CABLE BOLTS FOR LONGWALL GATE ENTRY SUPPORT

By Stephen C. Tadolini[1] and Jamie L. Gallagher[2]

ABSTRACT

Cable supports offer several advantages over traditional secondary support methods. Cable supports enhance stress redistribution to pillars and gob areas, minimize or eliminate timbers and cribs that reduce ventilation capabilities, eradicate material-handling injuries related to the placement of crib supports, and provide a cost-effective alternative to secondary support. The U.S. Bureau of Mines (USBM) has designed and installed cable supports to improve the stability of tailgate and bleeder entries in a western U.S. underground longwall mine. Design and installation techniques are presented for both cement and resin-grouted cable bolt systems. Three case studies discuss the design philosophy, describe measurement instrumentation, and present the quantifiable results to date.

INTRODUCTION

Cable bolts were introduced to the U.S. mining industry in 1970 as a method to reinforce ground prior to mining. Until recently, only four underground U.S. hard-rock mines had used cable supports. In the beginning, 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 in a 4.1- to 6.4-cm (1.625- to 2.5-in) borehole and grouted with cement. The USBM has also developed the use of resin as an anchorage material in a 2.5- and 3.5-cm- (1- and 1-3/8-in-) diameter 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 (29.5×10^6 psi). 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, "Monster Cables" measuring 1.78 cm (0.70 in) in diameter 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 cables are placed in a 3.5-cm- (1-3/8-in-) diameter hole. Monster 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 a diameter of about 1.2-m (4-ft) for handling. This flexibility is one of the primary advantages of cable supports, since the support length is not limited by the opening height.

For a cable support system to be effective, the loads must be successfully transferred from the rock to the cable through the grout. Laboratory and field investigations have determined that to achieve the ultimate cable capacity of 258.0 kN (58,000 lbf), depending on the water-cement ratio, the top 1.5 m (5 ft) of the cable should be grouted when using a cement product. When resin products are used, 1.2 m (4 ft) of grout will develop the ultimate capacity of the cable. Of course, adequate anchorage should be evaluated on a site-specific basis by performing a standard pull test. Laboratory and field results indicate that at least 6% elongation for the ungrouted portion of the cable can be expected (*1-2*).[3] This permits

[1]Mining engineer.
[2]Geophysical engineer.
Denver Research Center, U.S. Bureau of Mines, Denver, CO.

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

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.

Gate roads are the most hazardous areas in longwall mines. Keeping the gate road entries stable is a challenging task for the mine engineer. The 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 retreat, generally require secondary reinforcement for a distance outby the face using wooden posts, hydraulic jacks, or spot roof bolts 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 solely used for return and sometimes for intake air passages and travelways. Much higher ground pressures exist around the tailgate entries, which makes them very 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. Cribbing patterns and densities are varied to support difficult ground pressures. In 1992, the USBM evaluated the potential for supplementing or replacing conventional headgate and tailgate secondary supports in an underground longwall mine with cement-grouted cables. 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 (*3*). The successful application of this technology indicated the potential for several mining advantages. A direct benefit of cable systems is the installation of a stiff support system capable of strengthening and reinforcing roof members to transfer high pressures into the main and immediate roof and onto supporting structures away from the periphery of the entries. This will also reduce entry convergence. The indirect benefit of cables is the reduction or elimination of cribs.

A recent analysis revealed that in 1991-92 a total of 734 accidents in U.S. underground mines related to cribbing, timbering, and blocking were reported. The majority of the serious accidents occurred in western U.S. underground longwall mines. This directly relates to the height of the coal seam, which requires the use of ladders to build the cribs with heavy and cumbersome materials (*4-5*). The removal of cribs will also help to minimize the ventilation resistance in air passages, which results in savings in air pressure and increases airflow, while removing gas and dust from the working face more efficiently (*6*). This will create a safer, cleaner environment for underground mineworkers, while reducing mine ventilation costs for 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 hectares (248 acres) of select-cut prime timber. Eliminating or minimizing crib supports can reduce accidents related to support installation, improve ventilation, and reduce wood consumption.

CABLE SUPPORT DESIGN

Although cables have been used in U.S. hard-rock mines, the design 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 (*7-8*). The analytical and theoretical approach, ongoing during the investigation, is initially simplistic, but is being constantly modified based on field measurements.

The first step of the design procedure requires 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 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 by empirical methods or modeling should be included in the stability analysis. The weight of the rock mass should be expressed per linear 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

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

where F_w = weight of the rock per linear meter (foot), N/m (lb/ft),

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

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

and γ = average rock density to the parting plane, kg/m^3 (lb/ft^3).

This method is conservative, as 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 mine 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 (*9*). The weight of the material within the arch can be estimated with respect to the opening width and 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 the rock within the pressure arch per lineal meter (foot), N/m (lb/ft),

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

H_a = height of the pressure arch, m (ft),

and γ = rock density, kg/m^3 (lb/ft^3).

The behavior of the pillar under different loading conditions determines the effective opening width W_3, 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 (*10*). Wilson's equations are

$$W = 2\frac{m}{F}\ln\left(\frac{q}{p+p'}\right)_{\text{(rigid roof-floor)}} \qquad (3)$$

and

$$W = m\left[\left(\frac{q}{p+p'}\right)^{1/k-1} - 1\right]_{\text{(yielding roof-floor)}}, \qquad (4)$$

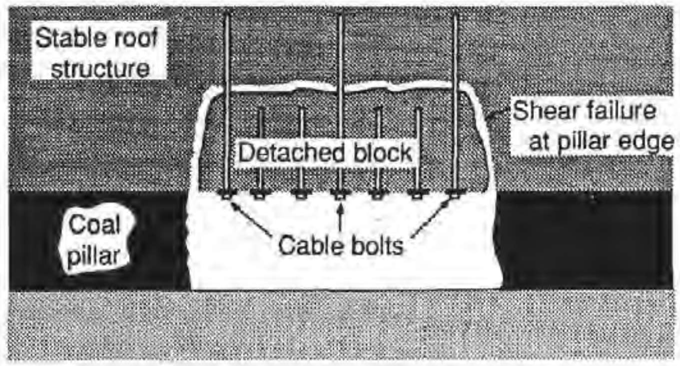

Figure 1.—Detached block failure being supported by cables.

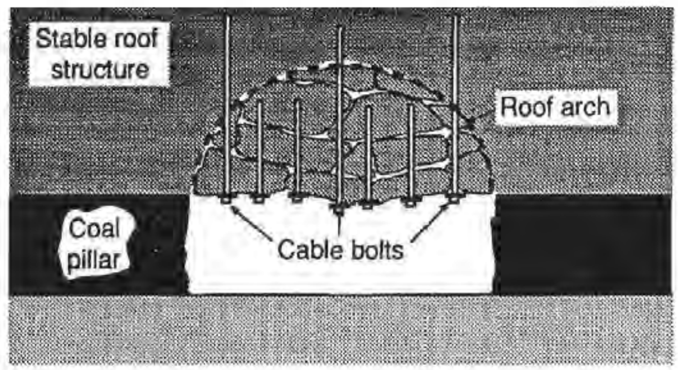

Figure 2.—Formation of a pressure arch of failed mine roof material.

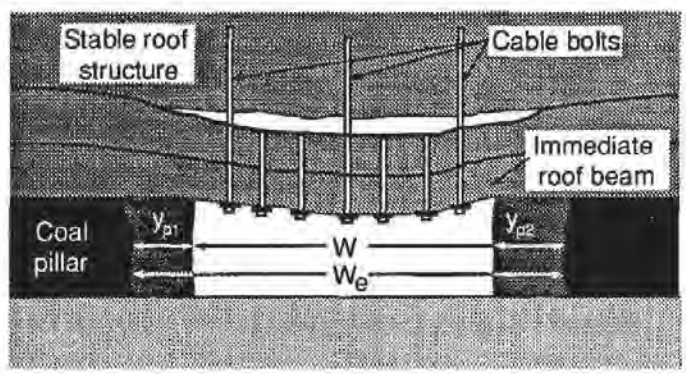

Figure 3.—Formation of a yield zone on the coal pillars. W_e = effective opening width; y_{p1} = yield zone pillar 1; y_{p2} = yield zone pillar 2; W - mined width of the opening.

where W = pillar width, m (ft),

m = seam height, m (ft),

q = overburden load, kN/m^2 or kPa (ton/ft^2),

p = artificial edge restraint (0 kPa (0 ton/ft^2)),

p' = uniaxial strength of fractured coal, 96.4 kN/m^2 (1 ton/ft^2),

k = triaxial factor $\frac{1 = \sin\phi}{1 - \sin\phi}$,

ϕ = angle of internal friction, deg,

and

$$F = \frac{k-1}{\sqrt{k}} + \frac{(k-1)^2}{k} \tan^{-1}\sqrt{k}, \qquad (5)$$

where $\tan^{-1}\sqrt{k}$ is expressed in radians.

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

$$W_e = W + Y_{p1} + Y_{p2}, \qquad (6)$$

where W_e = effective opening width, m (ft),

W = mined width of the 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 have been determined, it is 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/cable (58,000 lb/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 an effective opening width, W_e, of 7.6 m (25 ft) and a material weighing 2,403 kg/m^3 (150 lb/ft^3). 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.

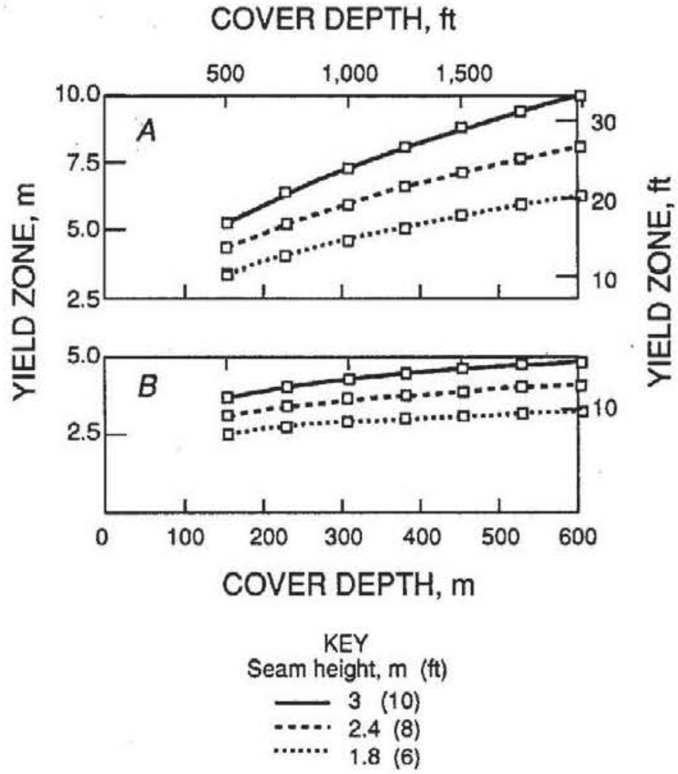

Figure 4.—Design chart to determine the yield zone width in coal pillars. *A*, Yielding roof; *B*, rigid roof.

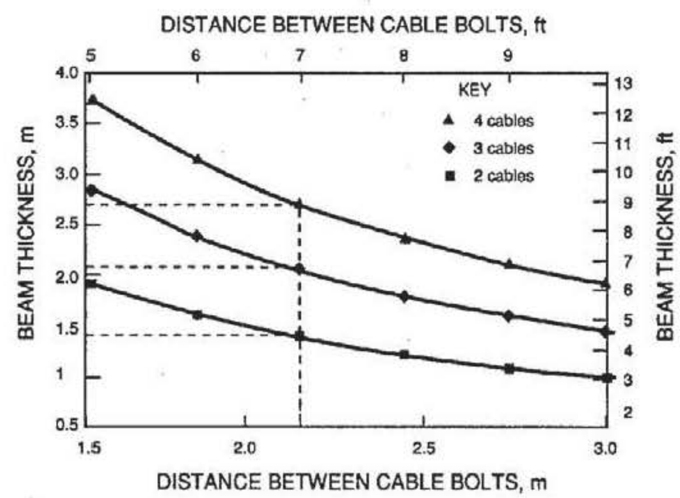

Figure 5.—Cable support design chart where W_e = 7.6 m (25 ft) and γ = 2,403 kg/m^3 (150 lb/ft^3).

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 where a cement cable system would be preferable. For example, if large voids or washout of the roof rock were 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. Cement grout would also be preferable when a full column of grout is desired 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 are 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, when the grout runs out of the tube, this 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 cement and water mixture at a ratio of 0.35 parts of water to 1 part of cement by weight. Several laboratory studies investigated the effects of water-cement ratios. The strength selected is site-specific and related to the available pumping equipment. The greater the cement-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 approximately 24 h later, 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 cable grips, are shown in figure 6.

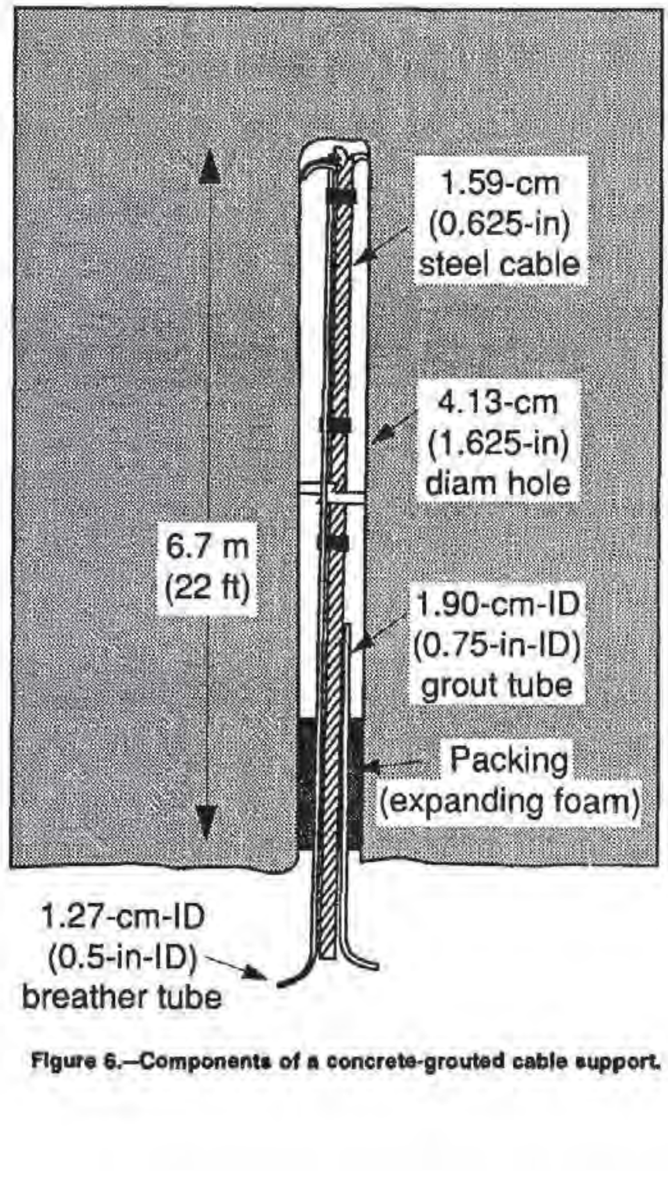

Figure 6.—Components of a concrete-grouted cable support.

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 resin-grouted cable bolt used in several USBM investigations is shown in figure 7. Each component (*A* through *E*) serves a specific function that contributes to the overall success of the cable bolt system.

Referring to figure 7, the end of the cables are clamped together using a swedged-on fitting (*A*). This ties all the cable strands together, including the kingwire, which is the center cable strand. The next swedged-on buttons (*B*) serve two functions. 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 are imbedded in the borehole. Secondly, the buttons help to mix the resin during placement by forcing it around the tight opening. The turbulence enhances the mixing and helps to detach the cartridge cover. The short button (*C*) generally serves 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 make the difference between adequate or 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 great, causing the cable to bend or kink. The cable stiffener rod (*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 (*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

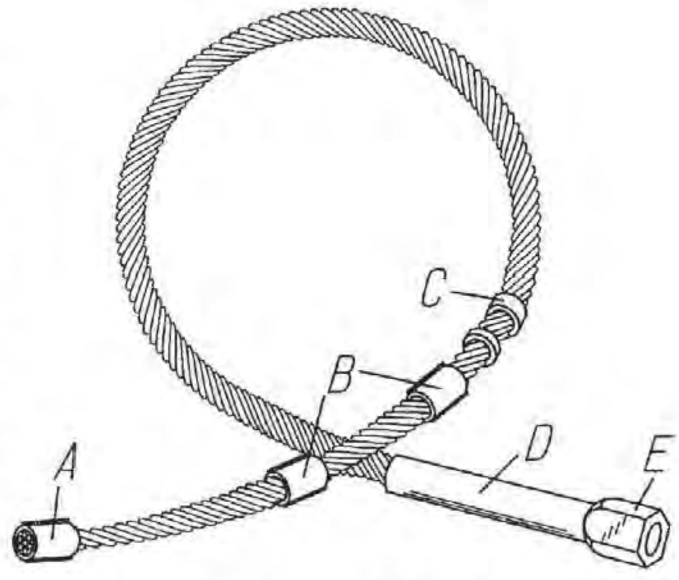

Figure 7.—Specific components of a resin-grouted cable bolt. *A*, swedged-on fitting to tie all cable strands together; *B*, wedged-on buttons for additional anchorages; *C*, short button to hold plastic seal in place; *D*, cable stiffener rod; and *E*, installation head or cable nut.

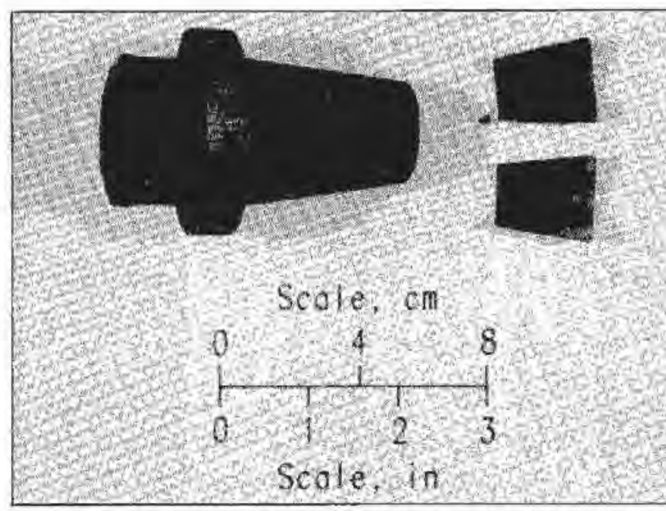

Figure 8.—Tensionable nut for a resin-grouted cable bolt.

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 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 work well, especially in instances 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 for the total amount of time recommended by the manufacturer. Be careful 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 mine roof. While rotating the cable clockwise will pull up the resin, it creates back pressure after the resin has been mixed and the cable is pushed up against the mine 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 mine 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 (*11*).

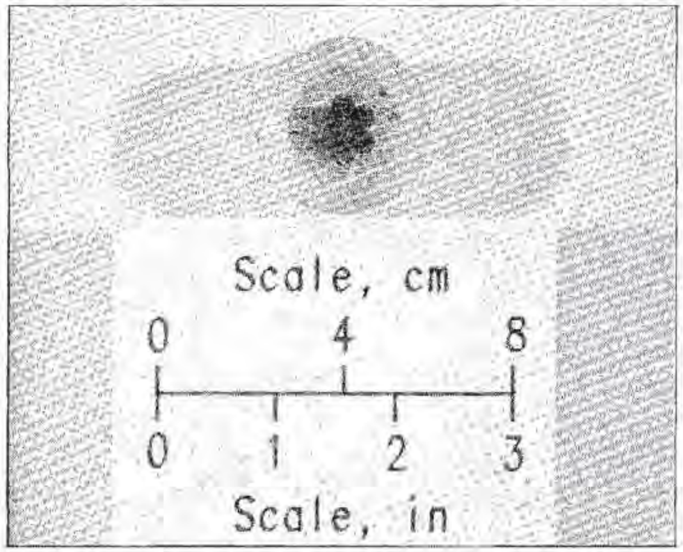

Figure 9.—Cross section of a 1.52-cm- (0.60-in-) diameter cable installed in a 2.5-cm- (1-in-) diameter hole.

5. If a tensionable unit has been installed, the cable is then ready for jacking to a predetermined load using special equipment.

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

FIELD EVALUATIONS

Several long-term field evaluations are underway at underground coal mines. In all of the cases, the cables are being used as secondary support systems in gate roads and bleeder entries. Current work is examining cable supports as primary support as a substitute for intersection trusses, and secondary support fixtures in difficult ground conditions.

CASE STUDY No. 1

Cable supports were installed to assess their effects on the stability of tailgate entries in a western U.S. underground longwall 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.

Site Description

The longwall gate roads and the test area are shown in figure 10. The final dimensions of the pillar nearest the panel were 39.6 by 39.6 m (130 by 130 ft), while those of the pillar between entries 2 and 3 were 39.6 m (130 ft) long by 30.5 m (100 ft) wide. 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 at 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 at a 2.1-m (7-ft) spacing, as shown in figure 10. The difference in cable length was attributed to the different loading conditions to which the supports would be subjected as the panel was extracted. The primary support system in the area, installed on initial development, were 1.8-m (6-ft), full-column, resin-grouted

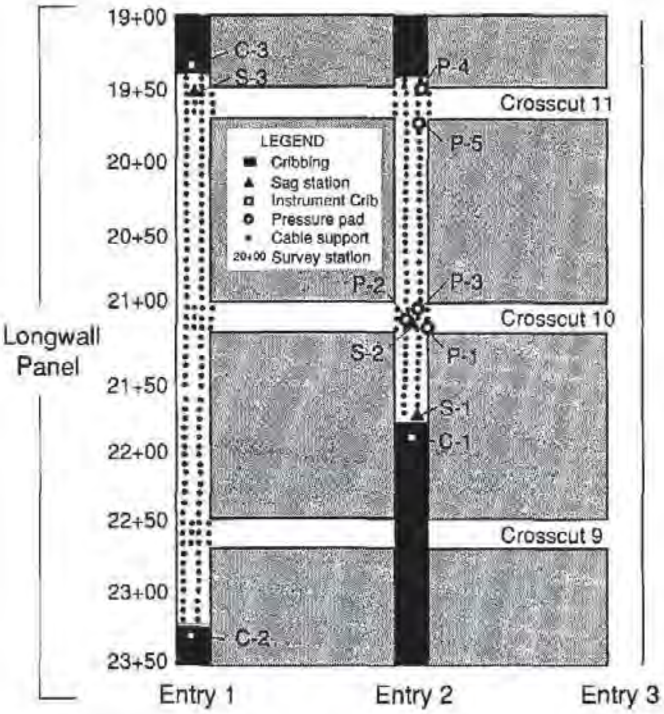

Figure 10.—Gate roads showing the cement-grouted cable test area and instrumentation layout at a western U.S. underground longwall mine.

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 mine 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 upward in grain size, with the top portion consisting of sandstone. The thickness of the sandstone in the test area was approximately 4.9 m (16 ft).

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 at 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.

Test Site Results

As the panel was extracted, 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 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 elected to provide additional supplementary support in the form of two 20.3-cm- (8-in-) diameter timber posts 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. At a distance of about 22.9 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 1.55 MPa (225 psi) on the timber post. When the face was directly adjacent to the posts, the load reached approximately 124.5 kN (28,000 lb) or 3.86 MPa (560 psi), then failed.

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. 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. The installed instrumentation indicated that very little active support can be expected from installed crib systems. Differential magnetic sag stations indicated that roof bed separations, even with the cable support stiff system, 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 124.5 kN (28,000 lb), then failed. The loading on the timber post, while appearing substantial, could have been supported by an additional cable installed in the roof of entry 1. Adding another cable with a 258.0-kN (58,000-lb) capacity in the middle of the entry may have compensated for these extra loads. 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 2.7 m (9 ft) above the coal seam. This investigation prompted the mine operator to investigate the effectiveness of cable supports in the bleeder entry behind the next longwall panel start room. The results of this investigation are described in greater detail in reference (3).

CASE STUDY No. 2

The first resin-grouted cable test area was established in a longwall bleeder entry immediately behind the 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, 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 measuring 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 mine 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 upward in grain size, with the top portion consisting of sandstone. The test area location is shown in figure 11.

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,

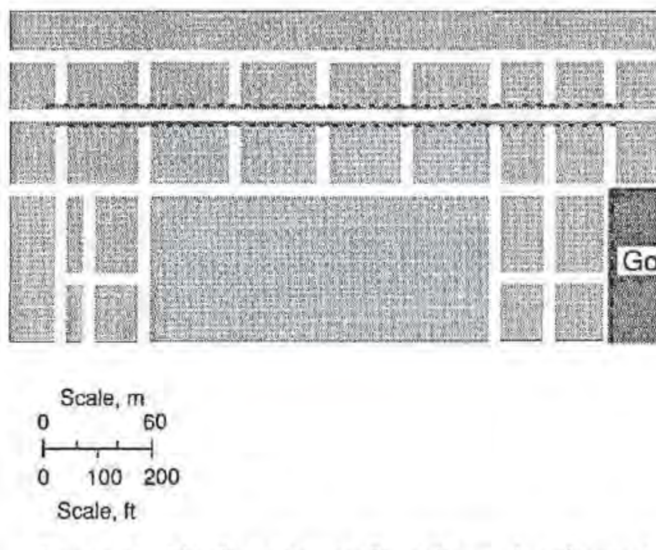

Figure 11.—Bleeder entry configuration and resin-grouted cable test area.

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 occurring about 1.2 m (4 ft) up from the roof. Additionally, the interbedded siltstones and shales could separate if the abutment forces became great 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). Considering these facts, the mine operator elected to install a 4.9-m- (16-ft-) long cable, which would intersect the sandstone at 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 at 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 12. In addition to 15.2- by 15.2-cm (6- by 6-in) bearing plates, the cables and primary support were also installed with "Monster Mats." These 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. The mats provide excellent mine roof support between the cables and maintain any failed material. Mats help prevent unraveling or progressive roof-type failures.

Instrumentation

Hydraulic U-cells were installed throughout the test area 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

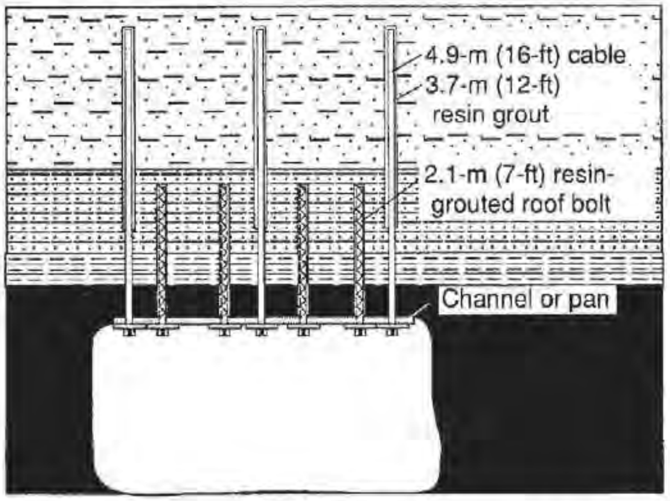

Figure 12.—Cross section showing the primary and cable support spacing and resin grout lengths.

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

Test Site Results

The test area has been monitored and evaluated 14 times. The panel has been completely extracted, and the adjacent panel is beginning to be mined. The cables will be subjected to higher loads during this period as the adjacent panel creates large spans of unsupported ground. When the immediate and main roofs cave, the support will be subjected to the highest loads. The results of this test area are expected to be published in more detail after the completion of the next panel loads have been measured and evaluated. The cables have performed as expected, with little or no sloughage of the immediate mine roof. The average load measured on the hydraulic load cells was 27.9 kN (6,230 lb) when the face had been completed. The minimum and maximum loads recorded during this same period were 5.6 kN and 129.0 kN (1,250 and 29,000 lb), respectively. The recorded loads were highly variable, with the higher readings mostly related to geologic features that occurred in the supported area.

The mine elected to continue supporting the bleeder system with resin-grouted cables. An additional 2,000 cables have been installed to maintain the bleeder behind the next set of developed panels. These areas have been instrumented to assist in evaluating the long-term stability of the bleeder entries and cable performance.

CASE STUDY No. 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 have been designed and installed to provide stability in a gate road to 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 to reduce the possibility of coal mine bumps or burst. The final yield pillar dimensions, next to the panel during the second panel mining phase, are 9.8 m (32 ft) wide by 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, are 30.5 m (100 ft) wide by 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. However, the laminated materials under this member were competent, and physical property tests indicated a compressive strength of about 103.4 MPa (15,000 psi) for tested specimens.

Cable System Design

After careful analysis of the anticipated pillar behavior and with the experience gained in the first cement-grouted test area, it was decided to test three variations of resin-grouted cables. The test area is shown in figure 13. The secondary supports were 4.9-m- (16-ft-) long, high-strength cable supports installed using three different support concepts. The first 91 m (300 ft) of the designated test area utilized 1.7 m (5 ft 8 in) of resin, leaving approximately 3.4 m (11 ft) of the cable ungrouted. This allowed the cable and the mine roof to yield and relax as abutment loads were redistributed to the gate road. The second area used a 3.7-m (12-ft) equivalent length column of resin, leaving only 1.2 m (4 ft) of the cable ungrouted. This provided a stiffer system that will resist yielding, yet still have enough ungrouted column to allow for separations at the interface between the coal, shale, and interbedded siltstone layers. The third type of support was a tensionable cable system. The cables were installed utilizing 1.7 m (5 ft 8 in) of resin grout and pretensioned using a specially designed jack system to about 35.6 kN (8,000 lb). 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.

All of the cable systems were installed with 15.2- by 15.2-cm (6- by 6-in) bearing plates and Monster Mats. The cables were placed four across the 5.5-m- (18-ft-) wide opening at a spacing of 1.5 m (5 ft). The cables at the ends were installed as closely as possible to the pillar and panel edges and angled at 80°. A cross section of the cable pattern is shown in figure 14.

Instrumentation

The test area was instrumented with 36 hydraulic U-cells and Goodyear pressure pads to evaluate individual cable loading trends and patterns. The mine roof behavior is being evaluated with 12 differential sag stations using magnetic extensometers at critical geologic interfaces. Individual cribs have also been instrumented ahead of the test area to measure the loads and stiffness of the wooden material. Additionally, the pillars and panels were instrumented with 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.

Test Site Results

The instrumentation in the test area has been read and evaluated eight times during critical phases of first panel mining. At the conclusion of the first panel mining, the instrumentation indicated that the cables have loaded an average of 24.2 kN (5,450 lb). The minimum recorded cable load was 0 kN (0 lb), where roof dilation has actually caused the cable to unload, while the maximum load was 148.6 kN (33,400 lb) where a localized separation above the primary support has caused the two adjacent cables to support the entire rock mass. The differential sag stations indicate roof separations between the coal and shale interfaces and at the shale and interbedded siltstones, which was expected. Figures 15 and 16 show a portion of the test area before and after the effects of first panel mining. The effects of second panel mining, when the area becomes a tailgate, will be evaluated as the next panel is extracted. The results of this test site are to be published in detail at that time.

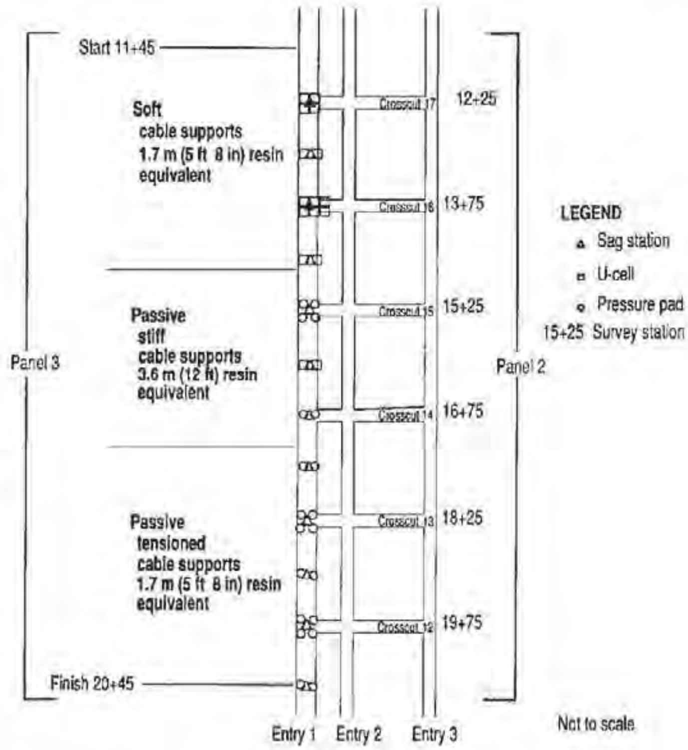

Figure 13.—Test area examining three different types of resin-grouted cable supports and respective instrumentation types and locations.

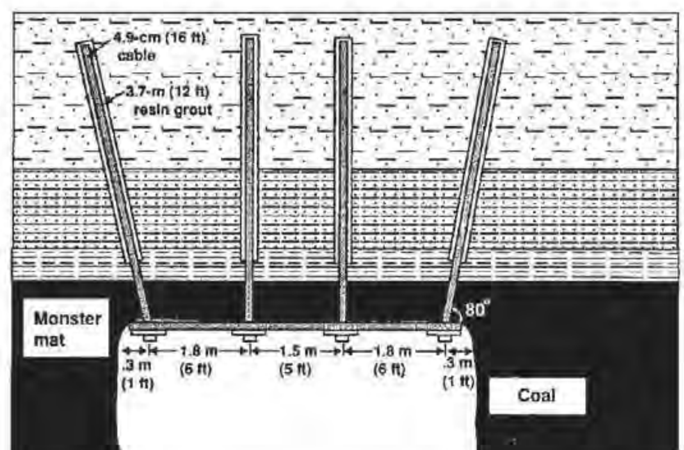

Figure 14.—Generalized cross section of the cable spacing and lengths used in the gate road test areas.

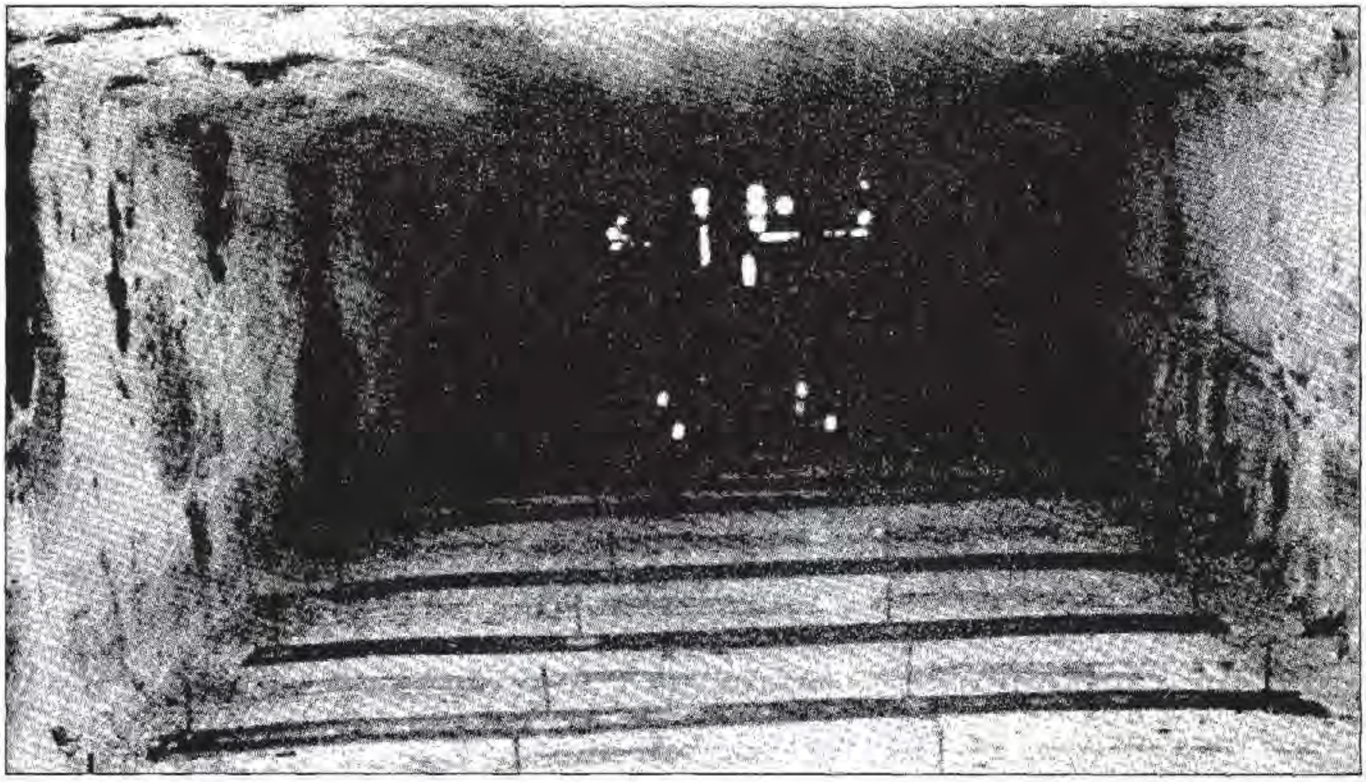

Figure 15.—Resin-grouted cable test area immediately after cable support installation (before first panel mining).

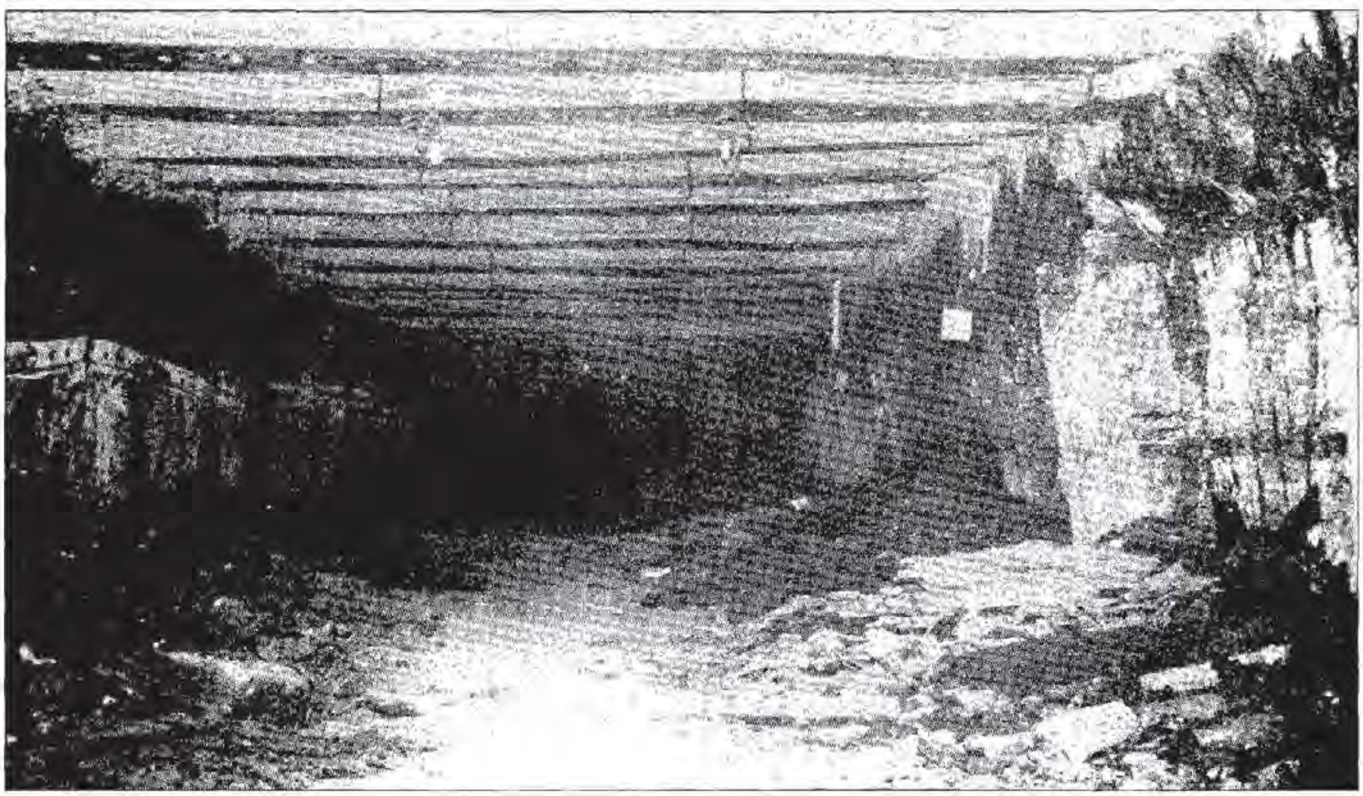

Figure 16.—Resin-grouted cable test area after the first panel has passed and the loads have been redistributed onto the pillars and next panel.

DISCUSSION 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 hazardous underground locations, 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 obtained. 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 ultimate goal of this investigation is to provide engineered and economically feasible support designs that will provide safer working areas under diverse and hazardous ground control conditions for the Nation's underground mineworkers.

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Proceedings: U.S. Bureau of Mines Technology Transfer Seminar

Compiled by Christopher Mark, Robert J. Tuchman, Richard C. Repsher, and Catherine L. Simon

UNITED STATES DEPARTMENT OF THE INTERIOR
Bruce Babbitt, Secretary

BUREAU OF MINES

Cover Photograph: The U.S. Bureau of Mines has developed highly practical technologies for maintaining effective ground control in the hazardous tailgate entries of longwall mining systems, which will significantly improve the safety of the Nation's underground mineworkers. *(Photo: Alan A. Campoli, Pittsburgh Research Center, U.S. Bureau of Mines)*

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