IC 9402

BUREAU OF MINES
INFORMATION CIRCULAR/1994

Cable Bolt Support Technology in North America

By J. M. Goris, S. D. Nickson,
and R. Pakalnis

UNITED STATES DEPARTMENT OF THE INTERIOR

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Bruce Babbitt, Secretary

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Library of Congress Cataloging in Publication Data:

Goris, J. M. (John M.), 1938-

Cable bolt support technology in North America / by J.M. Goris, S.D. Nickson, and R. Pakalnis.

p. cm. — (Information circular; 9402)

Includes bibliographical references (p. 46).

Supt. of Docs. no.: I 28.27:9402.

1. Mine roof bolting. 2. Cables. I. Nickson, S. D. (Simon D.). II. Pakalnis, Rimas Thomas, 1956- . III. Title. IV. Series: Information circular (United States. Bureau of Mines); 9402.

TN295.U4 [TN289.3] 622 s—dc20 [622] 94-8079 CIP

CONTENTS

CONTENTS—Continued

ILLUSTRATIONS

ILLUSTRATIONS—Continued

TABLES

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

cm	centimeter	L	liter
deg	degree	m	meter
GPa	gigapascal	mm	millimeter
h	hour	mm^2	square millimeter
kg	kilogram	mm/m	millimeter per meter
kg/m	kilogram per meter	MPa	megapascal
kg/m^3	kilogram per cubic meter	mt	metric ton
kN	kilonewton	pct	percent
kN/mm	kilonewton per millimeter		

CABLE BOLT SUPPORT TECHNOLOGY IN NORTH AMERICA

By J. M. Goris,[1] S. D. Nickson,[2] and R. Pakalnis[3]

ABSTRACT

Cable bolt supports are becoming an important ground control technique in underground mines in the United States and Canada. They show great versatility and are adaptable for use with many mining techniques, they have high load-carrying capacities, they can be installed in small areas with low roofs, and they are cost effective. Investigators in a number of research institutes in North America have conducted studies to assess the material and support properties of cable bolts, evaluate their support capabilities under various mining conditions, and provide design criteria for using cable bolt supports as roof control systems under varying mining conditions. While conducting these studies, personnel have collected important information on the history, application, performance, and economics of these supports. This U.S. Bureau of Mines report documents this information so that ground control personnel unfamiliar with cable bolt supports can gain an understanding and appreciation of their use. A comprehensive bibliography of technical information on cable bolt supports was selected by the authors and is also presented.

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INTRODUCTION

Cable bolt supports consist of a steel cable grouted into a drill hole (fig. 1). Their primary purpose is to prevent rock falls and slippage along joints.

Cable bolt reinforcements have a very long history of use in the construction of dams, tunnels, and other structures, especially where slopes must be stabilized. However, even though Garcia (1929) reported the use of cable slings to support the backs of underground coal mines as a convenient way of utilizing scrap hoist rope in 1929, cable bolt supports were only introduced to the mining industry during the 1960's as a means of reinforcing rock prior to excavation. One of the earliest applications in North America was at the Geco Mine in Ontario in 1963 (Gramoli, 1975). Here, discarded and degreased 25-mm-diam hoist ropes (fig. 2) were used to control sloughing and caving of backs and walls in large, blasthole open stopes.

Discarded rope was the preferred choice of most ground control personnel in a number of mines in the early days. Later, it was realized that not only was degreasing these ropes expensive, but using such rope could be dangerous because often its residual strength was unknown. Ground control engineers then turned to the prestressing cable used today.

Since the early 1970's, many improvements have been made in cable bolt technology. Various researchers in North America, including researchers from the U.S. Bureau of Mines (USBM), have conducted extensive studies at laboratories and at field sites to understand the behavior of cable bolt supports and to provide design criteria for their use. Such work supports the USBM's mission of investigating ways to increase safety in underground mines and has resulted in improvements in installation practices and performance of cable bolts. The information presented in this Information Circular (IC) documents both USBM research and that of other investigators so that ground control personnel unfamiliar with cable bolts can gain an understanding and appreciation of their use.

Today, the basic cable bolt support consists of a high-strength steel cable(s) placed in a drill hole and grouted with a neat cement grout (fig. 1) or, more recently, with resin grouts. These supports vary in length, but cables 20 m long or longer are common. The steel cable is very flexible and can be coiled into a roll about 1.2 m in diameter (fig. 3). This flexibility is one of the primary advantages of cable bolt supports because long supports, 20 m or longer, can be installed in a drift having less than 2.5 m of headroom. Consequently, cable bolts are used in such applications as prereinforcement of rock before mining, where they can provide support around an opening after a portion of the rock has been blasted.

Figure 4 illustrates how cable bolts are applied to breast mining in a cut-and-fill stope. The bolts are placed at specified intervals in a fan-shaped configuration. The ore is drilled horizontally, blasted, and then removed. During this operation, the rock in the back and the hanging wall is supported by the cable bolts. After the ore is removed, conventional rock bolts are installed to help support the immediate back. Once a lift has been mined out, the stope is backfilled and mining continues. Figure 5 shows a typical profile of a stope after the cable bolt and rock bolt supports have been installed; figure 6 shows how the roof of the stope would appear.

Cable bolts are also used with cable sling support systems (fig. 7) (Castle and Scott, 1989), as well as in surface mines to help stabilize highwalls. Figure 8 shows groups of cable bolts installed in a highwall of a surface mine in Quebec. The cables were installed and then tensioned after the grout had cured.

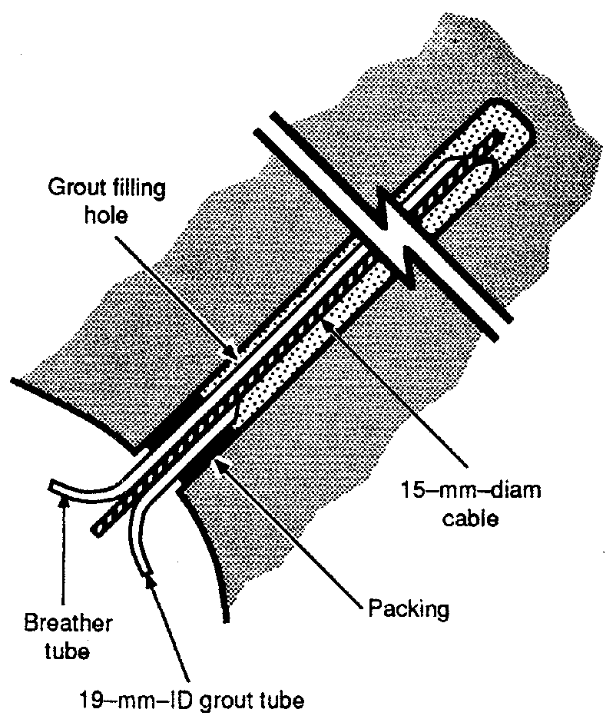

Figure 1.—Cutaway view of cable bolt support.

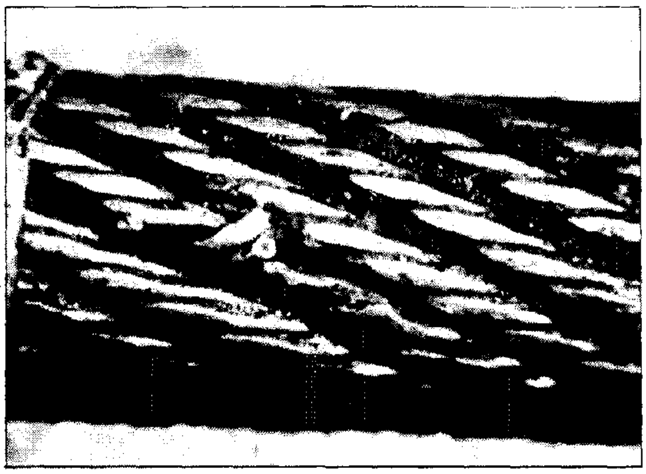

Figure 2.—Used wire rope before degreasing.

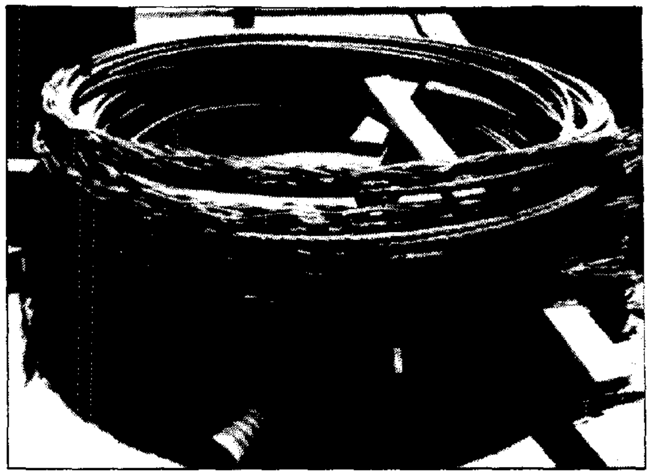

Figure 3.—Small coils of steel cable.

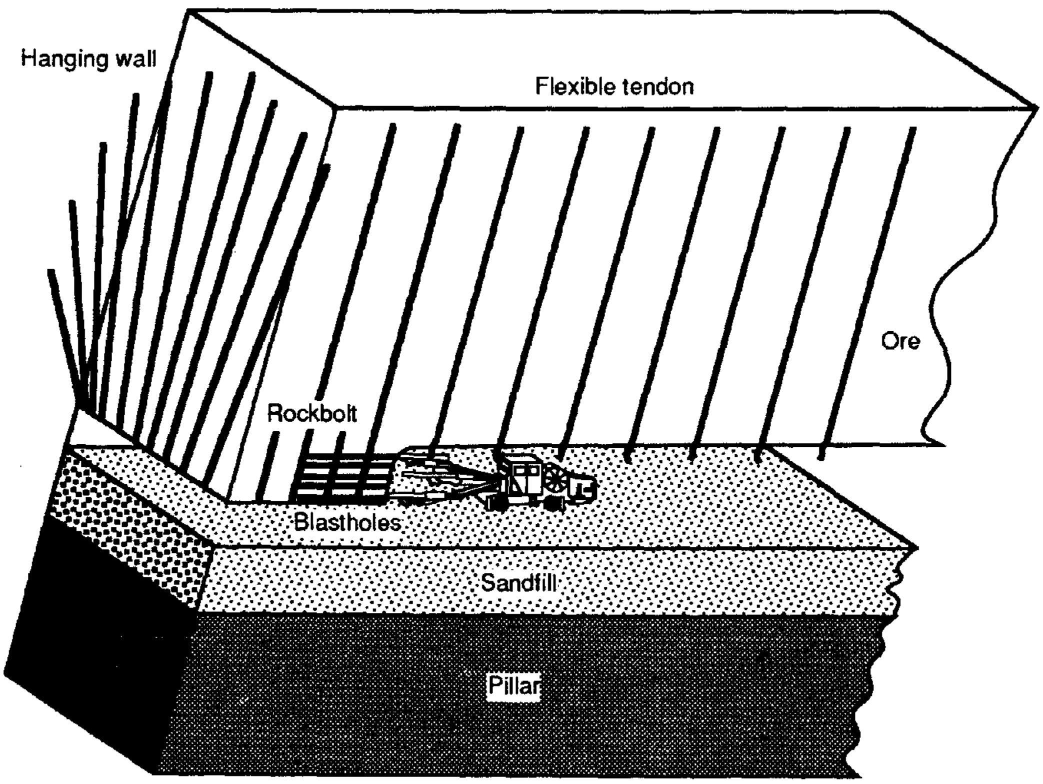

Figure 4.—Use of cable bolt supports in breast stope mining.

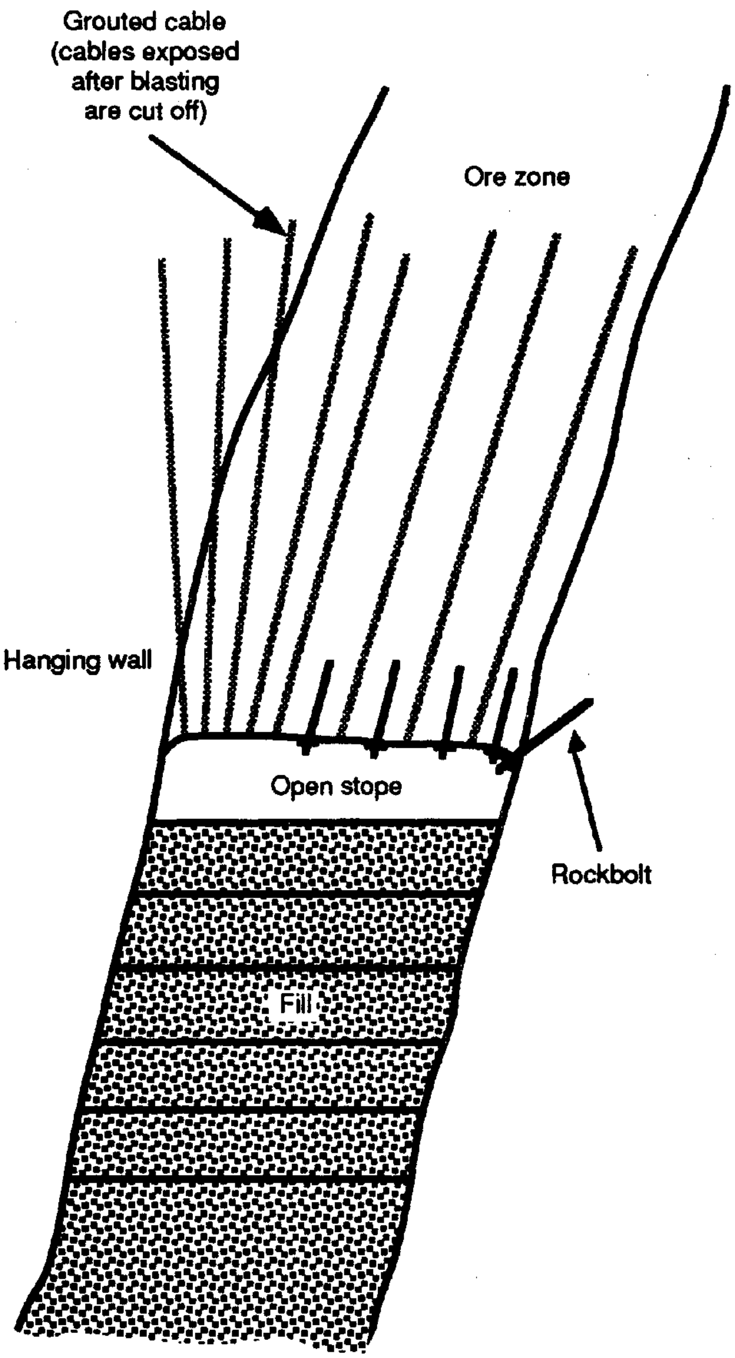

Figure 5.—Typical profile of stope after installation of cable bolt supports.

Figure 6.—Roof of stope after installation of cable bolt supports.

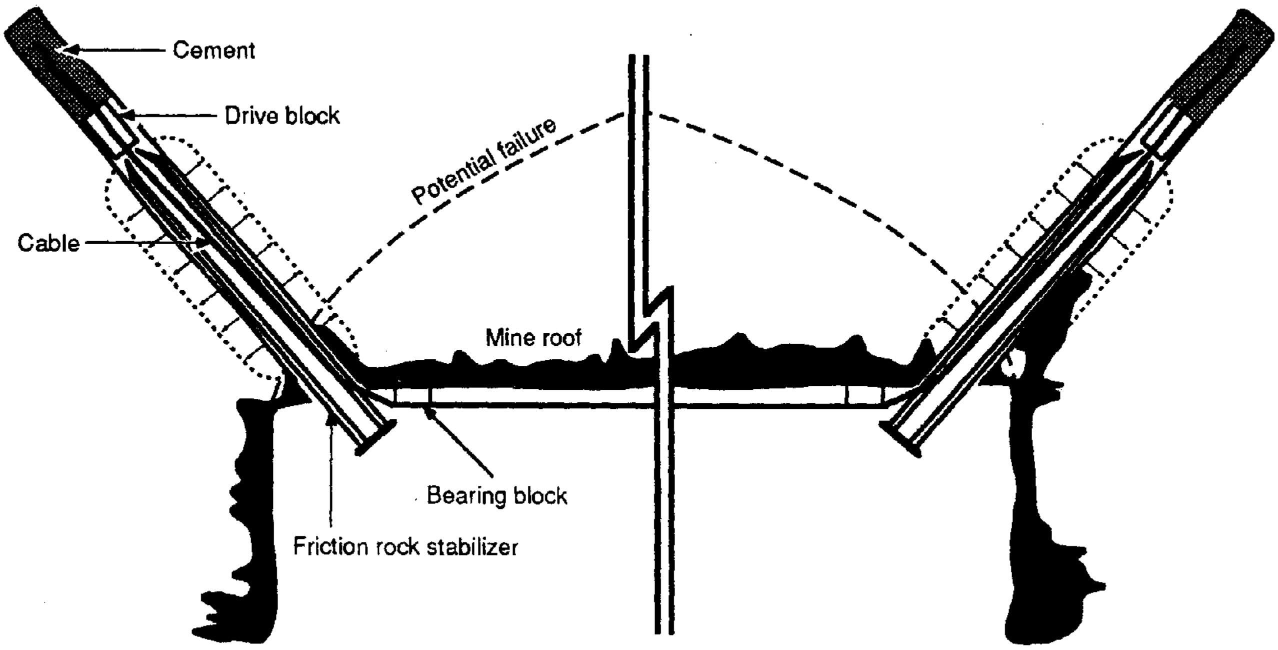

Figure 7.—Cable sling support (Castle and Scott, 1989).

Figure 8.—Cable bolt supports installed in highwall of surface mine.

TYPES OF CABLES

The surface condition and configuration of a cable have a great influence on its load-carrying capabilities, and for this reason, cables have been altered to improve performance. Six types of cable bolts are available (fig. 9).

CONVENTIONAL CABLES

Ordinary steel cables are the most common type used in the mining industry as cable bolt support. These cables have an average ultimate strength of 258 kN and a modulus of elasticity of approximately 2.03×10^5 MPa. They are usually 15 to 16 mm in diameter, have a cross-sectional area of 141.9 mm^2, and consist of seven wires (fig. 10). They are, however, susceptible to corrosion, and their load-carrying capacity can be adversely affected by grease and mud.

EPOXY-COATED CABLES

Recently, an epoxy-coated cable was marketed to provide corrosion resistance in prestressed concrete members. Initially, the coated cable did not provide enough shear resistance against pullout, so the manufacturer developed a method for embedding silica grit into the outer surface of the coating to provide this resistance (fig. 11). The shear resistance can be altered by varying the size and concentration of grit (Dorsten and others, 1984).

Pull tests were conducted on samples of these cables to determine if they could be used as supports in mines (Goris, 1991). The test results showed that pullout resistance was approximately 31 pct higher than for conventional bare cables. This means that, given the same embedment length of cable, the epoxy-coated cable would have a greater load-carrying capacity under identical conditions. Figure 12 shows one of the epoxy-coated cable samples cut in half longitudinally after it was tested. This cable was actually pulled through the grout for approximately 15 cm, but the surface contact between the grout and the epoxy coating was still excellent. The epoxy coating had not pulled away from the cable. All slippage took place between the cable and the grout.

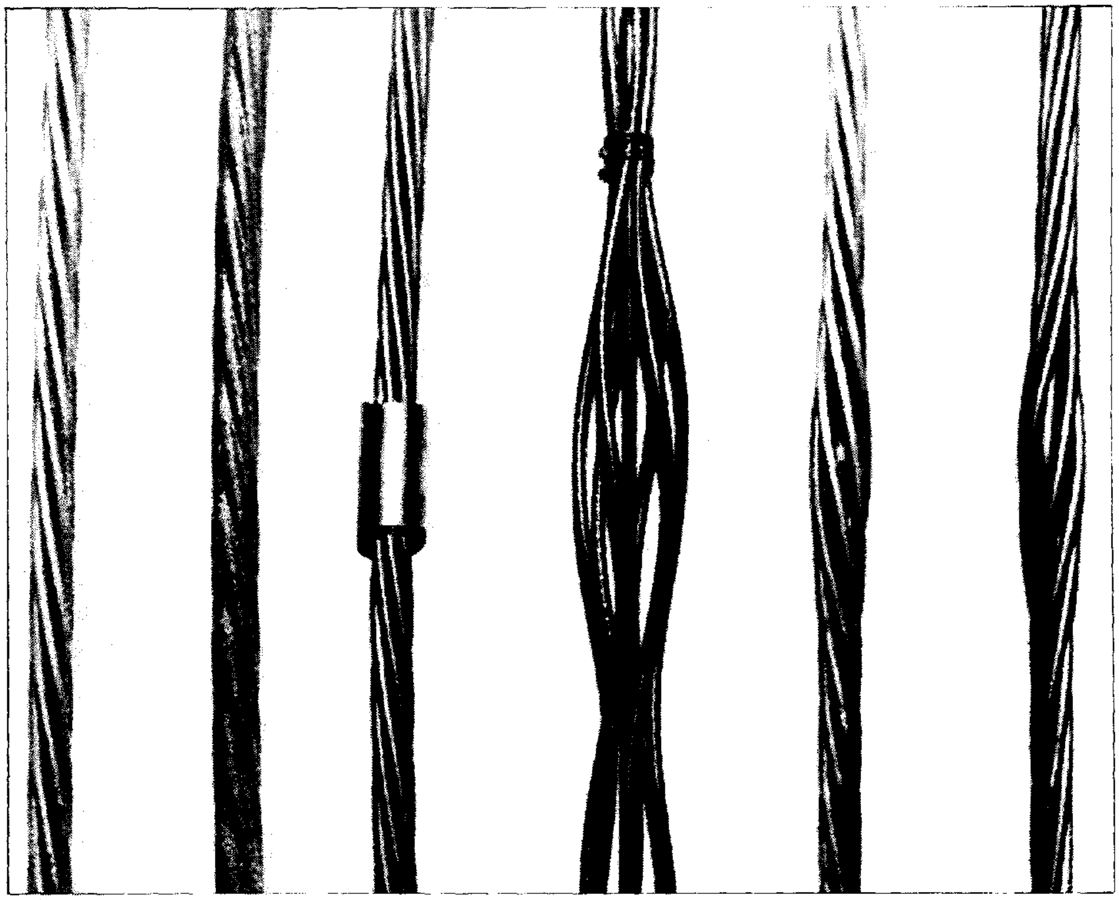

Figure 9.—Six types of cables. Left to right, conventional, epoxy-coated, conventional with button, birdcage, nutcase, and Garford bulb.

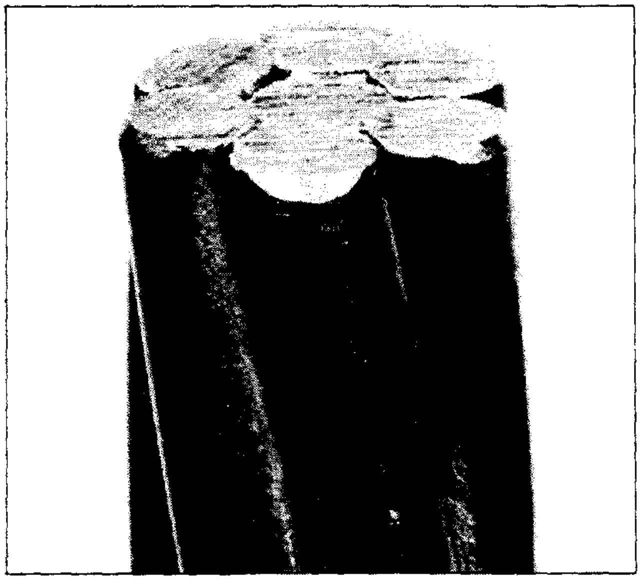

Figure 10.—Seven-wire steel cable.

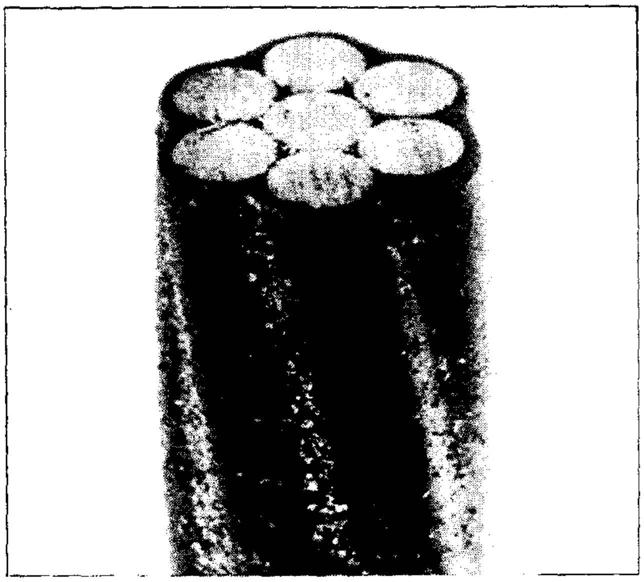

Figure 11.—Epoxy-coated cable.

Figure 12.—Cross section of epoxy-coated cable.

The manufacturer of the epoxy-coated cables had the product tested for chemical resistance, flexibility of the coating, abrasion resistance, and many other conditions, as required by the Federal Highway Administration,[4] and the cable passed (Dorsten and others, 1984).

CONVENTIONAL CABLES WITH STEEL BUTTONS

Conventional cables have been modified by pressing steel buttons onto them to help improve pullout resistance (fig. 13). To a great extent, resistance depends on surface conditions of the cable, properties of the grout and rock, direction of applied load, and location of the button on the

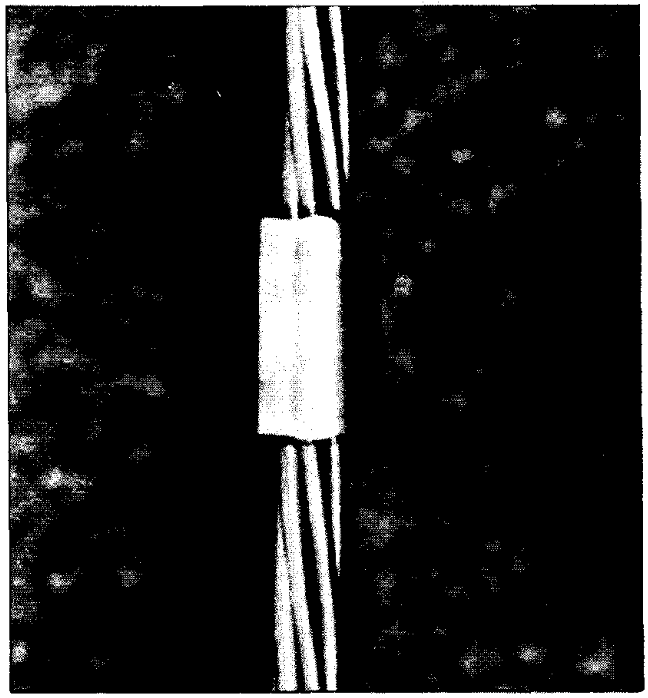

Figure 13.—Conventional cable with button.

cable. The developers of the button realized that the pullout resistance of cable bolts could be increased by adding a bearing surface perpendicular to the axis of the cable. The device selected was a steel button, 25 to 32 mm in diameter by 38 to 45 mm long; however, barrel-and-wedge grips used for tensioning cables can also be used for this purpose. The buttons are pressed onto the cables at specified intervals by a force of approximately 890 kN.

The spacing of buttons along the cable greatly influences pullout resistance. Factors to be considered are the number of fractures in the rock mass and their locations with respect to rock failure planes. If, for example, the button is located above a failure plane and not in the portion of the rock mass that is moving, the button will have no effect on the ability of the cable to support the rock mass. If the button is located within 25 mm or so of a failure plane, the grout column between the button and the failure plane would be too short to take much load. The farther the button is positioned away from the failure plane, the more load the grout column will be able to withstand.

Test data from laboratory pull tests indicate that the failure mechanics of a button-cable system are complex and that resistance to pullout is initially the result of mechanical interlock along the grout-cable interface (Goris, 1991). However, once slippage begins, resistance to pullout is affected by a combination of friction along the grout-cable interface and compressive force applied against

[4]Many of the conditions found during the installation of cables during highway construction are also found in mining. Such conditions include placement of cables in rock masses and soil embankments, exposure of cables to corrosive chemicals, flexing of cables during installation, and abrasion of cables when the rock mass being supported begins to move. Therefore, the results from tests required by the Federal Highway Administration to determine the capabilities of the cables are applicable to mining.

the grout column by the steel button. The use of buttons, therefore, significantly alters the behavior of conventional cables (Goris, 1991).

The location of the button within the grout column is an important key to success, and ensuring that the buttons are placed properly is a difficult task, which may be a major disadvantage to their use. Another disadvantage is that the cable must be precut so that the buttons can be placed on it. The cable must then be handled in separate pieces rather than in a continuous coil.

BIRDCAGE CABLES

Birdcage cables are another modification of conventional cables (fig. 14). The "birdcage" is formed by separating the seven wires of a conventional cable and then recombining them to form an open cable with a series of nodes and antinodes spaced at about 18-cm intervals. These nodes behave just like anchors and significantly increase the pullout resistance of the cable. In fact, laboratory pull tests conducted on birdcage cables showed an average increase in maximum pullout resistance between 36 and 79 pct over pullout resistance of conventional cables (Goris, 1991). Birdcaging can begin or end at any point along the length of the cable.

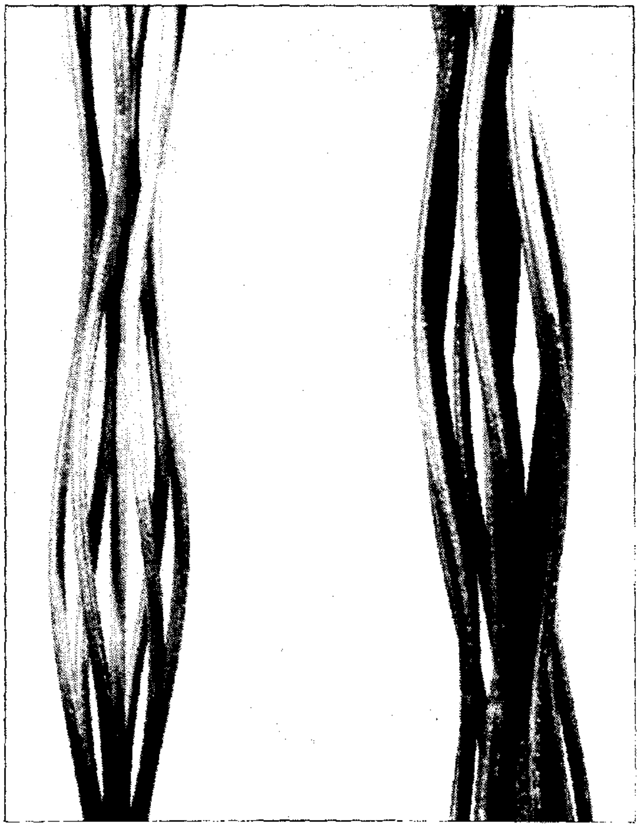

Figure 14.—Single (left) and double (right) birdcage cables.

The major advantages of birdcage cables are—

1. When birdcage cables are placed in drill holes at or near horizontal, almost 100 pct of the surface area of the cable can be grouted because only a small area of a few outer wires at the nodes will rest on the wall of the hole.
2. Birdcage cables are not as sensitive to grease, rust, mud, etc., on the wires as are conventional cables because the failure mechanism is different.
3. Birdcage cables do not contribute to grout bleeding.

Major disadvantages are that the cables must be made to specific lengths and therefore can not be handled in a continuous coil, they require a larger drill hole because of the nodes, and they are more difficult to push into a drill hole than are conventional cables.

The cost of birdcage cable in North America is approximately 76 pct greater than the cost of conventional cables; however, the cost of the cable represents only about 15 pct of the total cost of a cable bolt support. Therefore, the actual increase in cost of a birdcage cable support would be about 11.2 pct. This makes birdcage cable bolts cost effective, depending on installation costs.

NUTCASE CABLES

The nutcase cable is composed of a conventional seven-wire cable with a hexagonal steel nut placed on the center wire (kingwire). In producing this cable, a conventional cable is unwound, a hexagonal nut is placed on the kingwire, and the cable is rewound. This process forms a node (nutcase) along the cable (fig. 15) that behaves as an anchor and significantly increases the pullout resistance of the cable.

The nutcase cable concept was introduced in Canada in 1991, and both laboratory and field tests were conducted by researchers at Queen's University, Kingston, ON, to determine the support capabilities of these cables. Results from laboratory tests by Hyett and others (1993) indicate that—

1. The optimum size nut is 12.7 mm, making the diameter of the nutcase 23.02 mm.
2. The nutcases should be formed at 30- to 40-cm intervals along the cable.
3. The load-carrying capacity of nutcase cables is less sensitive to variations in water-cement ratios than is the load-carrying capacity of conventional cables.
4. The recommended water-cement ratio is 0.35 to 0.40 for mine installation.

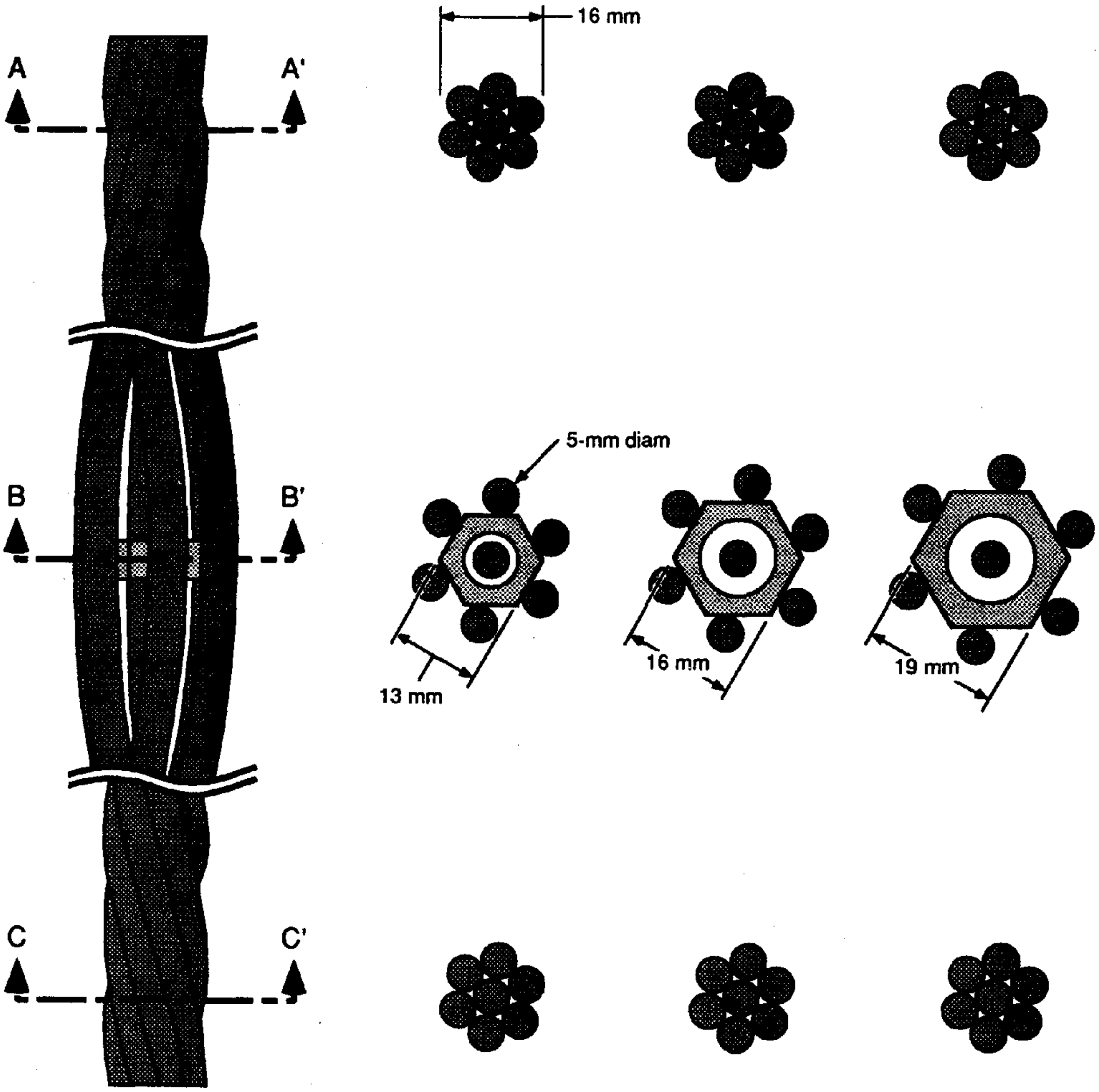

Figure 15.—Nutcase cable.

Results from field tests conducted at the Hemlo Golden Giant Mine in Canada by Hyett and others (1993) indicate that—

1. The nutcase cable achieved average increases in load-carrying capacity of approximately 97 and 54 pct over conventional cables in a hanging wall and an ore zone, respectively, and approximately 10 and 14 pct over birdcage cables in a hanging wall and an ore zone, respectively (table 1).
2. Nutcase cables performed very well in poor-quality, destressed, or failed rock masses.
3. Slippage of the nutcase cable occurred after 25 to 40 mm of axial deformation, indicating that these cables could be considered a yielding support system, especially in poor ground.

Table 1.—Cable bolt pull-test results from 4775 level of Hemlo Golden Giant Mine, load in metric tons

Location	Nutcase	Birdcage	Conventional
Hanging wall	30.4	27.7	15.4
Ore zone	27.6	24.2	17.9

Source: Hyett and others, 1993.

Cost figures for nutcase cables supplied by Thiessen Ltd. show that these cables are approximately 35 pct more expensive than conventional cables. However, field studies indicate that their load-carrying capacities are considerably greater than those of conventional cables.

GARFORD BULB SUPPORTS

Figure 16 shows the Garford bulb cable support,[5] which is manufactured in Australia. This support is similar in concept to a birdcage cable in that bulbs or nodes are formed along the cable to provide improved anchorage and increased shear resistance. The bulbs are formed mechanically by gripping the cable with two separate grips spaced at a given distance, usually 50 mm or more, and forcing the grips toward each other. This deforms the seven wires of the cable into a bulb (fig. 16) approximately 25 mm in diameter. The spacing of the bulbs along the cable is determined by anchorage requirements. The manufacturer recommends a spacing of three bulbs per meter for maximum effectiveness. When twin cables are used, the bulbs can be offset to minimize hole diameter size. The manufacturer recommends a minimum hole diameter of 28 mm for a single cable and 50 mm for twin cables. However, the drill hole diameter will have to be larger than recommended to accommodate grout and breather tubes.

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

Figure 16.—Garford bulb cable support.

Strata Control Technology in Australia conducted tests to determine the load characteristics of twin Garford bulb cables using different bulb spacings. The company also compared load characteristics of twin Garford bulb cables and twin birdcage cables (Tarrant and Fabjanczyk, 1990). Table 2 is a summary of test conditions.

Table 2.—Summary of test conditions for twin Garford bulb and twin birdcage cables

Test series	Cable type	Grout strength, MPa	Embedment length, cm	Bulb spacing, m	Maximum load, kN
1	Garford	61	38	3.9	483
2	Garford	61	38	3.9	500
3	Garford	55	38	5.3	519
4	Garford	[e]55	50	6	508
5	Birdcage	[e]55	50	NAp	516

[e]Estimated. NAp Not applicable.

Source: Tarrant and Fabjanczyk, 1990.

In these tests, the cables were grouted into two thick-walled steel pipes. Procedures were similar to those used by Goris (1990, 1991). Figure 17*A* shows plots of the load-displacement characteristics for test series 1 through 4. The results (Tarrant and Fabjanczyk, 1990) indicate that—

1. The onset of yield occurred at higher loads when there were more bulbs per cable length.
2. The displacement at yield was marginally higher when there were more bulbs per cable length.
3. After the onset of yield, the stiffness of the system was a function of bulb spacing.

Figure 17*B* compares plots of the load-displacement characteristics for birdcage cables (test series 5, table 2). Comparing the load-displacement characteristics of both birdcage and Garford bulb cables (fig. 17*A* and 17*B*) indicates that the performances of the 2 types of cables are similar; that is, a twin Garford bulb cable with 6 bulbs per meter is comparable in performance to a twin (14-wire) birdcage cable.

Cost comparisons for Garford bulb, birdcage, and conventional cables were not provided. However, the manufacturer of Garford bulb cables reported that the cost of the Garford bulb cable depended on the length of the bolt, and that in Australia the Garford bulb cable was marginally more expensive than conventional cable but less expensive than birdcage cable.

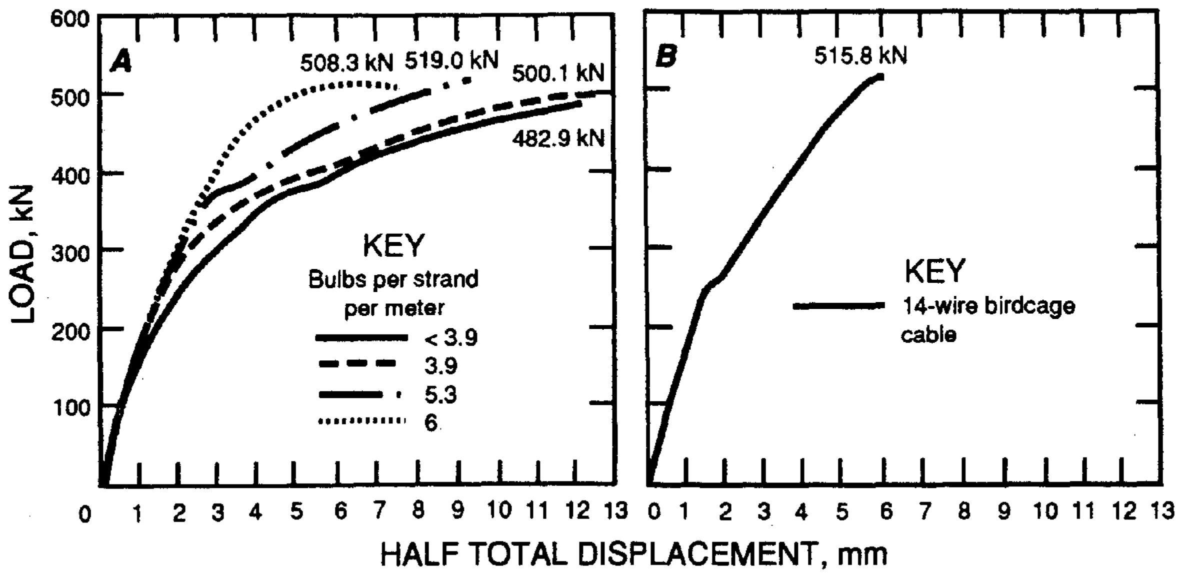

Figure 17.—Load-displacement plots for (*A*) twin Garford bulb cables and (*B*) twin 14-wire birdcage cable.

GROUTS

The grout used for cable bolts consists of portland cement and water. Cement-based grouts are mixed at ratios between 0.3 and 0.45 part of water to 1 part of cement by weight. Table 3 shows the required amount of water necessary to achieve a given water-cement ratio.

Table 3.—Amounts of water required to achieve given water-cement ratios using one sack of cement weighing 42.6 kg

Ratio	Water by weight, kg	Water by volume, L
0.3:1	12.8	12.8
0.35:1	14.9	14.9
0.4:1	17.0	17.0
0.45:1	19.2	19.2

Some mines use a grout with a high water-cement ratio so that pumping problems can be kept to a minimum. Unfortunately, large amounts of water reduce compressive and tensile strengths in grout, which in turn reduces the load-carrying capacity of the support (Goris, 1990). Extensive laboratory tests on grout samples (Goris, 1990) showed a direct correlation between grout water-cement ratios, grout strength, and maximum load-carrying capacity of cable bolt supports. The tests involved grouts made from Type I-II portland cement and water. The results are shown in table 4.

High-water-content grouts also cause increased water bleeding and cement particle sedimentation, a process in which cement particles in the grout settle and water rises to the surface. It is expressed quantitatively as the degree of settling in millimeters of grout per meter of grouted cable. This settling process results in less grout around the top portion of the cable, which reduces the total support length of the cable bolt. Test results show an average bleeding of 15.8, 32.5, 63.3, and 95.8 mm/m for grouts mixed with water-cement ratios of 0.3, 0.35, 0.4, and 0.45, respectively. On the basis of these data, a 20-m-long cable bolt embedded using a 0.45 water-cement ratio grout would have approximately 1.92 m of ungrouted cable at the top of the hole, compared with 0.32 m of ungrouted cable using a grout mixed with a water-cement ratio of 0.3 (Goris, 1990).

Table 4.—Correlation among water-cement ratio, grout strength, and maximum load-carrying capacity of cable bolts embedded 30 cm

Ratio	Compressive strength of grout, MPa	Maximum load-carrying capacity of bolt, kN
0.3:1	67.9	163.8
0.35:1	56.4	142.7
0.4:1	52.3	116.1
0.45:1	47.9	88.2

Thick grout (grout mixed with water-cement ratios between 0.3 and 0.35) gives grouting crews the option of either filling the cable bolt hole from the bottom up and using a breather tube, or filling it from the top down and not using a breather tube. In the latter case, the thick grout stays in the hole and does not run out. Another advantage of using a low water-cement ratio grout is improved quality control of the support system. If water is inadvertently added to a grout with a low water content, the resulting grout is still likely to have high strength and provide effective support.

Based on laboratory tests and observations from various mines, low water-cement ratio grouts offer several advantages for cable bolt supports (Goris, 1990). These include—

High load-carrying capacities.
High grout strengths.
Less water bleeding and particle sedimentation.
Greater flexibility in placing grout.
Improved quality control of grout.

A major disadvantage of a low water-cement ratio grout is that it is more difficult to achieve consistent mixes and to pump. It is recommended that if a low water-cement ratio grout is used, a grout tube with a rating of at least 1 MPa be used to avoid bursting tubes and possible injury to personnel.

LOAD-CARRYING CHARACTERISTICS OF CABLE BOLT SUPPORTS

For a cable bolt support system to be effective, the load must be transferred from the rock to the cable through the grout. Therefore, the capacity of the system is governed primarily by four factors:

1. Rock-to-grout bond.
2. Grout-to-cable bond.
3. Strength of the cable.
4. Strength of the grout.

The rock-grout and grout-cable bonds are primarily mechanical rather than adhesive bonds. The cables shown in figure 6 are considered to be passive; that is, they are not under load as yet. In this state, any loading will depend on movement of the rock, which will transfer the load of the rock through the grout to the cable. The resistance the cable offers to rock fall depends on mechanical interlocking between the grout and the cable and the grout and the rock. As the rock moves, loads on the cable will increase, the bond between the cable and grout will begin to break, and that segment of the cable closest to the rock joint where separation is taking place will begin to elongate. Depending on the mass of the rock, the load on the cable will increase rapidly after only 2 mm or so of displacement.

As the load increases, the bond between the grout and cable will be broken, first near the rock joint and then in both directions along the length of the cable away from the joint. Once again, depending on the mass of the rock, displacement may cease, or it may continue until eventually the rock slips from the cable or the weight of the rock breaks the cable.

Resistance to rock slippage along the cable is the result of several factors:

1. The grout-cable bond.
2. Friction between the steel and the grout.
3. Friction between the cable and the grout because of mechanical interlocking. This amount of friction will vary depending on the configuration of the cable bolt.
4. Dilation caused by grout movement along the cable. The resulting normal force pushes the grout and cable together.

Tests to investigate the load-displacement relationship between the cable and the rock were conducted at U.S. Bureau of Mines laboratories (Goris, 1990). A typical curve is illustrated in figure 18. A schematic of the test apparatus is shown in figure 19 and simulates a rock being supported by a single steel cable. As the lower rock begins to move, the load on the embedded cable begins to increase, as indicated by the curve in figure 18. This curve represents the relationship between load and rock movement (displacement). As the load increases between points *A* and *B* (fig. 18), the bond between the cable and grout begins to break near the joint and that segment of the cable closest to the rock joint begins to elongate in both directions. The slope of the curve at this point is approximately 14.6 kN/mm.

At point *B*, the bond along the entire 305-mm length of the cable breaks, and the rock and the grout column begin

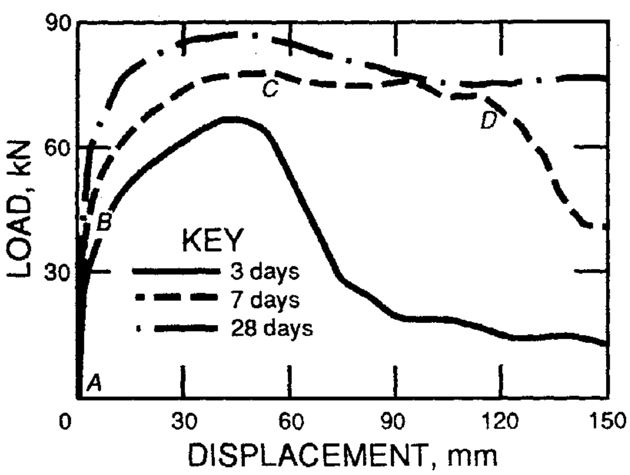

Figure 18.—Typical load-displacement relationship for conventional cable bolts. *A* through *D* are points of loading described in the text (Goris, 1990).

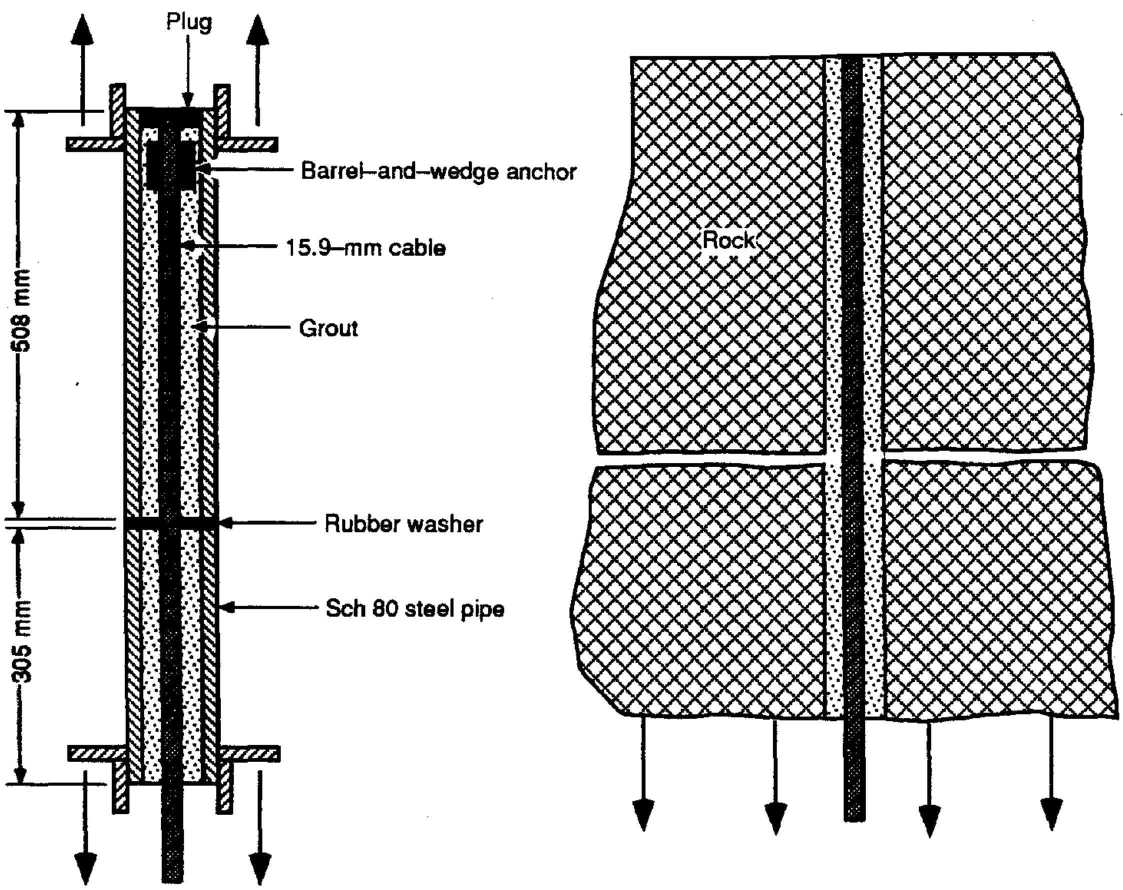

Figure 19.—Cable bolt pull-test apparatus.

to slip along the surface of the cable. The cable does not pull out of the upper rock because the embedment length is much greater here than in the lower rock.

As loading continues past point *B*, the lower rock continues to slip along the grout-cable interface. However, because neither the cable nor the grout column can rotate, the cable begins to shear the ridges of grout between individual wires. This causes an increase in pressure along the grout-rock interface (Goris and others, 1992) (fig. 20). These ridges are similar to riflings in the barrel of a rifle. As loading continues, the grout particles sheared from the ridges become wedged between individual wires of the cable, thereby increasing resistance to movement.

At point *C* (fig. 18), the maximum load-carrying capacity is reached; however, because of dilation of the sheared grout particles and confinement of the grout column, the frictional resistance and, consequently, the load remain high between points *C* and *D*. This is referred to as residual load-carrying capacity.

As displacement continues, friction causes load transfer between the cable and the grout, and the surface of the grout begins to smooth. At point *D*, the load begins to drop rapidly. Note, however, that displacement to this point is approximately 11.4 cm. This characteristic (high residual loads at large magnitudes of displacement) is an excellent attribute in many rock support situations because the supports allow the rock to deform, thereby redistributing loads to the surrounding rock.

The load-displacement behavior just described is quite typical for cable bolt supports. Major deviations from typical behavior can be caused by the use of different cable configurations (such as birdcage cables), buttons on the cables, and the number of cables placed in the hole.

Cable bolt supports work well in fractured ground because the entire length of the cable is bonded to the rock with grout. Also, cable bolt support patterns can be designed to respond to various types of ground movement. In highly stressed rock, single cables in each hole allow large amounts of rock deformation to occur, thereby redistributing load to the pillars. Double cables, on the other hand, have very high load-carrying capacities at low amounts of displacement, which restricts displacement for a given load.

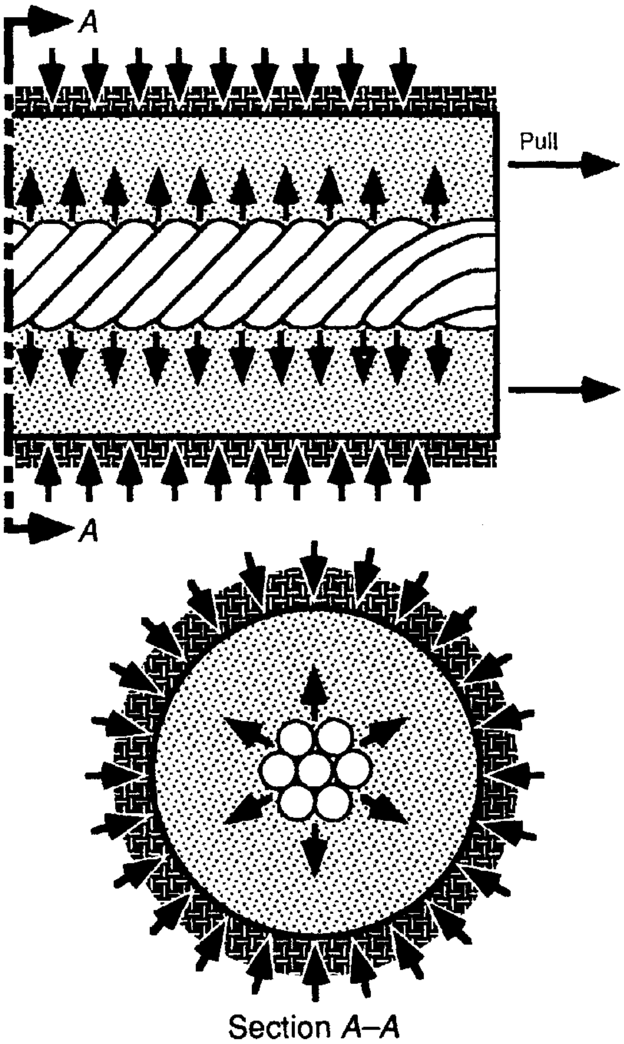

Figure 20.—Effects of pulling on grouted cable (Nickson, 1992).

EFFECTS OF EMBEDMENT LENGTH ON LOAD-DISPLACEMENT BEHAVIOR

The load-displacement relationship shown in figure 18 represents an embedment length of 30 cm and a corresponding maximum load-carrying capacity of approximately 88 kN after 28 days of curing. Varying the embedment lengths will influence the maximum load-carrying capacity of cable bolt supports; this was determined through laboratory pull tests conducted on 60 cable bolt samples (Goris, 1990). Each test sample contained a 15.9-mm-diam cable embedded in a cement-based grout with a water-cement ratio of 0.45. The embedment lengths ranged between 20.3 and 76.2 cm at increments of 5.1 cm.

The average load-displacement curve for each embedment length is shown in figure 21. Each of the 12 curves represents an average of 5 samples. It is obvious from these test results that the longer the embedment length, the greater the load-carrying capacity of the support. (The symbols representing each curve in figure 21 do not represent individual data points, but denote individual curves. Each curve is represented by approximately 600 data points). The maximum load-carrying capacity for reach of the 60 samples tested was determined and plotted against embedment length (fig. 22). As seen, none of the cable bolt samples were tested to the ultimate capacity of the cable (258 kN). The highest capacity achieved was approximately 192 kN at an embedment length of 76.2 cm.

CRITICAL EMBEDMENT LENGTH

Embedment length is an important factor in designing cable supports because the load-carrying capacity of these supports depends on the pullout resistance per unit length of embedded cable. The pullout resistance in turn depends on such factors as grout strength; cable geometry; rock strength; and number, orientation, and condition of the rock joints. The required embedment length is referred to as the critical embedment length (fig. 23). In figure 23, a rock exerting a force of 150 kN is supported by a conventional cable. In figure 23*A*, the embedment length is 0.75 m. Using the data presented in figure 22, it can be seen that the load-carrying capacity of a cable bolt embedded 0.75 m is approximately 188 kN, which exceeds the weight of the rock; therefore, the cable should hold the rock. Also, the weight of the rock does not exceed the average load-carrying capacity of the cable, which is approximately 258 kN, meaning that the weight of the rock will not break the cable. In figure 23*B*, the same rock has been rotated 90° and now the embedment length is 0.5 m. According to the data in figure 22, a 0.5-m embedment length will support only about 134 kN; however, the weight of the rock is 150 kN. Therefore, the rock will slide off the cable.

This example illustrates the importance of embedment length, which is usually dictated by joining within a rock mass, that is, where the rock mass is most likely to separate and form blocks. The size of these blocks must be considered in designing cable bolt supports.

Figure 24 shows a cross section of a cut-and-fill stope. The combination of bedding planes and rock joints determines the number and size of the rock blocks in the walls and back of the stope. Placing cable bolts in the back (fig. 24*B*) will provide support for the blocks, but the number installed may not be sufficient to support all the blocks. Block 1 (fig. 24*B*) will most likely be stable because the embedment length is long. Block 2, however, may not be stable even though three cable bolt supports have been placed. In this example, the combined embedment lengths of the three cables may not be adequate to prevent slippage along the grout and cable interface.

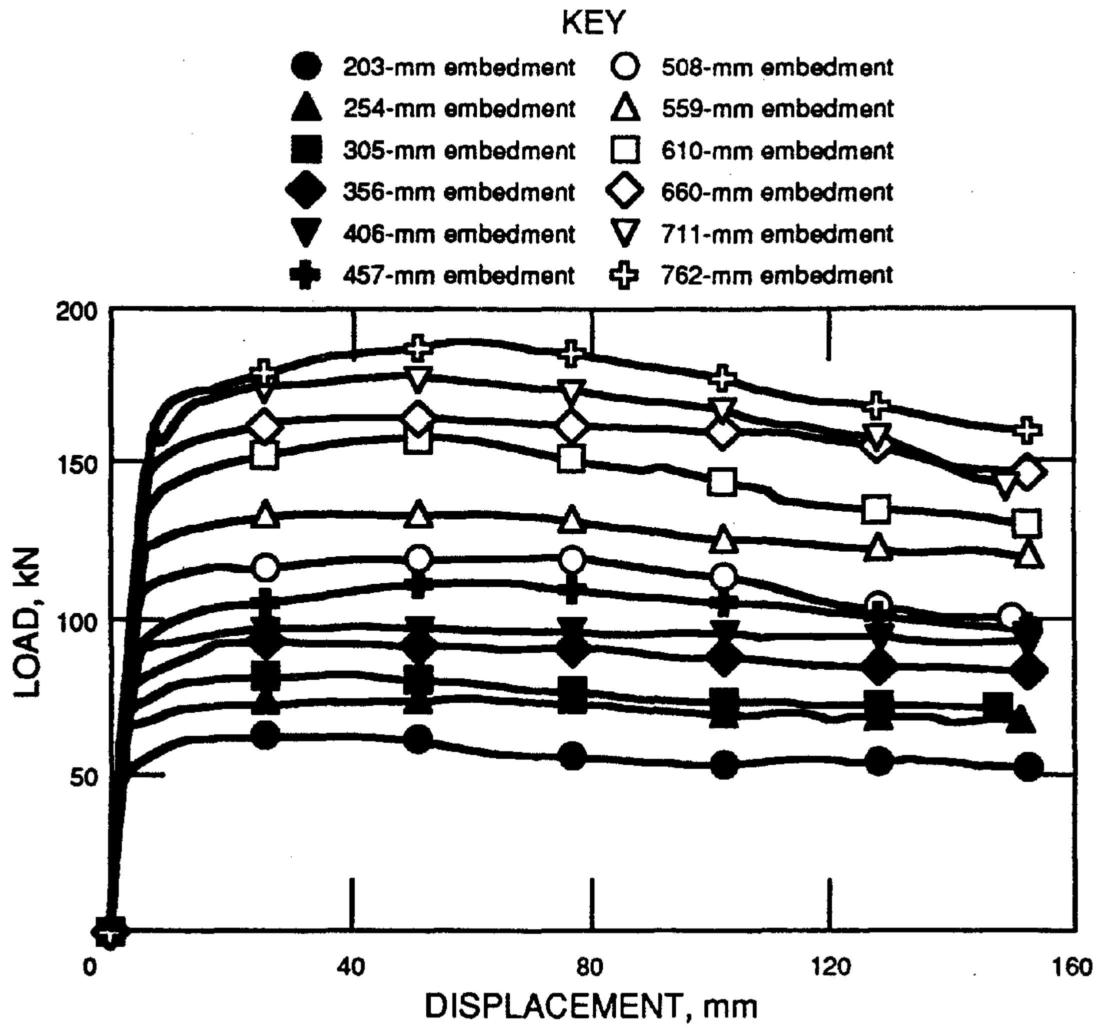

Figure 21.—Averaged 28-day load-displacement curves of cables at various embedment lengths (Goris, 1990).

Some ground control engineers attempt to design cable bolt support systems based on the maximum load-carrying capability of a cable plus an added safety factor. The critical embedment length will then be based on a maximum load-carrying capacity of 258 kN. The embedment length required to carry this load will depend on a number of factors. The quality of the grout is one of the most important. The curve in figure 22 shows a maximum load-carrying capacity of approximately 191 kN for an embedment length of 0.76 m. For purposes of discussion, extrapolating beyond the boundary of these data, a 258-kN load (which is the ultimate load-carrying capacity of a 15.9-mm-diam cable) would require approximately 1.07 m of embedded cable to achieve maximum pullout resistance when a grout with a water-cement ratio of 0.45:1 is used. Reichert and others (1992) also conducted pull tests to determine the relationship between cable bolt capacity and embedment length, but used a grout with a water-cement ratio of 0.3:1. A plot of their data is shown as figure 25. Extrapolation shows that a 258-kN load would require an embedment length of approximately 0.62 m to achieve maximum pullout resistance. The shorter embedment length required is the result of using a grout with a lower water-cement ratio, which has greater compressive strength. The relationship between water-cement ratio and pullout resistance for cable bolt supports is covered in this report in the section entitled "Grouts."

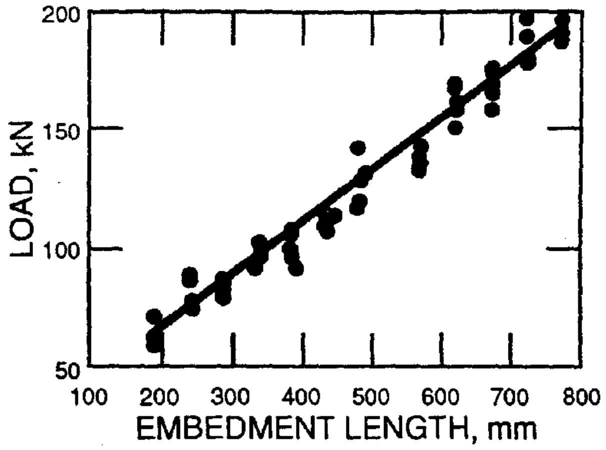

Figure 22.—Maximum load-carrying capacity versus embedment lengths (Goris, 1990).

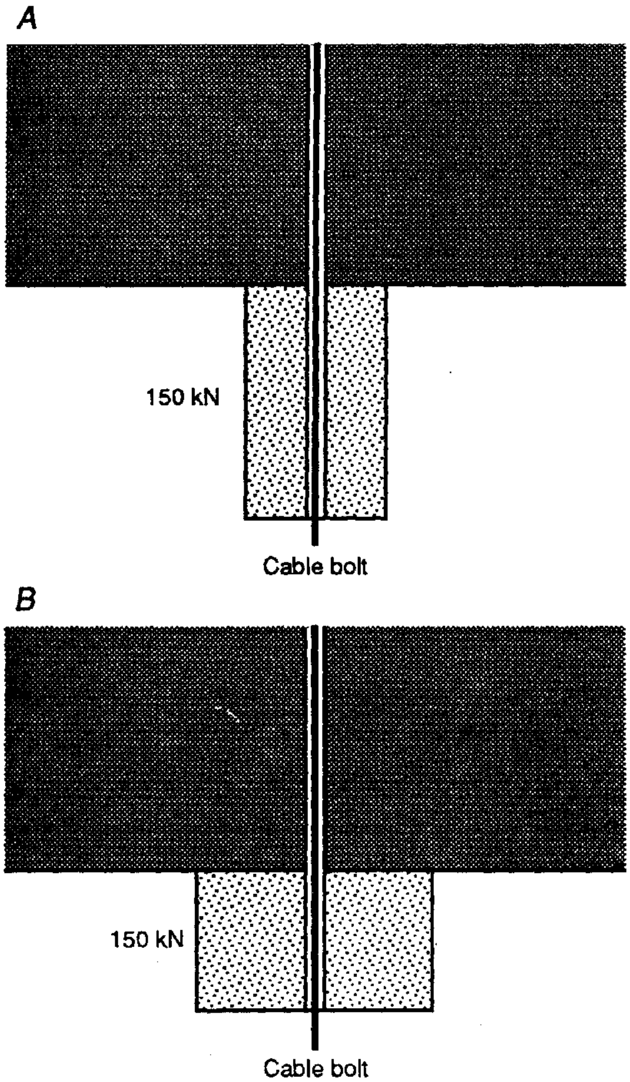

Figure 23.—Critical embedment length. *A*, Embedment length = 0.75 m; *B*, embedment length = 0.5 m.

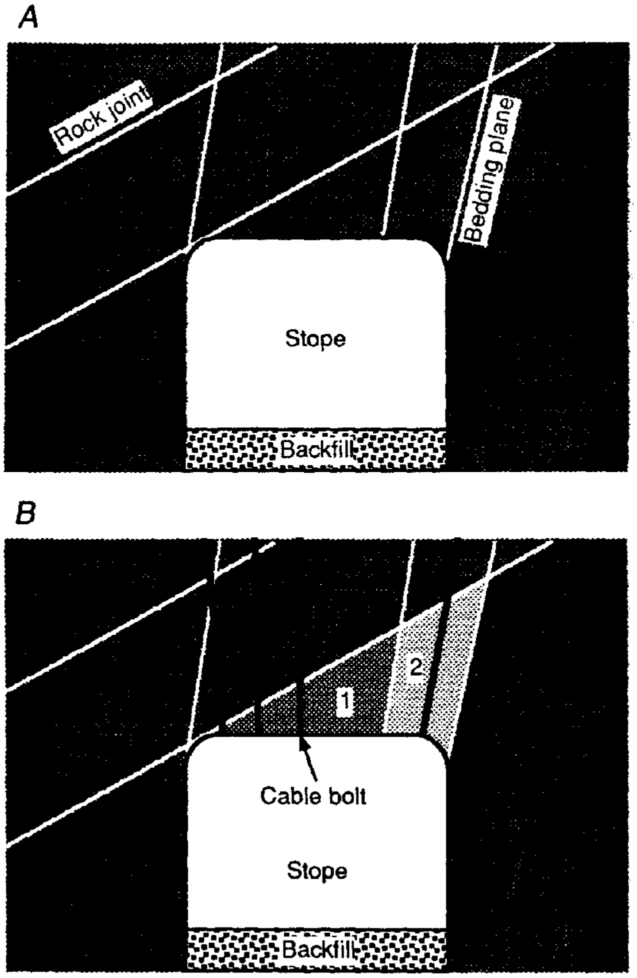

Figure 24.—Cross section of cut-and-fill stope. *A*, Unsupported stope; *B*, stope suported by cable bolts. 1 and 2 refer to rock blocks as described in text.

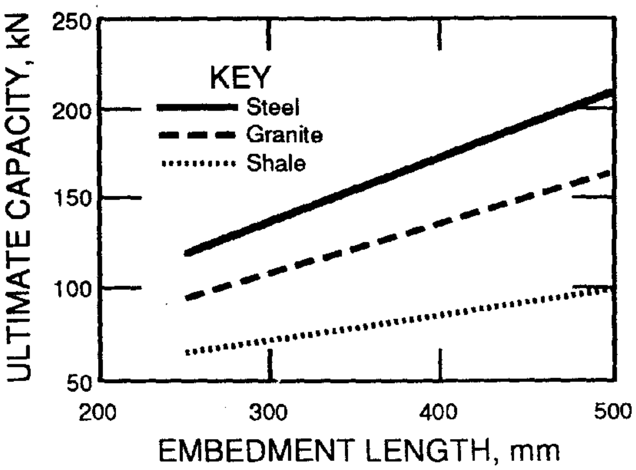

Figure 25.—Maximum load-carrying capacity versus embedment lengths using 0.3 water-cement ratio grout. (Reichert and others, 1992).

CABLE BOLTING EQUIPMENT

DRILLS

A popular drill used for drilling cable bolt holes is the percussive ring drill (fig. 26). The drill itself is mounted on a steel ring that allows angle holes to be drilled into the back or walls of the stope. This assures that the long holes (up to 20 m) will follow the ore vein or intersect the hanging wall at the appropriate angle.

CABLE COILS

Steel cables used for supports can be precut to a required length and then coiled for ease of shipment and handling (fig. 3). Some cable bolt crews prefer to handle the cable in large coils called reels (fig. 27) and then cut the cable to length after it is placed in the hole. These large reels usually contain about 2,620 m of cable. The cable can be cut using an abrasive wheel.

CABLE PUSHERS

Cables are placed in the hole either by hand or by a mechanical pusher (fig. 28). The cable is first inserted in the pusher, a breather tube is attached to the cable, and the two are inserted into the hole.

GROUT MIXERS AND PUMPS

Three main types of grout mixers and pumps are used in mines in North America. The most common is the pneumatic Spedel 6000 used in conjunction with a Spedel B3100 (Oliver, 1992). Figure 29 shows the Spedel system, which will mix about 42.3 kg of cement and the required amount of water at one time. The pump is placed into the mixing tub so that once the grout is mixed thoroughly, the pump can be turned on and the grout pumped into the hole. This pump will handle very stiff grouts having water-cement ratios between 0.3 and 0.35, but it is generally used with grouts that have water-cement ratios above 0.4.

The primary advantages of this mixer and pump are that they are light in weight and portable, so that one person can move them from site to site, and they will handle stiff grouts. Many grouting crews prefer this mixer and pump in areas with restricted access. A major disadvantage is that the cable bolt crews cannot mix a new batch of grout while pumping a previously mixed batch.

The next most common mixer and pump is the Minepro 3 (fig. 30), which is a hydraulic, positive displacement pump with an auger assembly in tandem with a moyno pump. This system will mix and pump stiff grouts with water-cement ratios as low as 0.3. The hydraulic motor is run by air, but there are pneumatic and electric versions of both the mixer and the pump. The mixing tank is mounted above the rotor-stator assembly and will easily hold 85 kg of cement and the appropriate amount of water. While the grout is being mixed, it is circulated through the rotor-stator back into the mixing tub. When the grout is ready to be pumped, a valve is turned and the auger forces the grout into a moyno pump connected to the grout line. The hydraulic oil in this system tends to

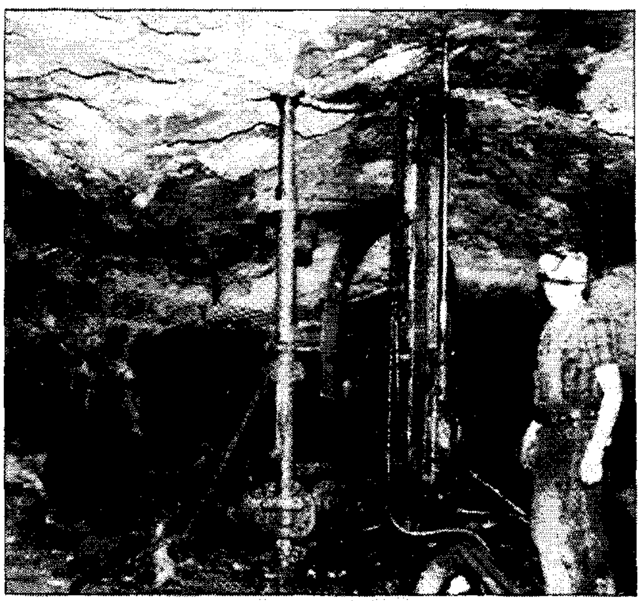

Figure 26.—Ring drill used to drill cable bolt holes.

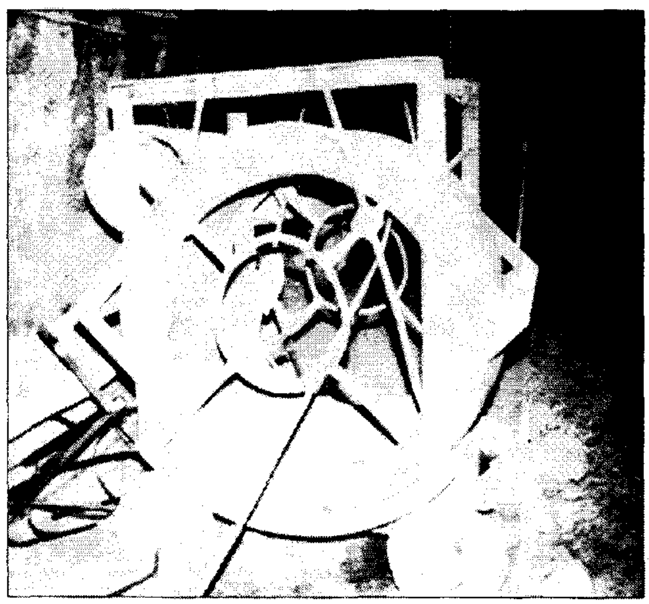

Figure 27.—Large reel of cable.

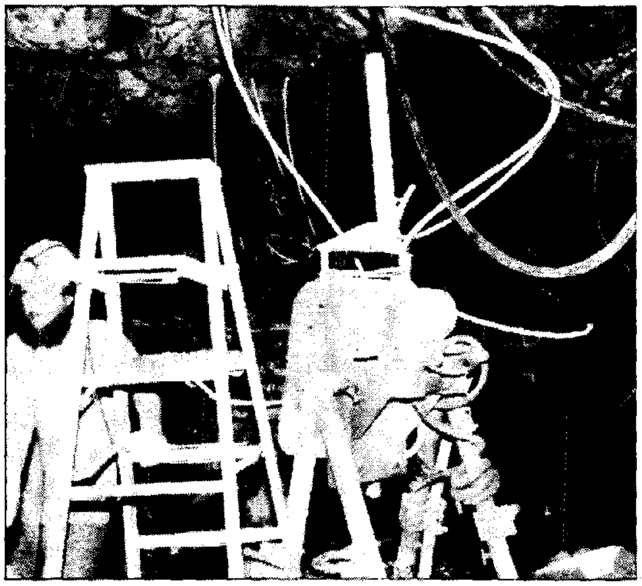

Figure 28.—Cable pusher.

Figure 29.—Spedel mixer and pump.

overheat when stiff grouts (those below a water-cement ratio of 0.4) are pumped. However, the manufacturer does offer a hydraulic oil cooler to prevent overheating.

Like the Spedel mixer and pump, the Minepro 3 involves batch mixing. This means that one batch must be mixed and pumped before a second batch can be mixed. Figure 31 shows a ChemGrout mixer that can mix and pump at the same time. This is a pneumatic mixer and pump that can handle 85 kg of cement plus the appropriate amount of water and can pump one batch while

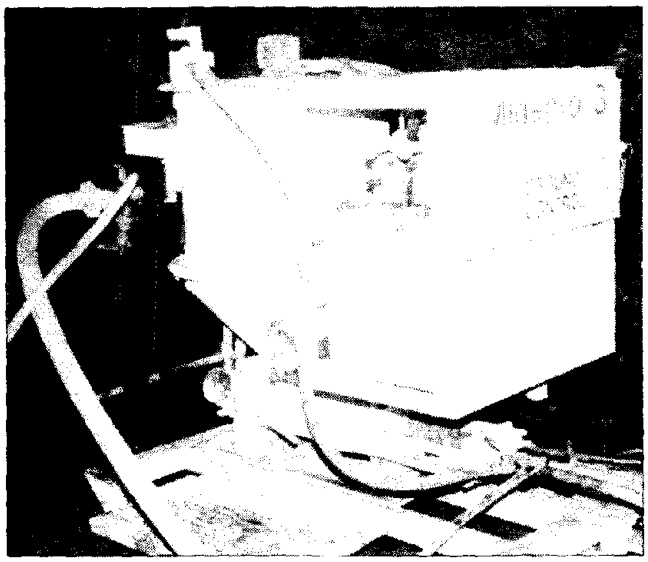

Figure 30.—Minepro 3 mixer and pump.

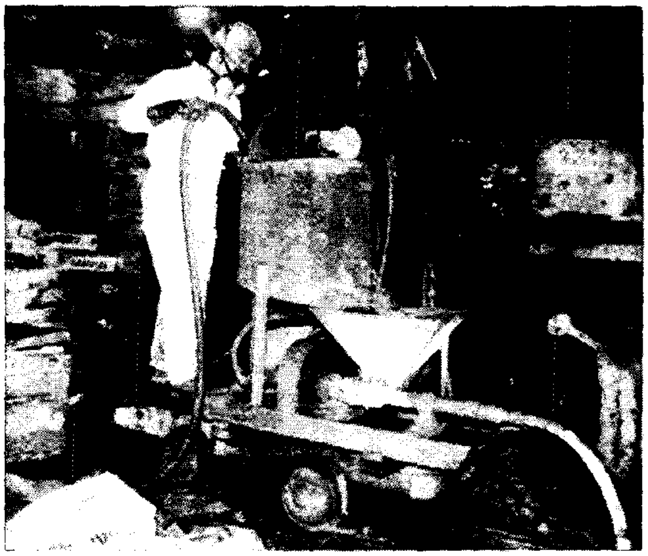

Figure 31.—ChemGrout mixer and pump.

Figure 32.—Mechanized cable bolt machine.

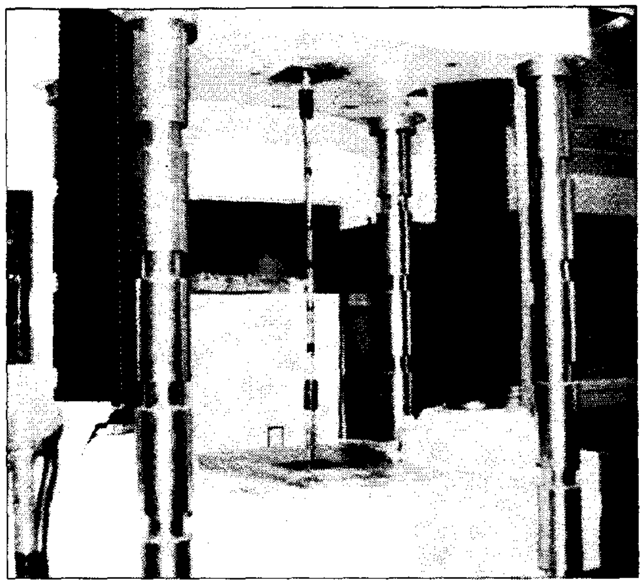

Figure 33.—Cable bolt strain gauge being tested in laboratory.

mixing another. This allows grout crews to continuously mix and pump grout. The mixing tub is mounted above a hopper and a moyno pump. Once a batch of grout is mixed, a valve is opened and the grout runs into a hopper and then into the moyno pump. The pump is gravity fed, which makes it difficult to force very stiff grouts into the funnel of the pump; consequently, this system will not handle grouts with water-cement ratios below 0.4. If stiffer grouts are required, it is recommended that a water-reducing agent be used to improve the viscosity of the grout so that it will flow into the pump.

MECHANIZED CABLE BOLTING EQUIPMENT

An equipment manufacturer in Finland is marketing an automated cable bolt machine that is self-contained and can be operated by one person (fig. 32). One operator on the machine can drill the holes, mix the grout, grout the holes, and insert the cables. Hole sizes range from 50 to 90 mm in diameter, and the machine holds 22 drill rods for a maximum hole depth of 40 m. This machine is capable of mixing a 250-kg batch of grout, which is sufficient for grouting 40 m of hole. It will handle stiff grouts (water-cement ratio of 0.3 to 1.0) and carries a 900-m reel of cable (Pearse, 1986).

MONITORING LOADS ON CABLE BOLT SUPPORTS

The loads placed on installed cable bolt supports can be monitored using specially built strain gauges. Figure 33 shows testing one of these gauges installed on a cable. The gauges function on the principle that changes in electrical resistance result when electrical wires are stretched, so by measuring the change in resistance, investigators candetermine the amount of load being placed on selected cables (Choquet and Miller, 1988).

These strain gauges are approximately 0.63 m long. A thin, nickel-chromium wire (0.25 mm in diameter) is wound into spiral grooves between the outer strands of the cable. This wire is protected by a plastic tube and is terminated at both ends by a molded rubber anchor bonded to the cable with quick-setting epoxy (fig. 34). Each anchor is then protected by a plastic sleeve to prevent grout from restricting movement when the cable is loaded.

Readouts from the gauges can be in either volts or strain. Figure 35 shows strain versus load behavior of two cable bolt strain gauges. Both gauges were attached to a conventional 15.9-mm-diam cable. In the first test, the cable was not embedded in grout. In the second test, both the gauge and the cable were embedded in a cement grout column and allowed to cure for 7 days before testing. The tests were conducted by placing the instrumented cable in a test machine, securing the ends of the cable with barrel-and-wedge anchors, and then loading the cable. Continuous readings of strain and load were recorded during the test. Both curves in figure 35 are linear, although the gauge embedded in grout showed a little less strain for a given load than the free-standing gauge. A curve for an unembedded gauge is provided by the manufacturer when gauges are purchased. The manufacturer will also install the gauges on the cables and ship the cable(s) and gauges to a site.

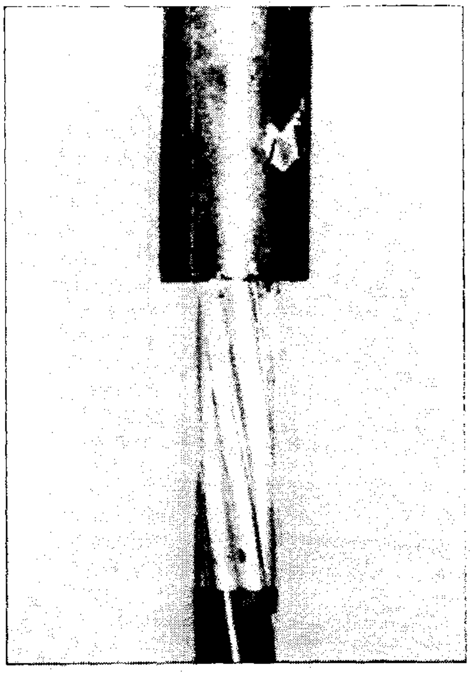

Figure 34.—Anchor of cable bolt strain gauge.

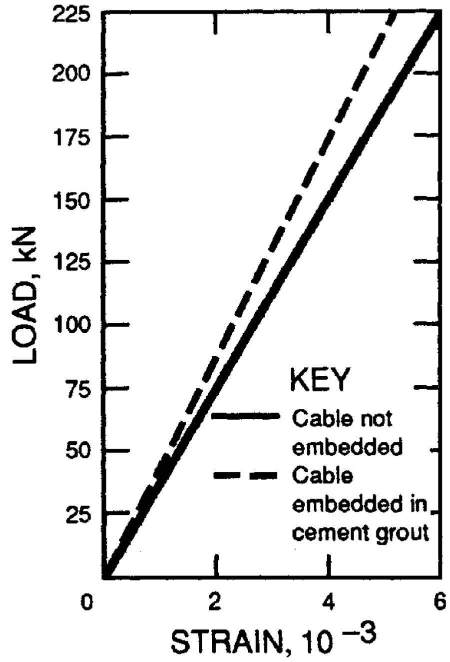

Figure 35.—Strain versus load for cable bolt strain gauge.

INSTALLING CABLE BOLT SUPPORTS

There are three basic techniques for installing cable bolt supports (fig. 36). The first technique (fig. 36*A*) involves drilling a hole 38 to 57 mm in diameter to a desired length and then inserting one or more cables and an attached 12.7-mm-diam plastic breather tube into the hole. The breather tube should have a pressure rating of at least 0.83 MPa. An anchoring device must be placed on the cable(s) to hold it in the hole until the grout cures. If the cable(s) is to be tensioned using a bearing plate and barrel-and-wedge anchor, approximately 0.5 m of cable should extend out of the drill hole. The hole is then flushed by injecting water up through the breather tube. To protect workers from injury from the sharp ends of the cables, protective covers should be placed on the cables (fig. 37).

A 19-mm-diam grout tube is then inserted about 1 m into the hole, and the hole is sealed with strips of burlap or shredded cotton. The use of expanding foam with the burlap and cotton waste will ensure a better plug. It is recommended that the foam be allowed to cure and harden for approximately 2 h.

Another method for plugging the hole is to force a rubber plug into the hole to form a seal (fig. 38) (Gagnon, 1983). A slot is cut into the side of the plug to allow a breather tube to extend out of the hole. Grout is then pumped into the hole through the grout tube. The hole is filled with grout from the bottom up, and the breather tube allows the air being displaced by the grout to escape. When the hole is filled, grout will run out of the tube. The ends of the two tubes are then folded over and tied off to prevent grout from draining out. These tubes then become a permanent part of the support system.

The second technique for installing cable supports is to insert a cable along with a 19-mm-diam grout tube into the hole (fig. 36*B*). A very thick grout (one with a water-cement ratio between 0.3 and 0.35) is pumped into the hole, and the hole is filled from the top down. With this method, it is important that the grout be thick to prevent it from running out. On occasion, some cable bolt crews will pull the grout tube from the hole as the hole is being filled so that the tube can be inserted into another hole, thereby reducing the delay time.

A third technique (fig. 36*C*) is to place a grout tube into the top of the hole and pull it out as the hole is being filled with grout. Once the hole is filled, the cable is shoved into the hole. This process can be very messy and wastes grout because the displaced grout comes out of the hole. However, if about 1 m of hole is left ungrouted at the collar of the hole, the grout will usually not run out. The advantage of the third method is that the hole diameter can be quite small because the cable and grout tube are not in the hole at the same time. A 15.9-mm-diam cable and a 19-mm-diam grout tube require at least a 34-mm-diam hole.

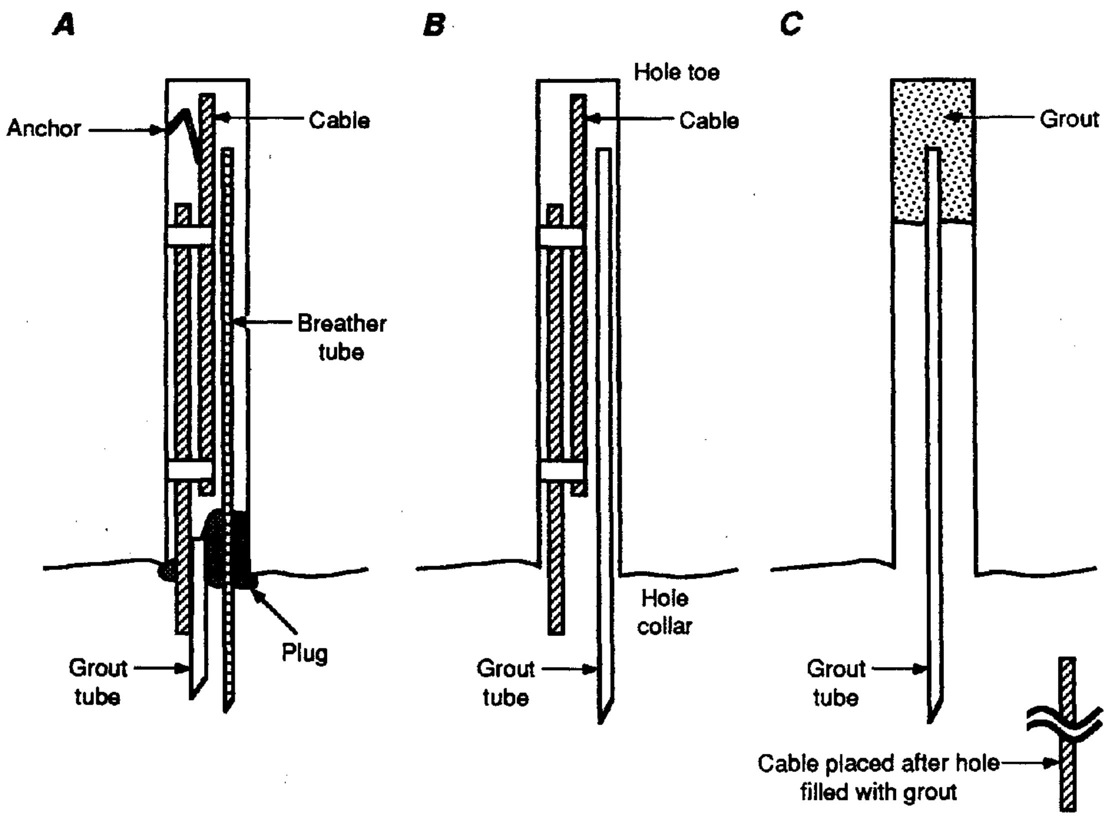

Figure 36.—Techniques for installing cable bolt supports. *A*, Breather tube and grout tube used to grout hole; *B*, grout tube alone used to grout hole; *C*, cable placed after hole grouted (Bourchier and others, 1992).

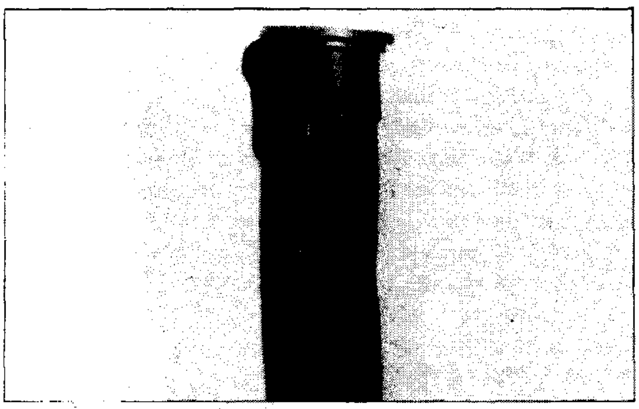

Figure 37.—Protective cover for cable ends.

On downholes, the third method (fig. 36*C*) is used, and the grout tube is pulled from the hole as the grout is placed. A breather tube is not required for downholes.

TECHNIQUES FOR ANCHORING CABLES IN DRILL HOLES

There are three basic methods for securing cables in the hole. The first is to bend one or more of the wires at the top end of the cable into a hook (fig. 39). Tests conducted by personnel at the Homestake Mine in Lead, SD, showed that the hook should be approximately 10 cm long and bent at an angle between 40° and 55°.[6] The pull tests also indicated that for cables 20 m long, a single-wire anchor resisted an average load of 113.4 kg, or approximately five times the weight of the cable (1.19 kg/m). Double-wire anchors on a 20-m cable resisted an average of 274 kg before slipping, which is 12 times the weight of the cable. Homestake personnel concluded that cables less than 12 m long only require one anchor, while cables 12 m long or longer require double anchors.

Another method for anchoring cables less than 12 m long is to bend one of the wires on the lower end of the cable up approximately 45°. This anchor is then positioned at the collar of the hole when the cable is inserted.

The second method for anchoring cables is to attach two steel strips (formed to resemble a cross) to the top end of the cable (fig. 40). The length of the steel strips can be varied to fit holes of different diameters. For a 50-mm-diam hole, the strips should be approximately

[6]This information was provided to the senior author by J. Pfarr, mining engineer, Homestake Mining Co., Lead, SD.

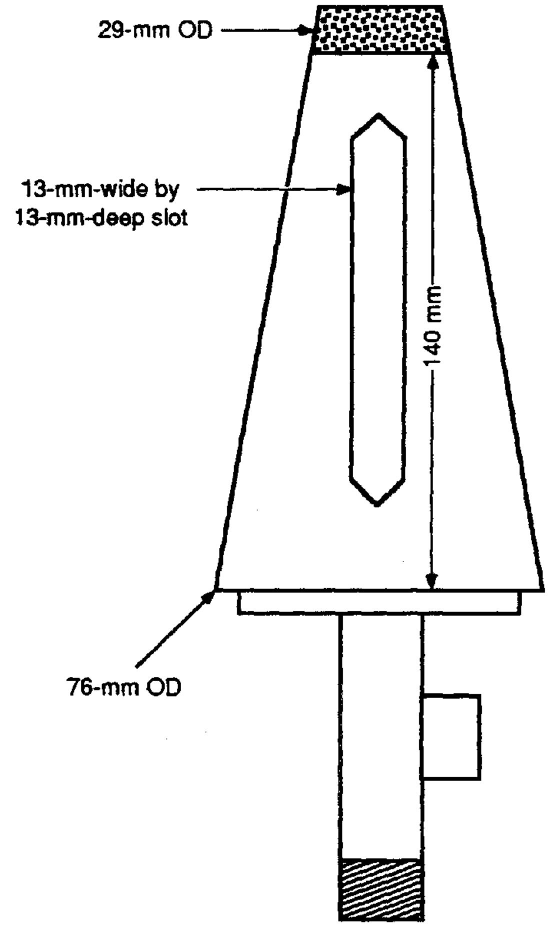

Figure 38.—Plug for grouting holes (Gagnon, 1983).

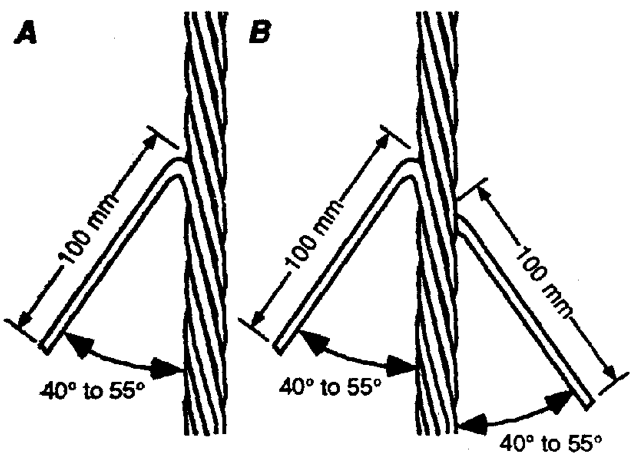

Figure 39.—Anchor for holding cable in hole. *A*, Single anchor; *B*, double anchor.

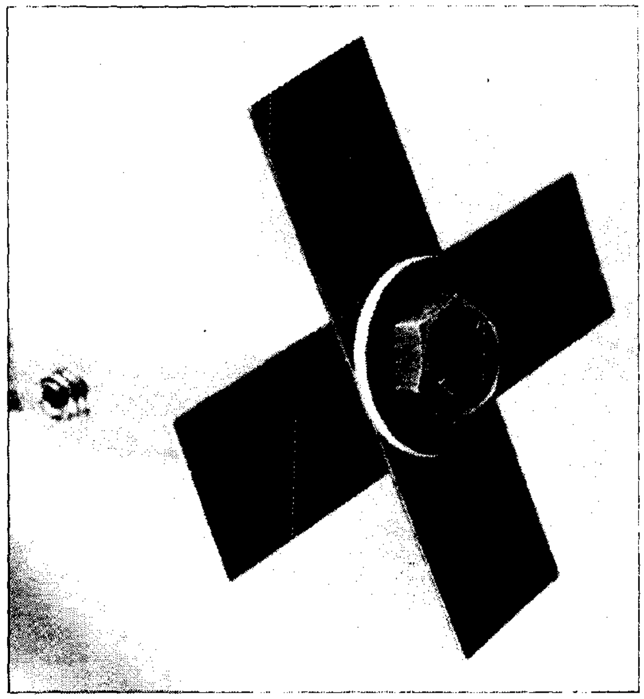

Figure 40.—Steel strap anchor for holding cable in hole.

57 mm long by 16 mm wide. For a 64-mm-diam hole, the strips should be approximately 76 by 19 mm.

The third method is to attach a stiff piece of cable about 30 to 40 cm long to the end of the cable. Several of the wires on the short cable are then bent into a hook. This is basically the same as the first method.

TENSIONING CABLE BOLT SUPPORTS

The cables shown in figure 6 were installed as passive supports; that is, they were not preloaded. Consequently, any load placed on them will depend on movement of the rock, which transfers load through the grout to the cable. However, the cable support can be preloaded to create an active support that helps reduce rock movement. This is accomplished by using a steel plate along with a barrel-and-wedge anchor (grip) (fig. 41).

The practice of using steel plates and grips and tensioning the cables has varied over the years. When cable bolts were first used in mining, they were usually tensioned shortly after installation, thereby providing an active support system. Years later, untensioned cable bolts were preferred because pretensioning was expensive; also, many ground control engineers did not think this practice was effective. In recent years, however, ground control engineers have been reconsidering pretensioning as an effective

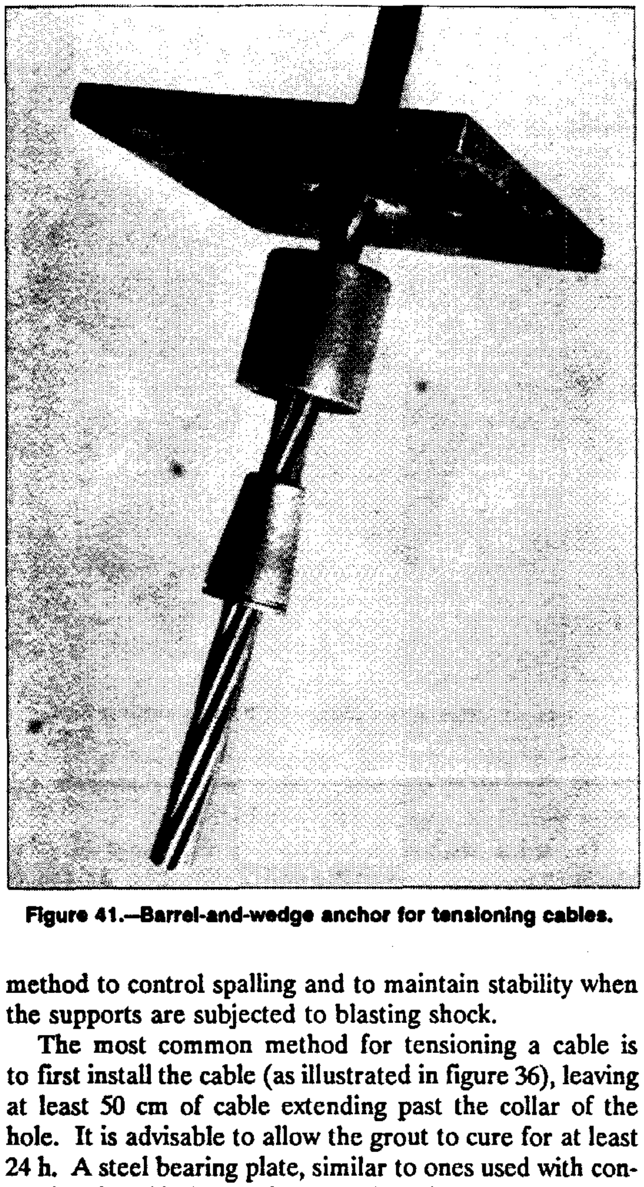

Figure 41.—Barrel-and-wedge anchor for tensioning cables.

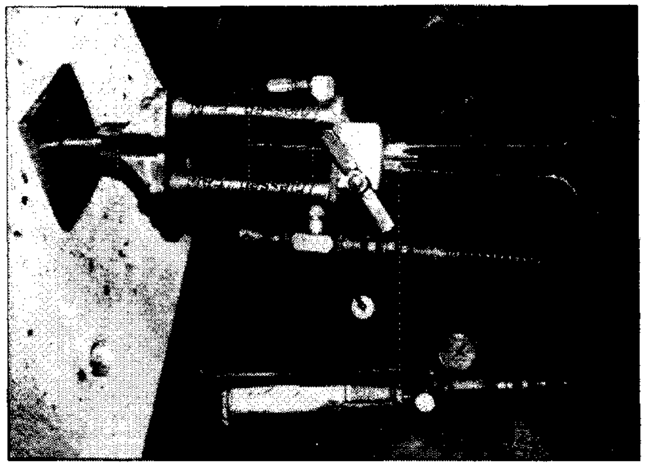

Figure 42.—Hydraulic jack for tensioning cable bolts.

method to control spalling and to maintain stability when the supports are subjected to blasting shock.

The most common method for tensioning a cable is to first install the cable (as illustrated in figure 36), leaving at least 50 cm of cable extending past the collar of the hole. It is advisable to allow the grout to cure for at least 24 h. A steel bearing plate, similar to ones used with conventional rockbolts, is placed against the rock, and a mechanical grip is placed on the cable. Figure 41 shows the bearing plates and grip on the cable. A hydraulic jack (fig. 42) is then placed on the end of the cable and load is applied. The hydraulic jack in figure 42 has a wedge grip at the rear of the jack and actually grips and pulls the cable while at the same time pushing against the forward grip up next to the bearing plate. A load of 1,800 to 2,700 kg is usually adequate to set the grip. It is advisable to pump the jack and achieve the desired load, bleed off the jack, and then pump it again to reset the grip.

Sometimes it is desirable to leave a long section of cable near the collar of the hole ungrouted so that it can be tensioned. The ungrouted portion of the cable will elongate more per unit length than grouted portions, thereby allowing more rock movement. This can be accomplished by placing a predetermined piece of 19-mm-diam grout tube over the cable like a sleeve and taping each end of the tube with electricians tape. This operation is done prior to inserting the cable in the hole. The entire cable is then placed in the hole and grouted. The grout will surround the tube but will not adhere to cable covered by the grout tube. The length of the cable to be left ungrouted will depend on the amount of cable elongation required.

SAFETY ASPECTS OF CABLE BOLT SUPPORTS

Installation of cable bolt supports involves the handling of steel cable, cement grouts, and breather and grout tubes. Also involved is the use of grout mixing and pumping equipment and, in some cases, cable-pushing devices. The use of safe procedures for handling and installing cable bolt supports is a critical component for ensuring a viable ground control system.

HANDLING CABLES

Cables are shipped to mines in large reels containing about 2,620 m of cable (fig. 27) or are precut to a specified length by the supplier, coiled, and then shipped to the mine. Each method requires a different procedure for safely handling the cables. A single, 15.9-mm-diam cable weighs about 1.1 kg/m. Therefore, a 2,620-m reel will weigh about 3,170 kg and requires a steel cage for handling and transporting underground. Once the reel is underground at the job site, the bands around the cable reel are cut, and the cable is pulled from the center of the reel and cut to a specified length. Note that the reel of cable has a great deal of "spring" energy; if the reel is not handled properly, a number of coils of cable could shoot out from the center of the reel and cause injury.

Precut cables are usually 10 to 20 m long and are shipped in 1.2-m-diam coils. Each coil is secured by banding the cable with either steel or plastic straps and then placing the coiled cables on shipping pallets. Once the cables arrive at the mine, they are usually shipped underground on the same pallet and uncoiled as needed. The straps used to band each cable are placed on the coil in sequence at the factory. These straps must be cut in the reverse sequence prior to use to limit the amount of cable released with each strap. This is a dangerous process because the cable tends to whip as the straps are cut. The recommended procedure is to lay the coil flat on the ground and then cut each strap in sequence while standing in the middle of the coil. This way a person can avoid being hit by the cable if it does whip. It is important to use a cutting device that is quick and effective. Cutting with an air-operated abrasive disk cutter is preferred over an axe or a hacksaw. It is also recommended that a protective cover be placed on the ends of the cables (fig. 37) to reduce injury in the event that someone is hit with the end of the cable. The use of leather gloves, safety glasses, and a protective guard on the cutter disk is essential for a safe operation.

CABLE ANCHORING

A common procedure for installing cable bolts in most mines is to insert a large number of cables in one pass and complete grouting at a later stage. If a cable slips from an uphole prior to grouting, it presents a danger to the bolting crew and other personnel in the area. A 20-m-long cable, for example, weighs about 22 kg and can cause serious injury. It is a good practice to insert and grout cables with as little delay as possible and to restrict access to the area during this phase of the operation. Cable slippage in the hole has been reported by a number of mines in the United States and Canada even when bent-wire anchors were used. It is recommended that cables less than 12 m long be anchored with a single bent-wire anchor and that cables 12 m or longer be anchored with a double bent-wire anchor. A double-wire anchor can be difficult to install manually, which prompted one operation to return to a single-wire anchor and a protective cover (fig. 37). If spring-steel anchors are used, additional strips of steel can be added to increase anchorage. Whatever the method of anchorage used, it is a good safety practice to use protective covers on the ends of all cables protruding out of the hole. These covers can be reused a number of times.

WORKING WITH CEMENT GROUTS

Extra care should be taken by grouting crews when handling cement. Although cement grouts are mixed with a mechanical mixer, the cement and water are added manually, which can lead to burns and respiratory problems for the grouting crew. Hunt and Askew (1977) commented that 80 pct of the injuries occurring during cable installation at one operation were the result of cement burns from grout. Schmuck (1979) also indicated that most injuries to cable bolting personnel were the result of cement burns and recommended that grouting crews use long gloves, eye goggles, and respirators. Skin-grout contact usually arises during the pumping phase of the operation as grout leaks from the hole or from the end of the grout tube. It is recommended that when grout does come in contact with skin, it should be immediately washed off with water. Protective overalls and waterproof suits are commonly used by operators in western Canada and by one operation in the United States, and this is viewed as a good safety measure by mine personnel.

GROUTING CABLE BOLT HOLES

Bursting of grout tubes and connectors between pump hoses and grout tubes has been frequently reported in North America, especially when low water-cement ratio grouts (less than 0.4 to 1) are used. The direct coupling of the pump outlet hose to a grouting assembly has been suggested (Bourchier and others, 1992) as a means of eliminating bursting connections. Some operators have adopted the use of high-pressure (1.7-MPa) tubing to reduce hose bursting. High pressures at the hole collar generated when using the breather tube method of installing cables can induce forces in fractured ground that encourage the release of loose rock. Proper scaling practice at the start of each shift will minimize ground falls resulting from grouting. After grouting is completed, line pressures will remain high, and care must be taken when disconnecting the pump hose from the grout tube.

QUALITY CONTROL

It is critical that quality control be maintained throughout the cable installation procedure. This has been discussed in detail throughout this report. While it is recognized that a low water-cement ratio is desirable, potential problems may result.

Using the toe-to-collar method of grouting (method *B*, figure 36) with a 0.35 water-cement ratio grout may result in an incomplete grout column along the length of the cable because of the flow characteristics of thick grout. Large-scale tests were conducted by Peterson (1993) to determine the effectiveness of the toe-to-collar method of installing cable bolt supports. Polyvinyl chloride (PVC) pipes, 6.1 m in length, were suspended vertically to represent a drill hole within a rock mass. Various series of tests were conducted using 4.8- and 5.8-cm-diam pipes, and single and double conventional cables as well as single birdcage cables.

The grout was pumped to the toe of each hole using a 2.2-cm-diam grout tube. Samples were allowed to cure for 7 days; then 5-cm-thick sections were cut from the pipe to display the cable and grout tube embedded in grout. Figures 43 through 45 show the grout coverage around single and double cables embedded in 4.8-cm-diam holes. These figures show incomplete grout coverage throughout the column of the hole. This is especially the case when a single cable is used. Figure 43 shows that voids in the grout constitute as much as 25 pct of the cross-sectional area of these samples. For double cables (fig. 44), the percentage of voids is less. Similar results are shown for the 5.8-cm-diam hole (fig. 45). The birdcage cable tested (fig. 45, bottom) shows complete coverage of the grout column.

A possible explanation for the different flow characteristics of the grout with conventional and birdcage cables is the interaction between thick fluids such as cement grouts, the walls of pipes, and objects in the path of the grout. A thick grout (water-cement ratio of 0.35) behaves like a Bingham fluid, which includes fluids such as coal slurries, emulsions such as paint, and suspensions of finely divided solids in liquid such as drilling mud. Figure 46 shows a flow profile for Bingham fluids as well as for grout around a cable. Normally, the grout would flow like a Bingham fluid, but when a cable is placed in the hole, frictional resistance is created. The seven individual wires of the birdcage cable are more evenly distributed across the hole, resulting in less turbulence in grout flow. While PVC pipe does not have the same frictional properties as rock, the above discussion shows that a potential problem may occur when using low water-cement ratio grouts to place cable bolt supports.

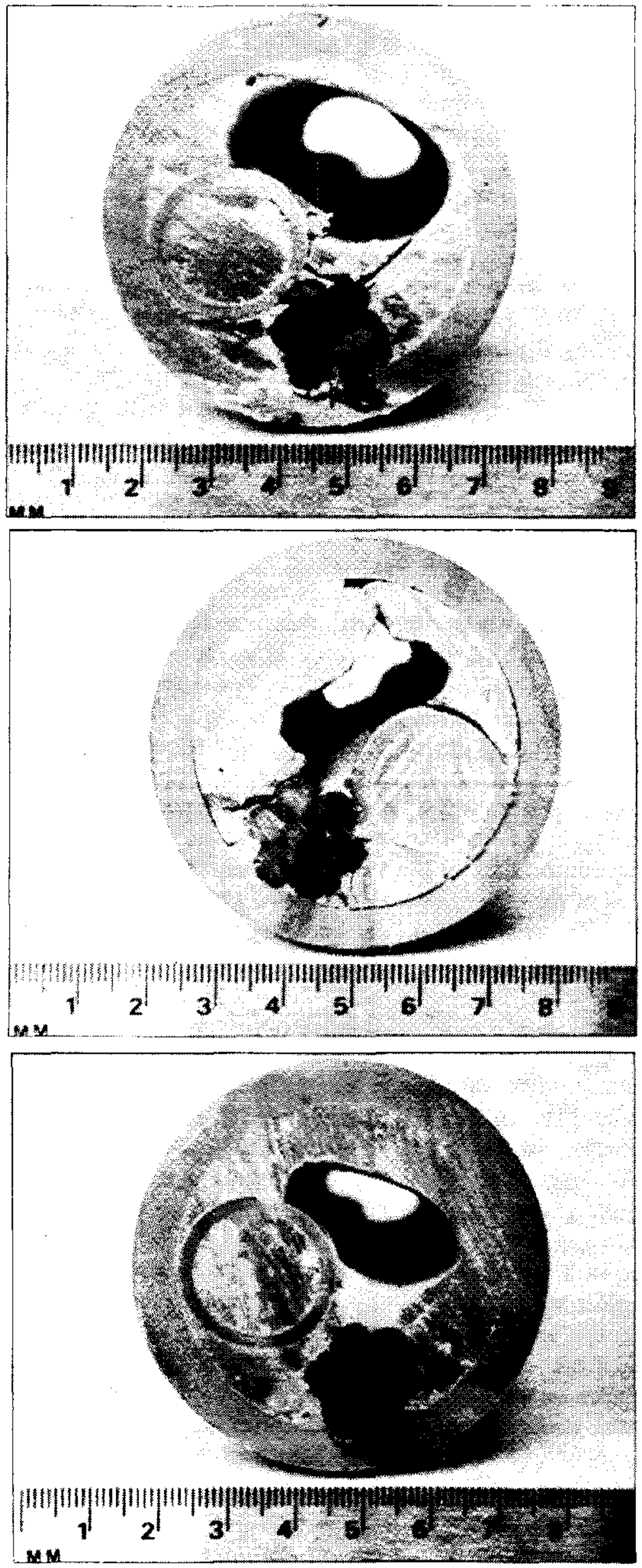

Figure 43.—Air voids in grout column of 4.8-cm-diam hole with single cable.

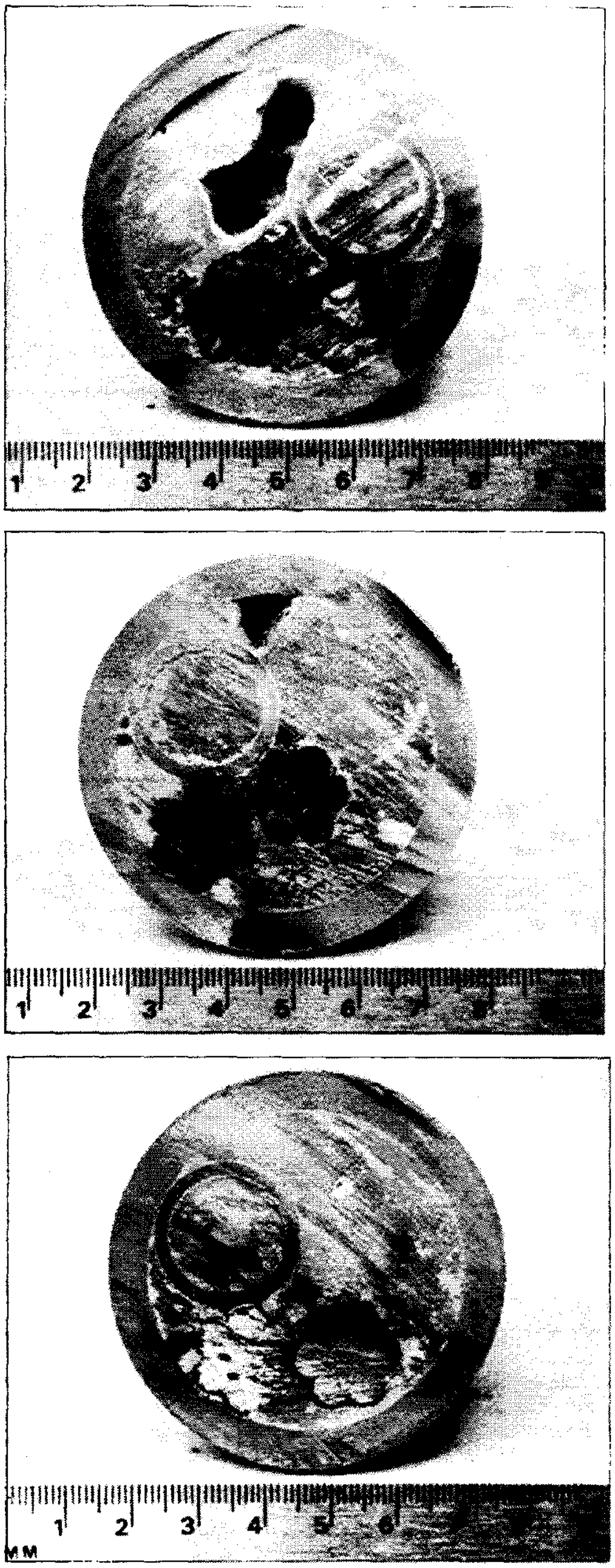

Figure 44.—Air voids in grout column of 4.8-cm-diam hole with double cable.

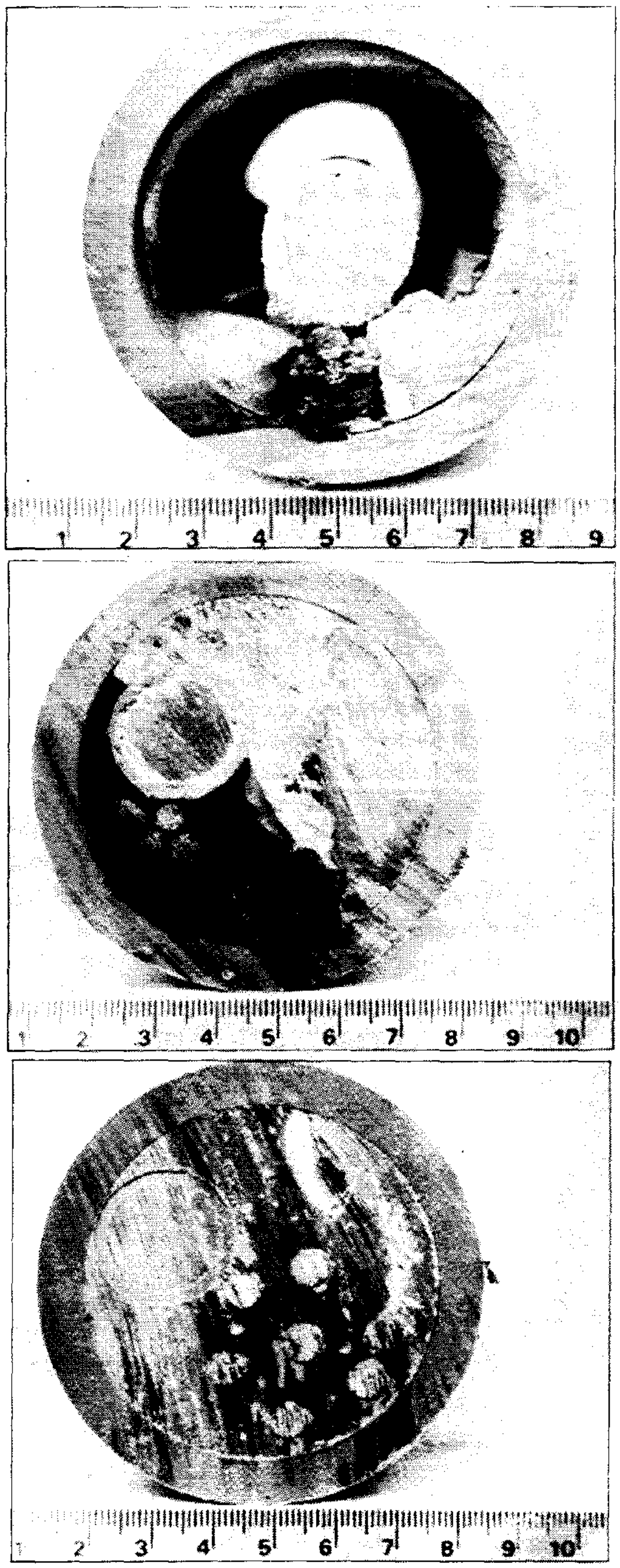

Figure 45.—Air voids in grout columns of 5.8-cm-diam holes with single (top), double (middle), and birdcage (bottom) cables.

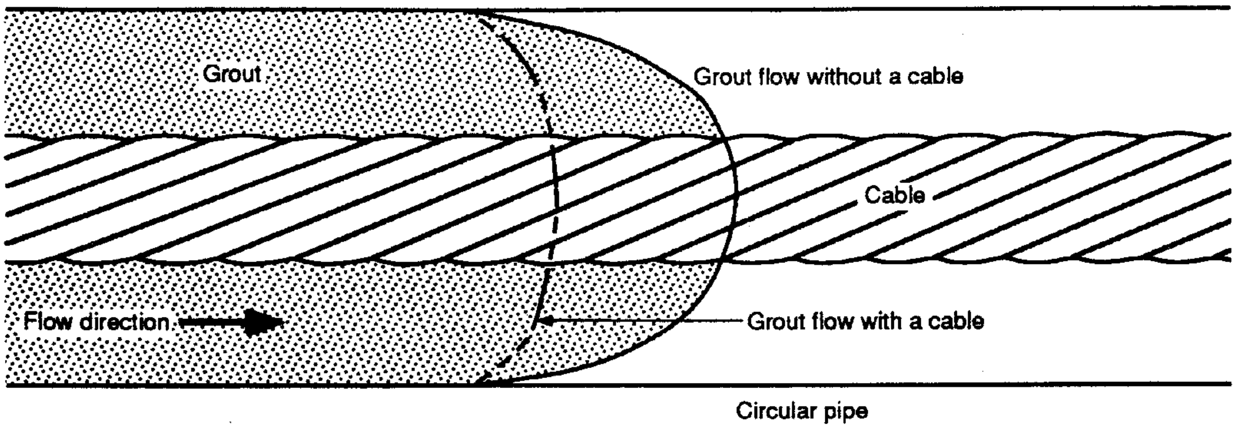

Figure 46.—Schematic of grout flow in circular pipe.

PRODUCTION RATES AND COSTS FOR CABLE BOLT SUPPORTS

Installation of cable bolt supports is usually accomplished in two separate operations, one for installing the cables and the other for grouting the holes. Production rates and costs for these operations obviously differ from one mine to the next. However, it is useful to compare information from several sources to establish guidelines. Tables 5 and 6 list production rates for both inserting cables and grouting holes for two mines in Canada and one in the United States. The mines listed in these two tables are the same mines.

Table 5.—Production rates for inserting cable bolts

Mine	Production rate per individual shift, m	Crew size	Cable length, m
A	[1]90	2	9 - 15
B	[1]69	2	15
C	[2]166	3	15 - 20

[1]8-h shift. [2]12-h shift.

Source: Goris and others, 1992.

Table 6.—Production rates for grouting cable bolt holes

Mine	Production rate per individual shift, m	Crew size	Cable length, m	Water-cement ratio
A	[1]180	2	9 - 15	0.32:1
B	[1]166	2	15	0.45:1
C	[2]230	3	15 - 20	0.32:1

[1]8-h shift. [2]12-h shift.

Source: Goris and others, 1992.

Table 7 gives total costs of cable bolt supports for seven mines in Canada and one in the United States. The mines in table 7 are not the same as those in tables 5 and 6. Table 8 gives itemized costs for installing a 12.2-m-long cable bolt support.

Table 7.—Total costs for cable bolt supports, Canadian dollars per meter

Mine	Cost	Comments
A	$29.46	0.3- by 0.3-m plates used.
B	19.71	Drilling consumables not included. Double cables used.
C	28.84	Double cables.
D	28.54	Double cables.
E	31.83	Single cables.
F	19.69	Single cables.
G	19.50	
H	28.00	
Average	26.84	

Source: Goris and others, 1992.

Table 8.—Itemized costs for 12.2-m-long cable bolt supports, Canadian dollars

Item	Cost
Hole drilling ($11.94/m incl. labor)	$145.67
Cables (12.2 m @ $3.61/m, double)	44.04
Cable anchor	4.25
Large 0.3- by 0.3-m steel plate	2.45
End anchor	3.30
12.2-m grout tube (15.2 m @ $0.82/m)	12.46
Cement (1 bag @ $10.00)	10.00
Labor ($4.27/m)	52.09
Total	274.26
Cost per meter	22.48

Source: Goris and others, 1992.

CABLE BOLT SUPPORT PATTERNS

An extensive study of 12 mines in Canada and the United States was conducted (Nickson, 1992) to review cable support practices. The majority of the mines were located in western Canada, but the data are applicable to many mines in North America. Some typical examples of cable bolt patterns found in this study are shown in figures 47 and 48.

SUPPORT PATTERNS FOR STOPE BACKS

Case studies of stope back supports by Nickson (1992) were made up largely of blasthole and vertical crater retreat open stoping situations, but some drift, cut-and-fill stope, room-and-pillar, and undercut-and-fill mining methods were included. Figure 47*A* shows typical cable installations in cut-and-fill mining. Cables up to 18 m long were usually installed in upholes to cover three or more mining lifts. Extra cables were often installed in the hanging wall for rock stability and to reduce ore dilution. Cables were also installed from an overcut but were restricted by how long a downhole could be drilled accurately. In narrow open stopes, bolts were installed in a fan pattern from footwall to hanging wall. Where the development area was large enough, cable bolts were installed on a square pattern and were sometimes angled into both the hanging wall and the footwall. Excessive ore dilution by rock from the hanging wall or footwall can undercut this type of back support and induce failure. Inadequate distribution of cables into a stope back often occurs as a result of limited access to the stope, or where the drill drifts are not slashed to the full width of the ore body. In this case, a square pattern is not possible, and the point-anchor approach is often adopted (fig. 48).

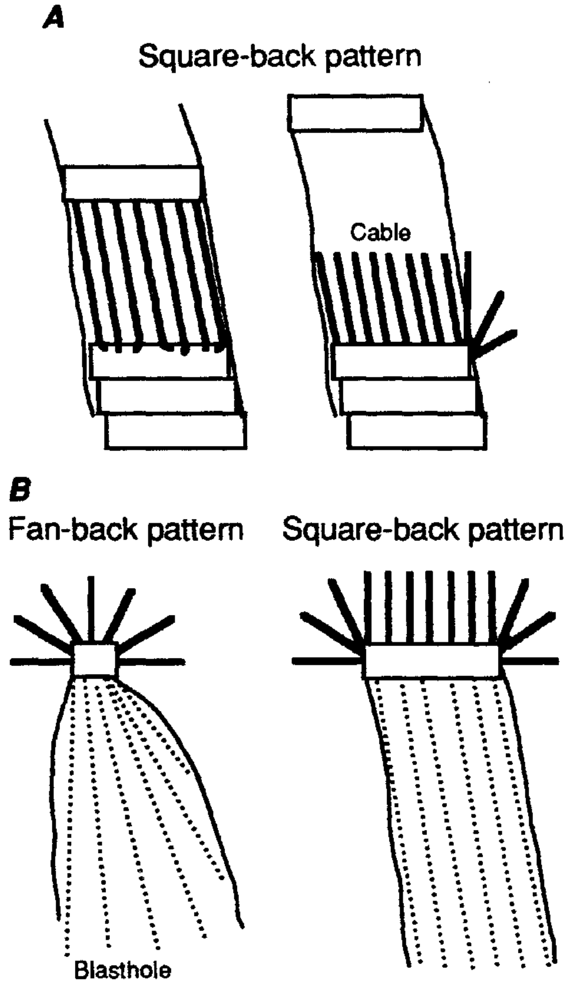

Figure 47.—Typical back support for (*A*) cut-and-fill mining and (*B*) open stope mining (Nickson, 1992).

SUPPORT PATTERNS FOR HANGING WALLS

Hanging wall cable bolt supports were installed mainly from a sublevel drill drift to act as point anchors. Figure 48 illustrates the point-anchor approach to this type of bolting. Cable bolt densities found on each sublevel ranged from two to seven bolts installed on rings spaced every 2.4 m along strike. The design strategy in some cases was not to stabilize the whole hanging wall, but to limit the effect of undercutting as mining advanced to the next lift. It is believed that the localized bolt density is not as important as the distance between each point anchor. Hanging wall cables can be evenly distributed over the supported surface by drilling holes from a hanging wall drill drift (hanging wall drift fan) or countersinking the bolts through the back of a sublevel drift (even hanging wall) (fig. 48). Cables installed from a separate hanging wall drift were found on only one occasion because the cost associated with the additional development work required is high.

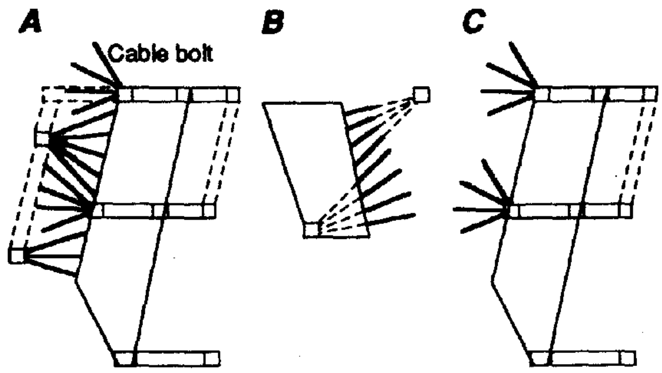

Figure 48.—Cable support patterns for hanging walls. *A*, Evenly distributed, point-anchored cable bolts placed from hanging wall drift; *B*, evenly distributed, countersunk cable bolts; *C*, point-anchored cable bolts placed from 0 sublevel drift (Fuller, 1983).

CABLE SLINGS

Cable slings were used in isolated cases of crown pillar recovery, bulkhead support, and pillar reinforcement. In the case of crown pillar recovery, cable slings supported timber mats below slag or tailings fill as mining advanced (fig. 49*A*). Slings were also used to reinforce the back and walls of a conventional sublevel development for an open stope slot. Figure 49*B* illustrates the use of a sling for support of an open stope. This type of support might be useful where cables are poorly distributed over the surface. The drill crosscuts were close enough to allow cable bolt holes to be drilled from one to the other. Cables could be installed and plated on each end in an attempt to sling the back between each drill crosscut. No patterns of this nature were encountered in practice, but they have been discussed as a design concept.

MANDOLIN CABLE BOLT SUPPORTS

Mandolin bolting is another method of cable support. Cable bolts are installed parallel to the stope hanging wall and attached to a second set of angled cables installed above the sublevel drill drift (fig. 50). The cables parallel to the hanging wall are angled less than the dip of the surface in order to place the end of the cable into competent rock. The sublevel drill drift may be shotcreted to protect the exposed portion of the cables. Several mines in Canada have considered using the mandolin bolting technique, but as yet have not tried it extensively.

In other countries, the mandolin approach has been used with success. In the early 1980's, ground control engineers at the ZC Mine in Broken Hills, New South Wales, Australia, placed cable bolts parallel to the stope across a large crack to stabilize a wall in a longhole open stope in blocky ground (fig. 50). The tension crack measured over 48 cm at one location. The support consisted of 11.4-cm-diam steel pipes grouted into 15.2-m-long holes running parallel to the wall of the stope. Three 15.9-mm-diam steel cables were then placed in the pipes, tied back into the rock at the top of the stope, and grouted. The purpose of the pipes was to increase shear resistance across major fractures. Twenty-six of these supports were installed in the wall of the stope and provided a great deal of flexible, yielding support capable of withstanding high deformation (Cutjar and others, 1985).

The use of cable bolts parallel to the wall of an open stope proved to be very effective at the ZC Mine. This same approach was also used successfully at the Outokumpu Pyhasalmi Mine in Finland. Figure 51 shows a profile of the X-22 stope at this mine, where cable bolts were installed on 1- to 1.5-m spacings (Lappalainen and Antikainen, 1987).

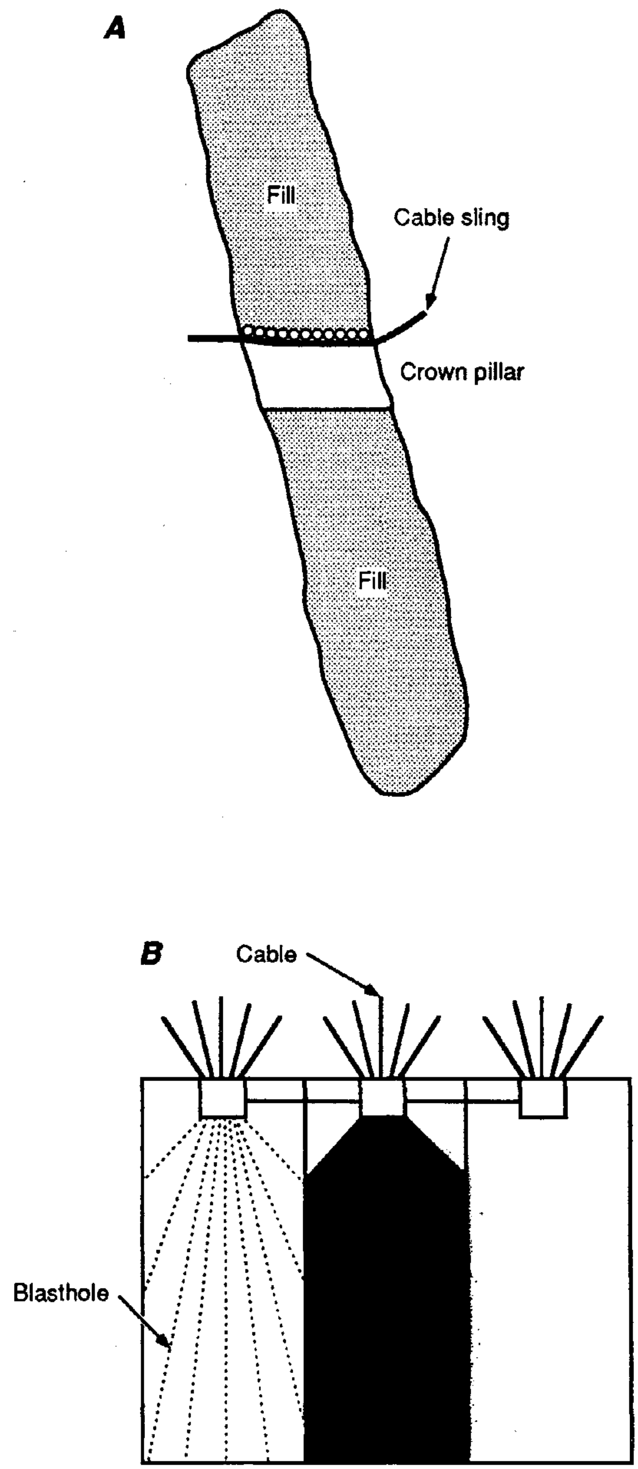

Figure 49.—Sling approach to open stope support. *A*, Crown pillar recovery; *B*, support of open stope (Nickson, 1992).

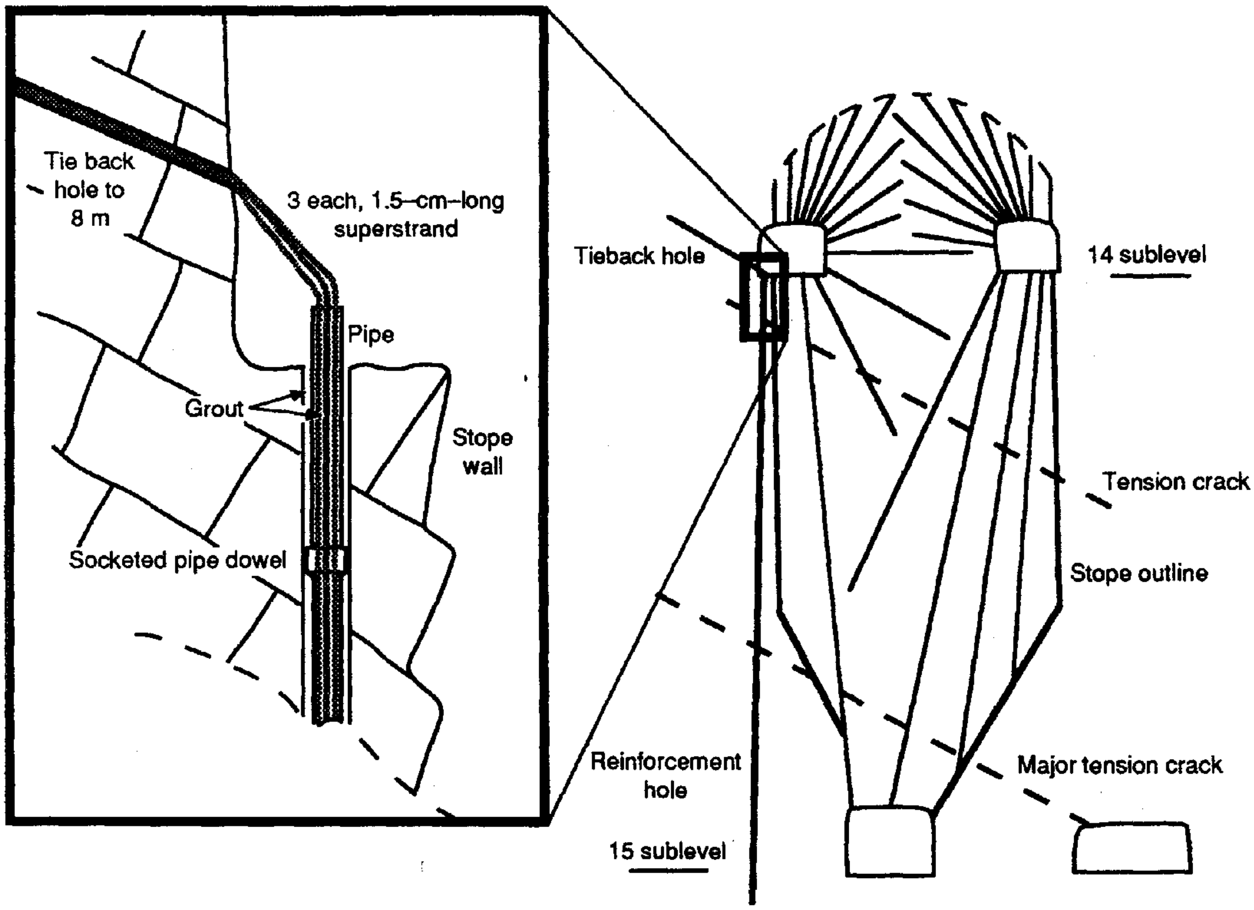

Figure 50.—Mandolin bolting approach in Australia (Cutjar and others, 1985).

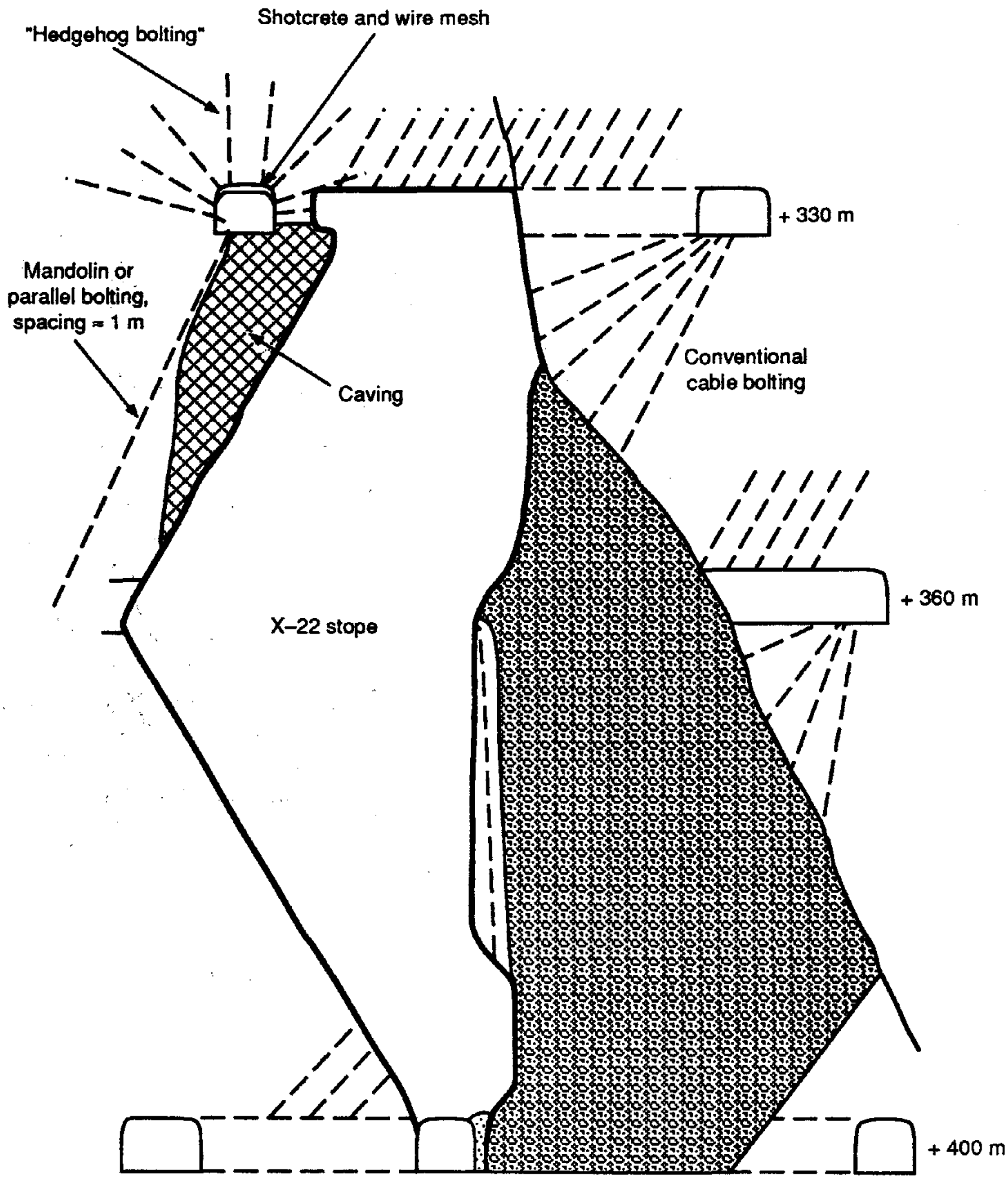

Figure 51.—Mandolin bolting approach in Finland (Lappalainen and Antikainen, 1987).

EXAMPLES OF CABLE BOLT INSTALLATION PROCEDURES

Cable bolt installation procedures (Nickson, 1992) were studied in 11 mines in Canada and 1 in the United States. The data collected provided an indication of the variations in grouting equipment, cable lengths, grout water-cement ratios, hole-plugging techniques, terminology, and other aspects of bolting practice. The procedures followed by six of the Canadian mines and one U.S. mine are summarized below. Table 9 lists the type of equipment, number of cables, and number of personnel used in cable bolting procedures at the seven mines.

Table 9.—Equipment, number of cables, and number of personnel used in cable bolting procedures

Mine	Type of equipment	Number of cables	Number in crew
Canada:			
1	Spedel 6000	2	2
2	Minepro 3	2	2
3	Spedel 6000	2	2
4	Spedel 6000	1	2
5	Minepro 3	1	2
6	Minepro 3	1	2
U.S. mine	ChemGrout	2	2

Source: Nickson, 1992.

CANADIAN MINE 1

Downholes: Use 19-mm grout tube to clean all holes prior to inserting cable bolts. Tape grout tube 15 cm from one end of cable using enough tube to run the whole length of hole and attach to grout pump. Lower cables into holes to be grouted with taped end of grout tube at toe of hole. Prepare grout with water-cement ratio of 0.35 and hook up pump to grout tube. Pump grout until it begins to come out collar of hole.

Upholes: Tape 9.5-mm breather tube to end of cable and insert into hole with taped end of breather tube at toe of hole. Insert grout tube 60 cm into collar of hole, leaving enough tube to reach pump. Plug collar of hole with rags. Prepare grout with water-cement ratio of 0.35 and hook up pump to grout tube. Pump grout until it comes out breather tube. Bend and tie off breather and grout tubes to prevent grout from leaking.

CANADIAN MINE 2

Upholes greater than 12 m long: Install steel strips on end of cable for anchorage. Tape 13-mm breather tube to end of cable with end of breather tube cut at 45°. Insert cable in hole and make sure that breather tube is not pinched. Connect water hose to breather tube and flush hole. Place grout tube 1 m into hole collar with end cut at 45°. Plug hole with Monofoam expanding foam sealant and allow to cure 24 h. Fill hole with 0.375 water-cement ratio grout. Make sure breather tube is completely full of grout.

Upholes less than 12 m long: Install steel strips on end of cable for anchorage. Tape 19-mm grout tube to end of cable. Insert cable into hole. Connect water hose to breather tube and flush hole. Fill hole with 0.35 water-cement ratio grout, making sure breather tube is completely full of grout. No plug is used at collar.

CANADIAN MINE 3

Upholes: Wash cables before installing. Install steel strips on end of cable for anchorage. Attach 9.5-mm breather tube 15 cm from end of cable with electrical tape. Wrap tube with tape at several locations along cable. Tape grout tube 2 m from the hole collar. Insert cable into hole. Seal hole collar with burlap and wooden wedges. Cut breather and grout tubes 3 m from collar. Attach grout tube to grout pump feeder hose and pump 0.4 water-cement ratio grout until it discharges from breather tube.

CANADIAN MINE 4

Upholes: Install steel strips on end of cable for anchorage. Tape 19-mm grout tube to top end of cable. Insert cable into hole and cut grout tube so that end reaches grout pump. Mix one sack (40 kg) cement with 12:1 water (water-cement ratio of 0.3). Pump grout until it comes out at hole collar. Plug collar with steel wool (four rolls). Pump grout until pump stalls. Crimp and tape grout tube at collar.

CANADIAN MINE 5

Upholes: Bend single wire at top of cable at 135°. Install cable in hole and push grout tube into hole until it reaches top; then pull back 15 cm. Tape grout tube to cable at collar of hole. Cut off grout tube, leaving enough to reach grout pump hose. Install wooden wedges in hole collar. Pump grout, bend end of grout tube, and tie off.

CANADIAN MINE 6

Upholes greater than 14 m long: Install steel strips on end of cable for anchorage. Tape 9.5-mm-diam (3-MPa) breather tube to cable in two or three places. Keep breather tube end within 15 cm of end of cable and cut at 45°. Install cable and breather tube in hole. Push a 19-mm-diam (1.7-MPa) grout tube 0.3 to 1 m into hole and leave 1-m-long tail out of hole. Wedge cable in hole with wooden wedge and plug collar with cotton waste and thick grout. Let collar seal set for 8 h. Mix grout (water-cement ratio between 0.33 and 0.40) and pump until grout flows out of breather tube. Bend end of grout and breather tubes 180° and tie off.

Upholes less than 14 m long: Install steel strips on end of cable for anchorage. Tape 19-mm (0.7-MPa) grout tube to cable in two or three places. Keep grout tube end within 15 cm of end of cable. Insert cable into hole. Wedge cable into hole with wooden wedge, making sure not to pinch grout tube. Mix grout (water-cement ratio of 0.3) and pump until it comes out hole collar. Bend and tie off grout tube. Grouts with water-cement ratios greater than 0.35 will require grout plug, so it is important to maintain a 0.3 ratio.

Downholes: Tape a 19-mm (0.7-MPa) grout tube within 15 cm of cable end. Insert cable and grout tube into hole and push to end of hole. Mix a 0.3 water-cement ratio grout and pump until grout appears at collar. Do not pull out grout tube while pumping, as tests have shown that air gaps may form in the grout column.

U.S. MINE

Upholes: Cut cables off cable reel to desired length, making sure that ends are square. Beware of stored spring energy in cable. Align pneumatic cable pusher with hole and feed cable through rollers on pusher until enough cable has been fed through to create anchor. Using flat-head screwdriver, twist single strand of wire from cable end protruding from pusher. Using ring drill starter steel, bend this wire over to form barb for anchoring cable. Barb should be between 7.6 and 11.5 cm long and be bent to an angle between 125° and 140° from vertical. If cable is longer than 12 m, bend over second barb with wire opposite first barb. Using cable pusher, run cable into hole.

When end of cable approaches rollers on pusher, attach second cable to first with sleeve and continue pushing cables until first cable is within 30 cm of hole collar. Disconnect sleeve and second cable and attach plastic end cap to end of first cable sticking out of hole. Before proceeding, pull on first cable to check anchorage. If there is any doubt, tie off cable to roof bolt before continuing. Create anchorage barb on second cable. Using electrical tape, securely fasten 12.7-mm-diam breather tube approximately 30 cm from end of second cable. Continue to push second cable into hole, leaving approximately 30 cm extending from the hole. Install 19-mm grout tube approximately 60 cm into hole. Leave enough tube hanging out of hole to reach grout tube coming from grout pump. Using cotton waste and expanding foam, fill lower portion of hole, tightly packing hole to prevent grout leakage. Grout hole.

Bend back one wire for cable anchorage if hole is shorter than 12 m, or two wires if hole is longer than 12 m. Use cable pusher if hole is more than 12 m.

CABLE BOLT DESIGN METHODS

Lang (1961) referred to rock bolting as "the designed use of rock bolts to reinforce and envelop the rock around an excavation into a structural entity." This concept takes advantage of the inherent strength of a rock mass and was first applied to cable bolts in cut-and-fill mining.

Most cable bolt designs have been developed using an even distribution of bolts in a regular square pattern. Where access is restricted, an even pattern may not be possible, and bolts are frequently installed at high densities in a fan-shaped configuration to act as point anchors. The point-anchor approach differs from pattern bolting in that large areas of the rock surface are left unsupported.

There are a number of methods for designing cable bolts supports. This section presents a brief overview of the most common methods: deadweight, arch theory, beam theory, past experience, Mathews method and bolting factor, and Potvin method. It is recommended that the original published document be reviewed prior to the use of any particular method.

DEADWEIGHT CONCEPT

Deadweight design is used at a number of mines when cable bolts are required to stabilize a specific block or wedge. In some cases, this method is used to design a general cable bolt pattern for stope backs. Most of the designs using a deadweight analysis assume that the tensile strength of the cable is equivalent to the bond strength of the cable-grout interface.

This method involves estimating the weight of the rock to be supported and using a factor of safety to determine the number of bolts required according to the following equation:

$$N = \frac{W \times FS}{T}, \quad (1)$$

where N = number of cable bolt supports,

W = weight of the rock, mt,

FS = factor of safety,

and T = bolt load (load-carrying capacity of each cable bolt), mt.

For example, assuming W = 100 mt, FS = 1.2, and T = 26.3 mt for each cable bolt, N would be

$$N = \frac{100 \times 1.2}{26.3} = 4.6.$$

Therefore, five cable bolts would be used.

The number of cable bolts would be distributed evenly over the area involved and would extend 1.5 to 4 m or more beyond the projected failure plane. If, however, the embedment lengths of the cables are limited, their load-carrying capacity may be reduced, resulting in a greater number of required cable bolts.

ARCH THEORY

Arch theory is also based on the concept of deadweight. Determination of the weight of rock to be supported is based on defining a zone of expected instability. This is not a well-defined procedure unless a fault or shear zone isolates an area of the rock mass. A natural arch is formed around an opening because of redistribution of stresses. The height of the arch can be determined through numerical modeling and has been related to span opening. The spacing of the cable bolt supports is given by the equation

$$BS = \frac{2T}{FS \times H \times D}, \quad (2)$$

where BS = bolt spacing, m^2,

H = height of the arch, m,

and D = density of the rock, kg/m^3.

This method assumes that the full bolt load-carrying capacity is used and that there is competent ground above the arch. Cable bolt lengths are typically determined by allowing for at least 4 m of anchorage into the arch.

Stheeman (1982) describes a procedure used at the Tsumeb Mine where the curvature of a natural arch above a cut-and-fill stope was approximated mathematically. The curved shape of fracture planes forming the walls of a cavity after a rock fall indicated the limits of the degree of self-support of such an arch. Fracture plane angles were noted at different positions and used to derive an elliptical relationship that supported the observations. This relationship was used to estimate the volume of rock below the natural arch that required cable support. The number of cables required was determined by dividing the weight of rock below the arch by the estimated breaking strength of the cable. The calculated cable density was subsequently increased to reflect a safety factor of 1.2. Bolt length was based on the coverage of five mining lifts, the height of the arch, the embedment length required to support the weight below the arch, and an allowance for grout bleeding.

BEAM THEORY

Beam theory has been applied to cable bolt design where the geologic structure is parallel to the surface to be supported. This is most commonly applied in a layered rock mass, where cable bolt supports are used to tie a number of layers together into a stable beam. The design method is similar to that used with the arch theory and is based on using the deadweight of the beam to determine cable bolt spacing. The volume of rock is approximated as a rectangular area, and the bolt spacing is calculated by the equation

$$BS = \frac{T}{FS \times L \times D}, \quad (3)$$

where L = the beam thickness, m.

Fuller (1983) describes a design approach for a point-anchor cable bolt pattern based on beam theory. A hanging wall in layered rock was assumed to behave as a beam, as shown in figure 52. Cable bolts installed in the hanging wall in a fan-shaped configuration at two intermediate sublevels were simulated by the reaction R_c. While stope abutments were fixed, the analysis allowed for some beam deflection at the intermediate points to represent the true action of cables taking load.

In this analysis, Fuller assumed that the beam experienced uniformly distributed loading (based on a rock specific gravity of 4) and related cable reactions to laboratory pull-test curves for different embedment lengths of 15.2-mm-diam double cables. The number of cables required along strike was related to beam thickness, as illustrated in figure 52. Thus, a minimum cable bolt density of four cables for each meter of strike length was determined for this particular stope. Beams less than 1 m thick were found to fail in tension.

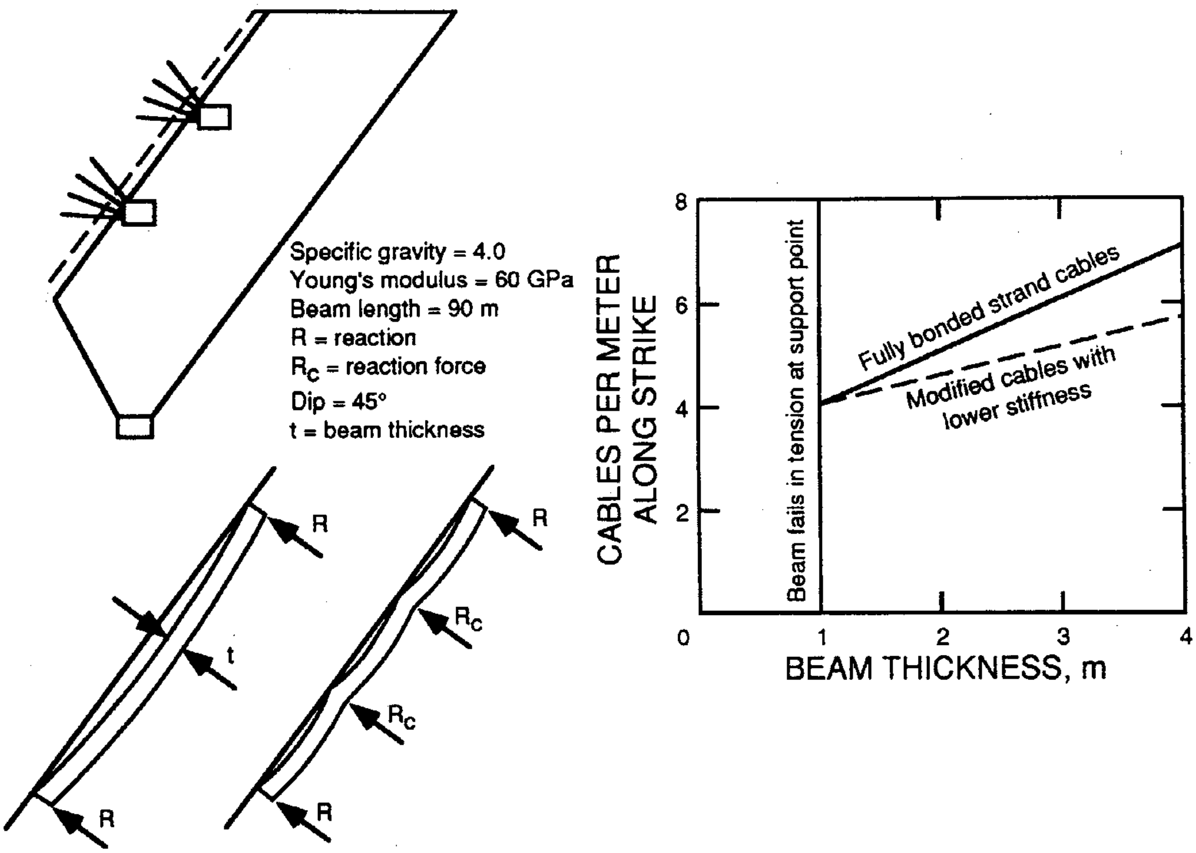

Figure 52.—Beam approach to hanging wall design (Fuller, 1983).

Case histories of hanging wall support collected from western Canada typically reflect major structures parallel to the surface. In most instances, however, there was a second minor joint set that contributed to failure between sublevels and limited the success of point-anchor cable bolting.

PAST EXPERIENCE

Past experience is the most common design method encountered in western Canada and the United States. This method relies on an initial trial-and-error process followed by adjustments based on performance. In many cases, the initial design is based on typical patterns used at other operations. The span of the opening created is usually the initial criterion that determines whether cable bolt supports are required. Therefore, larger spans dictate the use of a tighter cable pattern.

The use of this method is most noticeable where cut-and-fill mining is in progress. The cables are installed at lengths up to 18 m and cover several mining lifts. In these cases, the effectiveness of a cable bolt pattern can be evaluated fairly easily since cut-and-fill lifts are mined rapidly relative to open stoping situations.

This method has also been used in the design of open-stope back support, although the initial pattern selection is much more critical. Modifications to the installation after mining has begun are difficult, if not impossible. Cables are required in the back if the span exceeds a specified amount, which is determined by experience. This critical span is reduced if the quality of the rock mass deteriorates. In some cases, the cable bolt spacing is determined by the blasthole ring burden so that cable bolt holes can be drilled from existing setup points.

MATHEW'S METHOD AND BOLTING FACTOR

Mathews (Mathews and others, 1981) developed an empirical relationship between a stability number, N, and a shape factor, S, of a stope surface. The stability number can be evaluated as

$$N = Q' \times A \times B \times C, \qquad (4)$$

where Q' = Q-system rock mass rating with the stress reduction factor set to 1,[7]

A = stress factor,

B = rock defect orientation factor,

and C = design surface orientation factor.

The shape factor (S) is also called the hydraulic radius (HR) and is determined by the equation

$$S = HR = \frac{SA}{PR}, \qquad (5)$$

where SA = surface area of a stope back, m^2,

and PR = perimeter of the back, m.

The terms "span" and "length" are often applied in open-stope terminology, where the span of a surface can be defined as the minimum dimension and the length of that surface as the maximum dimension. A traditional longitudinal stoping sequence relates length to the distance along strike and span to ore body width. Hydraulic radius involves both the span and length of a particular surface. For square openings, HR is one-fourth the span, but as the ratio of span to length decreases, HR converges to half the span. Mathews proposed a design chart (fig. 53) that relates the stability number to the shape factor and defines zones of stability, potential instability, and potential caving. These zones are further described as follows:

1. Stable - The excavation will stand unsupported if some localized ground support is placed to control slabbing.
2. Unstable - The excavation will experience localized caving but will tend to form a stable arch. Cable bolt supports and modification of the extraction sequence are suggested as ways to make open stoping feasible.
3. Caved - The excavation will not stabilize until the void is full.

Figure 54 presents factors A, B, and C graphically. The rock stress factor A is related to σ_c:σ_i, which is the ratio of the uniaxial compressive strength of intact rock to the induced compressive stress parallel to the surface under consideration. Cases of high induced stress will reflect a lower σ_c:σ_i ratio and an overall reduction in the stability number through a reduction in factor A. The rock defect orientation factor B is based on the orientation of the most persistent joint set with respect to the stope surface. Structures perpendicular to the surface reflect the most favorable orientation and are given the highest rating. The design surface orientation factor C is based on the assumption that a vertical wall is eight times as stable as a horizontal surface under the effects of gravity.

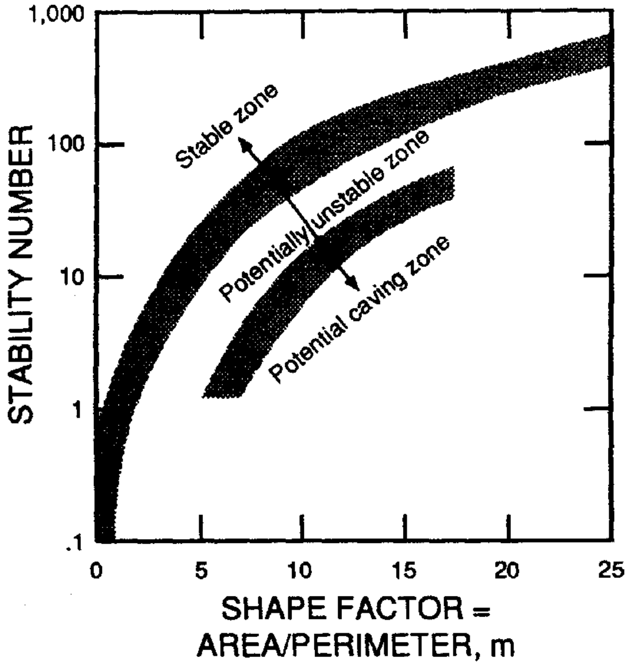

Figure 53.—Stability graph for underground excavations (Mathews and others, 1981).

POTVIN METHOD

The modified stability graph (fig. 55) was developed empirically from an analysis of 242 case histories of open-stoping situations in Canada (Potvin, 1988). The study involved the review of such essentials as mining method, rock conditions and properties, stope geometry, cable bolt density, and length of cable bolts. Based on these case studies and the concepts behind the Mathews design method, a stable zone and a caving zone were identified by relating a modified stability number, N', to HR of the surface. Potvin's (1988) approach described N' in terms of a block size factor, a compressive stress factor, a joint orientation factor, and a gravity factor. Therefore, if

[7]A discussion of the Q-system rock mass rating is presented in the section on "Potvin Method."

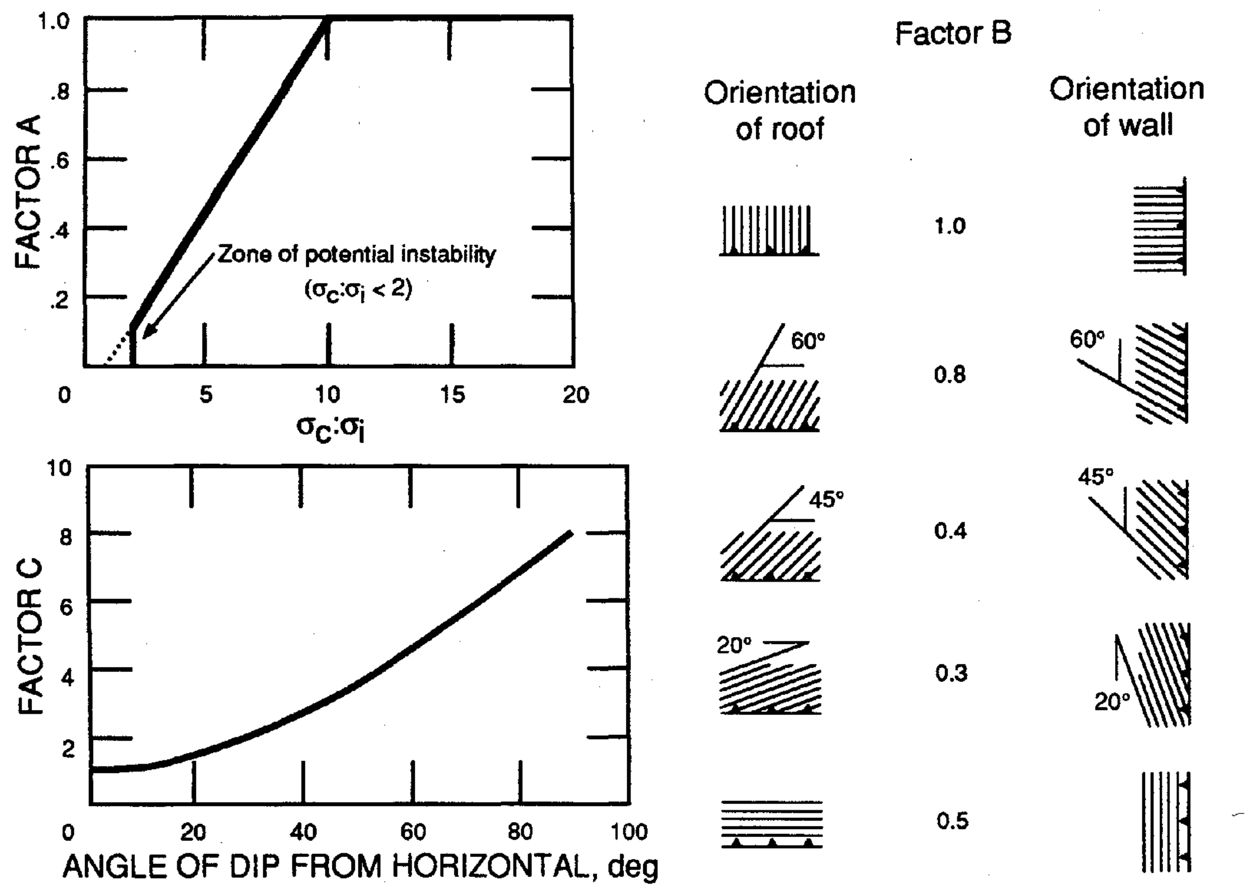

Figure 54.—Graphs for factors A, B, and C (Mathews and others, 1981).

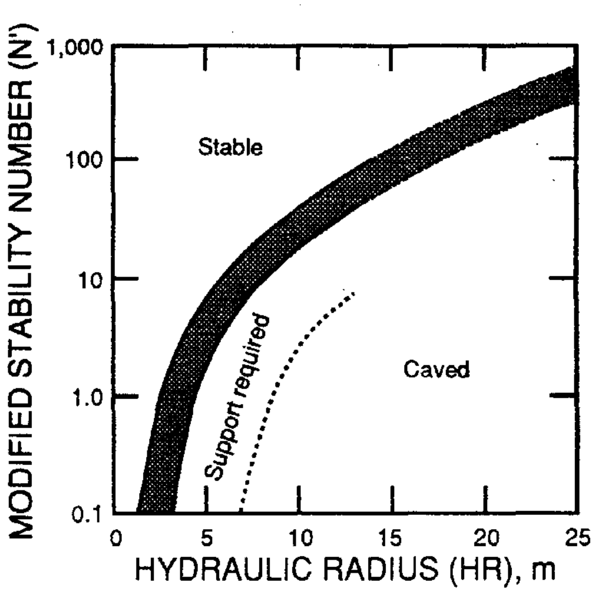

Figure 55.—Modified stability graph (Potvin, 1988).

$$Q' = \frac{RQD}{J_n} \times \frac{J_r}{J_a}, \qquad (6)$$

then $N' = Q' \times A \times B \times C$,

where RQD = rock quality designation,

J_n = joint set number,

J_r = joint roughness number,

and J_a = joint alteration number.

RQD and J_n, J_r, and J_a can be estimated from tables 10, 11, 12, and 13. Values for A, B, and C can be estimated from figure 56.

Once N' and HR are determined, the expected stability condition of the surface can be determined using figure 55.

Potvin (1988) suggested that the first stage in designing a cable bolt pattern is to complete a stability analysis to

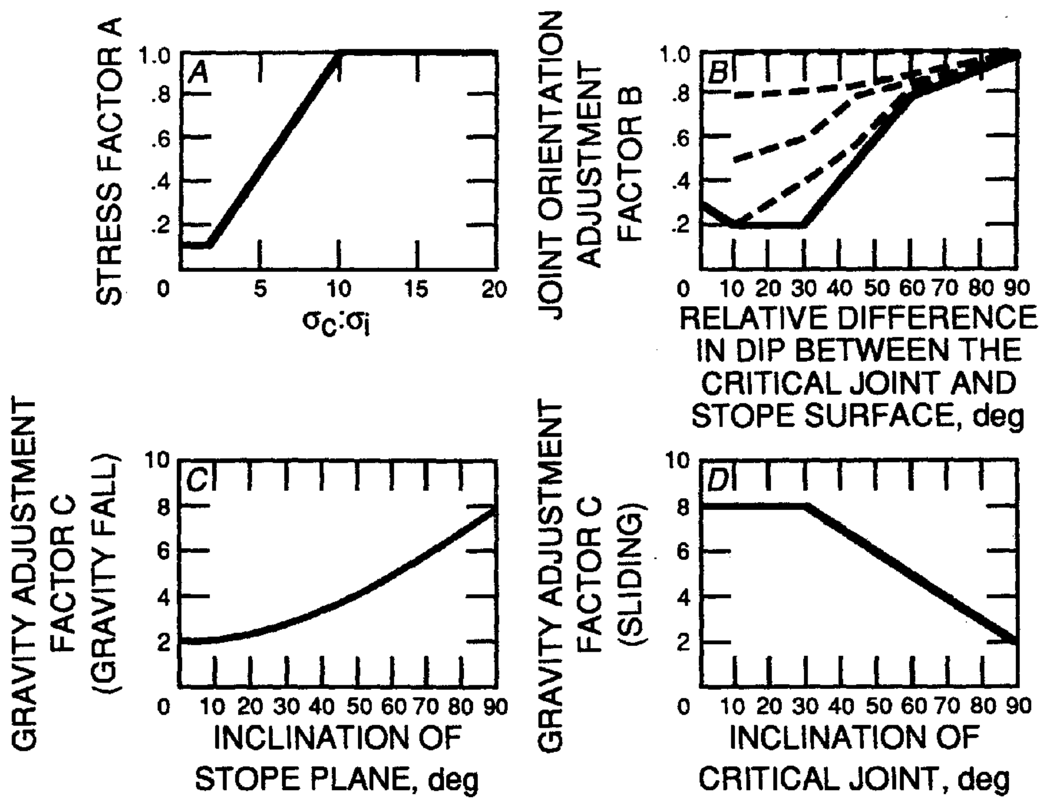

Figure 56.—Factors A, B, and C proposed by Potvin.

determine the expected stability condition of the surface. A zone is identified on the modified stability graph that is within the caving zone, but that could be classed as stable if cable bolt supports were added; that is, a stope that lies within this zone would require cable bolts to remain stable. Assuming that support is required, the density of cable bolt supports could be determined from the design chart for cable bolt density (fig. 57). This chart relates cable bolt density to a relative block size factor, RQD/(J_n × HR). It was originally developed on the basis of an analysis of 66 case histories of stope backs supported with cable bolts.

Table 10.—Rock quality designation (RQD)[1,2]

Designation	RQD
Very poor	0- 25
Poor	25- 50
Fair	50- 75
Good	75- 90
Excellent	90-100

[1]Where RQD is reported or measured as ≤10 (including 0), a nominal value of 10 is used to evaluate Q.

[2]RQD intervals of 5 (e.g., 100, 95, 90) are sufficiently accurate.

Source: Barton and others, 1974.

Table 11.—Joint set number (J_n)

Joint set	Corrected joint sets
Massive, none or few joints	0.5- 1.0
1 joint set	2
1 joint set plus random	3
2 joint sets	4
2 joint sets plus random	6
3 joint sets	9
3 joint sets plus random	12
4 or more joint sets random, heavily jointed, "sugar-cube-like"	15
Crushed rock, earthlike	20

NOTE.—For intersections, use three times the number of joints; for portals, use two times the number of joints.

Source: Barton and others, 1974.

The design techniques developed by Potvin (1988) are based on open stoping experience in Canadian mines. Nickson (1992) found that the modified stability graph (fig. 55) for stope design is used often in western Canadian operations. The design chart for cable bolt density (fig. 57) is rarely used to prepare the final design but is consulted before the desired pattern is determined. This chart (fig. 57) applies only in mines where there is an even distribution of cable bolts over the surface and is intended for use only where stope backs plot in the supportable zone of figure 55. Figure 57 is not intended for designing hanging wall supports or point-anchor back support. Potvin and Milne (1992) also suggest that weak material, slot raises or brow developments, and poor-quality grout will limit use of the cable bolt density chart.

Table 12.—Joint roughness number (J_r)

Joint description	J_r[1,2]
Rock wall contact before 10-cm shear:	
Discontinuous joints	4
Rough or irregular, undulating	3
Smooth, undulating	2
Slickensided, undulating	1.5
Rough or irregular, planar	1.5
Smooth, planar	1.0
Slickensided, planar	0.5
No rock wall contact when sheared:	
Zone containing clay minerals thick enough to prevent rock wall contact	1.0
Sandy, gravelly, or crushed rock thick enough to prevent rock wall contact	1.0

[1]Add 1.0 if the mean spacing of relevant joint set is greater than 3 m.

[2]J_r = 0.5 can be used for planar slickensided joints having lineations.

Source: Barton and others, 1974.

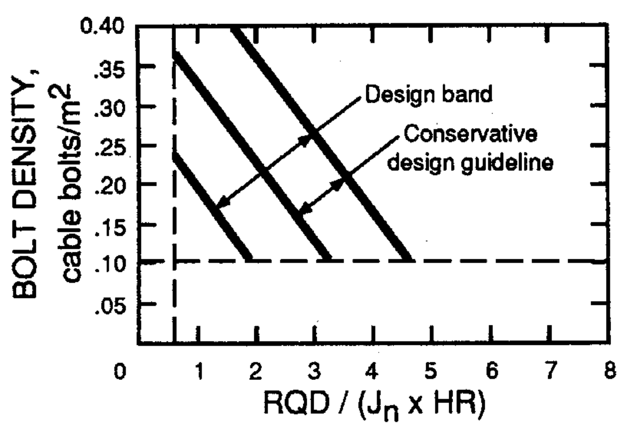

Figure 57.—Design chart for cable bolt density (Potvin, 1988).

Table 13.—Joint alteration number (J_a)

Joint description	J_a[1]	Residual friction angle, deg
Rock wall contact:		
Tightly healed, hard, nonsoftening, impermeable fillings, e.g., quartz or epidote	0.75	0
Unaltered joint walls, surface straining only	1.0	25-35
Slightly altered joint walls. Nonsoftening mineral of the alteration products, coatings, sandy particles, clay-free disintegrated rock	2.0	25-30
Silty or sandy clay coatings, small clay fraction (nonsoftening)	3.0	20-25
Softening or low-friction clay mineral coating, e.g., kaolinite, mica. Also chlorite, talc, gypsum, graphite, and small quantities of swelling clays. Discontinuous coatings, 1 to 2 mm or less in thickness	4.0	8-16
Rock wall contact before 10-cm shear:		
Sandy particles, clay-free disintegrated rock	4.0	25-30
Strongly overconsolidated, nonsoftening, clay mineral fillings. Continuous, <5 mm thick	6.0	16-24
Medium or low overconsolidated, softening, clay mineral fillings. Continuous, <5 mm thick	8.0	12-16
Swelling clay fillings, e.g., montmorillonite. Continuous, <5 mm thick. Value of J_a depends on percentage of swelling clay-sized particles and access to water	8-12	6-12
No rock wall contact when sheared:		
Zones or bands of disintegrated or crushed rock and clay	6, 8, or 8 to 12	6-24
Zones or bands of silty or sandy clay, small clay fraction (nonsoftening)	5	
Thick, continuous zones or bands of clay	5, 10, 13, or 13 to 20	6-24

[1]Values for friction angle are intended as an approximate guide to mineralogical properties of alteration products, if present.

Source: Barton and others, 1974.

UPDATED POTVIN METHOD

In 1992, Nickson (1992) completed a study that added more information to Potvin's database of cable bolt support practices. Twelve mines were visited to review cable bolt support practices. Eleven of the mines were located in western Canada, and one was in the United States.

The design methodology proposed by Potvin (1988) was based on cable bolt support of open stope backs. In Nickson's study (1992), particular emphasis was placed on collecting information on cable bolt supports placed in the hanging wall, which would expand the use of the modified stability graph.

Nickson's study included 13 cases of unsupported ground and 46 cases of supported ground. An "unsupported case" was defined as a stope that did not contain cable bolts. A "supported" case referred to a stope where the surface contained cable bolt supports. Tables 14 and 15 summarize data collected for both the unsupported and the supported stopes. These tables illustrate the range of variation in rock and stope conditions and densities and lengths of cable bolts.

Values for HR and N' in tables 14 and 15 were calculated and then plotted on the modified stability graph presented by Potvin (1988) (fig. 55). Overall, 85 pct of the case histories studied by Nickson agreed with the design ranges established by Potvin (1988). For the database of unsupported rock, 83 pct of the stable surfaces fell above the transition zone, while 80 pct of the caved surfaces fell below the transition zone (fig. 55). Also, 20 pct of the cases where the stopes were supported with cable bolts fell above the transition zone. This region is classified as "stable without support" and indicates a conservative approach to cable bolt support design by some operators.

Nickson's (1992) case histories of back support were then plotted on the design chart for cable bolt density (fig. 57). Seventy-nine percent of the cases were in agreement with the design ranges proposed by Potvin (1988). The database supported Potvin's observation that bolt densities of less than 0.1 bolt/m^2 are not used by mine operators. Unfortunately, the availability of data concerning rock falls at the mines in the study was limited; most of the case studies indicated stope backs were stable.

Nickson then combined his data with those collected by Potvin and conducted a statistical analysis to determine if modifications should be made to the modified stability graph (fig. 55) and the design chart for cable bolt density (fig. 57). The analysis resulted in a revision of the modified stability graph (fig. 58) (Nickson, 1992). The supportable region originally proposed by Potvin (1988) was divided into a stable-with-support zone and a supported transition zone. The combined databases increased confidence in the stable-with-support zone.

All of the cases plotted in the stable-with-support zone and the unsupported transition zone were stable and indicated a high degree of design confidence. Cable bolt supports should be considered when a design surface falls within the supportable region of the revised stability graph (fig. 58).

Table 14.—Summary of database for cases with no cable bolt supports[1]

Case	Surface	HR, m	Q'	A	B	C	N'	Stability
22	Back	6.2	13.3	0.1	0.2	2.0	0.53	Caved.
24	Back	5.2	13.3	0.1	0.2	2.0	0.53	Caved.
28	Hanging wall	10.3	10.0	1.0	0.3	5.0	15.0	Caved.
31	Hanging wall	16.4	5.9	1.0	0.2	5.5	6.5	Caved.
35	Hanging wall	7.0	13.1	1.0	0.2	8.0	21.0	Stable.
38	Hanging wall	5.2	7.2	1.0	0.2	5.0	7.2	Stable.
39	Back	1.3	15.8	0.1	0.2	2.0	0.63	Stable.
40	Hanging wall	6.1	21.5	1.0	0.3	6.0	38.7	Stable.
41	Back	1.8	15.8	0.1	0.2	2.0	0.63	Stable.
42	Hanging wall	6.1	21.5	1.0	0.2	5.0	21.5	Unstable.
44	Hanging wall	5.9	7.2	1.0	0.2	5.0	7.2	Caved.
51	Hanging wall	10.4	8.3	1.0	0.3	5.0	12.5	Stable.
58	Hanging wall	4.9	3.1	1.0	0.3	6.0	5.6	Unstable.

[1]HR = hydraulic radius; N' = modified stability number; see equation 4 for identification of other symbols.

Source: Nickson, 1992.

Table 15.—Summary of database for cases with cable bolt supports

Case	Surface	HR, m	Q'	A	B	C	N'	Stability	RQD/ J_n	RQD/ ($J_n \times$ HR)	Cable density, bolt/m[2]	Cable bolt length, m
1 ...	Hanging wall ...	10.0	11.7	1.0	0.2	6.0	14.0	Caved	11.7	1.17	0.018	9.1 - 18.3
2 ...	Back	2.3	11.7	1.0	0.2	2.0	4.7	Stable	11.7	5.09	0.130	6.1
3 ...	Hanging wall ...	11.7	2.5	1.0	0.3	7.5	5.6	Stable	6.7	0.57	0.021	6.1
4 ...	Hanging wall ...	19.1	2.5	1.0	0.3	7.5	5.6	Caved	6.7	0.35	0.018	6.1
5 ...	Back	2.6	11.7	1.0	0.2	2.0	4.7	Stable	11.7	4.50	0.160	9.1
6 ...	Hanging wall ...	17.1	27.3	1.0	C.3	7.5	61.4	Caved	13.7	0.80	0.022	6.1
7 ...	Hanging wall ...	10.9	2.5	1.0	0.2	7.5	3.8	Unstable ...	6.7	0.61	0.011	6.1
8 ...	Hanging wall ...	12.7	2.5	1.0	0.3	7.5	5.6	Unstable ...	6.7	0.53	0.020	6.1
9 ...	Hanging wall ...	13.2	23.7	1.0	0.3	8.0	56.9	Stable	11.8	0.89	0.018	6.1
10 ..	Back	5.0	18.8	0.2	0.3	2.0	2.3	Stable	12.5	2.50	0.130	22.0
11 ..	Hanging wall ...	10.8	30.0	1.0	0.3	7.5	67.5	Stable	15.0	1.39	0.023	6.1
12 ..	Back	1.6	18.8	0.1	0.2	2.0	0.75	Stable	12.5	7.81	0.580	7.6
13 ..	Hanging wall ...	3.6	11.7	1.0	0.2	2.0	4.7	Stable	11.7	3.25	0.116	6.1
14 ..	Hanging wall ...	4.3	0.6	1.0	0.4	2.0	0.48	Stable	1.67	0.39	0.180	14.0
15 ..	Hanging wall ...	7.6	0.5	1.0	0.3	3.0	0.45	Stable	2.5	0.33	0.280	8, 15
16 ..	Hanging wall ...	11.2	0.5	1.0	0.3	3.0	0.45	Caved	2.5	0.22	0.180	8, 12
17 ..	Hanging wall ...	8.6	0.6	1.0	0.2	2.0	0.24	Caved.	1.67	0.19	0.140	12, 15
18 ..	Hanging wall ...	4.2	13.3	0.1	0.3	2.0	0.80	Stable	13.3	3.17	0.290	9.8
19 ..	Hanging wall ...	5.2	13.3	0.1	0.3	2.0	0.80	Unstable ...	13.3	2.56	0.270	9.8
20 ..	Hanging wall ...	12.4	15.8	1.0	0.2	5.0	15.8	Caved	15.8	1.27	0.035	14.6
21 ..	Hanging wall ...	10.8	15.8	1.0	0.3	6.0	28.4	Stable	15.8	1.46	0.031	14.6
23 ..	Back	5.2	13.3	0.1	0.2	2.0	0.53	Stable	13.3	2.56	0.330	9.8
25 ..	Back	6.4	13.3	0.1	0.2	2.0	0.53	Caved	13.3	2.08	0.170	9.8
26 ..	Hanging wall ...	11.5	15.8	1.0	0.3	7.0	33.2	Stable	15.8	1.37	0.025	14.6
27 ..	Back	10.7	15.8	1.0	0.3	6.5	30.8	Stable	15.8	1.48	0.041	14.6
29 ..	Back	2.1	0.9	0.1	0.8	2.0	0.14	Stable	2.5	1.19	0.167	18.3
30 ..	Back	2.0	11.6	0.1	0.2	2.0	0.46	Stable	6.6	3.30	0.410	6.1
32 ..	Hanging wall ...	4.9	10.4	1.0	0.2	5.0	10.4	Stable	6.9	1.41	0.070	12.0
33 ..	Back	1.7	8.9	0.1	0.4	2.0	0.71	Stable	8.9	5.24	0.540	6.1
34 ..	Back	5.1	8.3	0.1	0.2	2.0	0.33	Unstable ...	4.7	0.92	0.300	9.1
36 ..	Back	1.8	29.2	1.0	0.5	2.0	29.2	Stable	11.1	6.17	0.550	6.1
37 ..	Back	2.3	12.3	0.1	0.2	2.0	0.49	Stable	5.8	2.52	0.410	12.2
43 ..	Back	2.4	11.1	0.1	0.2	2.0	0.44	Stable	6.25	2.60	[2]NAp	4.9
45 ..	Back	3.6	26.1	0.1	0.3	2.0	1.6	Stable	9.8	2.72	0.210	15.8
46 ..	Back	5.0	5.4	0.1	0.2	2.0	0.22	Stable	10.8	2.16	0.304	18.3
47 ..	Back	5.1	5.4	0.1	0.2	2.0	0.22	Stable	10.8	2.12	0.308	18.3
48 ..	Back	5.0	25.0	0.4	0.2	2.0	4.0	Stable	16.7	3.34	0.245	18.3
49 ..	Hanging wall ...	15.5	9.9	1.0	0.3	4.7	14.0	Stable	13.1	0.85	0.160	9.1
50 ..	Hanging wall ...	17.0	3.1	1.0	0.3	4.7	4.4	Caved	8.3	0.49	0.180	9.1
52 ..	Back	5.2	15.5	0.1	0.2	2.0	0.62	Stable	15.5	2.98	0.110	10.2
53 ..	Back	3.8	5.4	0.1	0.2	2.0	0.22	Stable	10.8	2.84	0.330	10.7
54 ..	Hanging wall ...	7.9	3.1	1.0	0.3	5.0	4.7	Stable	8.3	1.05	0.130	9.1
55 ..	Back	6.2	25.0	0.2	0.2	2.0	2.0	Stable	16.7	2.69	0.350	5.0
56 ..	Hanging wall ...	10.9	3.1	1.0	0.3	5.0	4.7	Stable	8.3	0.76	0.120	7.6 - 9.1
57 ..	Back	5.4	5.4	0.1	0.2	2.0	0.22	Stable	10.8	2.00	0.340	6.4
59 ..	Back	5.6	25.0	0.4	0.3	2.2	6.6	Stable	12.9	2.30	0.260	4.9

NAp Not applicable.

[1]HR = hydraulic radius; N' = modified stability number; RQD = rock quality designation; J_n = joint set number; see equation 4 for identification of other symbols.

[2]Back supported with rock bolts.

Source: Nickson, 1992.

Using the combined databases from Potvin (1988) and Nickson (1992), the design chart for cable bolt density was revised (fig. 59). This revised chart is recommended for determining cable bolt density for back support when cable bolts are evenly distributed over the surface. Square and fan-back patterns are the best application of this type of support, so they make up most of the cases in the combined database.

The design line in figure 59 represents the regression line obtained from a statistical analysis of the combined databases. It is recommended that cable bolt densities be above the design line for stope conditions that plot in the supportable region of figure 58. The upper limit of the 68 pct confidence band is suggested as a minimum design density for the supported transition zone.

CABLE BOLT LENGTH

Potvin and others (1989) related the length of cable bolts required to support a rock surface to HR of that surface (fig. 60) by conducting a regression analysis to define a linear relationship between cable length and HR where the stope backs were stable. The relationship is defined as cable bolt length = 1.30 + 1.84 × HR.

The regression line in figure 60 has a correlation coefficient of 0.495 and is very similar to the design line proposed by Potvin (1988). It is suggested that this line be used as a guide for determining the appropriate lengths for cable bolts for back support. The ratio of cable length to HR ranges from 3.1 to 2.0 and can be related to the span of the opening (table 16).

Table 16.—Relationship between cable length and span for back support

Ratio of length to span	Ratio of hydraulic radius (HR) to span	Cable length times span
1:1	0.25	0.5 to 0.8
2:1	0.33	0.7 to 1.0
4:1	0.40	0.8 to 1.2
9:1	0.45	0.9 to 1.4

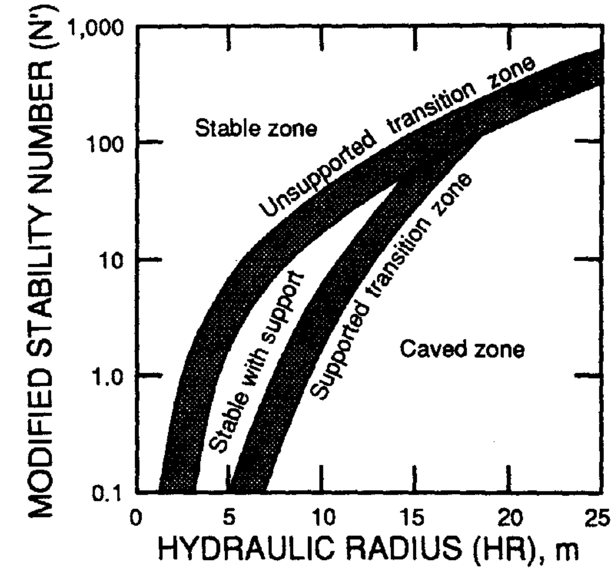

Figure 58.—Revised stability graph (Nickson, 1992).

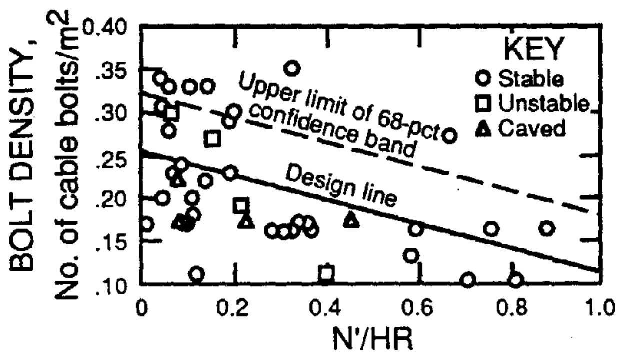

Figure 59.—Design chart for back cable support (Nickson, 1992).

POINT-ANCHOR APPROACH TO CABLE BOLT DESIGN

Fuller (1983) made the distinction between a localized and a uniform cable distribution in open stope hanging walls (fig. 48). The purpose of the localized, or point-anchor, approach to hanging wall support is described as dividing the hanging wall into smaller unsupported stable spans. The location of cable bolt supports is usually determined by sublevel development, and span is therefore related to the distance between sublevels. Thirteen cases of point-anchor support for hanging walls were assembled by Nickson (1992) and are summarized in table 17. Supported and unsupported spans have been defined as illustrated in figure 61. Unsupported spans for a particular case were variable, and the maximum unsupported span

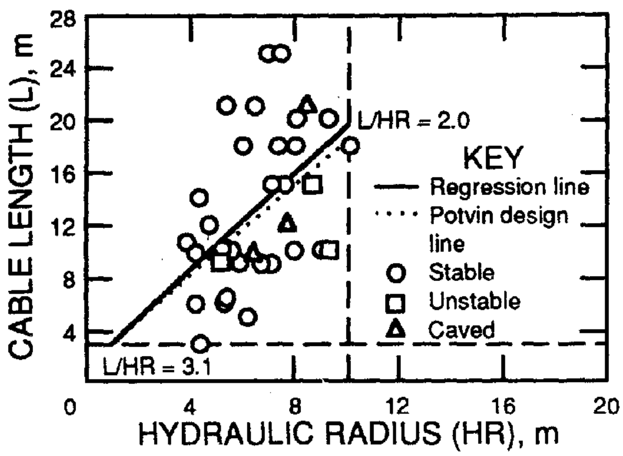

Figure 60.—Cable bolt length for back support (Nickson, 1992).

Table 17.—Database for point-anchor support for hanging walls[1]

Case	Stability	Supported HR, m	Average dip, deg	Strike, m	Supported span, m	Number of spans	Unsupported span, m			Supported HR, m			RQD/J_n	$RQD/(J_n \times HR)$	N'	Length, m	Max length, unsupported span, m
							Min	Max	Av	Min	Max	Av					
3	Stable	11.7	80	66.4	35.9	2	14.4	21.3	17.9	5.9	8.1	7.0	6.7	0.57	5.6	6.1	0.29
4	Caved	19.1	80	59.7	106.1	4	21.3	32.2	26.6	7.7	10.7	9.2	6.7	0.35	5.6	6.1	0.19
6	Caved	17.1	82	67.2	69.6	3	19.9	28.8	23.3	7.8	9.9	8.7	13.7	0.80	61.4	6.1	0.21
7	Unstable ...	10.9	85	32.6	66.3	2	20.1	46.4	32.2	6.2	9.6	8.2	6.7	0.61	3.8	6.1	0.13
8	Unstable ...	12.7	84	40.2	67.3	2	19.8	47.4	33.6	6.6	10.9	9.2	6.7	0.53	5.6	6.1	0.13
9	Stable	13.2	87	68.6	44.2	2	18.5	25.6	22.1	7.2	9.4	8.4	11.8	0.89	56.9	6.1	0.24
11	Stable	10.8	85	33.4	63.2	3	16.5	26.4	20.3	5.5	7.5	6.3	15.0	1.39	67.5	6.1	0.23
20	Caved	12.4	62	33.3	103.0	2	29.0	74.0	51.5	7.8	11.5	10.1	15.8	1.27	15.8	14.6	0.20
21	Stable	10.8	72	28.8	95.5	2	26.5	69.0	47.3	6.9	10.2	9.0	15.8	1.46	28.4	14.6	0.21
26	Stable	11.5	79	27.7	139.0	2	66.0	76.0	71.0	10.0	9.8	10.0	15.8	1.37	33.2	14.6	0.19
27	Stable	10.7	74	28.6	84.0	2	21.8	66.0	43.9	6.2	10.0	8.7	15.8	1.48	30.8	18.3	0.28
49	Stable	15.5	62	130.0	44.0	2	20.4	23.6	22.0	8.9	10.1	9.5	13.1	0.85	14.0	9.1	0.39
50	Caved	17.0	62	111.0	62.1	2	23.6	38.5	31.1	10.1	14.3	10.3	8.3	0.49	4.4	9.1	0.24

[1]HR = hydraulic radius; RQD = rock quality designation; J_n = joint set number.

Source: Nickson, 1992.

was used in this analysis. It is proposed that the success of the point-anchor approach to cable bolt support is related to the distance between sublevels and the size of the rock mass block. The design chart for point-anchor support for hanging walls is illustrated in figure 62. The chart relates the maximum unsupported HR to the relative block size factor expressed in terms of the supported HR. A statistical analysis of the point-anchor database was used to derive a support line for design. Underground mapping and stope planning will give an indication of the relative block size factor for design purposes. An acceptable design is indicated by projecting vertically up from the horizontal axis to the design line and reading an acceptable unsupported HR on the vertical axis. The unsupported HR was determined by considering the span and associated strike length for each sublevel interval. It is intended that this procedure be reversed to derive either an acceptable span or a strike length to be excavated.

The design chart for point-anchor supports for hanging walls is based on the assumption that the revised modified stability graph (fig. 58) can be used to determine if cable bolt supports are required. Additional case histories should be collected to improve upon this relationship. This design method also assumes that adequate support has been installed at each sublevel, and it is recommended that bearing plates be used. The average water-cement ratio in the point-anchor database ranges between 0.40 and 0.45.

Because of the large surface areas involved, it is difficult to relate bolt density to the number of bolts per square meter of surface area, as proposed in the design chart for back cable bolt support. Preliminary guidelines for bolt density and length can be based on current practice. Bolt density for point-anchor support is related to the number of bolts installed on each ring and the spacing between rings. The database in this study reflects an average of four cable bolts per ring and 2.4 m between rings. If double cables are treated as two separate bolts, then an average of slightly over five cable bolts per ring is reflected. It is suggested that the average values for ring spacing and the number of cable bolts per ring be used as preliminary design guidelines.

Nickson (1992) proposed that design bolt length for the point-anchor approach to cable bolt supports in hanging walls is related to distance between sublevels. The ratio of cable length to maximum unsupported span listed in table 15 can be used as a preliminary guideline for the determination of cable bolt length. These data indicate that cable length should exceed 25 pct of the maximum unsupported span.

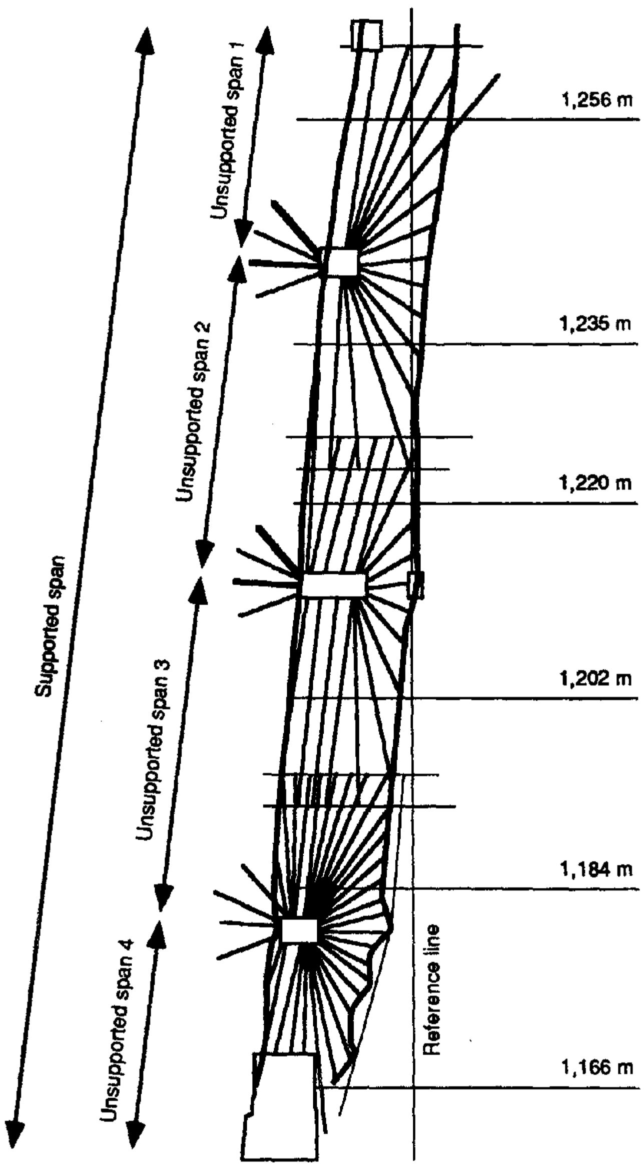

Figure 61.—Description of geometry for point-anchor approach to cable bolt support design.

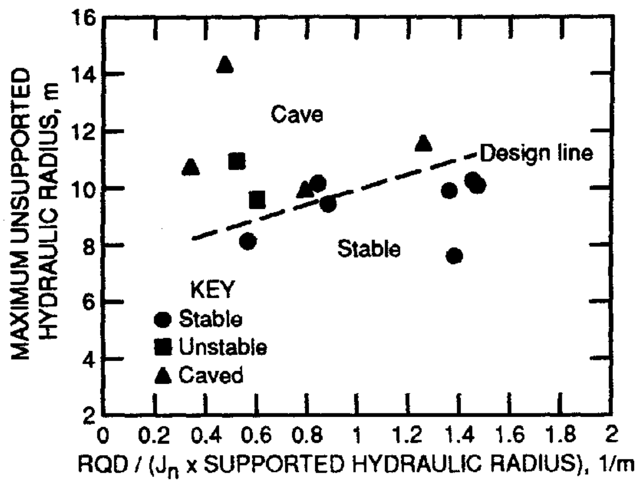

Figure 62.—Design chart for point-anchor support for hanging wall. Specifications: 0.40 to 0.45 water-cement ratio, 2.4-m ring spacing, five single cables per ring, plates recommended.

CONCLUSIONS

Cable bolt supports are an established ground control technique in the mining industry and offer a variety of approaches for reinforcement of rock masses. They can be installed as either passive or active supports; they can behave as stiff or yieldable reinforcement, depending on their configuration; they can be placed at any angle; and they contribute to safe, productive mining without the need to make changes in existing mining methods. Properly installed, these supports offer an effective ground control system for the mining industry.

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