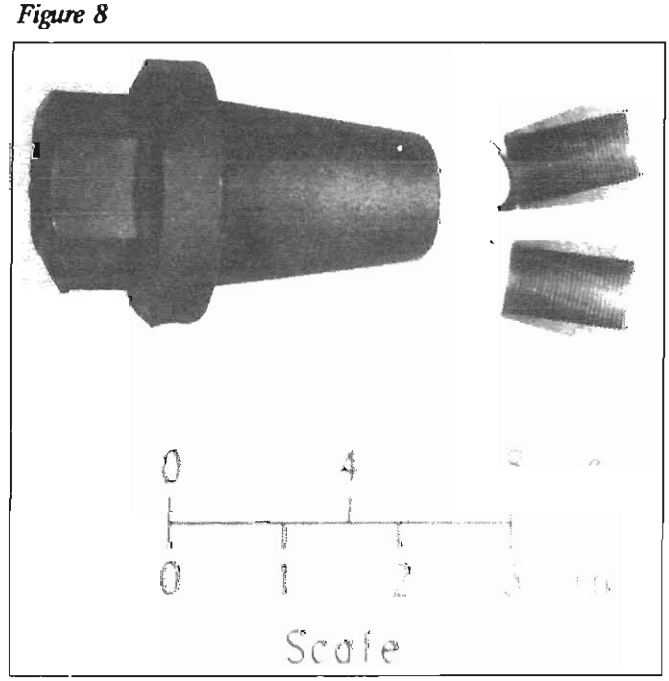
Several variations of this system are becoming available. One system of particular interest allows tension to be applied to the cable after the resin has cured. This component is shown in figure 8. High degrees of tension may be an important consideration in a laminated roof material where any separation may lead to progressive-type failures. If a high-tensioned system is desired, it is important to realize that the resulting ultimate support capacity is

*Figure 7*




***Specific components of resin-grouted cable bolt. A, Swedged-on fitting to tie all cable strands together; B, swedged-on buttons for additional anchorages; C, short button to hold plastic seal in place; D, cable stiffener rod; E, installation head or cable nut.***

*Figure 8*

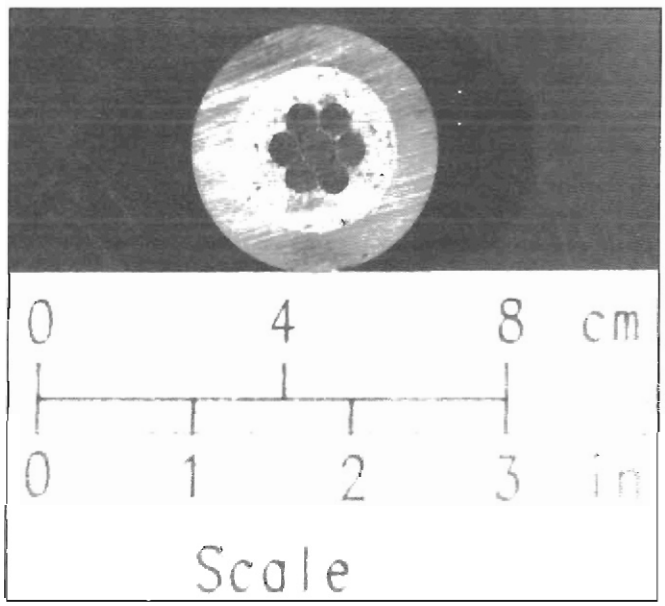



***Tensionable nut for resin-grouted cable bolt.***

lowered by the tensioned amount. For example, if the system is pretensioned 44.5 kN (10,000 lb), the remaining capacity of the system to support roof loads is 213.5 kN (48,000 lb). To install the cables with resin grout, the following steps are taken:

1. Drill the prescribed hole 2.5 to 5.1 cm (1 to 2 in) longer than the cable to be installed. The holes can be drilled with a water or vacuum drilling system. Hole diameters ranging from 2.5 to 3.5 cm (1 to 1-3/8 in) have been used successfully.
2. Place the resin cartridges into the borehole. An installation technique that appears to be working well, especially in the case where more than one cartridge of resin is required, is the placement of a faster setting cartridge at the top of the hole. This permits fast installation, instantaneous anchorage, and immediate support.
3. As the cable is pushed up through the resin cartridges, it is rotated slowly to enhance the mixing of the resin. When the cable is approximately 7.6 to 10.2 cm (3 to 4 in) from the back of the hole, the rotation is increased and the resin is mixed the total amount of time recommended by the manufacturer. (Care should be taken not to overspin the bolt.) The mixing time begins when the cartridges are punctured by the insertion of the cable. Field investigations have revealed that the cable should be rotated counterclockwise. This tends to screw the cable into the hole, getting a positive contact between the bearing plate and the roof. Rotating the cable clockwise will pull up the resin, creating back pressure after the resin has been mixed and the cable is pushed up against the roof. When the bolting stinger is removed, the cable relaxes and pushes out of the hole, voiding any plate roof contact that may allow separation and progressive failure to occur.
4. When the resin has been adequately mixed, the cable is pushed up against the roof with the full force of the bolter and held in place until the resin has cured. This provides an active bearing plate pressure and pretensions the cable (*13*).
5. If a tensionable unit has been installed, the cable is then ready for jacking to a predetermined load, using special equipment.

*Figure 9*




***Cross section of 1.52-cm (0.60-in) diam cable installed in 2.5-cm (1-in) diam hole.***

A cross section of a 1.52-cm (0.60-in) diam cable is shown in figure 9. The resin is tight against the six outer strands and there are traces of resin around the center strand. This demonstrates how resin grout provides an excellent mechanical interlock, creating a high degree of cable anchorage.

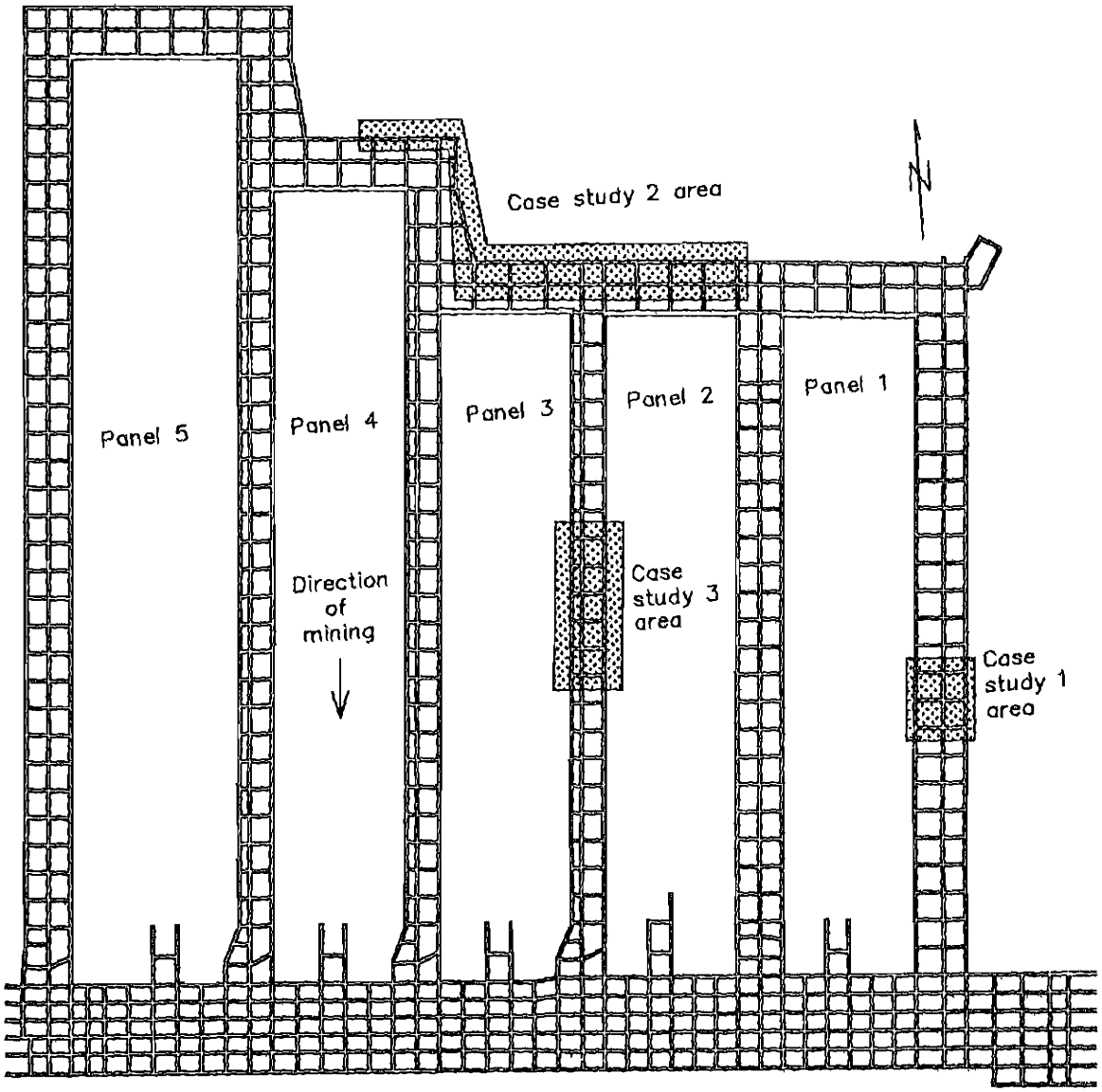
## FIELD EVALUATIONS

Several long-term field evaluations were undertaken at various locations in the West Elk Mine. In each of the three case studies, the cables were being used as secondary support systems in gate roads and bleeder entries. Case study 1 involved cement-grouted cables in a bleeder and gate road entry; case study 2 involved advancing to resin-grouted cables installed in a bleeder entry system; and case study 3 was an evaluation of resin-grouted cables as the sole means of secondary support in a longwall tailgate entry. Figure 10 shows the general mine layout with the location of the three case study areas.

### CASE STUDY 1

Cable supports were installed to assess their effects on the stability of tailgate entries in an underground Western U.S. coal mine. Two entries closest to the longwall panel were supported with high-strength cables installed with cement grout. Instrumentation was used to monitor and assist in the evaluation of the effectiveness of the support system and the general response of the mine openings.

*Figure 10*




***General mine layout showing case study locations.***

## Site Description

The longwall gate roads and the test area are shown in figure 11. The final pillar dimensions for the pillar nearest the panel were 39.6 by 39.6 m (130 by 130 ft), and the pillar between entries 2 and 3 was 39.6 m (130 ft) long and 30.5 m (100 ft) wide. Basically, with modifications in the intersections, 109.7 m (360 ft) of roadway nearest the longwall panel, entry 1, was supported with 6.7-m (22-ft) long cables installed on a 2.1-m (7-ft) spacing. Approximately 76.2 m (250 ft) of the second entry, longwall bleeder entry 2, was supported with 5.5-m (18-ft) long cables installed on a 2.1-m (7-ft) spacing, as shown in figure 11. The difference in cable length was attributed to the different loading conditions that the supports would be subjected to as the panel was extracted. The primary support system in the area, installed on initial development, was 1.8-m (6-ft), full-column, resin-grouted bolts on a 1.5-m (5-ft) pattern installed with pans and complete wire meshing.

An examination of the mining horizon indicated 4.2 m (14 ft) of minable coal seam, but the gate road entries were driven only 2.9 m (9.5 ft) high. A generalized summary of the immediate roof in the area included about 1.1 m (3.5 ft) of top coal overlain by about 0.45 m (1.5 ft) of a silty shale. The roof above the silty shale grades vertically into interbedded units of siltstone, shale, and sandstone. This unit coarsens upwards in grain size, and the top portion consists of sandstone. The thickness of the sandstone in the test area was approximately 4.9 m (16 ft) thick. A geological column of the test area, obtained with a borehole television camera, is shown in figure 12.

## Instrumentation

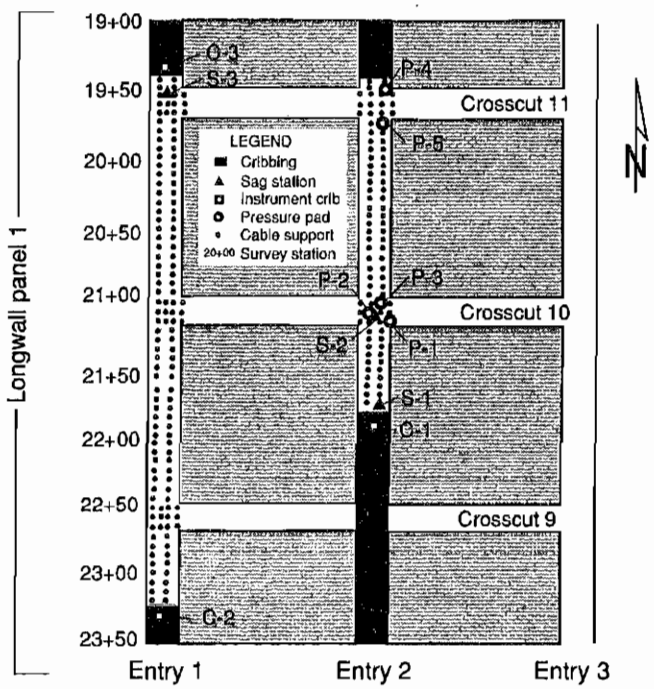
Individual cable loads were monitored with hydraulic U-cells and Goodyear pressure pads. The mine roof behavior was simultaneously monitored with differential magnetic sag stations and closure meters on 7.6-m (25-ft) spacings along the entry axis during the development of the longwall panel to evaluate the response of the immediate and main roofs. Additionally, hydraulic flatjacks and closure gauges were installed on wooden cribs on the inby and outby sides of the test areas to determine the crib stiffnesses.

## Test Site Results

The test area was installed 229 m (750 ft) outby the longwall panel face to ensure that the instrumentation and cables had properly stabilized and cured before mining. The last crib (C-3) set to maintain abutment and tailgate loads was installed near station 19+35. Hydraulic flatjacks were installed between the 20.3- by 20.3-cm (8- by 8-in) timber blocks, and the closure of the crib was monitored to establish the in situ stiffness of the crib supports and to determine a baseline for the amount of load the cables would be subjected to during panel extraction. The cribs remained virtually unloaded until the face was approximately 15.2 m (50 ft) inby the crib. The crib was subjected to a calculated load of approximately 138 kN (31,000 lb) before becoming inaccessible for subsequent readings. The stiffness of the crib changed as a function of time and subsequent increases in loading. Figure 13 shows how the calculated stiffness started at 126 kN/cm (72,000 lb/in) and then decreased to a final value of about 61 kN/cm (35,000 lb/in). This clearly indicates that timber supports are a soft system and do not provide resistance to subsequent main roof loading.

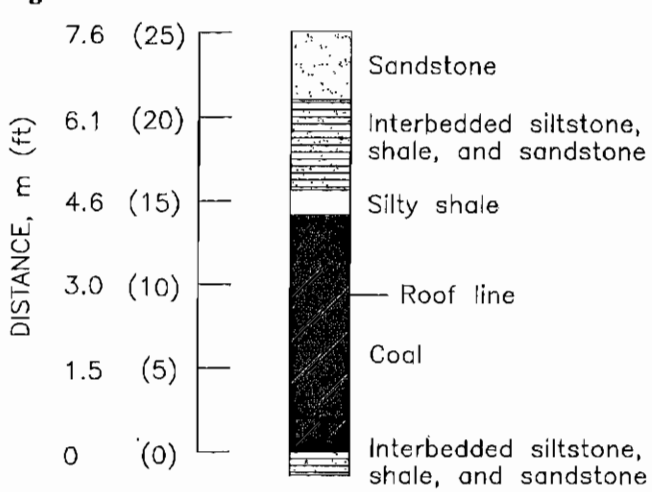
The roof movement was carefully monitored using magnetic anchors, established at various locations of the test area at approximately 0.61-m (2-ft) intervals along the length of a 3.7-m (12-ft) borehole. Sag station 3, located in entry 1 of the tailgate (figure 11), is the only one that indicated bed separation and subsequent roof movements. The sag station was located at approximately construction station 19+48. The roof strata remained stable until the panel face was approximately 2.4 m (8 ft) inby. The immediate roof then began to show signs of movement and separation. The largest separations occurred between the 0.61- and 1.22-m (2- and 4-ft) anchors, the top coal and

*Figure 11*




*Gate roads showing cement-grouted cable test area and instrumentation layout at a Western U.S. underground longwall mine.*

*Figure 12*




*Geological column from test area.*

shale interface, and the 1.8- to 2.4-m (6- to 8-ft) anchors, the transition zone between the interbedded siltstone and the sandy shale. The roof between the 2.4- and 3.0-m (8- and 10-ft) levels experienced minor separation. Individual anchor displacements and an accumulation of the total displacement are shown in figure 14. The entire roof was subjected to approximately 5.7 cm (2.25 in) of total separation over a 3.6-m (12-ft) length.

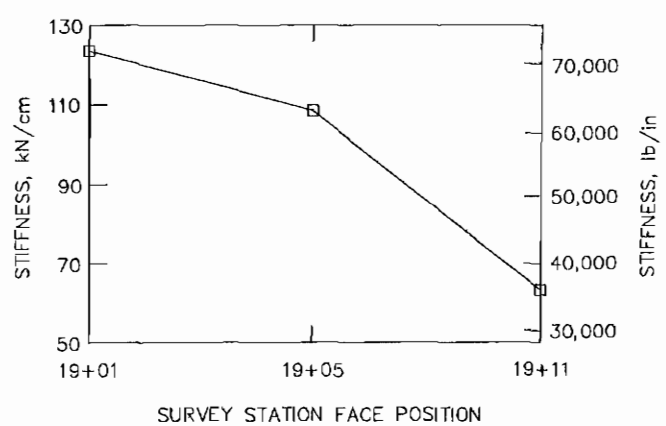
Two pressure pads were located near the intersection in crosscut 11 of bleeder entry 2, shown in figure 11. Again, the installed loads remained constant until the face was outby approximately 4.6 m (15 ft). As the panel was mined to station 20+88, approximately 36.6 m (120 ft) outby crosscut 11, pressure pad P-5, near the right rib, loaded to about 28.9 kN (6,500 lb), and then maintained that load for the duration of the test. Pressure pad P-4, located in the intersection in crosscut 11, experienced only minor changes, 2.7 kN (600 lb), during the entire panel extraction. The behavior of pads P-4 and P-5 is shown in figure 15.

The loading behavior of the pressure pads located in bleeder entry 2 at crosscut 10 (figure 16) showed a profile that included minor roof dilation until the face was approximately 15.2 m (50 ft) inby the crosscut. The cables then loaded approximately 4.5 kN (1,000 lb), and then unloaded as the face was mined past the intersection and the abutment loads and pressures dissipated. The differential sag stations in the same area indicated only a minor separation, 0.381 cm (0.15 in), between the coal and siltstone interface.

Closure meters were positioned at 7.6-m (25-ft) intervals along the length of entry 1. The data were fairly consistent; an example is shown in figure 17. The entry showed only minor signs of closure, less than 1.27 cm (0.5 in), until the face was approximately at 23 m (75 ft) inby. At that time, the closure increased to about 5 cm (2 in). This pattern remained fairly consistent unless the face sat idle for extended periods of time. The data indicated that the closure, 15.2 m (50 ft) outby the face, exceeded a total of 10.2 cm (4 in) when the face sat idle for more than 2 days. Another interesting observation in the data was the roof response, consistently recorded between closure stations up to 15.2 m (50 ft) apart. As shown in figure 17, the closure on inby station 20+00 corresponded to an upward movement at station 20+50. This occurred when the longwall face sat idle for an extended period of time and may be related to time-dependent deflections of the immediate roof.

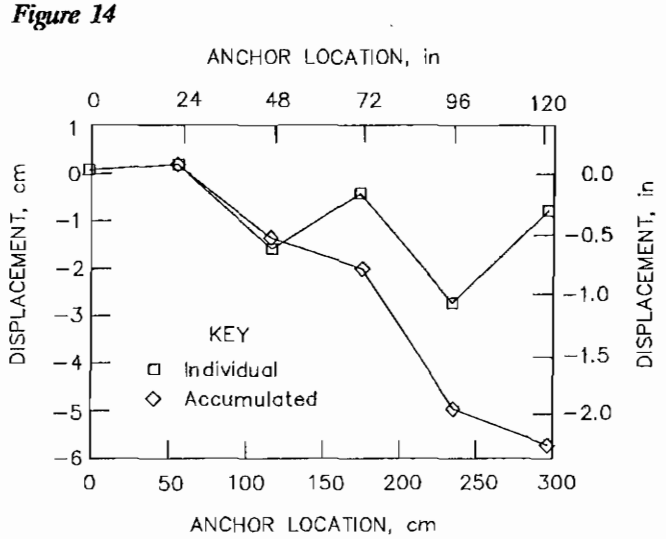
As the panel was extracted to approximately construction station 19+98, excessive forward abutments and side loading caused two cables installed near the pillar side of entry 1 to fail at a height of approximately 2.7 m (9 ft). Cable anchorage was strong enough to cause the cable to fail about 0.3 m (1 ft) below the fall of ground. The fall extended at an angle to construction station 20+18, about 6.1 m (20 ft) in front of the face. Face ventilation was never disrupted, and the panel was mined through the fall area without major additional incidence. The mine operator elected to provide additional supplementary support in the form of two 20-cm (8-in) diam timber posts, from station 21+50 to station 22+50, spaced on 1.5-m (5-ft) centers for the remainder of the test area. Five timber posts were instrumented with hydraulic flatjacks to assess their contribution to the total support of the entry. Again, at a distance of about 23 m (75 ft) outby the face, the posts began taking load. When the face was approximately 6.1 to 9.1 m (20 to 30 ft) inby, the instrumented posts were carrying an average of 48.9 kN (11,000 lb), which translates to about 1,550 kPa (225 lbf/in²) on the timber post. When the face was directly adjacent to the posts, the load reached approximately 125 kN or 3,860 kPa (28,000 lb or 560 lbf/in²) and then the posts failed.

*Figure 13*




***Crib stiffness at different intervals of face advance.***

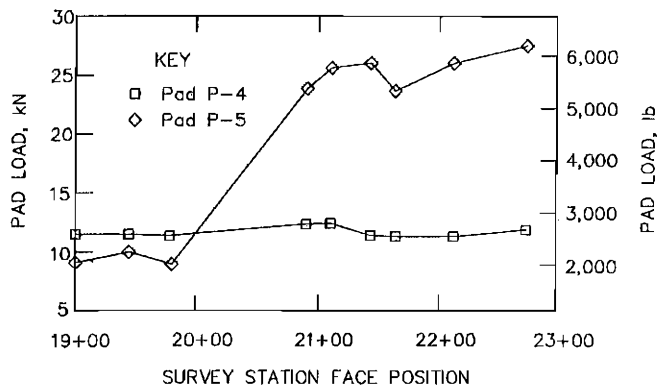
*Figure 14*




***Individual and accumulated roof anchor displacements.***

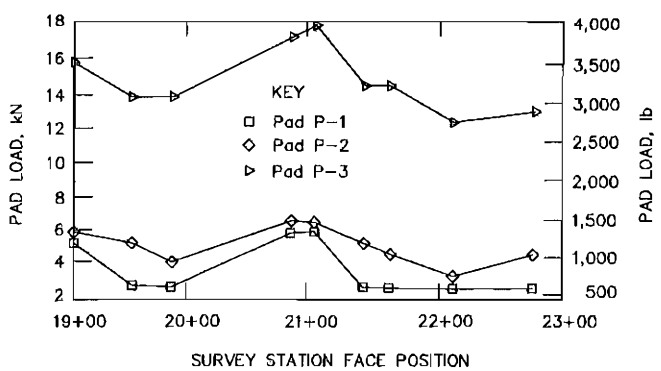
The cable test area in entry 1 was successfully mined through without further incidence of caving or roof falls ahead of the face. Bleeder entry 2 showed only minor signs of loading, and differential roof sag stations indicated insignificant movements or separation in the roof.

*Figure 15*



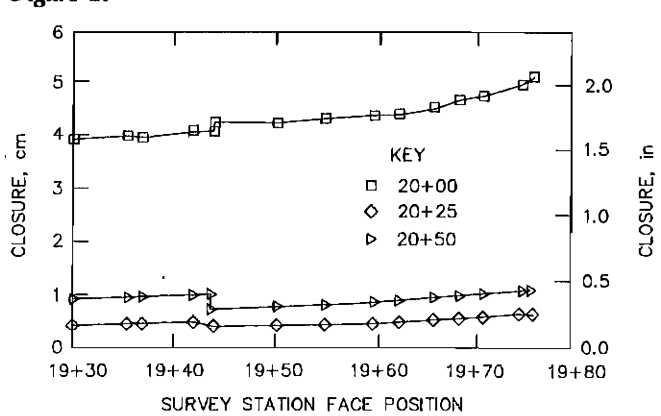

***Pressure pad loading behavior with face advance in crosscut 11.***

*Figure 16*




***Pressure pad loading behavior with face advance in crosscut 10.***

*Figure 17*




***Entry closure and roof response.***

## Discussion

Given the pillar layout and site-specific roof conditions and strengths at the test mine, cable bolt systems were successfully used to maintain the longwall tailgate entries. The installed instrumentation indicated that very little active support can be expected from installed crib systems. Even with the stiff cable support system, differential magnetic sag stations indicated that roof bed separations still occurred between geological interfaces, which eventually led to loading beyond the designed capacity. Timber posts, installed as additional secondary support in the tailgate entry, were subjected to loads of approximately 125 kN (28,000 lb) and then subsequently failed. The loading on the timber posts, while appearing substantial, could have been supported by an additional cable installed in the roof of entry 1. Adding another cable with a 258-kN (58,000-lb) capacity in the middle of the entry may have compensated for these extra loads. A review of the design chart shows that a three-cable system, spaced on 2.1-m (7-ft) centers, would have been sufficient to carry the loads generated by the observed roof separation at a height of 9 ft above the coal seam.

## CASE STUDY 2

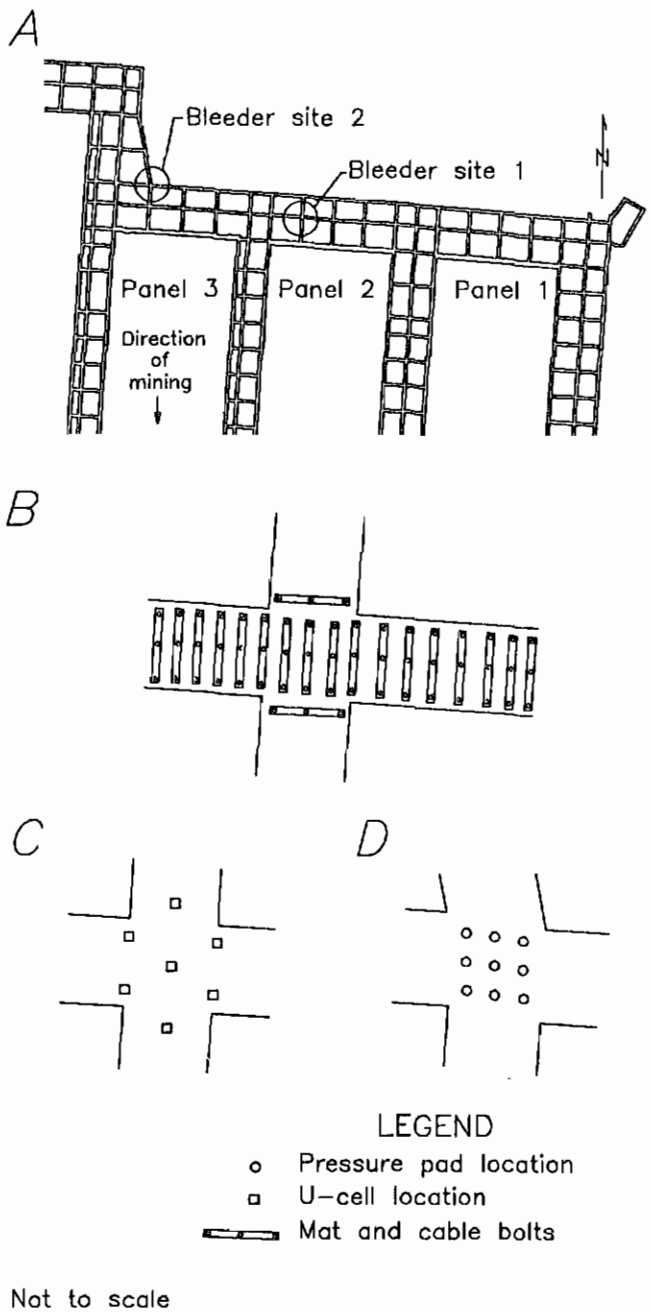
The first resin-grouted cable test area was established in a bleeder entry system immediately behind a longwall panel. A longwall bleeder system is a designated set of special entries developed and maintained as part of the mine ventilation system. These entries are designed to continuously move air-methane mixtures from the gob, away from the active workings, and deliver the mixtures to the return air courses. Because these systems are critical to safe ventilation and provide emergency escapeways, they must be maintained to permit adequate airflow and travel. To accomplish this, most mines elect to install crib supports along the entire length of the bleeder entry system. As an alternative secondary support system, approved by the U.S. Mine Safety and Health Administration (MSHA), resin-grouted cables were selected to support the bleeder entry.

### Site Description

A three-entry system was driven behind the longwall system. The first entry was used as the longwall setup room. The second set of entries were the bleeder entries. The two entries were separated by pillars that measured 57.9 m (190 ft) in length and 45.7 m (150 ft) in width. The 6.1-m (20-ft) wide entries were primarily supported with 2.4-m (8-ft) full-column, resin-grouted bolts and steel mesh. The third set of entries, adjacent to virgin coal, resulted in a final pillar dimension of 57.9 m (190 ft) in length and 47.5 m (150 ft) in width. An examination of the mining horizon indicated 4.3 m (14 ft) of minable coal seam, with the entries driven only 2.9 m (9.5 ft) high. A

generalized summary of the immediate roof in the area included about 0.8 m (2.5 ft) of top coal overlain by about 0.5 m (1.5 ft) of a silty shale. The roof above the silty shale grades vertically into interbedded units of siltstone, shale, and sandstone. This unit coarsens upwards in grain size, and the top portion consists of sandstone. The test area location is shown in figure 18.

*Figure 18*




***Bleeder entry configuration and resin-grouted cable test area. A, Location of bleeder sites; B, general layout of cable bolt supports in bleeder entries; C, bleeder site 1 detail; D, bleeder site 2 detail.***

Mine management decided to move the bleeder entry to the third entry for panel 4. The intersection for this transfer is where bleeder site 2 is located. The general arrangement of the cable bolt supports in the bleeder entries is shown in figure 18*B*, and the details of the two bleeder instrumentation sites are shown in figure 18, *C* and *D*. Cable supports were installed in the second entry along the entire width of panels 2 and 3 and in the third entry behind panel 4.

## Cable System Design

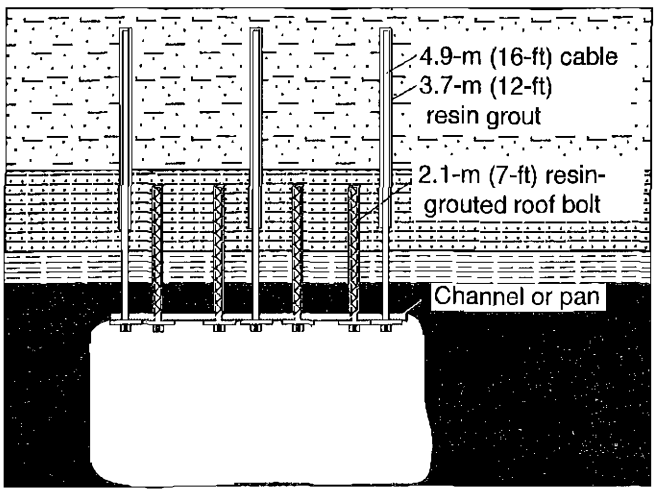
Based on the caving results from a previous panel, the site lithology, and the loading generated on the crib supports in the previous bleeder, a cable system was designed to support the loads generated on the roof when the panel was extracted and the immediate and main roofs caved. The entries were protected by the abutment-size pillars, but the transfer of stress over those pillars warranted secondary support to ensure an adequate ventilation and escapeway entry. It was believed that if separation did occur, it would most likely be in the layers of silty shale that occurred about 1.2 m (4 ft) up from the roof. Additionally, the interbedded siltstones and shales could separate if the abutment forces became large enough or the immediate roof was lost. Yield zones on the pillars, both calculated and observed in the mine, indicated that the effective roof span would be approximately 7.9 m (26 ft). With these facts, the mine operator elected to install a 4.9-m (16-ft) long cable, which would intersect the sandstone 1.8 m (6 ft). The cables were installed with 3.7 m (12 ft) of resin grout to ensure a strong cable anchorage and also to help hold the lower silty shale member intact. The design specified three cables across the entry on a 1.5-m (5-ft) spacing. The cable supports would support a complete roof separation—the worst case scenario—if it occurred at a depth of 2.7 m (9 ft). A cross section of the support system is shown in figure 19. In addition to 15.2- by 15.2-cm (6- by 6-in) bearing plates, the cables and primary support were also installed in conjunction with Monster Mats.[5] The mats provide excellent roof support between the cables and maintain any failed material. Mats help prevent unraveling or progressive roof-type failures.

## Instrumentation

Hydraulic U-cells and Goodyear pressure pads were installed throughout the test areas on individual cables to determine the actual loading that occurred during various phases of panel extraction. Differential sag stations were installed to monitor any roof separations that occurred in the first entry to establish the expected failure surface and

[5]These mats are 0.48-cm (3/16-in) thick steel pans that are 35.6 cm (14 in) wide and 4.9 m (16 ft) long, manufactured by Rocky Mountain Bolt Co., Salt Lake City, UT.

*Figure 19*




***Cross section of test area showing primary and cable support spacing and resin grout lengths.***

to estimate the volume of roof material that may have to be carried by the cable supports.

## Test Site Results

The test areas were monitored and evaluated 14 times throughout panels 2 and 3 mining. Panels 2 and 3 have now been completely extracted. The cables installed behind panel 2 have been subjected to increased loads as panels 2 and 3 mining created large spans of unsupported ground. Data from bleeder site 1, shown in figure 20, indicate the load profiles for the cable-supported intersection during panel 2 and part of panel 3 mining. The initial measurements, shown in figure 20*A*, recorded approximately 35 days after the entry was supported with cables but before panel 2 extraction began, show signs of general loading across the middle of the intersection. More load was observed on the cables closest to panel 2. When panel 2 had been mined to a distance of 337 m (1,105 ft) from the setup room, shown in figure 20*B*, the loading pattern was nearly identical; overall load on the cables increased, with the largest loading still on the cables toward panel 2. The main cave for panel 2, as reported by the miners, took place when the panel 2 face had been extracted 37.5 m (120 ft). The cable loads in figure 20*B*, averaging 16.6 kN (3,730 lbf), represent the effects of main roof loading from panel 2 mining transferring to the bleeder abutment pillars. Figure 20*C* shows the loading profile at bleeder site 1, resulting from the complete extraction of panel 2 and 365 m (1,198 ft) of panel 3. Again, the load pattern across the intersection is similar to that in figure 20, *A* and *B*, and the cable loads increased slightly during the initial mining of panel 3. Loads measured on individual cables ranged from 14.7 to 149.0 kN (3,295 to 33,500 lbf), averaging 18.8 kN (4,225 lbf) for the seven hydraulic U-cells. Differential sag stations placed in the roof indicated very little separation in the interface between the shale and sandstone. The entire bleeder entry behind panel 2 remained extremely stable, with no roof falls and very little signs of roof spalling or separation. Additional recorded cable loads varied throughout the entire panel 2 bleeder area, with higher readings mostly related to geologic features that occurred in the cable-supported area.

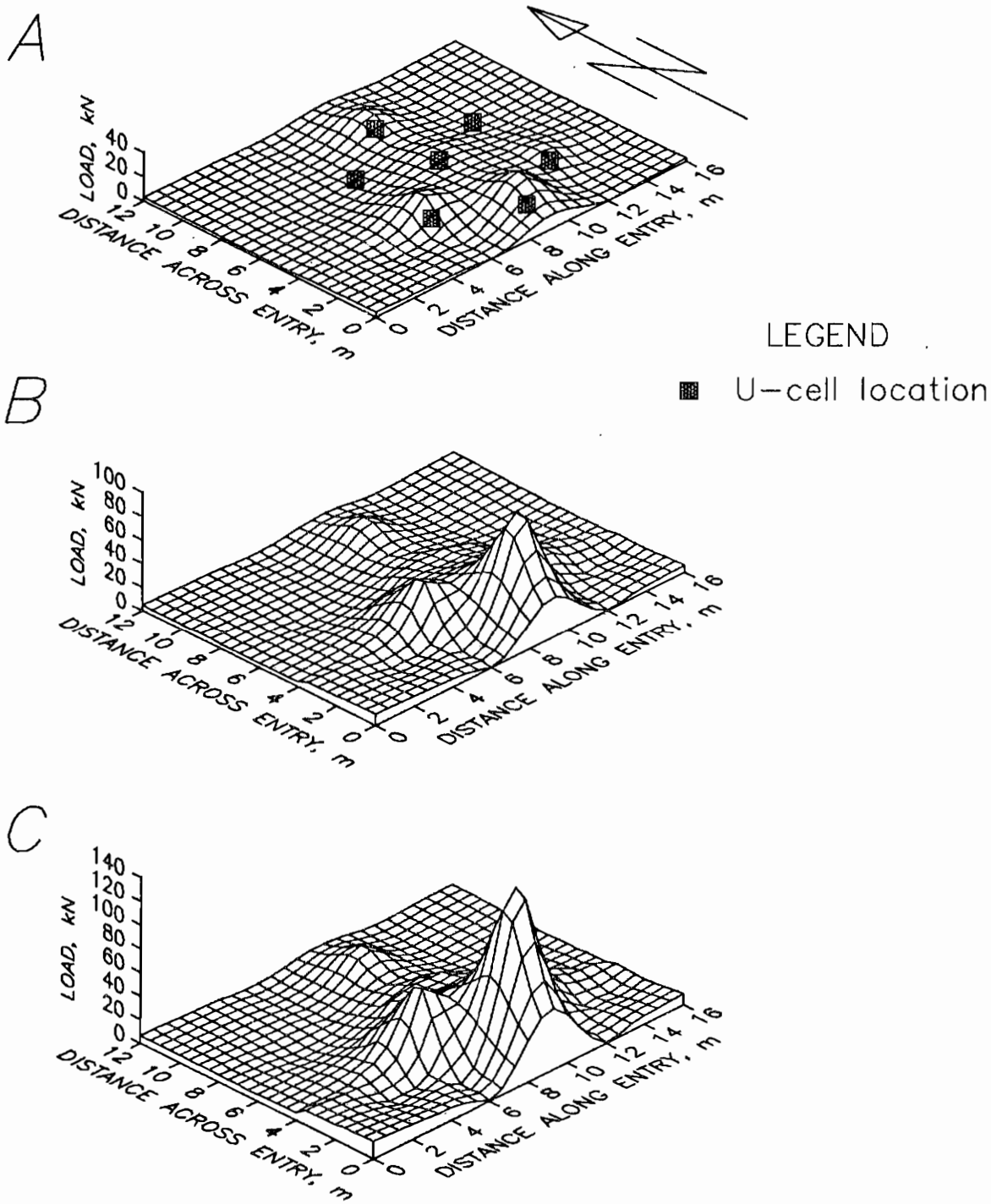
The behavior in bleeder site 2 area was very similar to the behavior of bleeder site 1. The main difference was that loading on the test area occurred shortly after panel 3 started mining and then stabilized as if the system had reached equilibrium, with the cable supports experiencing no additional load increases. The initial cable loads and the subsequent loads recorded when the panel 3 face had been extracted to approximately 91 m (300 ft) and 365 m (1,198 ft) are shown in figure 21. The pillars in this area showed more rib sloughage and spalling than the area behind panel 2. However, the roof remained competent with no visual signs or recorded loads that would indicate unanticipated separation. The loads measured on the eight Goodyear pressure pads in bleeder site 2 ranged from 18.0 to 119.2 kN (4,120 to 26,800 lbf) and averaged 73.8 kN (16,600 lbf). Subsequent measurements recorded in the panel 3 bleeder entry system showed little or no cable bolt load increase as panel 3 was extracted. The results from both bleeder test areas demonstrated the effectiveness of cable supports as an alternative to other forms of secondary support. Ground conditions remained good in the bleeder entry during the mining of two complete longwall panels, and none of the cable loads exceeded 50% of failure strength. The vast majority of the cables did not exceed 10% of the ultimate cable load capacity.

## Discussion

Results from this bleeder entry cable support study demonstrated the effectiveness of cable supports with resin-grouted anchorage systems. The bleeder entry test area at the study mine, utilizing cable supports as the sole means of secondary support, experienced the effects from two longwall panels with very good results, no roof problems, and good bleeder entry conditions. A couple of areas in the bleeder entry experienced significant loading. Still, the cable and Monster Mat[6] arrangement maintained the roof integrity while keeping the entry clear of crib material.

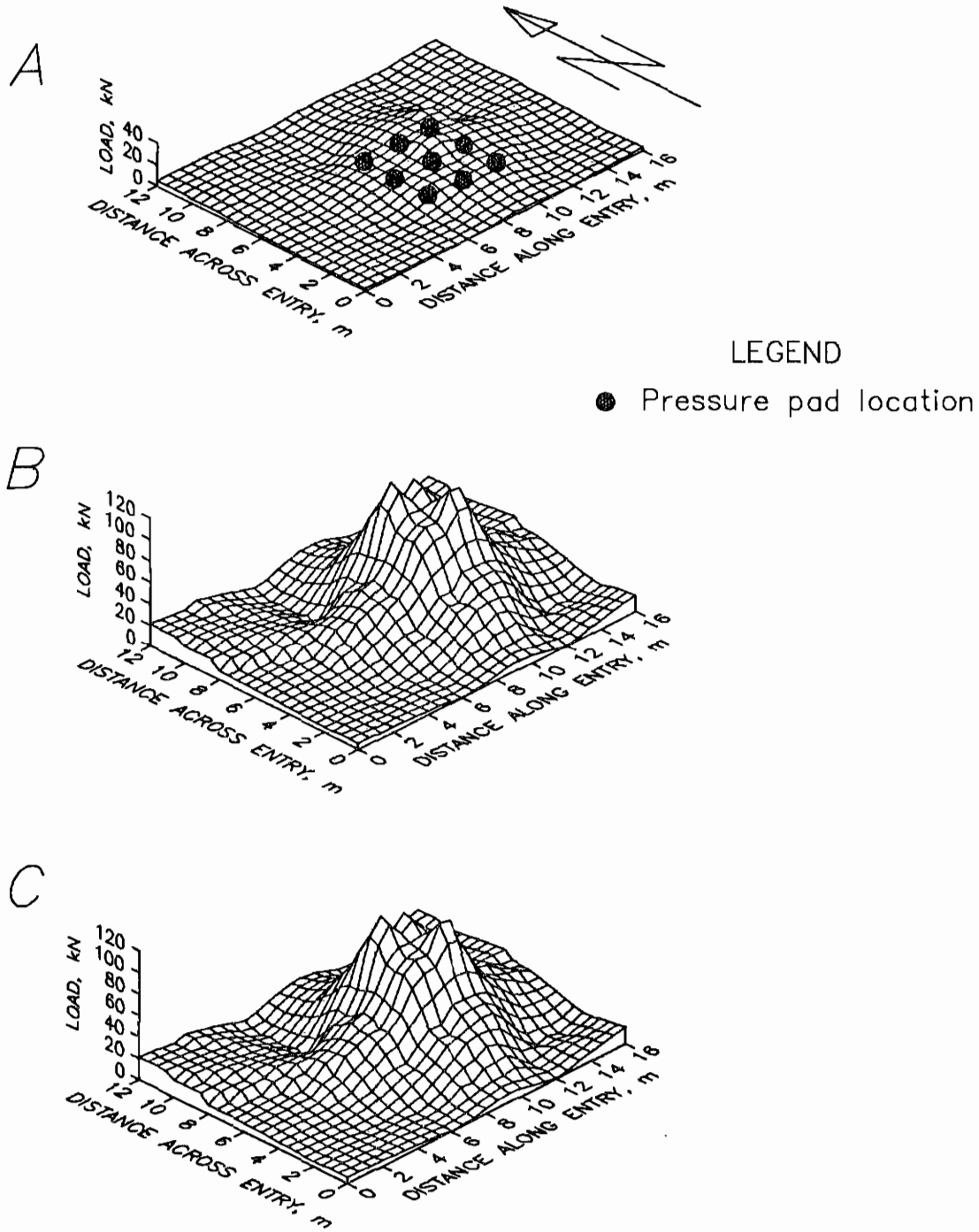
[6]See footnote 5.

*Figure 20*




***Bleeder site 1 profiles during panels 2 and 3 mining. A, Panel 2 face position 0+00 with U-cell instrument locations; B, panel 2 face position 11+05; C, panel 3 face position 11+98.***

*Figure 21*




***Bleeder site 2 profiles during panel 3 mining. A, Panel 3 face position 0+00 with pressure pad instrument locations; B, panel 3 face position 3+00; C, panel 3 face position 11+98.***

## CASE STUDY 3

This last case study describes the most aggressive attempt at installing and evaluating resin-grouted cables for secondary support in a longwall gate road. Cable support systems were designed and installed to provide stability in a gate road that will be utilized for two longwall panels, first as a headgate and then a tailgate.

### Site Description

The instrumented test area, approximately 274 m (900 ft) long, was initially supported with 2.1-m (7-ft) full-column, resin-grouted bolts. The three-entry system is utilizing a yield-abutment pillar configuration to minimize the pressure on the entries when they become tailgates and reduce the possibility of coal mine bumps or burst. The final yield pillar dimensions, next to the panel during the second panel mining phase, measure 9.8 m (32 ft) wide and 39.6 m (130 ft) long. The final abutment pillar dimensions, designed to absorb the first panel abutment stresses and protect the integrity of the yield pillar, measure 30.5 m (100 ft) wide and 39.6 m (130 ft) long. The geology was similar to the bleeder test area with one minor exception—the competent sandstone layer was located about 3.0 m (10 ft) into the roof. The roof consisted of top coal, clay-shale, sandstone-shale, and a fine-grained competent sandstone layer. The laminated materials under the sandstone member were competent, and physical property tests indicated a compressive strength of about 103.4 MPa (15,000 lbf/in$^2$) for tested specimens. Five rock spars crossed the tensionable test area, which broke up and weakened the immediate roof about 1.2 m (4 ft) on either side, as shown in figure 22.

### Cable System Design

After careful analysis of the anticipated pillar behavior and the experience gained in the first cement-grouted test area, it was decided to test three variations of resin-grouted cables (*6*). Based on the caving results from a previous panel, the site lithology, and the loading generated on the crib supports in the previous gate road, a cable system was designed to support the loads generated on the roof when the longwall panels were extracted and the immediate and main roofs caved. If roof-bed separation did occur, it would most likely be in the layers of silty shale that occurred about 1.2 m (4 ft) up from the roof. Additionally, the interbedded siltstones and shales could separate if the abutment forces became large enough or the immediate roof was lost. Initial yield zones on the pillars, both calculated and observed in the mine, indicated that the effective roof span would be approximately 7.9 m (26 ft). With these facts, the mine operator elected to install a 4.9-m (16-ft) long cable as the sole means of secondary support, which would penetrate the sandstone a distance of 1.8 m (6 ft).

The tailgate test area is shown in figure 22. The cables installed in the first 91 m (300 ft) of the tailgate test area (passive test area) utilized 1.7 m (5-ft-8-in) of resin, which left approximately 3.1 m (10.3 ft) of the cable ungrouted. This allowed the cable and the roof to yield and relax as abutment loads were redistributed to the gate road. The second 91-m (300-ft) area (stiff test area) used a 3.7-m (12-ft) equivalent column of resin, leaving only 1.2 m (4 ft) of the cable ungrouted. This provided a stiffer support system that would resist yielding, but still have enough ungrouted length to allow for separations at the interface between the coal, shale, and interbedded siltstone layers. The third and final zone (tensionable test area), also 91 m (300 ft) long, was supported with a tensionable cable support system. The cables were installed utilizing 1.7 m (5 ft-8 in) of resin grout and pretensioned, using a specially designed jacking system, to about 35.6 kN (8,000 lbf). The tensionable system would help resist any downward movements, but still have enough ungrouted portion of cable to accept large deformations before reaching the ultimate capacity levels.

The support systems in both the passive and stiff support areas were preloaded by thrusting the bolt into the roof, using the force of the bolter, before the resin had cured. After the resin cured and the load was removed, tension, ranging from 6.7 to 22.2 kN (1,500 to 5,000 lbf), was recorded on the instrumented bolts.

All of the cable systems were installed with 15.2- by 15.2-cm (6- by 6-in) bearing plates and Monster Mats.[7] The mats provide excellent mine roof support between the cables and maintain any failed material in place. Mats also help prevent unraveling, rotation of failed blocks, or progressive-type failures. The cables were placed four across the 5.7-m (19-ft) wide opening at a row spacing of 1.5 m (5 ft). The cables at the ends, dictated by the holes in the mats, were installed as close to the pillar and panel edges as possible and angled at approximately 80° from horizontal. A cross section of the cable pattern is shown in figure 23.

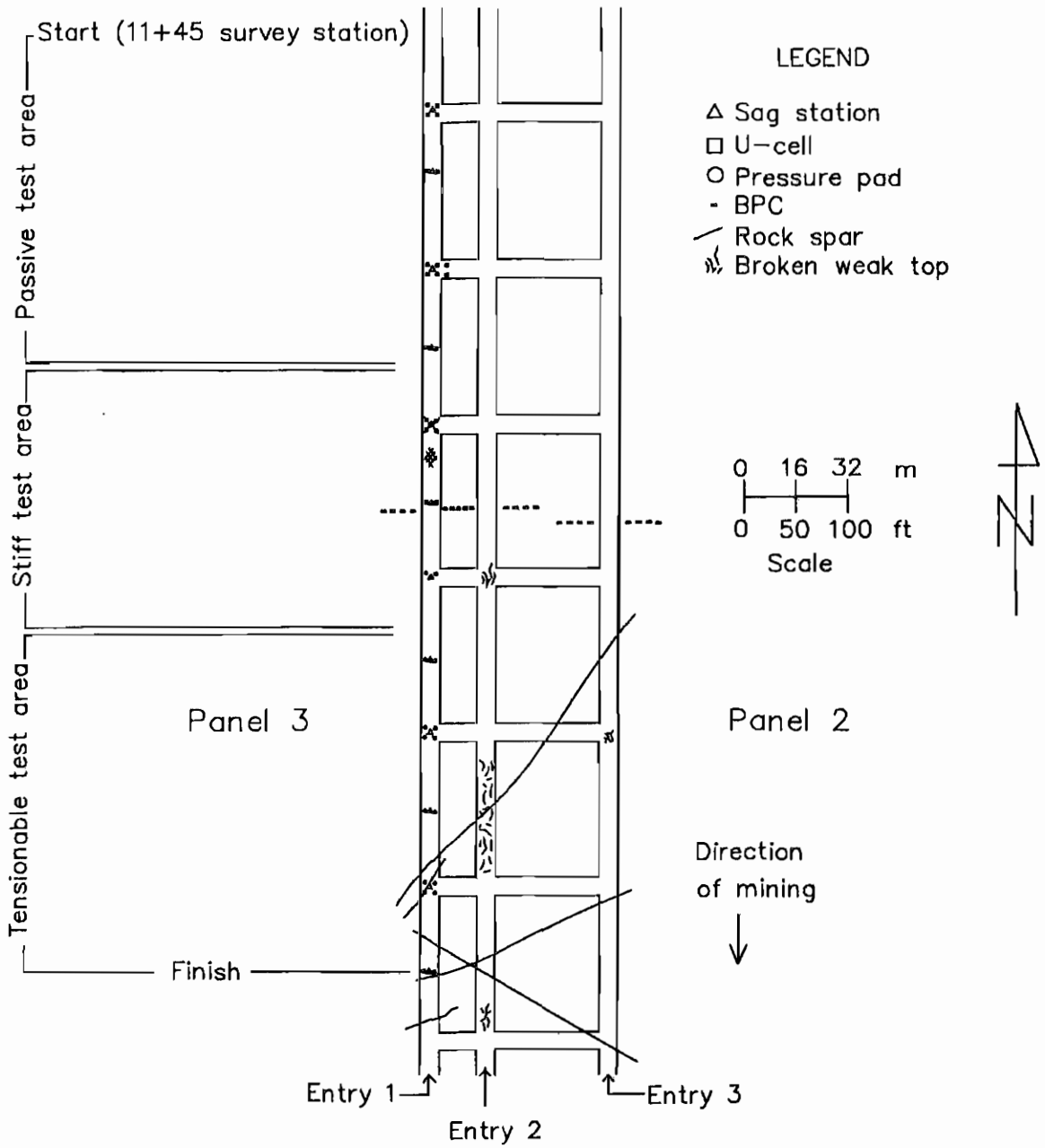
### Instrumentation

The test area was instrumented with 36 hydraulic U-cells and Goodyear pressure pads to evaluate individual cable loading trends and patterns. The extent of the test area and the instrumentation locations are shown in figure 22. The roof separations and movements were monitored with magnetic differential sag stations, wire-type extensometers, and a closure-rate meter. In addition to

[7]See footnote 5.

this instrumentation, crib loading inby the test area was evaluated with hydraulic flatjack cells to measure the loads and stiffness of the wooden material. Instrumentation was concentrated in the intersections, where one would expect the highest degrees of loading, and approximately at the midpoint of the entries between intersections. Stress redistribution and pillar performance were evaluated using borehole pressure cells (BPC's). The BPC's will help evaluate the effects and stresses generated by first panel mining, determine the core of the yield pillar after first panel mining, and provide insight into the effects of a stiff support system on pillar behavior.

*Figure 22*




*Case study 3 test area examining three different types of resin-grouted cable supports and respective instrumentation types and locations.*

## Test Site Results

The instrumentation in the test area was read and evaluated eight times during critical phases of panel 2 mining and daily during panel 3 longwall production shifts, starting when the test area was 15.2 m (50 ft) inby the cable support test area and ending when the face was mined past the test area.

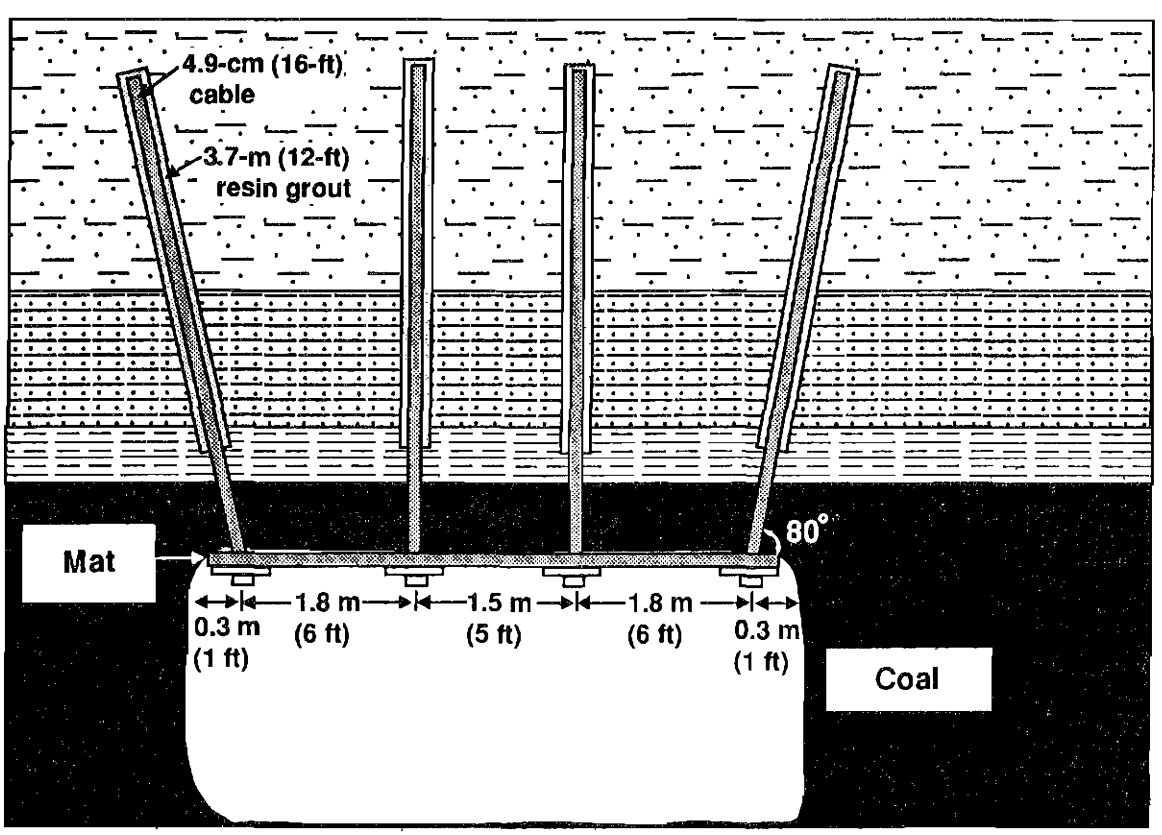
### Panel 2 Mining

At the conclusion of panel 2 mining, the instrumentation indicated that the cables loaded an average of 24.2 kN (5,450 lbf). The minimum recorded cable load was 0 kN (0 lbf), where roof dilation actually caused the cable to unload, and the maximum load was 148.6 kN (33,400 lbf), where a localized separation above the primary support caused the two adjacent cables to support the entire rock mass. The differential sag stations indicated minor roof separations between the coal and clay-shale interfaces and at the shale and sandstone-shale interfaces; this was expected. Minor floor heave, less than 15.2 cm (6 in), was present in all three test areas, predominantly in the intersections. Basically, with the exception of minor sloughage caused by increased pillar loading, the areas were extremely stable, and no roof control problems were noted. Figure 24 shows the resin-grouted cable test area after installation. Figure 25 shows the cable test area after panel 2 mining and prior to panel 3 mining.

### Panel 3 Mining

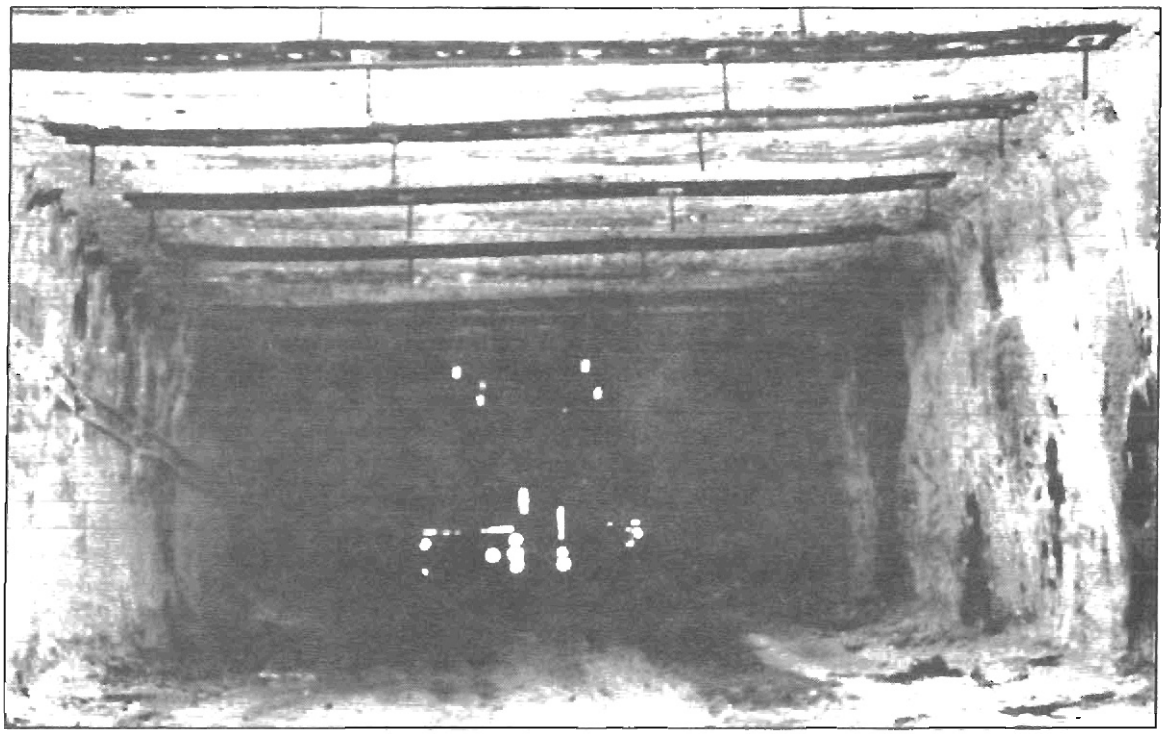
Panel 3 mining progressed for approximately 40 days before the longwall face entered the test area. The tailgate entry, apart from the test area, was typically secondary supported with two staggered rows of 1.8- by 1.2-m (6- by 4-ft) cribs installed with a 1.2-m (4-ft) wide walkway between rows. Crib spacing in each row was 3.6 m (12 ft), center to center. In anticipation of the forward and adjacent abutment loads and to quantify crib loading in the cribbed portion of the tailgate, two cribs located 25.9 and 6.1 m (85 and 20 ft) inby the test area were instrumented with hydraulic flatjacks. The flatjack pairs were installed at opposite corners of individual cribs between the timber material to determine the amount of load subjected on the cribs. Both crib supports performed similarly. After panel 2 was extracted, the cribs averaged 266.9 kN (60,000 lbf) of total load. When the face on panel 3 was 91.4 m

*Figure 23*



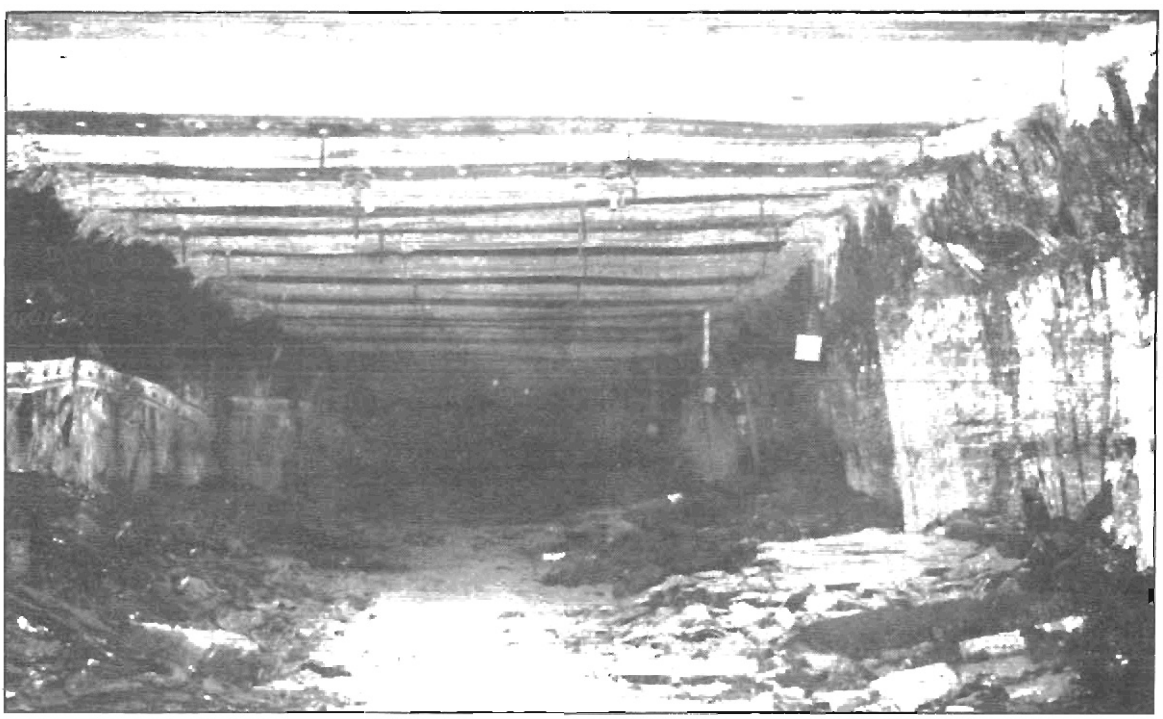

*Generalized cross section of cable spacing and lengths used in gate road test areas.*

*Figure 24*



*Resin-grouted cable test area immediately after cable support installation (before first panel mining).*

*Figure 25*



*Resin-grouted cable test area after first panel passed test area and loads were redistributed onto pillars and next panel.*

(300 ft) inby the cribs, the load had increased to about 445 kN (100,000 lbf). As the panel 3 face advanced to within 7.6 m (25 ft) of the cribs, the load increased to 1,112 kN (250,000 lbf) and remained at that level when the longwall passed and the cribs were inby the panel 3 face approximately 7.6 m (25 ft). The large loads were also attributed to the floor heave that occurred throughout the area, but the values still served as an indication of the loads that could be anticipated on the cable support systems.

*Passive Support System Test Area*

At the conclusion of panel 2 mining, the instrumentation in the passive support system test area was read and evaluated; the loads on the 14 hydraulic U-cells ranged from 0 to 31.1 kN (0 to 7,000 lbf) and averaged 13.2 kN (2,975 lbf). The four differential sag stations, installed in the intersections and entries, showed very few signs of roof separation. As the panel 3 longwall face approached the test area, minor separations began to occur between the coal-shale interface. Figure 26 shows the condition of the test area when panel 3 was approximately 243.8 m (800 ft) inby the test area. When the face entered the test area, the loads on individually monitored cables ranged from 0 to 106.8 kN (0 to 24,000 lbf), and the average increased to 18.8 kN (4,227 lbf). The roof remained intact and very little roof separation occurred. The differential sag station located in the first intersection (figure 27) showed a maximum of 0.6 cm (0.25 in) of total separation in the roof when the active face was only 2.7 m (9 ft) inby.

The roof remained extremely stable in the passive support test area throughout the retreat of panel 3. The only difficulty encountered was the floor heave adjacent to the yield pillar, as shown in figure 28. The floor began to move upward when the face was approximately 30.5 (100 ft) from the area. Measurements made with a rate-closure meter indicated that when the face was 7.6 m (25 ft) inby the measurement location, the floor moved upward, relative to the roof, at 0.43 cm/h (0.17 in/h), then leveled off. The heave never forced the entry to close tighter than 1.8 m (6 ft) or interfered significantly with ventilation. The roof of entry 1 behind the face remained intact and open an estimated 30 to 45 m (100 to 150 ft) and then caved gently. The roof behind the shields continued to cave, and no large overhangs were noted.

*Stiff Support System Test Area*

At the conclusion of panel 2 mining, the loads on the instrumentation in the stiff support system test area ranged from 3.4 to 37.3 kN (775 to 8,390 lbf), and the average for the 10 pieces of instrumentation (a combination of load cells and pressure pads) was 20.6 kN (4,640 lbf). The average load on the instrumentation was 25.9 kN (5,840 lbf) when the face entered the test area. Cable load measurements, recorded when the longwall face was immediately across from or just inby the test area, were 71.2 and 115.6 kN (16,000 and 26,000 lbf), respectively. This indicates that a separation, combined with forward and side abutments, loaded the cable supports to approximately 50% of their ultimate capacity.

The differential sag stations indicated that a minor separation occurred between the first and second extensometer anchors, which would have placed it between the coal and clay-shale interface. Typical roof behavior, shown in figure 29, as observed from a sag station located in the first intersection, indicated that a slight separation also occurred between the 135- and 178-cm (53- and 70-in) anchors. The yield pillar and panel sloughed as the forward abutment of panel 3 approached within 91.4 m (300 ft), as shown in figure 30. Similar to the first test area (passive area), floor heave forced the floor to within 1.8 m (6 ft) of the roof, starting approximately in the middle of the entry and propagating toward the yield pillar, indicating a stiff panel edge transferring stress into the floor. The stiff test area was mined through without problems, and the roof remained intact until the panel was well past the support system. Figure 31 shows the caving profile of the roof as it remains standing approximately 30.5 m (100 ft) behind the panel 3 longwall face, shown on the left side of the photograph.

*Tensionable Support System Test Area*

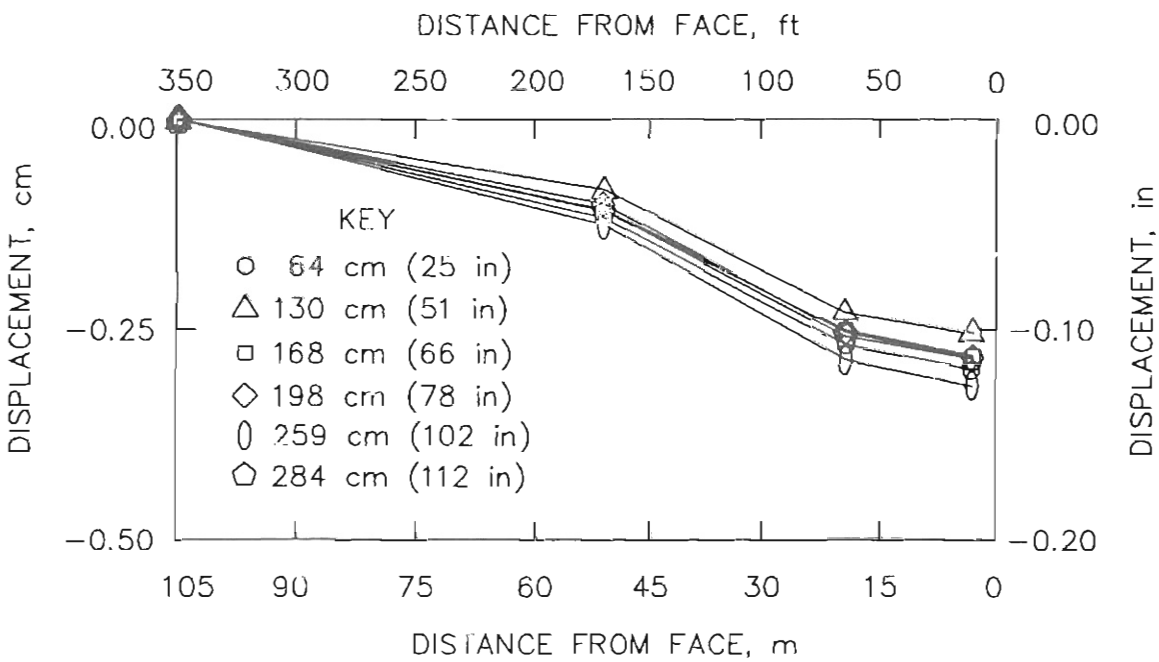
The tensionable support system test area provided the most difficult test conditions because the zone with cables contained several geologic features that created roof instability prior to panel extraction. A zone containing rock spars and sedimentary dikes had disrupted the continuity of the immediate roof, weakening it. Additionally, the discontinuities may have been associated with a changing roof lithology. The mine roof was heavily fractured in the areas traversed by rock spars and dikes. This roof condition resulted in some heavy loading almost immediately after panel 2 had mined past the site. In one specific area, approximately 3 m (9.9 ft) long and 5.5 m (18 ft) wide, the cables were loaded to 142.3 kN (32,000 lbf) and the roof appeared broken and fractured. Unfortunately, the differential sag station installed in the area was lost because of the broken roof. Calculating the roof separation height based on pressure pad loads, suggests that the loads measured on the cable supports could be the result of the roof separating at a height of about 3.7 to 4.0 m (12 to 13 ft). However, the cables, combined with primary support, Monster Mats,[8] and wire mesh, kept the immediate roof in place. The average loads on the instrumented cables at the conclusion of panel 2 mining ranged from 18.2 to 151.2 kN (4,100 to 34,000 lbf) and averaged 50.7 kN (11,400 lbf) for the 12 instrumented cables. The tensionable cable support area, when the panel 3 face is approximately 152.4 m (500 ft) inby, is shown in figure 32.

[8]See footnote 5.

*Figure 26*

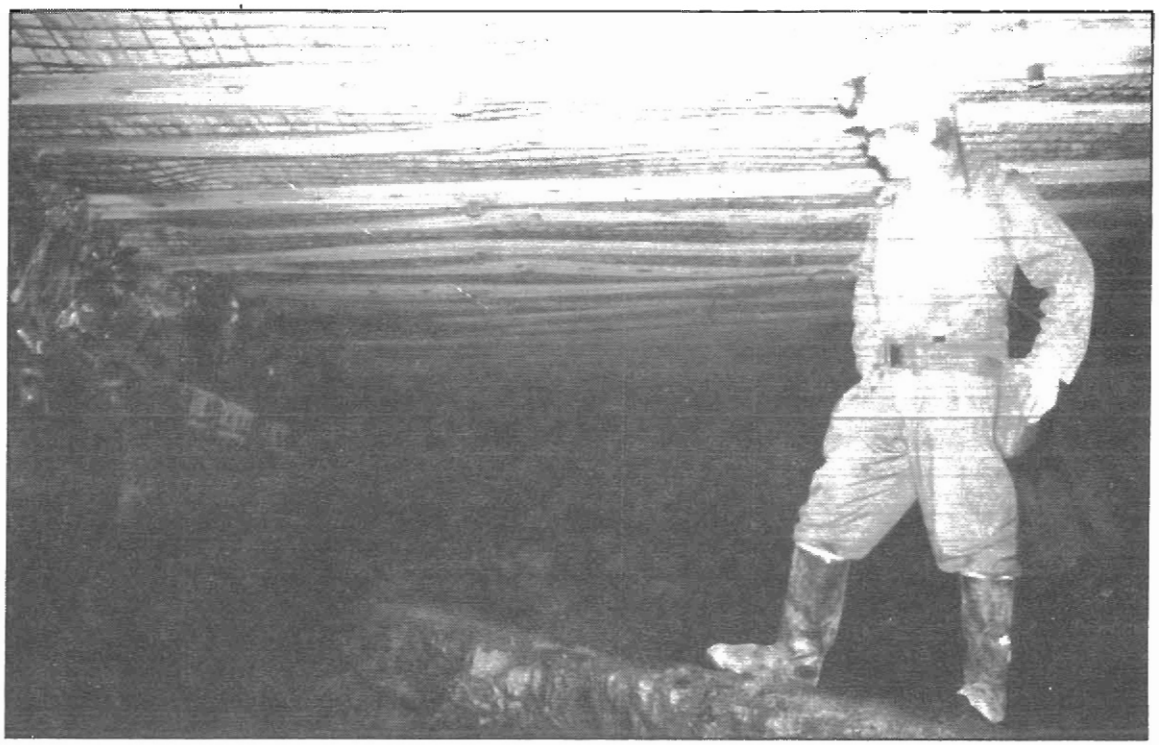
*Resin-grouted cable test area when panel 3 face was 243.8 m (800 ft) inby test area.*

*Figure 27*



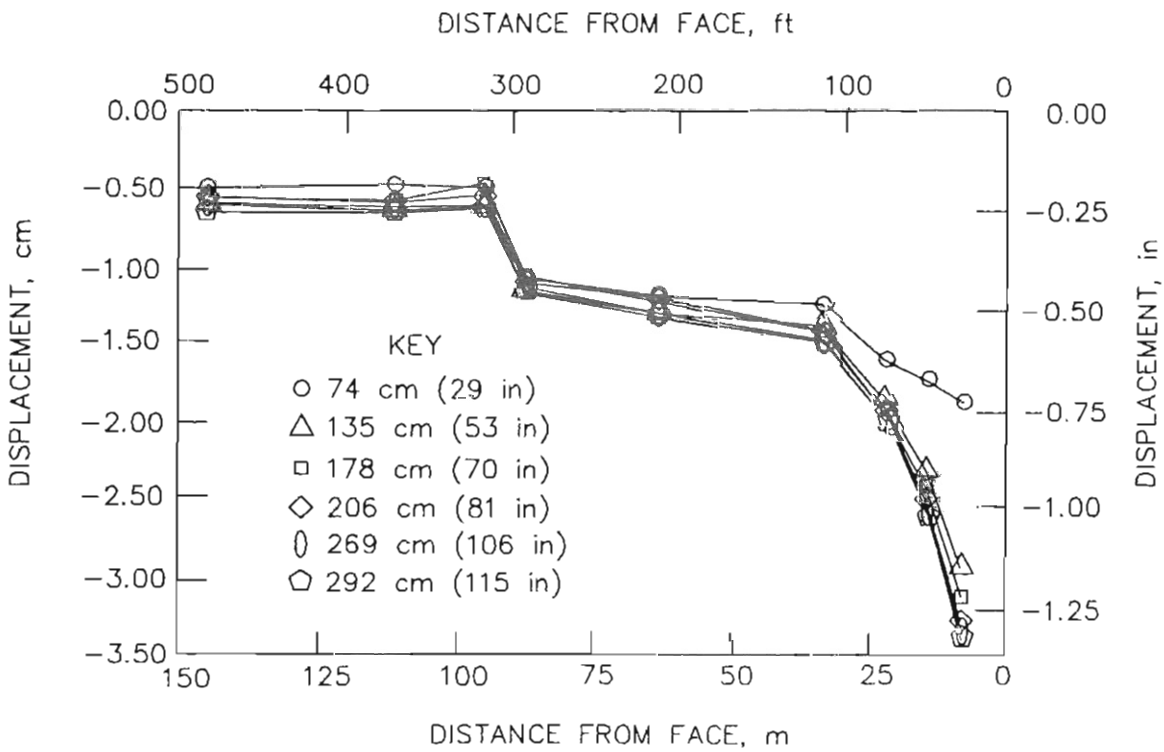

*Results from differential sag station being located in first intersection of passive test area.*

*Figure 28*



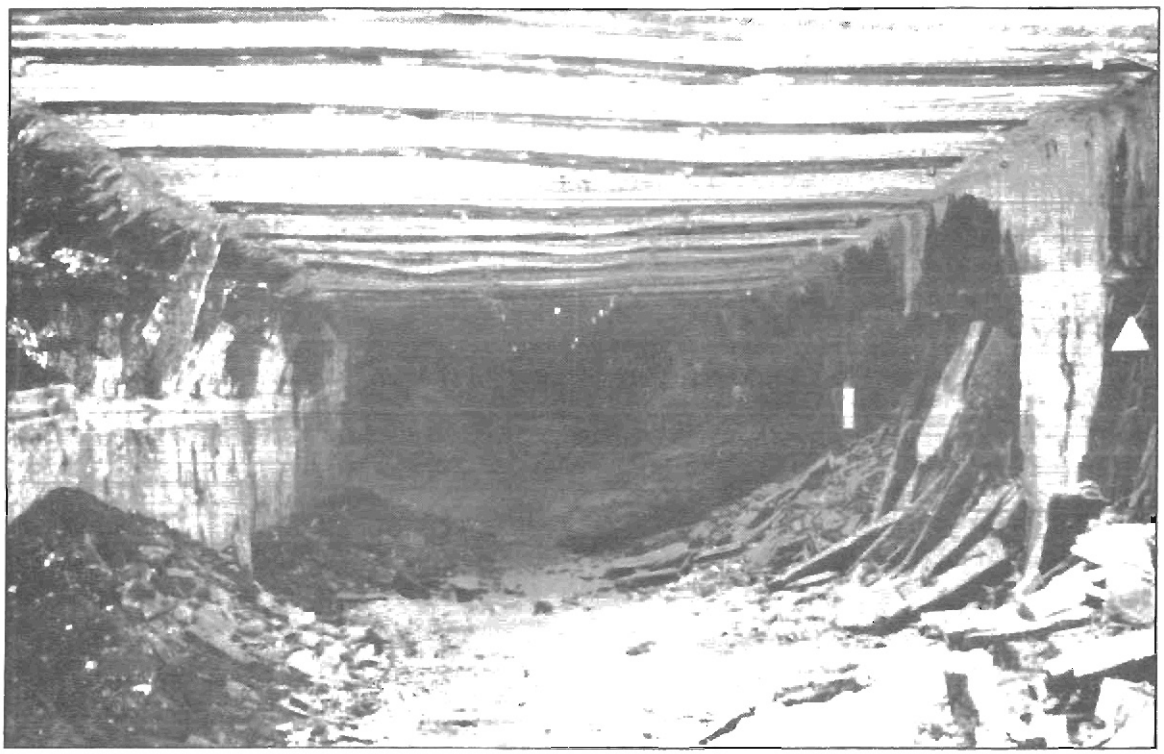
*Floor heave in passive support system test area.*

*Figure 29*



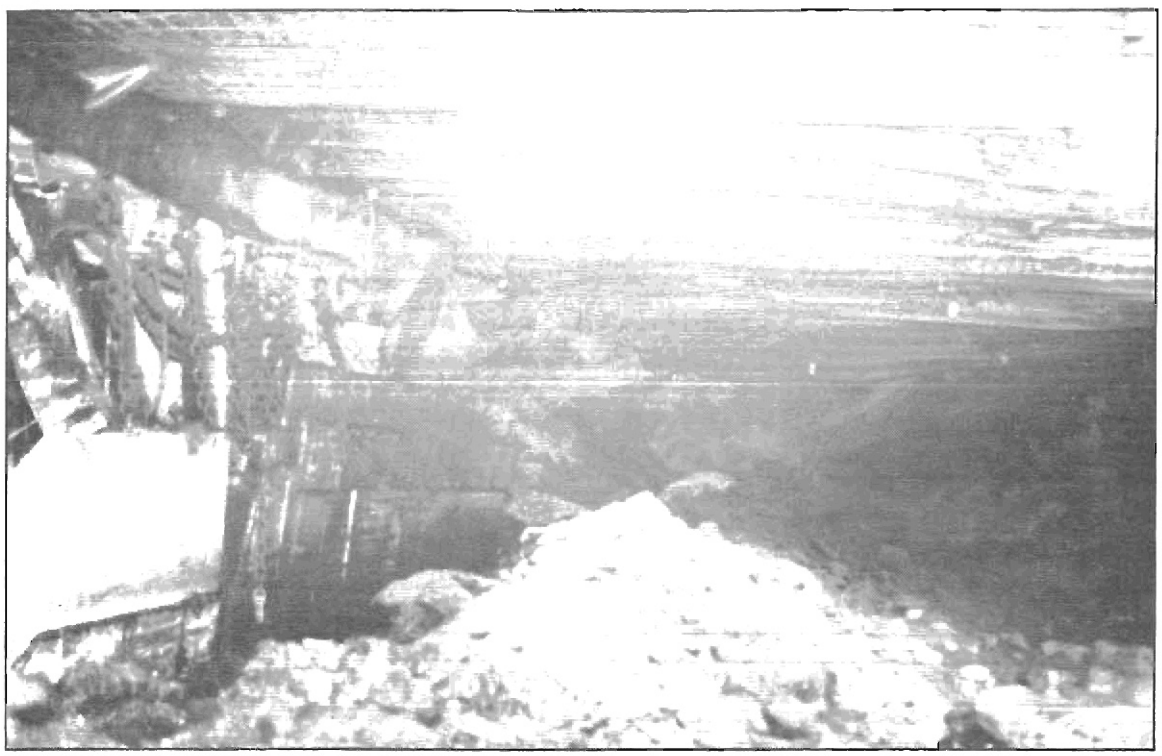

*Results from differential sag station being located in first intersection of stiff support system test area.*

*Figure 30*



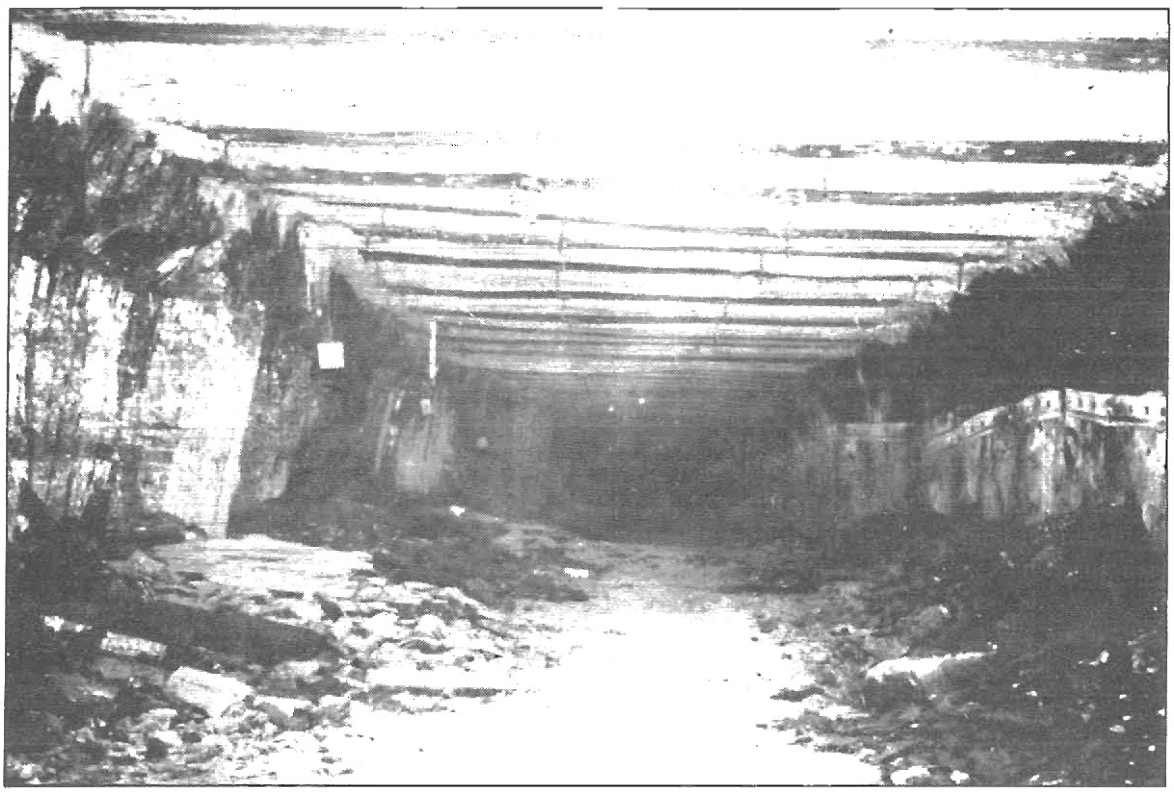
*Stiff support system test area condition during panel 3 mining.*

*Figure 31*



*Roof condition inby panel 3 longwall face.*

*Figure 32*



***Tensionable support system test area condition during panel 3 mining.***

The differential sag station results, shown in figure 33, are from the sag station located in the second intersection of the tensionable support system test area. The results indicate four distinct separations between the instrumented levels, with a major separation forming between the 160- and 188-cm (63- and 74-in) anchors in the roof. The roof below this level separated approximately 2.54 cm (1 in) when the face was approximately 9.4 m (31 ft) from the station. The separations occurred between geological interfaces, with the largest at the expected level. However, the cables maintained the immediate roof, which permitted reading the sag station when the face was 2.1 m (7 ft) past the station. The major separation had a displacement of approximately 1.3 cm (0.5 in), and two minor separations occurred between the 221- and 282-cm (87- and 111-in) anchors.

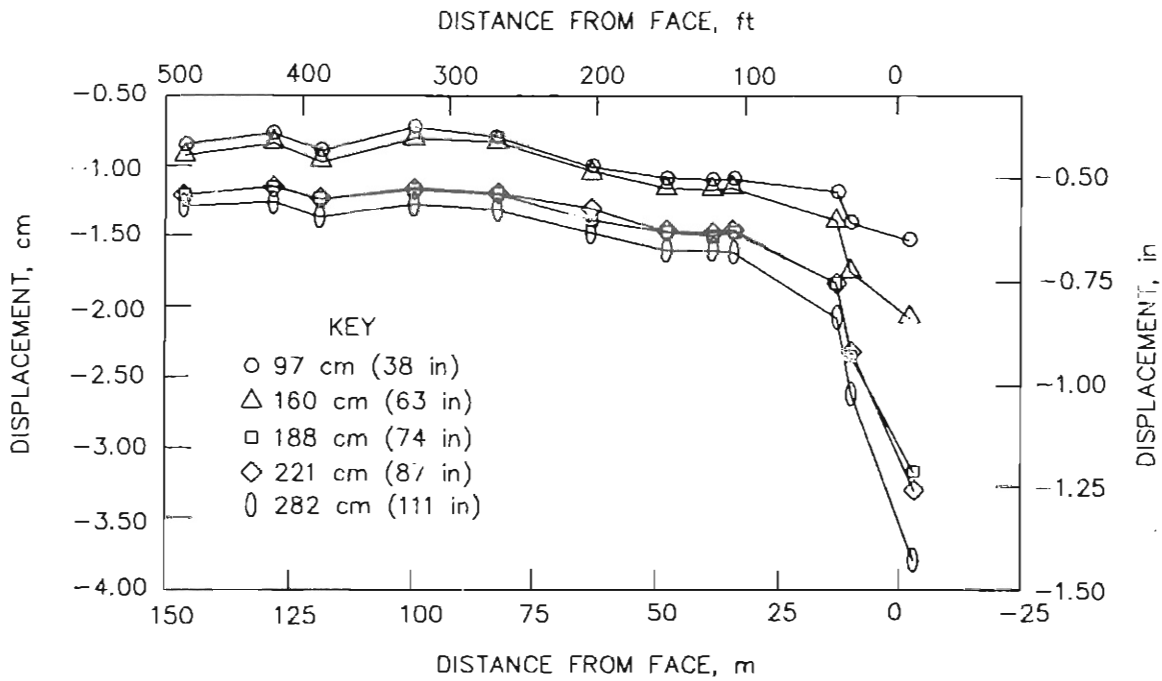
Floor heave also occurred in the tensionable support system test section with rates similar to the previous two areas. Closure instrumentation and the general condition of the test area when the face was approximately 45.7 m (150 ft) inby are shown in figure 34. As shown, if the cables failed or appeared to have taken considerable load, a timber post was placed adjacent to the cable. Additionally, some cable nuts appeared to be slipping under heavy load, preventing all seven strands from actively resisting movement. The area was mined through without incident, and a total of nine timber posts were set in the areas subjected to high loading. These areas were usually associated with the geological features as described earlier. The face and the integrity of the immediate roof are shown in figure 35. This type of roof condition, encountered throughout the entire test area, provided a safe and accessible travelway and maintained the ventilation airway.

*Pillar Performance*

The performance of the pillars and panels were evaluated to examine the support, if any, that the yield pillar or panel edge would provide for the immediate and main roof. This factor, which affects overall entry width, can dramatically change the cable support system design. The gate road pillar layout consisted of a three-entry system. The stiff pillar was adjacent to panel 2 and between entries 2 and 3. The yield pillar was adjacent to panel 3 and between entries 1 and 2.

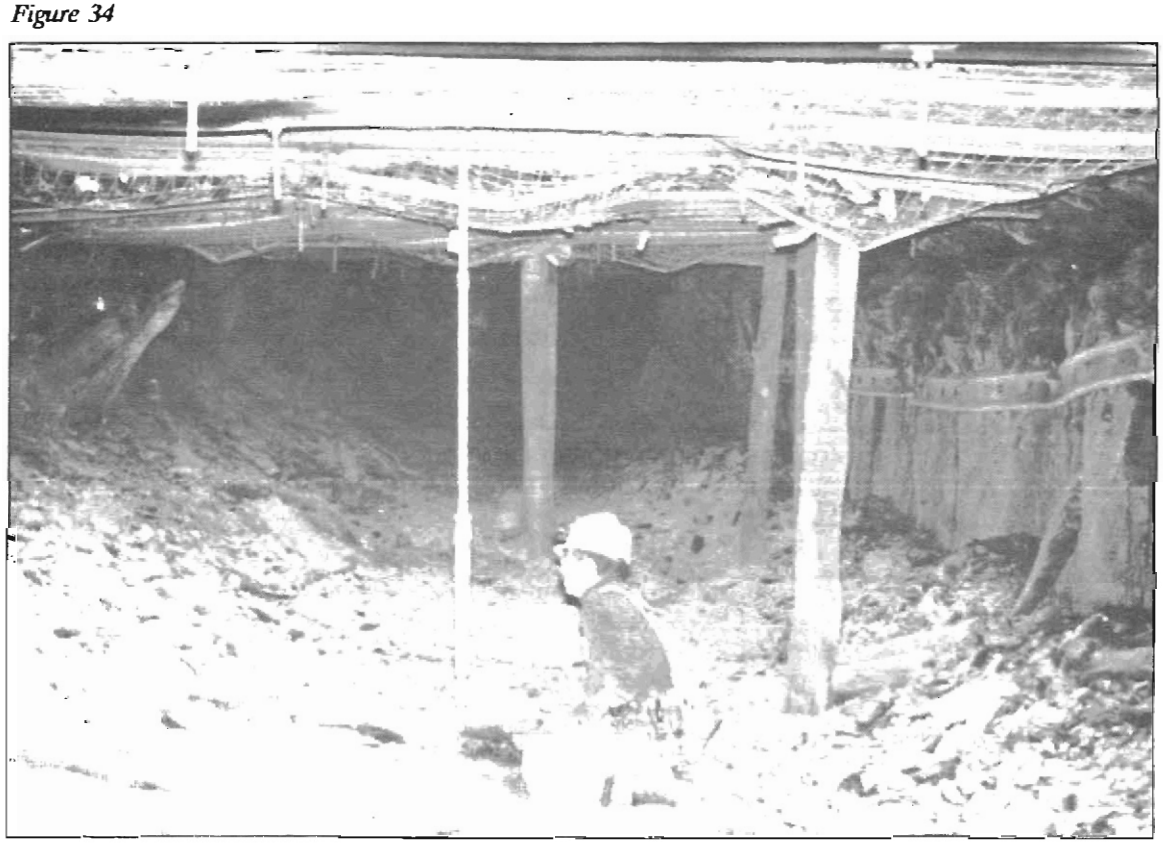
BPC's were installed in the pillars and adjacent panels to measure the vertical ground pressure changes as the face advanced. The BPC analysis included forward abutment pressures, pillar-panel load transfer, and the general yield behavior of the instrumented pillars. BPC's do not provide absolute pressures, but simply an understanding of the relative pressure increases and decreases, which may be an indication of stress change and coal failure.

*Figure 33*



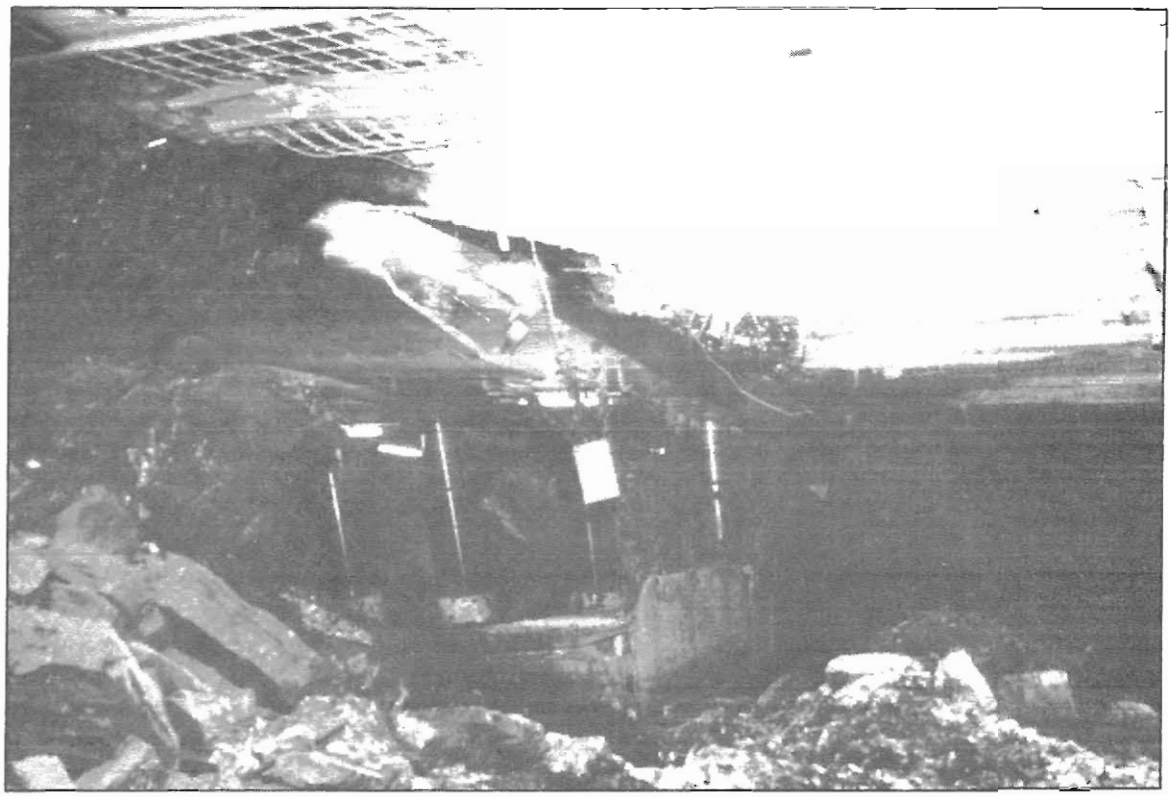

***Results from differential sag station being located in second intersection of tensionable support system test area.***

*Figure 34*



*Closure-rate instrumentation and condition of tensionable support system test area.*

*Figure 35*



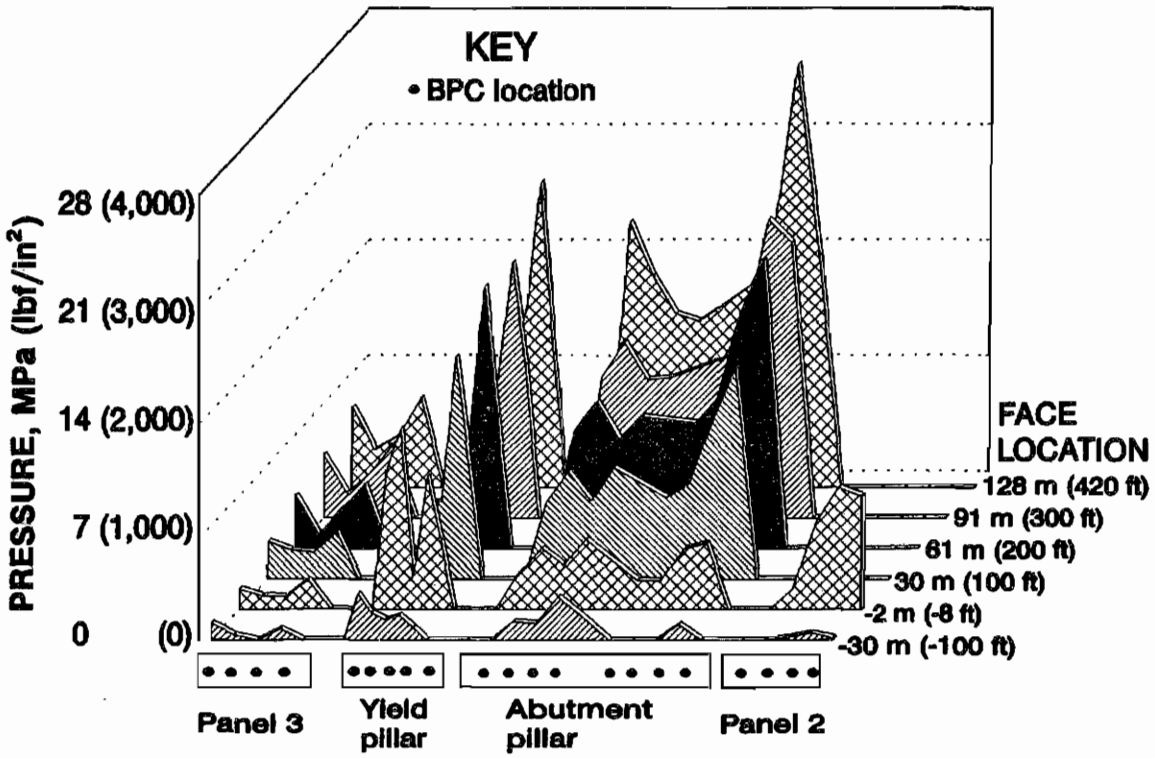
*Roof integrity and stability of tensionable support system test area.*

The forward abutment pressures during panel 2 mining were characterized by monitoring vertical pressure in the longwall panel 2 headgate, as shown in figure 36. For the purposes of discussion, the BPC pressure changes from equilibrium pressure are presented. All pressure cell data are plotted versus distance from the cell location to the active mining face. The onset of the forward stress abutment was first observed when the panel 2 face was approximately 30.5 m (100 ft) inby the BPC locations. A minus sign (-) indicates the panel face was inby the cells. An increased rate of cell pressure rise on the panel was observed when the face was 2.4 m (8 ft) inby the pressure cells, resulting in an average increase of 6.5 MPa (940 lbf/in²). At this time, the abutment pillar, adjacent to the panel, experienced an average increase of 3.0 MPa (435 lbf/in²). The yield pillar, adjacent to panel 3, realized an increased average cell pressure of 6.6 MPa (965 lbf/in²), with the major stress concentration occurring adjacent to entry 1 on the panel 3 side. Panel 3 cells experienced a minor loading change of about 1.2 MPa (170 lbf/in²) as panel 2 was mined through the instrumentation site.

When the panel 2 face was 91.4 m (300 ft) outby the instrumented pillars and panel, the pressure increases averaged 11.4 MPa (1,650 lbf/in²) on the abutment pillar, with the major concentrations of load indicated next to the extracted panel 2, as expected. The average increase across the yield pillar was 11.2 MPa (1,620 lbf/in²), with the major loads being observed on the entry 2 side of the yield pillar. The cells in the panel 3 edge were beginning to load slightly, increasing approximately 2.9 MPa (425 lbf/in²).

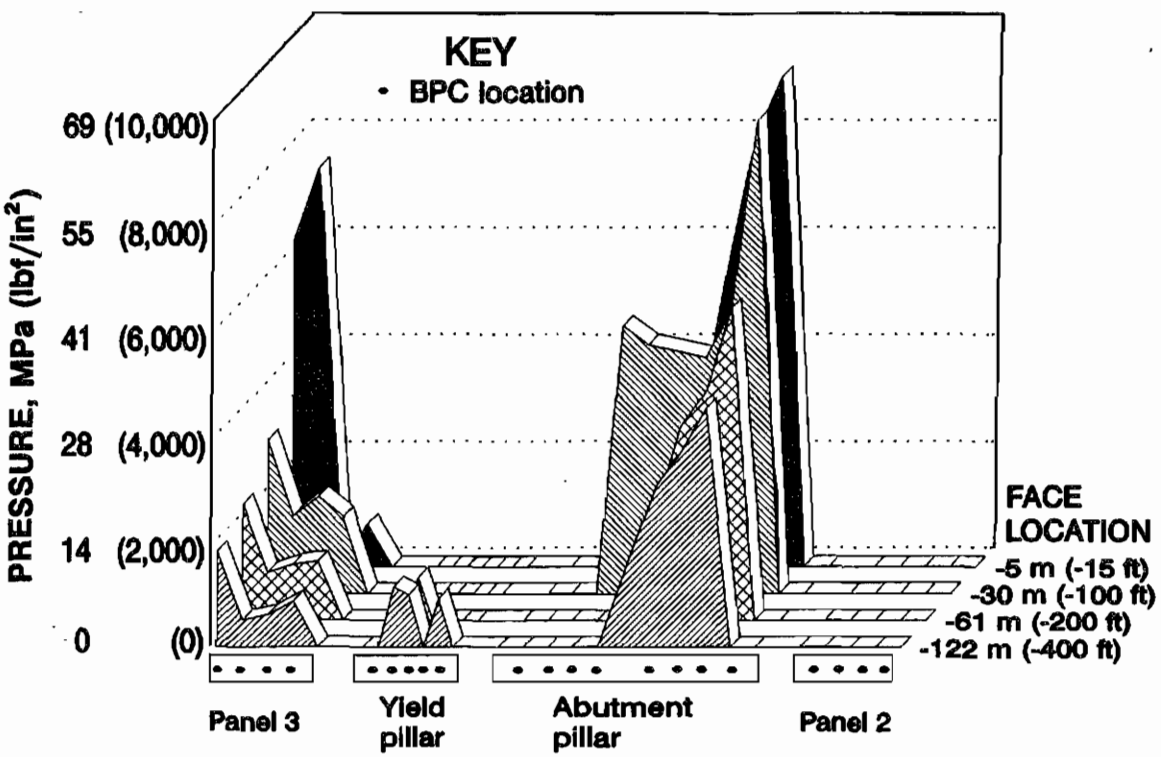
The analysis of the data started again after panel 2 was completed and the panel 3 face had mined to a position approximately 121.9 m (400 ft) inby the instrumentation, as shown in figure 37. At this point, the abutment pillar edge adjacent to entry 2 had yielded to the cell established 19.2 m (63 ft) inside the pillar from entry 2. The abutment pillar had also yielded about 3.0 m (10 ft) from entry 3 next to previously mined panel 2. The yield pillar remained intact with the exception of the outer 3.6 m (12 ft) of the pillar on the entry 2 side. This area, heavily loaded during panel 2 extraction, had yielded by the time the panel 3 face approached the site. When the panel 3 face was 61.0 m (200 ft) inby the instrumentation, the yield pillar core lost pressure. Even though the yield pillar appeared to be intact with only minor sloughage, the instrumentation indicated that the yield pillar loading had decreased. Load from the yield pillar was transferring to panel 3 and the core of the abutment pillar. The abutment pillar and panel 3 experienced increased load.

*Figure 36*




*Pillar and panel loading during panel 2 extraction.*

*Figure 37*




*Pillar and panel loading during panel 3 extraction.*

When the panel 3 face was 30.5 m (100 ft) from the site, the yield pillar core appeared to be intact, but no load increases were noted in the BPC's; the load-carrying capacity of the pillar had diminished. The abutment pillar cells had pressure increases in excess of 60.7 MPa (8,800 lbf/in²), and the average on the remaining core cells was 39.0 MPa (5,650 lbf/in²). The instruments in the panel 3 edge loaded to an average stress increase of 9.0 MPa (1,300 lbf/in²). The final data were recorded when the panel 3 face was 4.6 m (15 ft) inby the installed instrumentation. The BPC in panel 3, 12.2 m (40 ft) from entry 1, was loaded to about 51.7 MPa (7,500 lbf/in²). The BPC 9.1 m (30 ft) from the entry had dropped pressure, indicating the panel edge was yielding. Several bounces were recorded in the area of the abutment pillar, and the loads on the remaining abutment pillar core cells averaged 53.1 MPa (7,700 lbf/in²).

The yield pillar did not quit taking load until panel 3 was approximately 61.0 m (200 ft) inby the site. The pillar provided an element of roof support, but the load-carrying capacity had diminished before the stress levels became large or dangerous.

Even though entry 1 experienced floor heave 30.5 m (100 ft) outby the face that was caused by the apparent strength of the panel, which did not yield at the panel edge until the face was within 4.6 m (15 ft) from the instrumentation, the roof structure remained intact, and the cable supports held the immediate roof in place; very little roof-bed separation was recorded. The cable system between the yield pillar and panel at the BPC instrumentation site were the "stiff" cables with the longest column of resin grout. Whether or not this assisted in the stress transfer to panel 3 and protected the integrity of yield pillar cannot be determined. There were no visible differences in pillar performance in other portions of the tailgate test areas.

### Discussion

Resin-grouted cable support systems were extremely successful at this test site in maintaining a longwall gate road during first and second panel mining. Data were recorded from 120 instruments and evaluated 30 times during this 18-month investigation. The data provide an adequate understanding of the successful application of resin-grouted cable supports in a longwall tailgate. Three different cable concepts were evaluated—passive, stiff, and tensionable—to examine any differences in roof and subsequent pillar response. The data were inconclusive in determining if any one cable design was better in preventing roof separation. Differential sag-station measurements indicate that all three systems allowed some separation in the immediate roof at geological interfaces. However, the significant point is that the cables, in conjunction with primary support, mesh, and mats, provided enough resistance to prevent the roof from progressively failing beyond these initial separations. The largest separation recorded in the entire test area was less than 3.8 cm (1.5 in). This is minimal for the types of loading occurring in a coal mine gate road.

The loads generated by roof separation and movement could not account for the loads measured on the cribs immediately inby the test area. The additional loads were the result of the floor heave that occurred along the length of the entire gate road.

The BPC's indicated that the yield and abutment pillar behavior was adequate for maintaining the integrity of the roof while minimizing coal mine bounces or bumps. The yield pillar width of 9.7 m (32 ft) was sufficient to maintain an active core, which aided in roof performance, until the panel 3 face was approximately 30.5 m (100 ft) away. This was noted in the individual cable loads and reflected in the differential sag-station data.

## FUTURE RESEARCH

The underground success of resin-grouted cables at the West Elk Mine has prompted the mine operator to support the entire next gate road (tailgate entry 1 between panels 4 and 5) with resin-grouted cables. The final selection, based on test site performance and ease of installation, was similar to the design used in the passive test area. Cables 4.8 m (16 ft) long, anchored with a 1.7-m (5-ft-8-in) resin column, are being used in conjunction with Monster Mats[9] and high-capacity, dome-bearing plates. The test results from this gate road should be available in early 1995, at current mining rates. Additionally, cable supports are being evaluated as primary support, secondary support in difficult intersections where trusses were traditionally required, longwall setup rooms, and recovery chutes.

Resin-grouted cable support technology has proven to be a practical and economic alternative for timber supports in certain ground conditions and may be the next major shift in roof support and ground control technology. USBM personnel, with the continued support of industry, are continuing to examine the effectiveness of cable supports under a wide range of difficult geological and mining conditions.

[9]See footnote 5.

## SUMMARY AND CONCLUSIONS

Cable supports have been successfully installed with both cement- and resin-grouted anchorage systems. Cables have shown their ability to stabilize ground conditions in adverse situations, such as gate roads and bleeder entries. Design principles have been presented to permit initial simplistic cable configurations and spacings. These design schemes will be modified and updated as the results from several ongoing investigations are completed. USBM personnel, with the assistance of industry, are continuing to examine the effectiveness of cable supports under a wide range of geological and mining conditions. The improvements to ventilation and entry passage, although not specifically studied in this investigation, only add to the benefits to be received from cable supports.

## ACKNOWLEDGMENTS

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