CABLE SUPPORT GUIDELINES

FOR

UNDERGROUND HARD ROCK MINE OPERATIONS

by

SIMON DAVID NICKSON

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The University of British Columbia
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ABSTRACT

The objective of this thesis is to expand upon the existing database of cable support practice, and develop revised design criteria for the support of underground openings. A new empirical database of 46 supported case histories was assembled from a six month field study, involving visits to 13 mines in Western Canada, the United States, and Ireland. A comprehensive review of current cable support theory, practice and design is presented. A statistical methodology is introduced to define zones of stability from an empirically collected database. Existing guidelines for the design of underground support are reviewed and calibrated. Current design criteria for cable support are often based on an even distribution of bolts over the supported surface. The point anchor approach involves the use of a high concentration of cables, installed into large open stope hangingwalls from sublevel access drifts. This thesis proposes revised cable design guidelines for even distributions in open stope backs, and develops an approach for the point anchor design of hangingwall surfaces.

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To Michael Anthony Nickson

CHAPTER 1

INTRODUCTION

1.1 GENERAL

The use of cable bolts in underground mines has evolved from the need for a long, flexible, high capacity support system, that could be installed in advance of mining. The design of a cable bolt system is often a subjective process. The mining engineer can design a cable pattern based on dead weight, but lacks guidelines for cases where dead weight considerations may not be cost efficient. Potvin (1988) proposed a method for cable bolt design that was based on the empirical analysis of back support derived from sixty-six case histories of Canadian support practice. This thesis reviews the method proposed by Potvin (1988) and develops a revised design methodology for cable bolt support.

1.2 OBJECTIVE

The objective of this thesis is to expand upon the existing database of cable support practice and develop revised design criteria for the support of underground openings. Six months were spent in the field during the data gathering stage of this study, and visits were made to thirteen mines to review cable support practice. The majority of the mines were located in western Canada, but the application of cable support was also observed in the western United States and Ireland. A total of 13 unsupported and 46 supported case histories have been collected during this study. The terms "supported" and "unsupported" refer to the use of cable support. A supported case history incorporates cable bolts, while an unsupported case history does not.

An unsupported case history may however incorporate short pattern bolting that is installed to provide primary support during the development phase. Stope backs usually incorporate short pattern bolting based on operational or legal standards. Hangingwalls are occasionally not bolted during the development phase due to limited surface exposure. The design methodology proposed by Potvin (1988) was directed at the cable support of open stope backs. Cable bolts are also frequently used to support open stope hangingwalls, although limited design techniques are currently available. In this study, particular emphasis was placed on collecting a database of hangingwall cable support that would aid in the interpretation of related design criteria.

1.3 INTRODUCTION

The first half of this thesis reviews the current state of cable support theory, practice and design. The discussions are based on a literature review that has been complemented by observations from the field study phase of this project. A review of cable support practice is presented in Chapter 3, and has been designed as a guide for the development of operational procedures. Chapter 4 presents a review of current cable design methods, based on observed practice and techniques described in literature. The design methodology proposed by Potvin (1988) is reviewed in Chapter 5, and introduces the development of revised cable design criteria in the second section of this thesis. The database assembled in this study is presented and compared to the Potvin (1988) design proposals in Chapter 6. A statistical approach is used to develop relevant relationships for use in the design of cable support. Chapter 7 introduces revised design guidelines for back support and proposes a new technique for use in hangingwall cable design.

1.4 EMPIRICAL DESIGN

Empirical analysis provides a tool for the mining engineer to use in the design of underground openings and support. *Empirical* is defined as "Relying or based on practical experience without reference to scientific principles" (Webster's New World Dictionary of American English 1991). Scientific principles cannot be ignored in proper empirical analysis and ideally are used to complement results suggested through observation and experience. Empirical charts or graphs developed from observational data are frequently used for the design of underground openings and associated rock support. Several examples of such charts (Barton, Lien, and Lunde 1974; Mathews et al. 1981; Potvin 1988) will be reviewed later in this thesis, and can be a valuable asset to the mining engineer if used with proper care. It is recommended that the original published document be reviewed prior to the utilization of a particular design chart or graph. This permits a full understanding of the original objectives associated with the chart, and provides a means of assessing its applicability. For example, the *Design Chart for Cable Bolt Density* produced by Potvin (1988), relates cable bolt density to a relative block size factor. This chart was developed from a database that is made up of cable bolted backs, and was not intended for use in hangingwall design.

Charts can be custom built or modified to suit particular characteristics of one operation. Greer (1989) reviewed both the Mathews (1981) and Potvin (1988) empirical design approaches, to determine which displayed the best correlation with a database assembled from the Manitoba Division of Inco Ltd. Based on this data, the Mathews (1981) approach was suggested as the best guideline for hangingwall design, but back design was better suited to the Potvin (1988) methodology. Greer (1989) recommended that the database be expanded as mining progressed to allow for further calibration of design criteria. This is a good illustration of the modification of empirical design methods to reflect on site operating experience. Cullen (1991) discusses a site specific rock classification system developed for use at the H-W mine of Westmin Resources Ltd. The system involved a quantitative assessment of drill core quality, degree of schistosity, rock hardness and total gouge to produce a qualitative description of the rock mass that varied from *very poor* to *very good* ground. The classification was used as an indicator of ground conditions on geological sections, and related to planned excavations. Hoek and Brown (1980, 131) discuss the transition from intact rock material to a heavily jointed rock mass. Laboratory testing can be applied to determine the properties of intact rock but in terms of stope design, it is necessary to consider the rock mass. The empirical design approaches that are discussed in this thesis attempt to quantify rock mass conditions for practical use in the design of underground openings.

CHAPTER 2

CABLE SUPPORT PRINCIPLES

2.1 INTRODUCTION

Cable bolts as a means of rock support were first introduced to the mining industry in the 1960's. Gramoli (1975) describes the installation of discarded locked coil hoisting rope as a support system adopted by the Geco Mine in 1963. This is one of the earliest references to cable bolting on a large scale, but Garcia (1929) suggests that the installation of cable slings to support the back of a coal mine was a convenient way of utilizing scrap hoist rope. The initial application of cable support often involved the use of discarded hoist rope that was installed in a borehole, tensioned and subsequently grouted. Due to its flexibility, a long cable could be installed into a drill hole from a drift that had limited height. This was an advantage over traditional methods of support and quickly lead to widespread use of the cable bolt in the 1970's. Windsor (1992) describes the progression of cable bolting from the use of discarded hoist rope to pre-stressing wire and eventually to a pre-stressing strand. Hoist rope required extensive cleaning to remove grease and was abandoned in favour of a seven wire steel strand. The cable strand is made up of one central wire surrounded by six slightly smaller outside wires, and typically has a diameter of 15.2 to 15.9 mm (0.6" to .625"). The cable wires are made from high-strength steel with a modulus of elasticity of approximately 203.4 GPa (29.5 x 10^6 psi) and an ultimate strength of approximately 26.3 tonnes or 58,000 lbs (Goris 1990). There is a small variation in reported cable breaking strengths noted in literature due to different steel specifications and frequent conversion between metric and imperial units. In this thesis, the ultimate strength of a cable bolt is considered to be approximately 26.3 tonnes, but will be subsequently noted as 26 tonnes.

Cable breaking strength is not commonly attained in the laboratory and reference is often made to the pull out strength or load carrying capacity of the cable. These terms refer to loads at which the cable fails or pulls out of a grout column, and can be well below the steel breaking strength. Fuller (1983a) notes that steel failure was not common in a review of Australian supported open stopes. Gendron et al. (1992) indicate that cable bolts are typically left dangling from the back in failed cases observed at several Canadian mining operations. These comments suggest that the full use of steel breaking strength is not reflected in current cable bolt practice. The objective of cable bolt design should be to maximize the load carrying capacity of the cable so that it approaches the breaking strength. It is important however to recognize the distinction between breaking strength and pull out load in the design of cable support. This chapter discusses the factors involved in the determination of cable load carrying capacity.

2.2 INSTALLATION

A discussion of the common installation practices for cable bolts may be helpful as a preamble to the review of support principles. Three methods of installation are currently practiced as illustrated in Figure 2.1. Methods A and B refer to cable bolts installed in upholes while method C is applicable to downholes. For all methods, the cable is inserted into the hole prior to grouting. The breather tube method involves the use of a 9.5 mm (3/8") to 12.7 mm (1/2") diameter polyethylene tube that is installed to the toe of the hole. A 19.1 mm (3/4") diameter grout tube is inserted just beyond the hole collar, and grout is pumped from the collar to the toe. Trapped air is exhausted through the breather tube as grout is pumped into the hole. Grout coming out of the breather tube is an indication that the process is complete. The grout tube method does not require a breather tube to exhaust air, as grout is pumped from the toe to

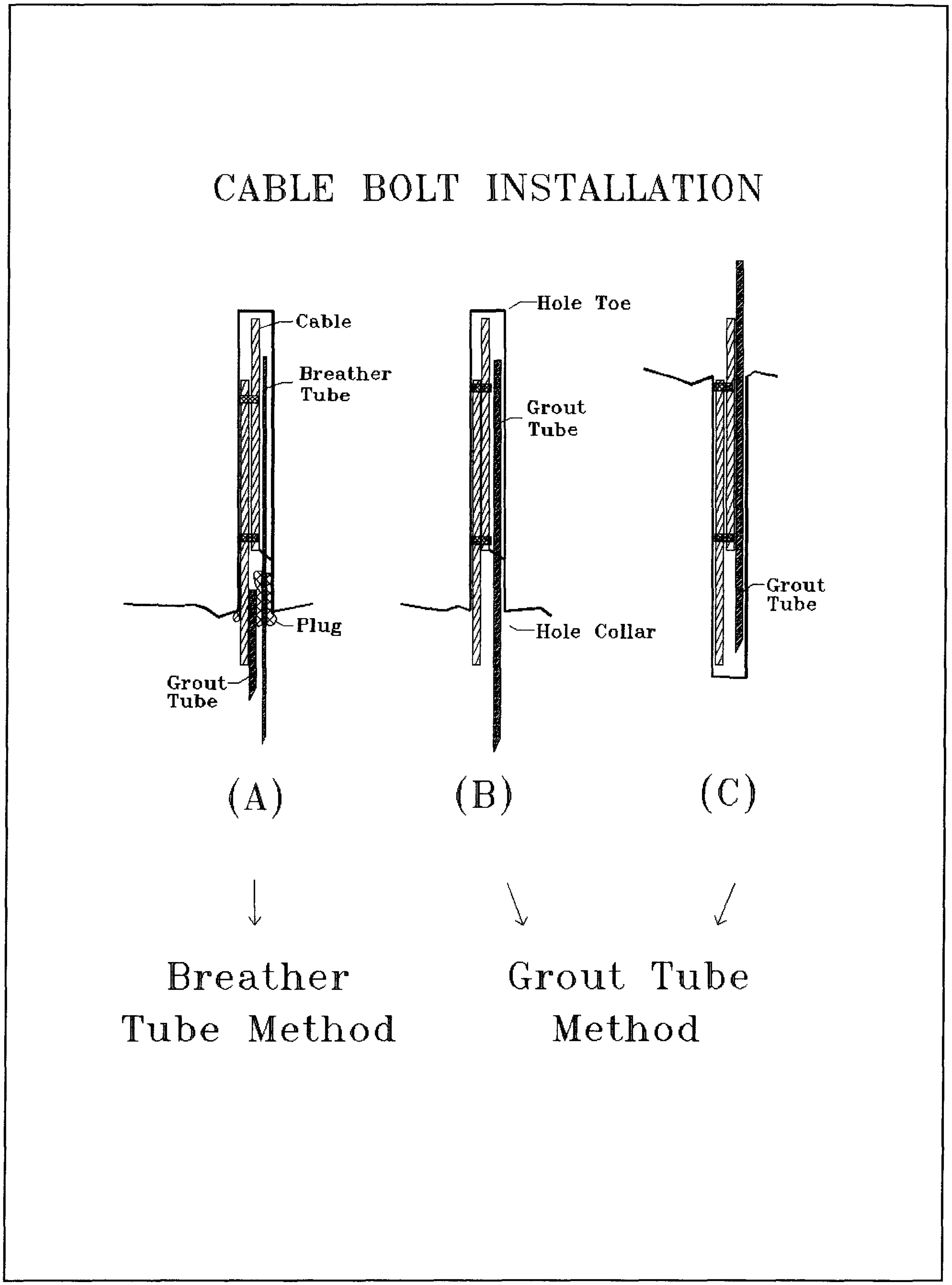


Figure 2.1: Cable bolt installation methods

the collar. Air is naturally exhausted through the drillhole column. This method can be used in upholes providing that the grout is thick enough to remain in the hole. The breather tube method requires a collar plug while the grout tube method does not. A more detailed discussion on installation methods along with some of the problems encountered will be covered in Chapter 3.

2.3 FAILURE MODES

Jeremic and Delaire (1983) identified four possible failure modes (Figure 2.2) that apply to cable bolt support. Mode A describes the failure of the grout-rock bond, which is not encountered often in practice due to the roughness of the rock surface and the large contact surface area. The failure of the grout-cable bond is shown in mode B, while modes C and D involve failure of the grout and rock respectively by internal friction. Rock failure is generally not a concern in hard rock mining practice but could be encountered in soft rock situations. Jeremic and Delaire (1983) suggest that the most frequent failure mode in practice involves failure of the grout, but in fact the most frequent mechanism is thought to be a combination of mode B and C. Failure of the steel cable is a fifth possible failure mode but is not commonly encountered in practice.

2.4 LABORATORY TESTING

2.4.1 Commonwealth Scientific & Industrial Research Organization Testing

Fuller and Cox (1975) completed pull tests on 7 mm stress relieved round wire grouted in a portland cement paste at a 0.45 water:cement ratio by weight. A typical load-displacement curve is shown in Figure 2.3. The peak load was reached with little displacement and was

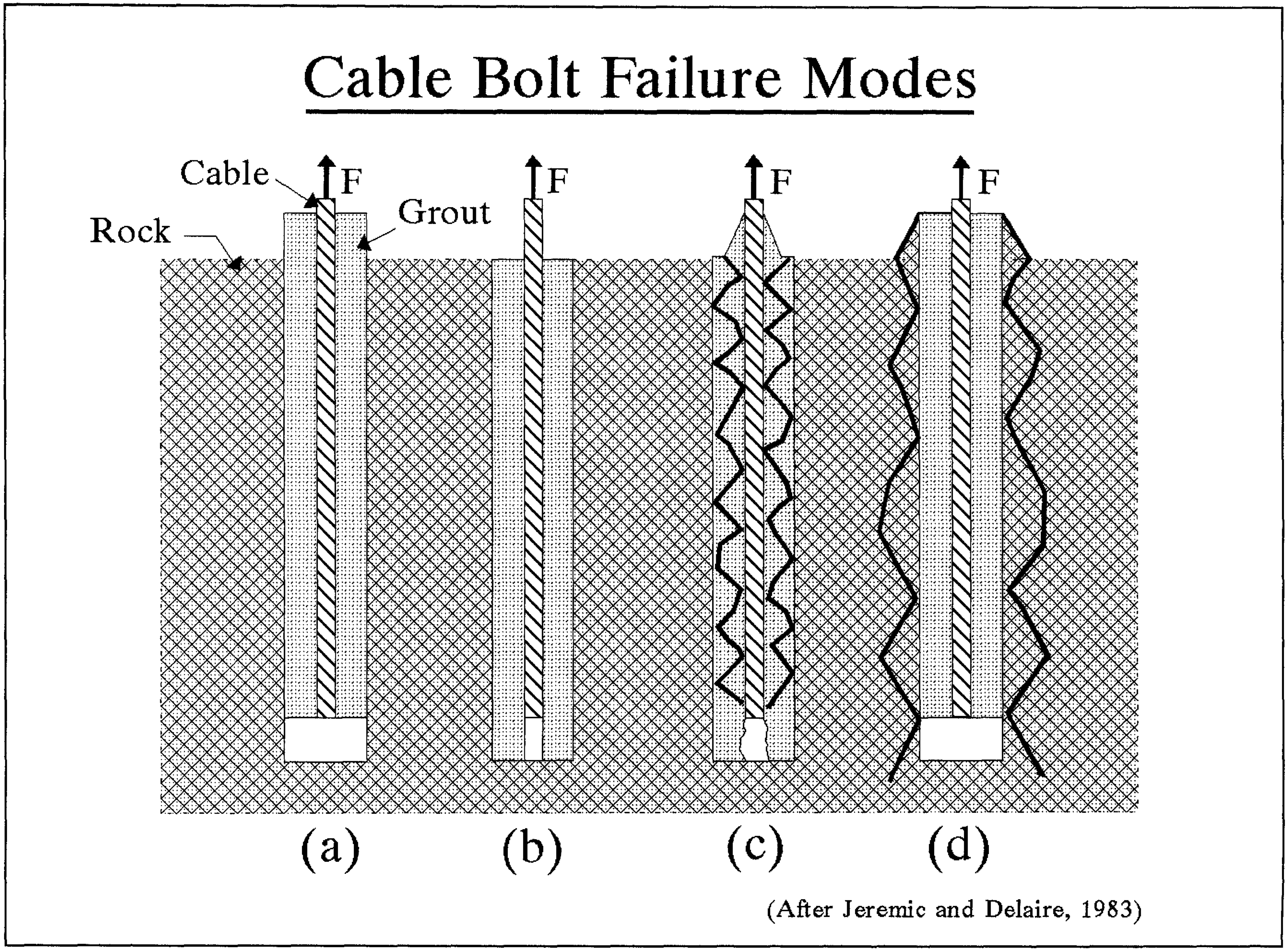


Figure 2.2: Cable bolt failure modes

followed with a gradually reducing residual load. The fluctuations in residual load were thought to be a result of small variations in wire diameter. The embedment length was varied from 100 to 700 mm during these tests and the peak load was found to increase linearly with increasing embedment length. Fuller and Cox (1975) proposed that the grout-steel bond initially failed progressively along the embedment length of the wire. This process occurs prior to the peak load being attained and results in limited wire displacement. The peak load occurs when the complete bond is broken along the embedment length, and frictional resistance continues to generate residual load as the wire moves out of the grout column. The frictional resistance is gradually overcome as the wire moves relative to the grout, and the residual load would be expected to decrease. A second series of tests were completed with rusted wires at an embedment length of 700 mm. The peak load at this embedment was found to be three times higher than smooth wire. It was proposed that increased frictional resistance due to the presence of rust was responsible for this increase in peak load. A third series of tests used indented wire at a 700 mm embedment length. The load-displacement curve for these tests was very similar to Figure 2.3, except that the peak load was approximately twice as high. The residual load fluctuations were still present and occurred at the same interval as the indentations on the wire. As displacement occurred, it was proposed that the indentations changed the failure mechanism and forced the wire to crush the grout. Rust on the indented wire surface was found to improve the peak strength approximately 30%. Pull tests were also conducted on 12 mm diameter 7 wire stress relieved strand, with an embedment length of 450 mm. A typical load-displacement curve for the strand pull testing is shown in Figure 2.4. The peak load attained was just over 11 times the load for a single smooth wire at a similar embedment. One marked difference in the strand load-displacement curve is the increase in post peak residual load. The grout-steel bond was expected to be higher for strand since the steel surface area was 2.3 times greater than a single 7 mm

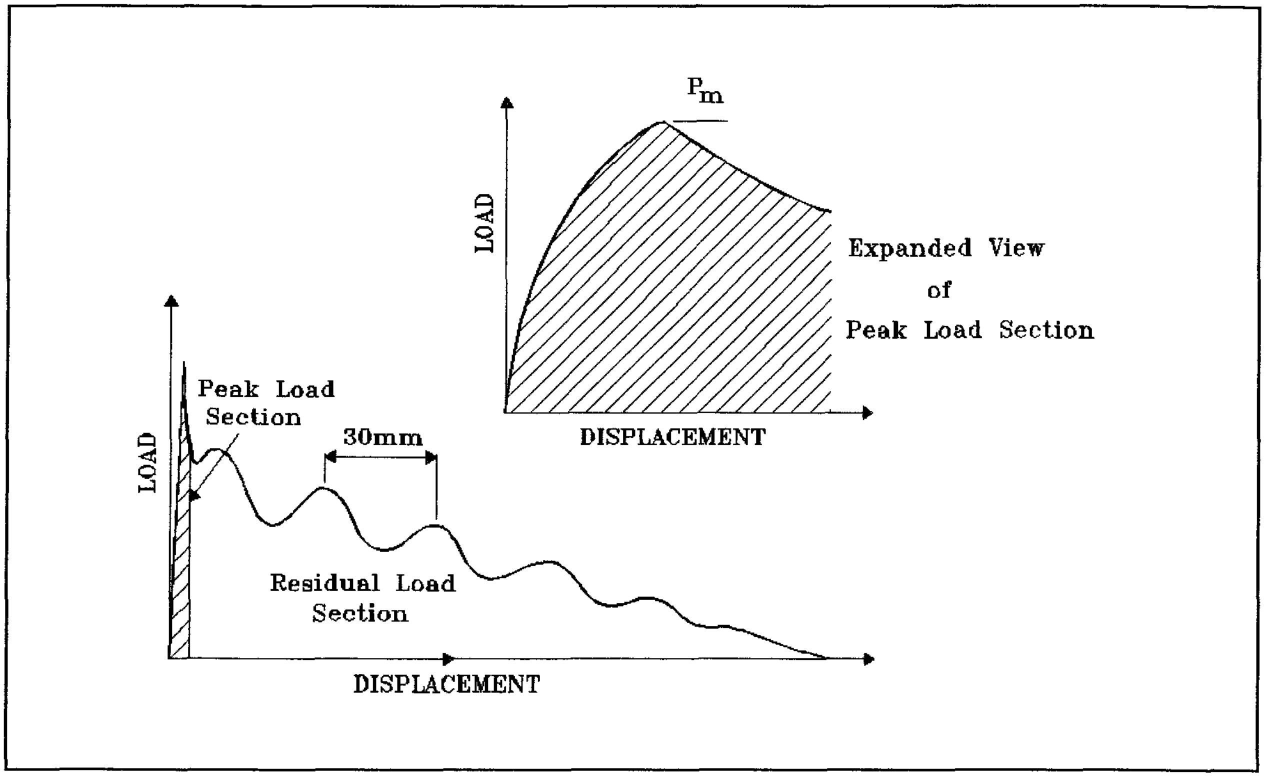


Figure 2.3: Typical single wire load-displacement curve (After Fuller and Cox 1975)

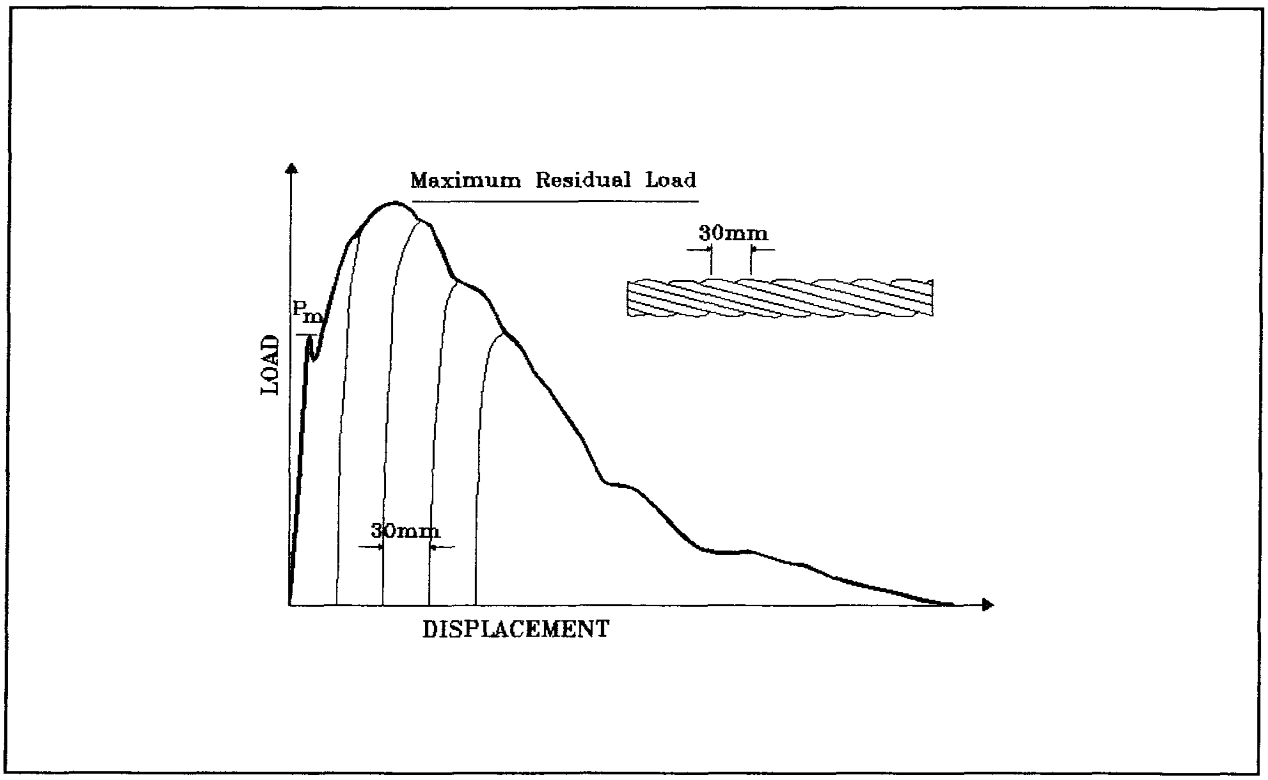


Figure 2.4: Typical strand load-displacement curve (After Fuller and Cox 1975)

wire. The additional increase in strength was thought to be caused by the individual wires having to fracture the grout in order to pull out. This failure mechanism takes advantage of the full grout compressive strength, which is higher than the grout-steel bond strength. Fuller and Cox (1975) proposed that grout fracturing caused the additional increase in peak load, and increased frictional resistance resulted in higher post peak residual loads. This testing illustrated that load transfer between steel and grout is dependent on the geometry and condition of the steel.

2.4.2 United States Bureau of Mines Testing

The United States Bureau of Mines conducted a series of laboratory studies on the support properties of cable bolts (Goris 1990, 1991). Thirteen series of pull tests were completed and the results are itemized in Table 2.1. The cable to be tested was grouted into two segments of 66.5 mm diameter steel pipe that were separated by a rubber washer, as illustrated in Figure 2.5. The portion of the cable installed within the 305 mm pipe was the length that was actually being tested. The "fixed" end of the cable was anchored in a 508 mm length of pipe with a barrel and wedge anchor. The sample was pulled apart by a hydraulic test machine and the cable would fail in the shorter section of pipe. The rate of displacement was set at 0.6 inches/minute until approximately 6" of total displacement was reached. These tests simulate a rock slipping off the end of a cable, as shown in Figure 2.5. Series 1 tests were set as the standard, and consisted of a single 15.9 mm (5/8") cable strand grouted in a 0.45 water:cement ratio by weight, with no breather tube or additives, and an average embedment length of 287 mm (11.3"). As shown in Table 2.1, the maximum load for the Series 1 test was 9.0 tonnes, and all other tests are compared as a percentage of this result. The average grout strength was also recorded for each test and is shown to be quite consistent, except where the water:cement ratio was varied.

A typical load displacement curve for the standard test is shown in Figure 2.6. It is very

Table 2.1: Summary of 28-day USBM Test Results

TEST SERIES	VARIABLE	MAX. LOAD (lb)	MAX. LOAD (tonnes)	PERCENT OF STANDARD	GROUT STRENGTH (psi)	GROUT STRENGTH (MPA)
1	Standard (0.45 w:c/11.3" Embedment)	19820	9.0	100	6940	47.8
2A	8" Embedment	14700	6.7	74	7291	50.3
2A	10" Embedment	18960	8.6	96	7291	50.3
2A	12" Embedment	19300	8.8	97	7291	50.3
2A	14" Embedment	21600	9.8	109	7291	50.3
2A	16" Embedment	23120	10.5	117	7291	50.3
2A	18" Embedment	25360	11.5	128	7291	50.3
2A	20" Embedment	28840	13.1	146	7291	50.3
2B	22" Embedment	31020	14.1	157	7175	49.5
2B	24" Embedment	36320	16.5	183	7175	49.5
2B	26" Embedment	37920	17.2	191	7175	49.5
2B	28" Embedment	41360	18.8	209	7175	49.5
2B	30" Embedment	43040	19.5	217	7175	49.5
3A	1/4" Breather (full)	19650	8.9	99	7109	49.0
3A	3/8" Breather (full)	18982	8.6	96	7109	49.0
3A	1/2" Breather (full)	19200	8.7	97	7109	49.0
3B	1/4" Breather (full)	19888	9.0	100	7265	50.1
3B	3/8" Breather (full)	20050	9.1	101	7265	50.1
3B	1/2" Breather (full)	19934	9.0	101	7265	50.1
3C	1/2" Breather (MT)	17660	8.0	89	7258	50.0
4	0.30 W:C Ratio	36820	16.7	186	9844	67.9
4	0.35 W:C Ratio	32080	14.6	162	8175	56.4
4	0.40 W:C Ratio	26100	11.8	132	7580	52.3
4	0.45 W:C Ratio	19820	9.0	100	6940	47.8
5	Cured 127 degrees F.	22900	10.4	116	N/A	N/A
6A	1.4:1 Sand/Cement Grout	27920	12.7	141	7600	52.4
6B	6A & 0.25lb additive/100lb cement	27592	12.5	139	7490	51.6
6C	6A & 0.45lb additive/100lb cement	27195	12.3	137	7605	52.4
7	Two Cables – 9.36" Emb.	41080	18.6	207	6748	46.5
8	Two Cables & 1/2" tube – 9.33" Emb.	43058	19.5	217	7103	49.0
9A	Steel Button @ 2"	26500	12.0	134	7375	50.8
9B	Steel Button @ 4"	55840	25.3	282	7337	50.6
9C	Steel Button @ 6"	53950	24.5	272	7470	51.5
10A	Birdcage(A-Node @ Pipe) – 10" Emb.	33960	15.4	171	7600	52.4
10B	Birdcage(Node @ Pipe) – 10" Emb.	25760	11.7	130	7357	50.7
11A	Two Birdcage(AN @ Pipe) – 10" Emb.	77300	35.1	390	7775	53.6
11B	Two Birdcage(N @ Pipe) – 10" Emb.	79750	36.2	402	7250	50.0
12	Epoxy Coated Cable	27875	12.6	141	7094	48.9
13	Two Epoxy Coated Cables – 10" Emb.	57550	26.1	290	7061	48.7

Reference: (Goris 1990, 1991)

Standard: Single 5/8" diameter cable with 0.45 w:c ratio and 11.3" embedment length.

Bleeding:

W:C Ratio	Bleed (in/ft)
0.45	1.15
0.40	0.76
0.35	0.39
0.30	0.19

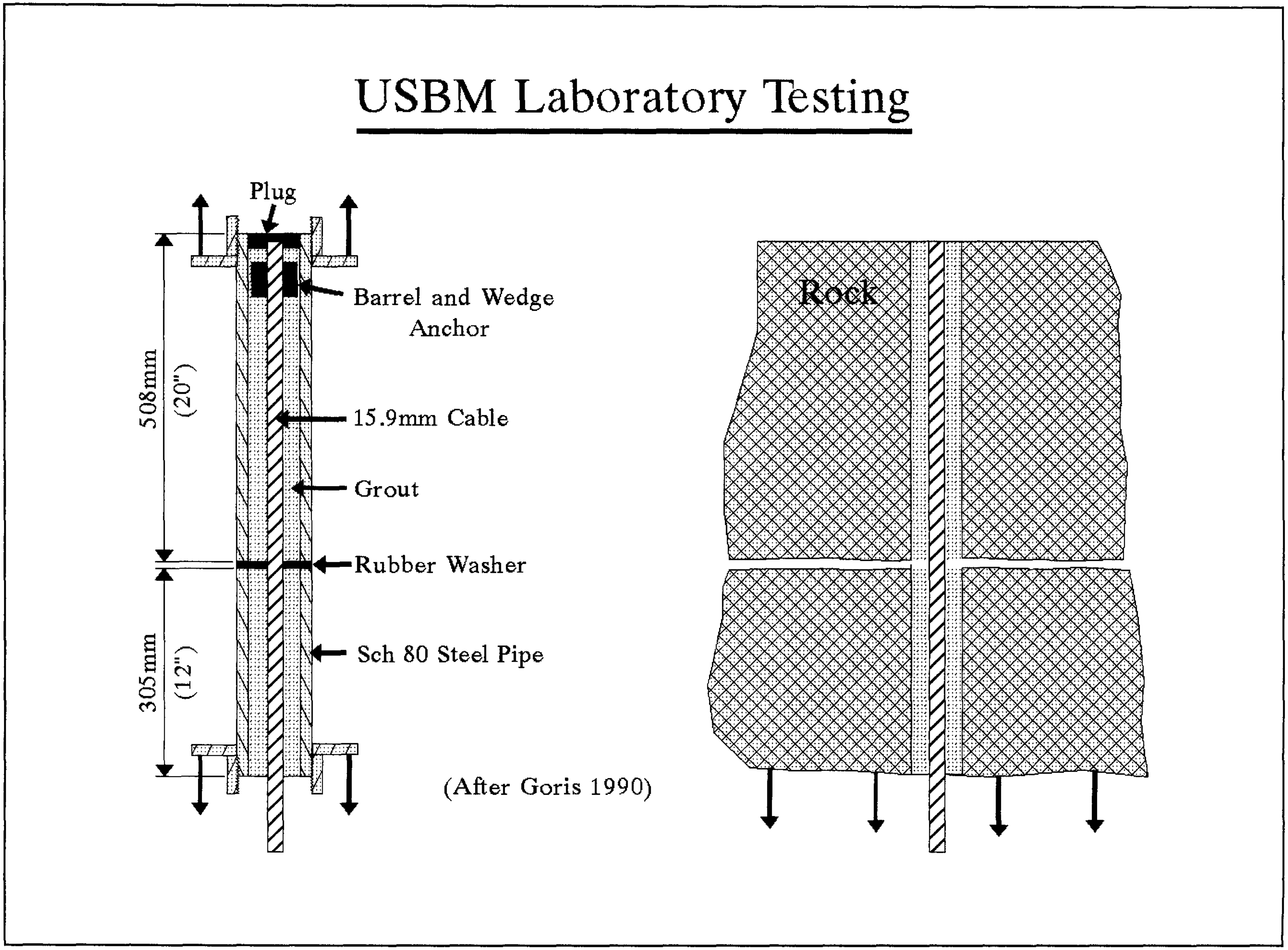


Figure 2.5: USBM pull test apparatus (After Goris 1990)

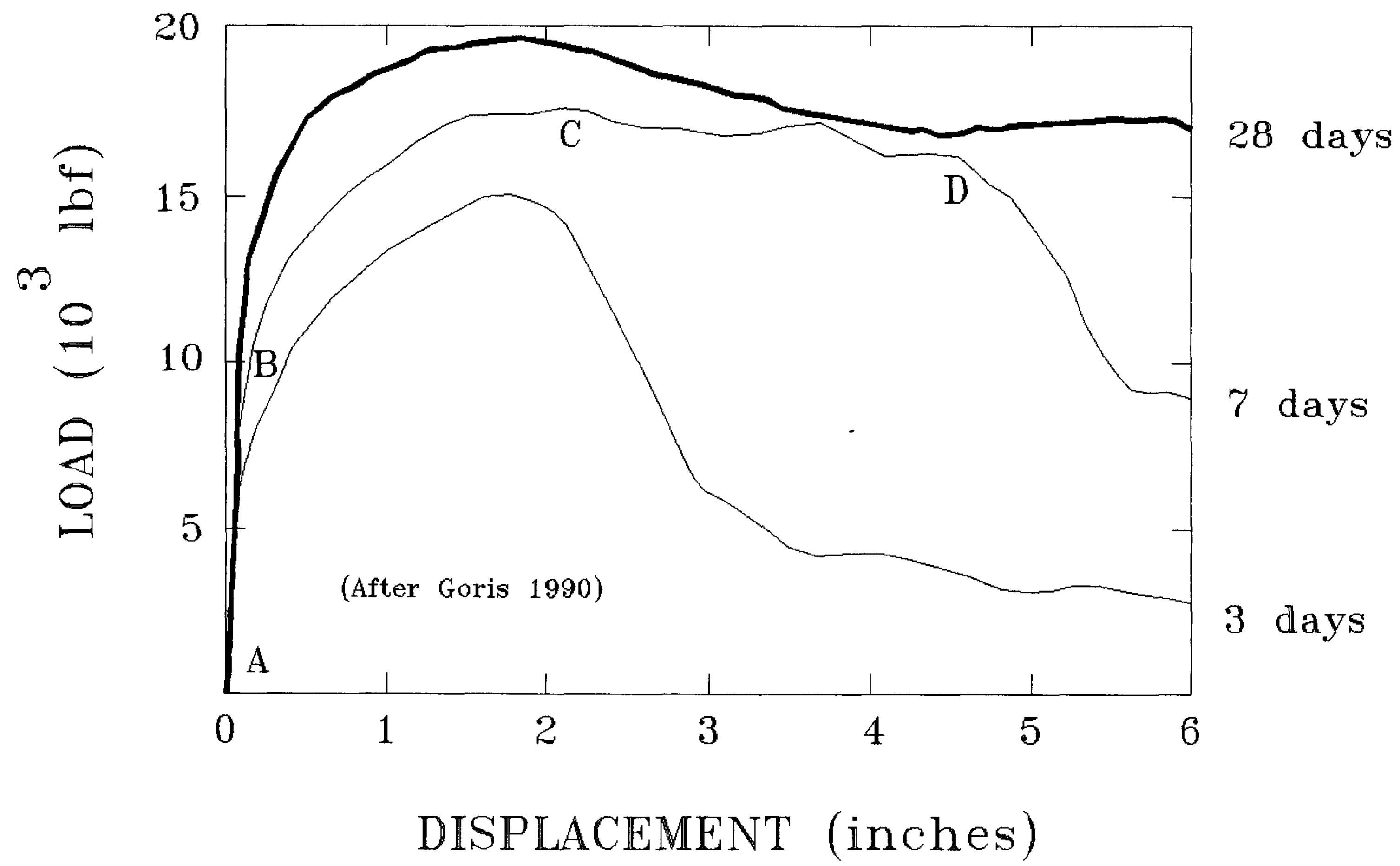


Figure 2.6: Averaged 7, 14, and 28 day load-displacement curves for test series 1 (After Goris 1990)

similar to the curve produced by Fuller and Cox (1975) for a single strand cable (Figure 2.4). The points on the seven day test curve represent the actions of the cable as it is pulled out of the grout. Between points A and B, the bond is broken between the steel and the grout, starting at the pipe joint and propagating to the other end. Between B and C the cable is free to move, but must break the ridges of grout that are between the individual wires of the strand. Broken particles from these ridges provide increased resistance to movement and result in a continual increase in load until a maximum is reached at point C. Displacement continues between C and D, but dilation, or expansion of grout particles, maintains a high residual strength. The effect of pulling on a cable, as illustrated in Figure 2.7, is to generate a normal force into the grout column as a result of grout particle dilation. Assuming that the rock stiffness is sufficient to counteract this force, the frictional resistance will remain high, until the strand wires are able to shear through the grout ridges.

Series 2 tests varied the embedment length from 203 mm (8") to 762 mm (30"). Goris (1991) found that the load was linearly related to the embedment length as shown in Figure 2.8. An extrapolation of these results indicates that 1.06 m (41.9") of embedment is required to mobilize the full 26 tonne breaking strength of the cable. The load-displacement curve for each embedment length is illustrated in Figure 2.9 and clearly shows the limited displacement as the cable-grout bond is progressively broken. The importance of embedment length to cable design is illustrated in Figure 2.10. A 20 tonne block of rock is supported by a single cable that is grouted with a water:cement ratio of 0.45. Based on the USBM test results, the cable will support the block with an embedment length of 1 m. However, if the block is rotated 90°, the embedment length changes to 0.5m and the block can potentially slide off the cable. The jointing within a rock mass or the geometry of a particular block are thus important considerations in the design of cable support.

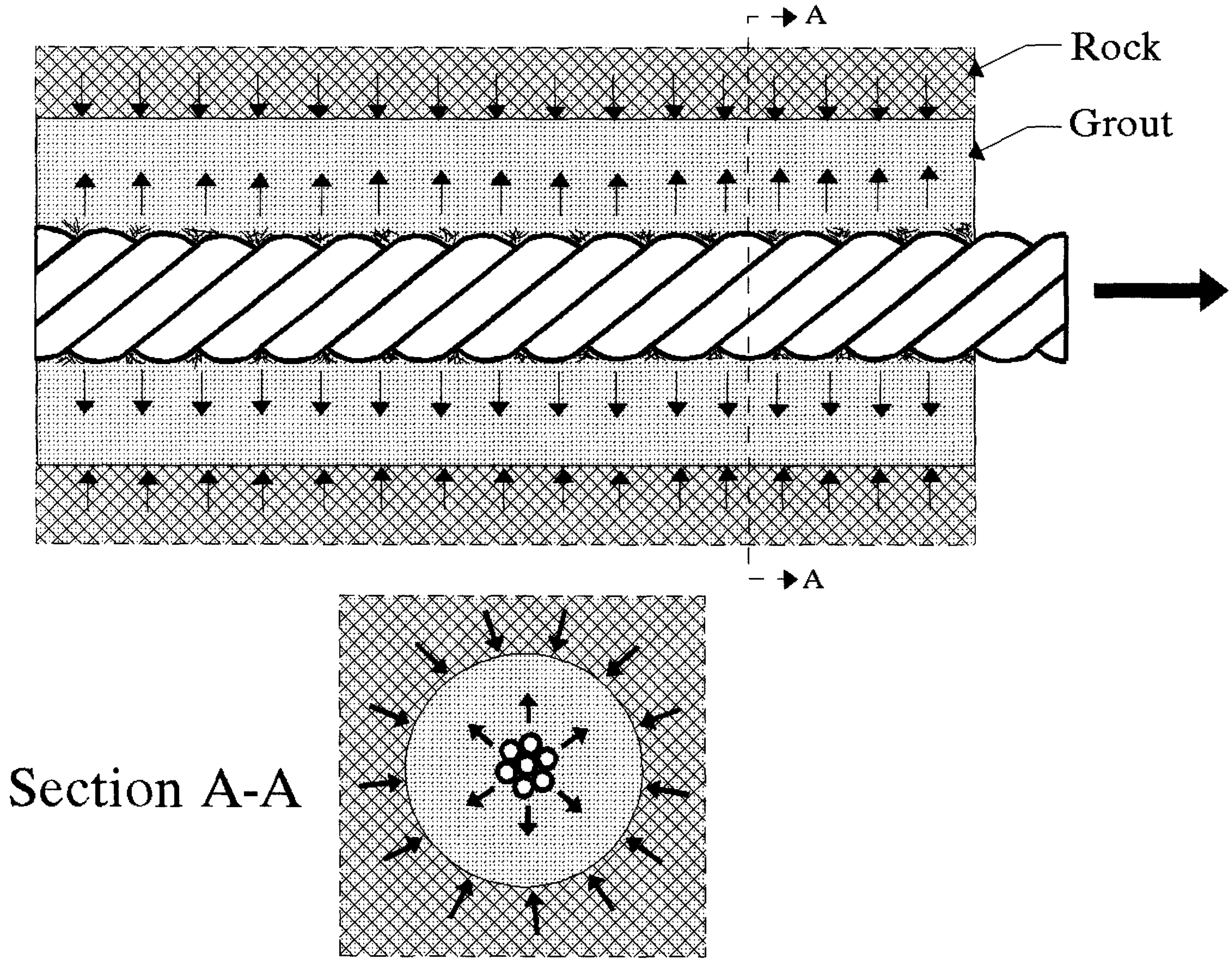


Figure 2.7: The effect of pulling on a grouted cable (After Noranda Technology Center 1990)

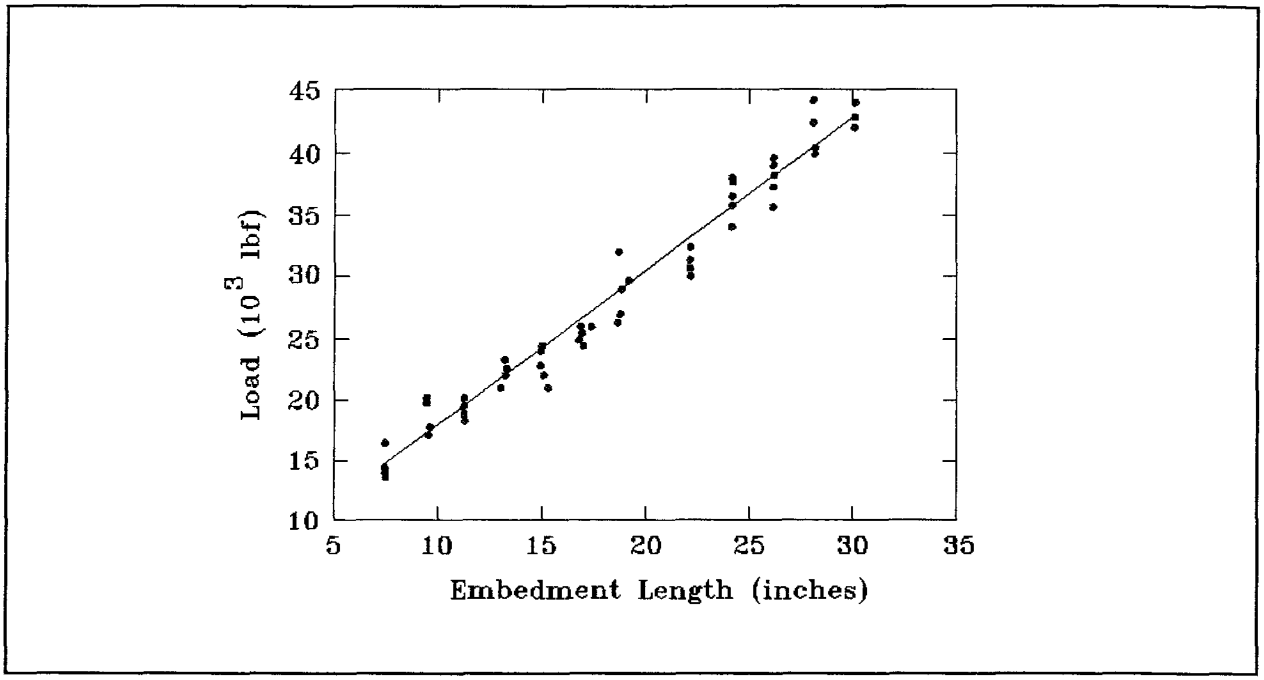


Figure 2.8: Maximum load versus embedment length (After Goris 1990)

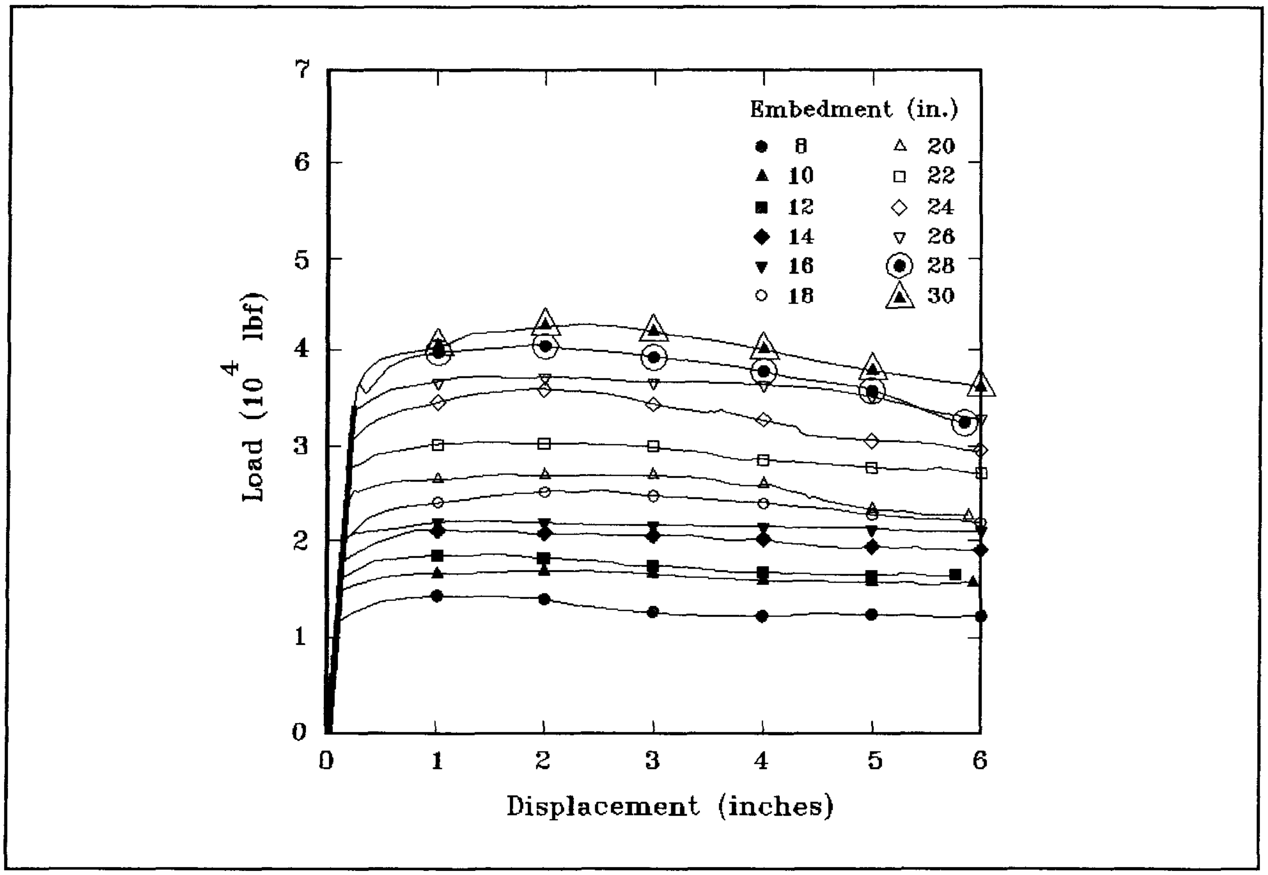


Figure 2.9: Averaged 28-day load-displacement curves (After Goris 1990)

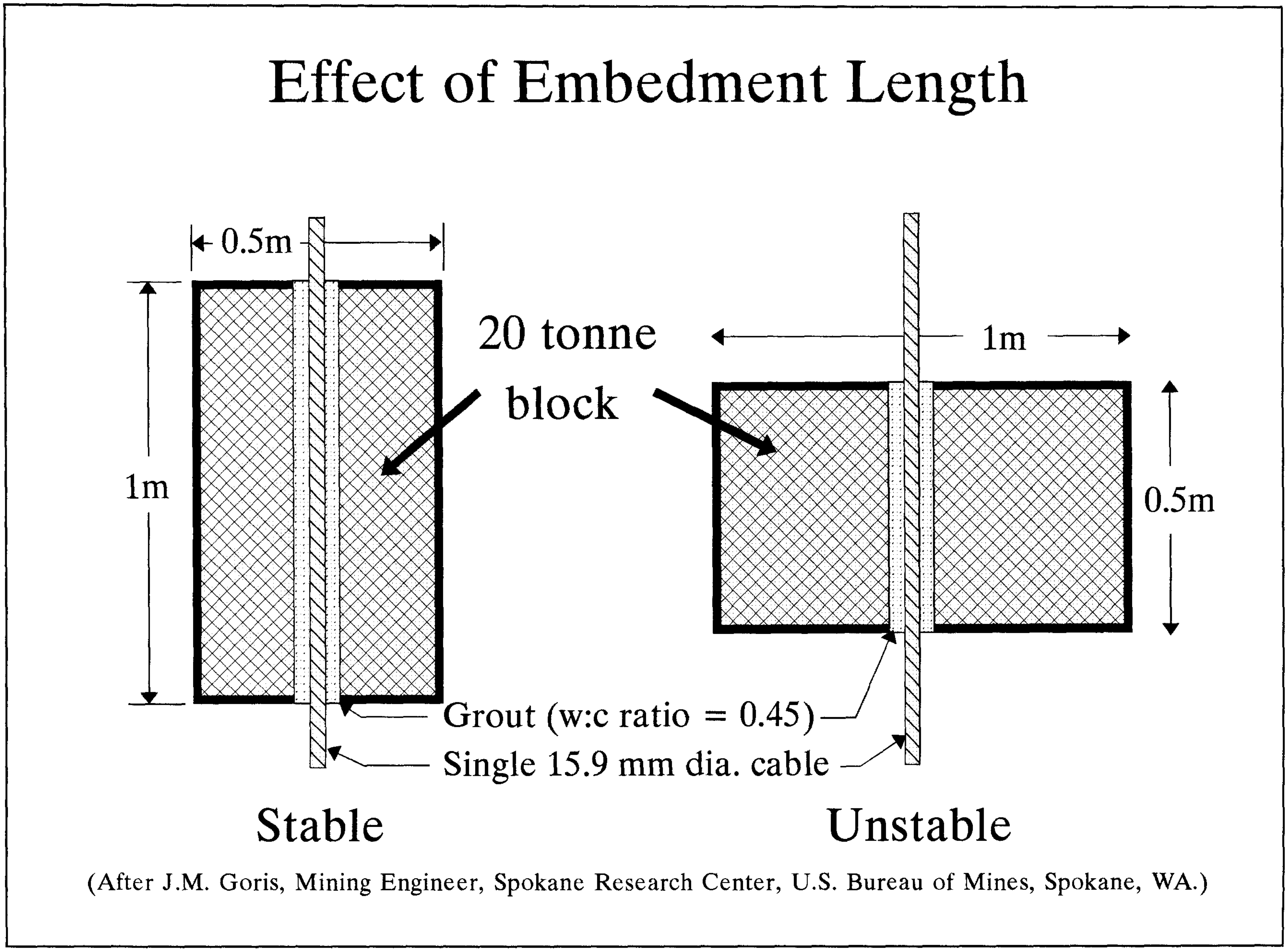


Figure 2.10: The effect of embedment length

Since a breather tube is commonly left in the hole after grouting, the effect of different sized tubes were analyzed in the third test series. The tube size was varied from 6.4 mm (1/4") to 12.7 mm (1/2") in diameter. The results revealed that the presence of a breather tube in the grout column had no effect on the cable strength, as long as the tube was filled with grout. When the breather tube was empty, the maximum load dropped slightly but the residual load carrying capacity was significantly reduced. As the cable was displaced, the grout was able to move into the void created by the empty breather tube, and frictional resistance was reduced.

The water:cement ratio was varied from 0.30 to 0.45 in the next series of tests. The maximum load increased as the water:cement ratio decreased. This indicates that an increase in grout strength as a result of reducing water:cement ratio, will increase the load carrying capacity of a cable bolt. A decrease in water:cement ratio from 0.45 to 0.3 approximately doubles the pull out strength of the cable. Cement particles in a grout column will settle immediately after placement in a process that is referred to as bleeding. Water rises to the surface as the cement particles settle. It was found that a grout column with a water:cement ratio of 0.45 would result in 96 mm/m (1.15"/ft) of grout bleed. The amount of bleed decreased as the water:cement ratio was reduced, as shown at the bottom of Table 2.1. This is important in terms of cable design since a vertical uphole that is 18 m in length, would have 1.7 m of water at the toe of the hole, if the water:cement ratio was 0.45.

Series 5 samples were cured at 127° F to determine the effect of high rock temperatures on a cable bolt installation. The higher temperature resulted in faster curing of the cement and high early strength. Adequate water was thought to be available to complete the hydration process since a grouted cable bolt is well contained, and there is little chance of evaporation. In series six, a 1.4:1 mixture of sand and cement was found to increase the maximum load by 41%. The sand particles are thought to provide an interlocking force and therefore greater frictional resistance as the cable displaces. Goris (1990) noted handling and flow problems with the sand:cement grout that would limit its underground application at this stage.

The remaining tests varied the geometry of the cable and resulted in significant increases in cable loads. The embedment lengths in some of these tests were reduced, since there was some concern about the capacity of the testing equipment. The load-displacement curves for double cables (Figure 2.11) reveal nearly twice the single cable maximum load. Higher maximum loads are reached with very little displacement, indicating that double cables offer a stiffer support system than single cables. Steel buttons were found to greatly increase the load carrying capacity of a cable, providing that the button is located greater than 102 mm (4") from the pipe joint. The failure mechanism changes from Figure 2.7, since the grout is forced to fail in compression in order for the cable to displace. When the button was located 51 mm (2") from the pipe joint, the resulting short grout column was easily fractured and the maximum load was significantly reduced. Laboratory testing indicates that button location with respect to rock mass jointing will control the load carrying capacity of the cable. Due to the difficulty of predicting this relationship, buttons are not commonly encountered in practice. They do however have the potential of significantly increasing the cable load carrying capacity, and allow large displacements at high loads.

Birdcage cable refers to an ordinary 7 wire strand that has been untwisted. The result is an open wire cable with a series of nodes and antinodes at 178 mm (7") intervals. Single birdcage cable was tested in Series 10 and indicated a 30% to 70% increase in load over ordinary strand. The open wires of a birdcage cable increase the steel surface area exposed to grout and result in a higher bond strength. In addition, grout is able to penetrate inside the birdcage configuration and each node acts as an anchor. As the cable displaces, the antinode puts the grout in compression and results in high loads but offers a much stiffer system than conventional

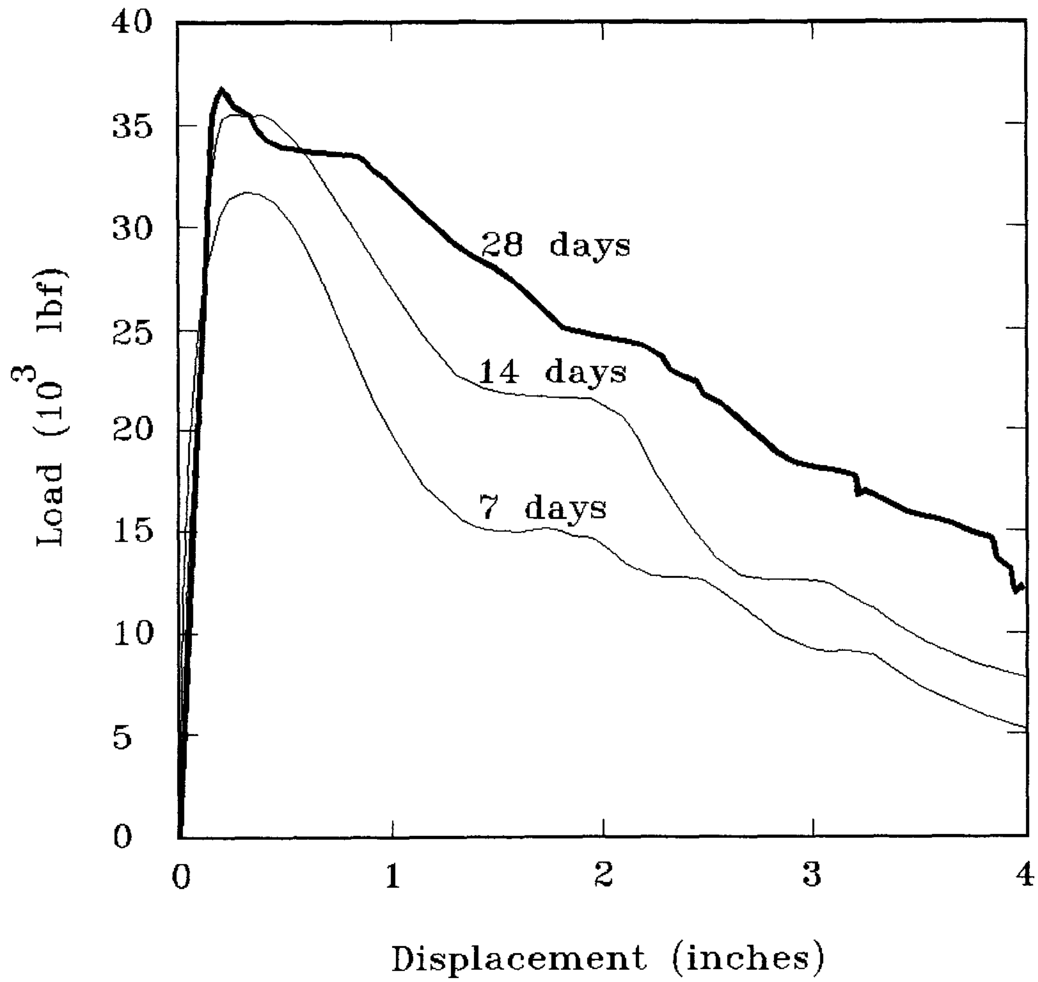


Figure 2.11: Load-averaged displacement curves for double cables (After Goris 1990)

strand.

The final test series looked at single and double epoxy coated cables with silica grit embedded in the epoxy. The silica grit provided increased frictional resistance and mobilized a 40% increase in pull out strength for a single cable. Epoxy coated cable was also found to significantly reduce the amount of grout bleed. The configuration of a regular steel strand allows water to enter into the wire arrangement and subsequently move to the top of the hole through internal voids. An epoxy covering does not allow water transmission as the strand arrangement is isolated from the grout mixture.

2.5 CRITICAL BOND LENGTH

The critical bond length refers to the length of grouted cable that is required to mobilize the full breaking strength of steel. An extrapolation of the USBM testing indicates that the critical bond length for a single 0.625 mm cable grouted in a water:cement ratio of 0.45, is just over 1 m. Observation of underground cable bolt systems frequently reveal that many failures leave the cable intact, signifying that the full strength of the bolt is not fully mobilized prior to failure. Much of the research into cable support has been directed at reducing the critical bond length to increase the potential of utilizing the full cable breaking strength. The research described in Section 2.4 illustrates several ways of decreasing the critical bond length. These involve:

1) increasing grout strength and stiffness

2) improving the cable geometry

3) increasing frictional resistance by altering the cable bolt surface.

Other factors that will increase the critical bond length include reduced grout stiffness due to empty breather tubes or voids and reduced confinement offered by low rock stiffness.

2.6 GROUT

Reichert, Bawden, and Hyett (1992) found that grout unconfined compressive strength increased with decreasing water:cement ratio. Grout strengths were tested between a water:cement ratio of 0.28 to 0.60 and the results are shown in Figure 2.12. It was noted that the variability in strengths is much larger at lower water:cement ratios. Reichert et al. (1992) suggest that a water:cement ratio of 0.3 does not have adequate flow characteristics for use in cable bolt systems, and propose the use of 0.35 to 0.40 as a practical compromise. Goris (1991) also noted that a 0.3 water:cement ratio grout would not flow without the addition of a water reducing agent. Gendron et al. (1992) found that laboratory grout strengths also varied as a function of the mixing method, as shown in Figure 2.13. Hand mixing of grout produced the lowest strengths, while a drum mixer and Mix and Inject (MAI) system produced progressively higher strengths. The Mix and Inject system (Reichert, Bawden, and Hyett 1992) utilizes a rotor-stator pumping assembly with a continuous mixing process. Underground grout samples were found to plot towards the lower range of the laboratory curves.

2.7 CABLE GEOMETRY

The geometry of a cable support system can be improved by increasing the exposed surface area and improving the grout-steel bond. Birdcage and double cables both significantly increase the exposed surface area. The surface condition of a cable can be altered by the use of an epoxy coating with embedded silica grit to improve the frictional resistance. The use of buttons and birdcage change the failure mechanism to take advantage of the full grout compressive strength. Noranda has conducted research on a cable grip (Gendron et al. 1992)

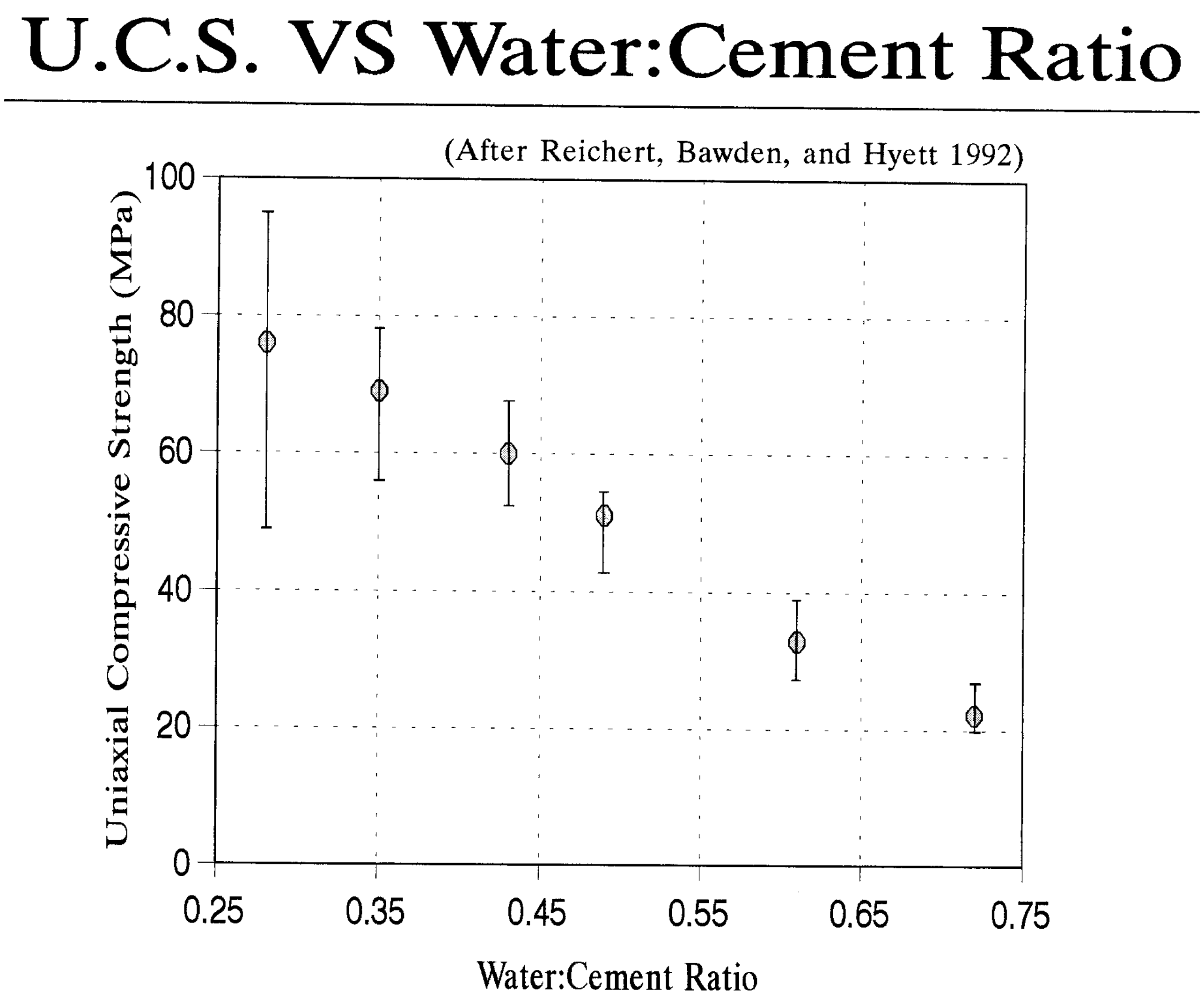


Figure 2.12: Grout unconfined compressive strength versus water:cement ratio (After Reichert, Bawden, and Hyett 1992)

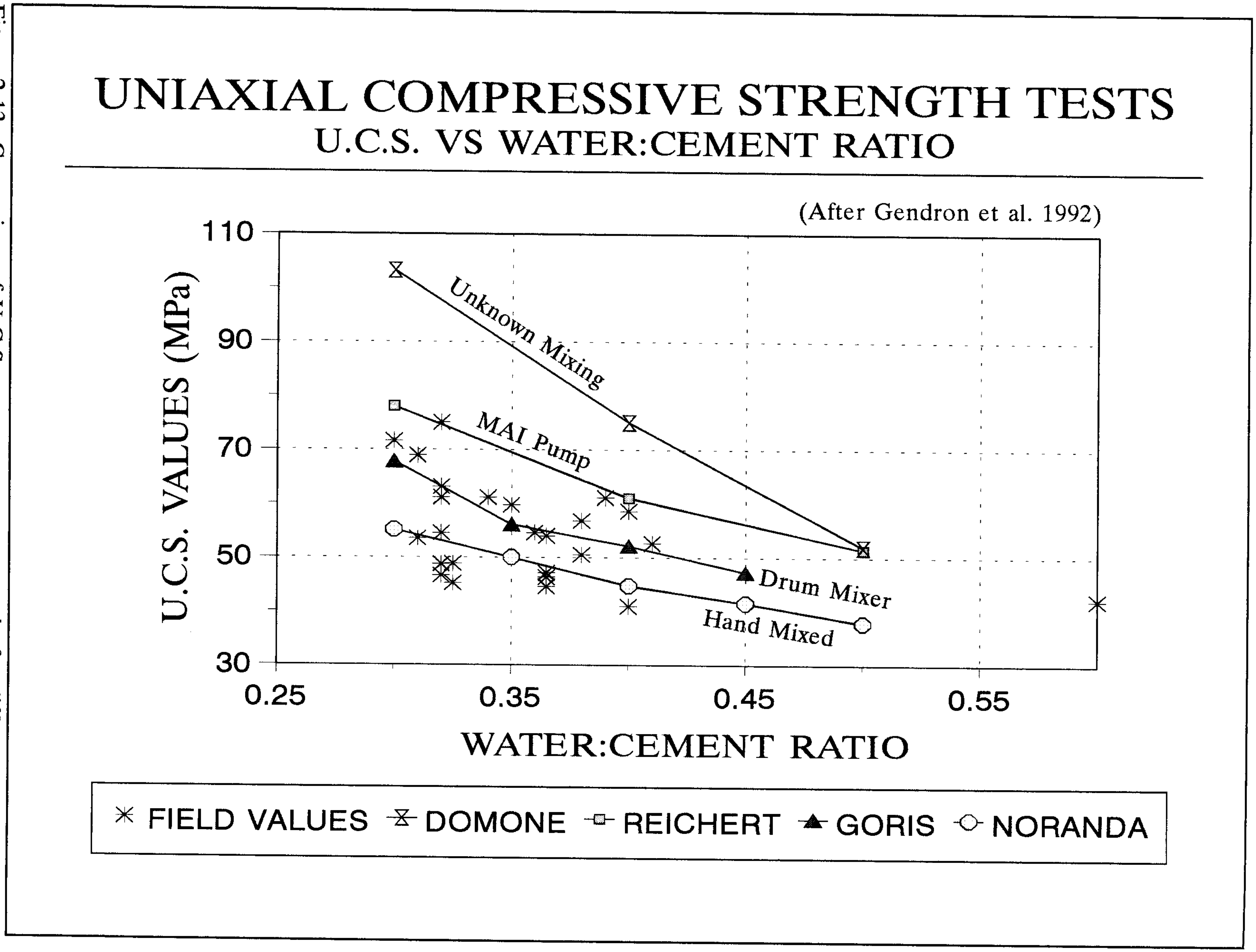


Figure 2.13: Comparison of U.C.S. versus water:cement ratio for different mixing methods (After Gendron et al. 1992)

made from the wedge portion of standard barrel and wedge anchors. The wedge is placed on the cable and covered with a plastic sleeve prior to grouting. Grout surrounding the sleeve functions as a cement version of the barrel, and test results have shown similar strengths to birdcage cable with reduced stiffness. The nutcase cable bolt is a recent cable configuration (Bawden, Hyett, and Cortolezzis 1992) that has been developed, and is currently being evaluated. The nutcase geometry involves placing a hexagonal nut over the central wire of a cable strand and rewinding the six outside wires so that each wire rests on one edge of the nut. The result is a more tightly wound version of the birdcage cable bolt. Preliminary testing (Bawden, Hyett, and Cortolezzis 1992) has indicated slightly higher strengths than the birdcage geometry and further study is planned to evaluate the ability of low water:cement ratios to penetrate the tighter geometry.

2.8 CONFINEMENT

The failure mechanism proposed by Goris (1990) generates a force normal to the cable due to grout dilation as a result of strand displacement. Reichert, Bawden, and Hyett (1992) have demonstrated that the stiffness of the material surrounding the grout will affect the cable strength. A series of laboratory and field pull tests were conducted to relate different levels of confinement to the load carrying capacity of a cable bolt. The concept of radial stiffness is introduced to compare the confinement offered by different pipe materials and rock types. Radial stiffness is described (Reichert, Bawden and Hyett 1992) as the amount of pressure that is required to induce a specified internal radial deformation, expressed in units of MPa/mm. The radial stiffness of pipes was calculated based on thick wall cylinder theory and a borehole dilatometer was used to evaluate radial stiffness for different rock types. The borehole dilatometer is a high pressure inflated packer that records the deformability of 76 mm diameter holes. The relationship between cable load carrying capacity and embedment length for steel, granite and shale is shown in Figure 2.14. Steel pipe exhibited the highest radial stiffness, followed by granite and then shale. The results illustrated in Figure 2.14 suggest that a stiffer rock mass improves the load carrying capacity of a cable bolt by providing increased confinement to dilation of the grout column. This in turn results in an increase in the level of frictional resistance to cable movement.

2.9 STRESS

Stress change is to be expected within an active mining area and has been considered in cable design by Kaiser, Maloney, and Yazici (1991). The previous discussion on confinement has indicated that the resistance of a rock mass to grout dilation can have an effect on the cable bolt strength. Kaiser et al. (1991) have proposed that a change in the field stress will result in a corresponding change in the pressure at the grout-steel interface. An increase in stress perpendicular to a cable bolt hole, as might occur in the back of a cut and fill stope, would increase the cable bolt strength by increasing the confinement around the steel. A stress decrease would similarly decrease the cable bolt strength. Research in this area is currently limited but future work is planned to study the implications of this concept.

2.10 CABLE ORIENTATION

The preceding discussion has concentrated on axial loading of cable bolts, as limited research has been conducted on the effects of shearing on cable bolt behaviour. Miller (1984) has suggested that cables are most efficient when oriented parallel to the shear direction and inclined between 17° and 27° from the joint. Fuller (1983a) suggested that this angle should

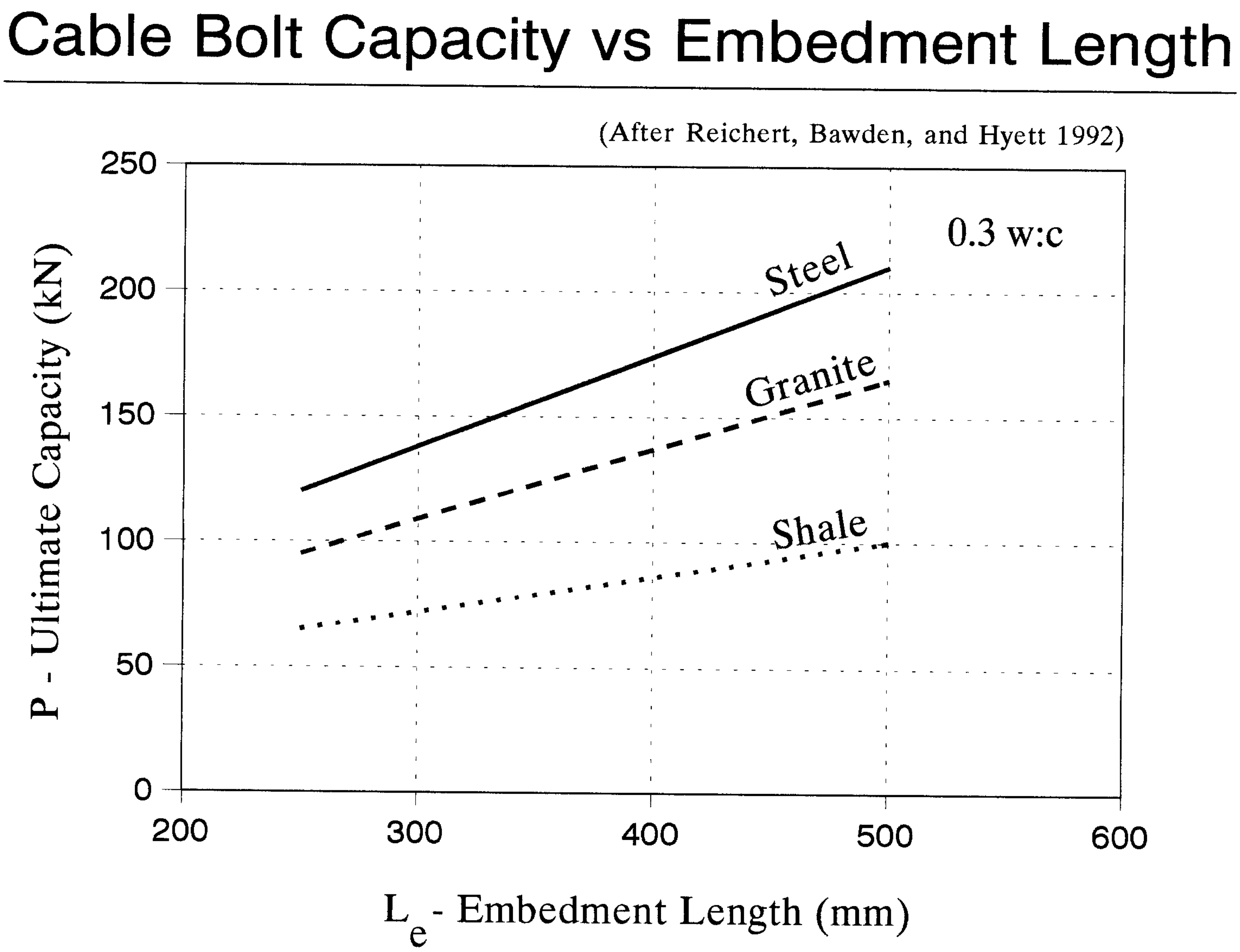


Figure 2.14: Cable bolt capacity vs embedment length at 0.3 w:c (After Reichert, Bawden, and Hyett 1992)

range between 15° and 30° for maximum efficiency. In situations where the restriction of joint opening is required, the most efficient orientation is at 90° to the joint.

2.11 SUPPORT STIFFNESS

Potvin, Hudyma, and Miller (1989) suggest that it is desirable in most situations to match the rock mass and support stiffness. The support stiffness refers to the amount of deformation that is permitted prior to failure. The response of an ungrouted cable placed in tension can be evaluated by considering the induced strain and the modulus of elasticity of the steel. In this situation the entire length of the cable is used to determine the induced strain. Crack dilation between the ends of a grouted cable bolt causes gradual debonding of the grout-steel interface away from the crack in both directions. The induced strain is related to the debonded length and can result in much higher loads than in the free cable situation. The relationship between the amount of debonding that occurs for a particular load has not been well defined and is still under investigation. Matthews, Tillmann and Worotnicki (1983) describe the application of an artificial debonding procedure designed to reduce the stiffness of a cable support system in a highly stressed crown pillar. The technique involved installing plastic tubing over the cable between button type anchors placed along the length of the strand. The plastic tubing prevents the formation of the grout-steel bond and maximizes the strain potential of the cable. The support stiffness of a cable system can be increased by tensioning the individual bolts. Aside from the application of small loads to seat plates at the hole collar, the tensioning of cables is rarely encountered in practice.

2.12 DISCUSSION

Based on laboratory observations it is evident that the full breaking strength of a cable bolt is not always mobilized, but is in fact dependent on many variables. Some of these variables are related to different laboratory test results in Table 2.2. Very few cases of steel failure are noted and most results are related to pull out loads. Villaescusa, Sandy, and Bywater (1992) report on recent laboratory testing that involve mobilization of the full breaking strength of birdcage, double and combination cables. Combination cables refer to the use of one standard strand and one birdcage cable. These results are included in Table 2.2 and suggest that double cables compare very closely to a single birdcage at a 0.55 water:cement ratio. In terms of cable design, laboratory results can be useful in an assessment of the inherent strength of a cable bolt. An estimate of grout quality combined with a review of laboratory test results can provide an indication of expected cable bolt strength. For example, if a 0.5 m thick block is to be supported and the grout water:cement ratio is estimated at 0.45, Table 2.1 suggests that one cable bolt will support approximately 12 tonnes. Additional cables are required if the block is greater than 12 tonnes. This type of design application is difficult at this stage since there are many variables to consider. This thesis advocates an empirical design tool that relates support to the rock mass based on operational experience.

Table 2.2: Comparison of Laboratory Test Results

Reference	Support Type	Embedment Length (mm)	Water:Cement Ratio	Maximum Load (tonnes)	Steel Failure	Displacement at Maximum Load (mm)	Testing Medium	Grouted Diameter	Curing Time
Goris 1990	Single	288	0.45	9.0	No	45	Steel Pipe	51 mm	28 days
Villaescusa et al. 1992	Single	500	0.55	10.0*	No	8*	Steel Pipe	68 mm	7 days
Goris 1990	Single	289	0.40	11.8	No	60*	Steel Pipe	51 mm	28 days
Goris 1990	Single	300	0.35	14.6	No	41*	Steel Pipe	51 mm	28 days
Goris 1990	Single	299	0.30	16.7	No	68*	Steel Pipe	51 mm	28 days
Villaescusa et al. 1992	Single	1000	0.55	15.3*	No	25*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Single	500	0.30	17.1*	No	43*	Steel Pipe	68 mm	7 days
Goris 1990	Single	763	0.45	19.5	No	61*	Steel Pipe	51 mm	28 days
Villaescusa et al. 1992	Single	1000	0.30	26.1*	No	42*	Steel Pipe	68 mm	7 days
Goris 1990	Single	1064	0.45	26.3	Projected		Steel Pipe	51 mm	28 days
Villaescusa et al. 1992	Double	500	0.55	17.8*	No	20*	Steel Pipe	68 mm	7 days
Goris 1990	Double	238	0.45	18.6	No	3	Steel Pipe	51 mm	28 days
Oliver 1992	Double	305	0.30 (additive)	52.6	Yes		Rock		24 hours
Villaescusa et al. 1992	Double	1000	0.55	30.0*	No	31*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Double	500	0.30	40.8*	No	40*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Double	1000	0.30	51.0*	No	50*	Steel Pipe	68 mm	7 days
Goris 1991	Button @ 2"	305*	0.45	12.0	No	18*	Steel Pipe	51 mm	28 days
Goris 1991	Button @ 6"	305*	0.45	24.5	No	131*	Steel Pipe	51 mm	28 days
Goris 1991	Button @ 4"	305*	0.45	25.3	No	120*	Steel Pipe	51 mm	28 days
Goris 1991	Birdcage–Node	254	0.45	11.7	No	8	Steel Pipe	51 mm	28 days
Goris 1991	Birdcage–ANode	254	0.45	15.4	No	7	Steel Pipe	51 mm	28 days
Villaescusa et al. 1992	Birdcage	500	0.55	18.4*	No	17*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Birdcage	1000	0.30	24.3*	Yes	14*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Birdcage	500	0.30	24.4*	Yes	18*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Birdcage	1000	0.55	26.9*	Yes	23*	Steel Pipe	68 mm	7 days
Goris 1991	Double Birdcage–ANode	254	0.45	35.1	No	120	Steel Pipe	51 mm	28 days
Goris 1991	Double Birdcage–Node	254	0.45	36.2	No	15	Steel Pipe	51 mm	28 days
Villaescusa et al. 1992	Combination	500	0.55	32.5*	No	16*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Combination	1000	0.55	45.6*	Yes	29*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Combination	500	0.30	44.1*	Yes	24*	Steel Pipe	68 mm	7 days
Villaescusa et al. 1992	Combination	1000	0.30	50.1*	Yes	25*	Steel Pipe	68 mm	7 days

* Approximated

CHAPTER 3

CABLE SUPPORT PRACTICE

3.1 INTRODUCTION

Cable bolt practice was noted in detail at all of the operations visited, and when the opportunity arose time was spent with the cable bolt crews. The database will be discussed in detail in Chapter 6, but the particulars of cable bolt practice for each case study are summarized in Table 3.1. Table 3.2 summarizes the installation procedures for each minesite, and a more detailed description is available in Appendix C. This chapter will present a review of current cable bolt practice based on field observations and a literature review.

3.2 GROUT

Grout quality was found to vary quite extensively at each mine. An attempt was made to log an estimate of the grout quality for each case study based on visual observation of the grouting process. On average, the grout water:cement ratio was estimated to be in the 0.4 to 0.45 range but varied from 0.35 to 0.55. Laboratory testing completed by Goris (1990) relates the grout water:cement ratio to the maximum load. The results show a maximum pull out load of 9 tonnes for a 0.45 water:cement ratio and 16.7 tonnes for a 0.30 water:cement ratio, based on a 287 mm (11.3") embedment length with a single 15.9 mm (5/8") cable and no breather tube. In similar tests, other authors (Reichert, Bawden, and Hyett 1992) have shown an increase in pull out load with decreasing water:cement ratio. Goris (1990) also found that grout bleed decreased from 96 mm/m (1.15"/foot) of cable at a 0.45 water:cement ratio to 16 mm/m (0.19"/foot) at

Table 3.1: Case Study Support Practice

CASE #	Stability	Surface	Support Pattern	Mining Method	Hole Dia (mm)	Number of Cables	Anchor	Bolt Length (m)	Plates	Straps	Other Support	Grout Pump
1	Caved	HW	HW Drift Fan	Blasthole	51	2	Downholes	9.1 – 18.3	No	No	No	Spedel 6000
2	Stable	Back	Square	Drift	51	2	Bent Wire	6.1	No	No	No	Spedel 6000
3	Stable	HW	Point Anchor	Blasthole	51	2	Bent Wire	6.1	305 x 305 mm	No	No	Spedel 6000
4	Caved	HW	Point Anchor	Blasthole	51	2	Bent Wire	6.1	305 x 305 mm	No	No	Spedel 6000
5	Stable	Back	Square	Blasthole	51	2	Bent Wire	6.1	No	No	No	Spedel 6000
6	Caved	HW	Point Anchor	Blasthole	51	2	Bent Wire	6.1	No	No	No	Spedel 6000
7	Unstable	HW	Point Anchor	Blasthole	51	2	Bent Wire	6.1	305 x 305 mm	No	No	Spedel 6000
8	Unstable	HW	Point Anchor	Blasthole	51	2	Bent Wire	6.1	305 x 305 mm	No	No	Spedel 6000
9	Stable	HW	Point Anchor	Blasthole	51	2	Bent Wire	6.1	No	No	No	Spedel 6000
10	Stable	Back	Square	Cut & Fill	51	2	Downholes	22	No	No	2.4m Swellex	Spedel 6000
11	Stable	HW	Point Anchor	Blasthole	51	2	Bent Wire	6.1	No	No	No	Spedel 6000
12	Stable	Back	Fan	Drift	51	2	Bent Wire	7.6	305 mm dia.	Steel Sets	3.7m Sw/1.5m RB	Spedel 6000
13	Stable	Back	Square	Blasthole	51	2	Bent Wire	6.1	No	No	Rockbolts	Spedel 6000
14	Stable	Back	Point Anchor	Blasthole	51 & 57	2	Bent Collar Wire	14.0	No	No	Rockbolts	Spedel 6000
15	Stable	Back	Square	Blasthole	51 & 57	2	Bent Collar Wire	8 & 15	No	No		Spedel 6000
16	Caved	Back	Point Anchor	Blasthole	51 & 57	2	Bent Collar Wire	8 & 12	No	No		Spedel 6000
17	Caved	Back	Square	Blasthole	51 & 57	2	Bent Collar Wire	12 & 15	No	No		Spedel 6000
18	Stable	Back	Square	Blasthole	51	2		9.8	254 x 254 mm	No	Rockbolts	Spedel 6000
19	Unstable	Back	Square & HW Fan	Blasthole	51	2		9.8	254 x 254 mm	No	Rockbolts	Spedel 6000
20	Caved	HW	Point Anchor	Blasthole	51	1		14.6	102 x 102 mm	No	No	Spedel 6000
21	Stable	HW	Point Anchor	Blasthole	51	2		14.6	102 x 102 mm	No	No	Spedel 6000
23	Stable	Back	Square & HW Fan	Blasthole	51	2		9.8	102 x 102 mm	No	Rockbolts	Spedel 6000
25	Caved	Back	HW/FW & Square	Blasthole	51	2		9.8	102 x 102 mm	No	Rockbolts	Spedel 6000
26	Stable	HW	Point Anchor	Blasthole	51	1		14.6		No		Spedel 6000
27	Stable	HW	Point Anchor	Blasthole	51	1		14.6		No		Spedel 6000
29	Stable	Back	Square	Cut & Fill	57	2	Bent Toe Wire (1 & 2)	18.3	No	No	Rockbolts	Moyno 3L3
30	Stable	Back	Fan	Blasthole	51	1	Spring Steel	6.1	152 x 152 mm	Yes	2.4m RB/Scr/Ex	Spedel 6000
32	Stable	HW	Quasi – Mandolin	Blasthole	64	1	Spring Steel	12.0	152 x 152 mm	Yes	No	Spedel 6000
33	Stable	Back	Fan	VCR	51	1	Spring Steel	6.1	152 x 152 mm	No	2.4m RB/Screen	Spedel 6000
34	Unstable	Back	Point Anchor	Blasthole	51	1	Spring Steel	9.1	152 x 152 mm	No	2.4m Rebar/Ex	Spedel 6000
36	Stable	Back	Square	Drift	51	1	Spring Steel	6.1	152 x 152 mm	No	2.4m RB/Ex	Spedel 6000
37	Stable	Back	Fan	VCR	51	1	Spring Steel	12.2	152 x 152 mm	Yes	2.4m RB/Scr/Ex	Spedel 6000
43	Stable	Back	Fan	VCR		Rockbolts		4.9	152 x 152 mm	Yes	2.4m RB/Scr/Ex	N/A
45	Stable	Back	Square	Cut & Fill	51	1	Bent Toe Wire/Wedge	15.8	No	No	2.4m RB/Swellex	Minpro 3
46	Stable	Back	Square	Cut & Fill	51	1	Mess Wire/Wedge	18.3	No	No	Rockbolts	Minpro 3
47	Stable	Back	Square	Cut & Fill	51	1	Mess Wire/Wedge	18.3	No	No	Rockbolts	Minpro 3
48	Stable	Back	Square	Cut & Fill	51	1	Mess Wire/Wedge	18.3	No	No	Rockbolts	Minpro 3
49	Stable	HW	Point Anchor	Blasthole	51	1	Mess Wire/Wedge	9.1	No	No		Minpro 3
50	Caved	HW	Point Anchor	Blasthole	51	1	Mess Wire/Wedge	9.1	No	No		Minpro 3
52	Stable	Back	Square	Blasthole	51	1	Mess Wire/Wedge	10.2	No	No	RB/Resin	Minpro 3
53	Stable	Back	Square	Blasthole	51	1	Mess Wire/Wedge	10.7	Yes	No	Rockbolts	Minpro 3
54	Stable	HW	Even	Blasthole	51	1	Mess Wire/Wedge	9.1	No	No	No	Minpro 3
55	Stable	Back	Point Anchor	Blasthole	51	1	Mess Wire/Wedge	5.0	No	No	Rockbolts	Minpro 3
56	Stable	HW	Even	Blasthole	51	1	Mess Wire/Wedge	7.6 – 9.1	No	No	No	Minpro 3
57	Stable	Back	Square	Blasthole	51	1	Mess Wire/Wedge	6.4	No	No	Rockbolts	Minpro 3
59	Stable	Back	Square	Room & Pillar	64	2	Bent Toe Wire (2)	4.9	No	No	2.1m Swellex	Minpro 3

Table 3.2: Cable Bolt Installation Procedure Summary

Mine	Pump	Installation Method	Anchor	W:C Ratio	# Cables	Length (m)
Mine #1	Spedel 6000	Breather	Bent Wire	0.45	2	6.1–22.0
Mine #2	Minepro 3	Breather > 12.2m Grout < 12.2m	Spring Steel	0.375 0.32	2	8.0–15.0
Mine #3	Spedel 6000	Breather	Spring Steel	0.45	1 & 2	9.8–14.6
Mine #4	Moyno 3L3	Breather	Bent Toe Wire (1 or 2)	0.50	2	18.3
Mine #5	Spedel 6000	Grout	Spring Steel	0.40	1	6.1–12.0
Mine #6	Spedel 6000	Grout	Spring Steel	0.4–0.45	1	12.2
Mine #7	Minepro 3	Grout	Bent Toe Wire	0.35–0.40	1	15.8
Mine #9	Minepro 3	Breather > 13.7m Grout < 13.7m	Messenger Wire & Wedge	0.33–0.40 0.30	1	5.0–18.3
Mine #10	Minepro 3	Grout (Retrieved)	Bent Toe Wire (2)	0.35–0.40	2	4.9
Mine #11	Cabolt	Grout (Retrieved)	Cable Kink	0.30–0.35	2	8–10

a water:cement ratio of 0.30. Bleed has been found to be an important consideration in cut and fill mining where cables are typically installed at lengths of up to 18.3 meters. The last lift may not contain fully grouted cable if the water:cement ratio is too high (Cluett 1991).

Limited monitoring of the grout quality was found to exist and is recommended as an important component of any established cable bolt procedure. A test kit (Gendron et al. 1992) from the Noranda Technology Center was made available for this study in order to collect grout samples to be submitted for strength testing. Unfortunately, samples were collected on only one occasion due to the uncertainty of arriving on site while cables were being grouted. The kit comes in a metal box and contains tubes that can be filled with grout and sent to the laboratory for testing. It is also possible to visually estimate the grout quality, but this requires some exposure to the flow characteristics of grout at different water:cement ratios. This is best done

Table 3.3: Grout Characteristics (After Hyett, Bawden, and Coulson 1992)

W:C RATIO	GROUT CHARACTERISTICS AT END OF GROUT HOSE	HANDLING CHARACTERISTICS
<0.30	Dry, stiff sausage structure	Sausage fractures when bent Grout too dry to stick to hand Can be rolled into balls
0.30	Moist sausage structure 'Melts' slightly with time	Sausage is fully flexible Grout will stick to hand Easily rolled into wet, soft balls
0.35	Wet sausage structure Structure 'melts' away with time	Grout sticks readily to hand Hangs from hand when upturned
0.40	Sausage structure lost immediately Flows viscously under its own weight to form pancake	Grout readily sticks to hand but can be shaken free
0.50	Grout flows readily and splashes on impact with ground	Grout will drip from hand - no shaking required

by preparing samples and observing how each sample flows into the mixing container and from the grout hose. Hyett, Bawden and Coulson (1992) present a description of different water:cement ratios (Table 3.3) that describe the observation of grout quality. Normal portland cement was used at all the mines visited in this study, but the use of high early strength cement was noted on a few occasions. Admixtures were found to be limited to anti-bleed and water-reducing chemical agents that were used on an inconsistent basis. The admixtures encountered required typical dosages of approximately 1% by weight of cement, and were added during the grout mixing phase. Gendron et al. (1992) suggest that grout admixtures create quality control problems that overshadow any possible benefits. Premixed additives are available but result in increased material costs.

3.3 GROUT PUMPS

Two main types of grout pumps were found in use in western Canadian mines. The most common pump was the air powered Spedel 6000 reciprocating piston pump that was used in conjunction with a Spedel B3100 mixer (Oliver 1992). It was found that the use of a Spedel pump was generally associated with the use of a grout water:cement ratio above 0.4. This does not appear to be a limitation of the pump as 0.35 water:cement ratio grout was observed to be successfully pumped, and Oliver (1992) discusses successful pumping of 0.3 water:cement ratio grout in a 6.1 meter length of plastic tubing. Industrial experience however indicates that the Spedel 6000 is limited to pumping 12 meters for a vertical hole at 0.35-0.40 water:cement ratio. Oliver (1992) describes the reduction in Spedel pump performance as a result of inadequate pump cleaning. The Minepro 3 pump is also used at a number of operations, but is still relatively new on the market. The Minepro is a positive displacement pump with a rotor-stator assembly that is designed to pump grout down to a 0.3 water:cement ratio. Practical applications observed in this study indicated that pumping problems occurred below a water:cement ratio of 0.35 in vertical holes longer than 14 meters. Both a pneumatic and electric version of the Minepro 3 pump are available and it can also be operated from the hydraulics of a host vehicle. The internal parts of the rotor-stator assembly are easy to remove and clean. The mixing tank is mounted within the pump hardware and offers a more effective shear mixing mechanism than the Spedel pump. The Minepro pump however is larger and more difficult to mobilize, whereas the Spedel pump is preferred for areas of restricted access. Most of the mining operations are moving towards the Minepro 3 pump in an attempt to improve the quality of grout obtained. The Minepro seems to handle the mine environment well but has yet to establish a lengthy operating history.

Both the Minepro and Spedel pumps involve a batch mixing process where a fixed quantity of grout is prepared and pumped prior to returning to the mixing phase. Continuous mixing presents additional quality control problems and has not yet been efficiently adapted to cable bolt applications. A recirculation system (Oliver 1992) is required during the grouting phase to permit continual flow during the pumping of a prepared batch. It was found in this study that shutting down the pump with grout still in the system frequently lead to line blockage, especially utilizing a water:cement ratio below 0.40. When establishing a cable bolting procedure it is recommended that consideration be given to the hardware involved. A special vise for removing the rotor from the stator for cleaning was found useful at one operation. Minimizing restrictions in the line from the pump to the grout tube can help in reducing line blockages, and adapting quick coupling mechanisms to the hose ends can improve efficiency. Pagel (1987) describes the use of reusable rubber plugs to block hole collars while grouting with the breather tube method. These plugs are held in place during pumping and incorporate a slot for the breather tube. In addition, the grout plug incorporates a 25 mm nipple that permits the coupling of a 25 mm hose directly from the pump. Typical practice in Western Canada involves coupling the pump discharge directly to the 19.1 mm grout tube and generates frictional resistance that seems to limit pump performance. Bourchier, Dib and O'Flaherty (1992) report on the use of a grouting assembly that also permitted coupling of a grout hose directly from the pump. This method was devised to pump 0.35 water:cement ratio grout with a Spedel 6000 pump using the breather tube method and 15.2 meter vertical holes. The grouting assembly was found to reduce the frictional head losses and eliminate bursting hoses but was only reported successful when the grout tube was recessed 8 meters from the hole toe.

3.4 PLATES AND STRAPPING

Plates were used at four of the operations visited but only on a consistent basis at one location. The plates varied from 102 mm x 102 mm (4" x 4") to 305 mm x 305 mm (12" x 12") in size and 6.4 mm (0.25") to 9.7 mm (0.38") in thickness. The majority of the case histories collected did not incorporate plates or straps within the design. The use of plates is important where blocks have the ability to unravel around the cables or where tensioning may be desirable. Plates also mobilize the full strength of a cable bolt by preventing blocks from sliding off the strand. Additional restraint can be provided by tying cables together with strapping or screen. In one case, cables were designed to be three meters longer than the holes so that they could be laced together using wire rope clips. This idea may have some merit but in practice the lacing was not completed due to production requirements. The use of plates in conjunction with straps was successfully observed at several operations. A strap thickness of 4.8 mm (3/16") was selected in one case to allow for some ground movement to occur. In another case, 102 mm x

102 mm (4" x 4") square mesh screen was used in conjunction with cables and straps. Both plates and straps were attached to cables with the use of barrel and wedge anchors. The barrel and wedge anchor consists of a segmented tapered steel wedge that fits inside a tapered steel barrel (Thompson 1992). The whole assembly slides over the cable with the tapered end of the barrel against the plate or strap. Rock movement pushes the barrel against the wedge, which in turn provides a positive interlock against the cable. The barrel and wedge anchors encountered in practice were flat on both ends, but cast domed anchors are also available to improve the load distribution at the plate-anchor junction. The installation of barrel and wedge anchors was frequently done by hand and rarely resulted in plates being tight to the rock. A hollow drill rod was used on one occasion in conjunction with the deck of a scissor lift to tighten plates. Two mines used a hydraulic jack to apply approximately 2 to 4 tonnes to plated cable ends. This was found to be the most effective method of installing plates and straps, but results in reduced productivity. The installation of plates and straps was generally left until two or three days after cable grouting. Hyett, Bawden, and Coulson (1992) suggest that plates can be installed 16 to 24 hours after grouting without damaging the cable-grout interface.

3.5 CABLE GEOMETRY

Ninety percent of the mines visited used 15.9 mm (5/8") cables that were supplied in pre-cut lengths. One operation ordered their cable in coils that were shipped underground and cut by the cable bolt crew as required. This permits the installation of variable cable lengths, but continuous cable is susceptible to tangling and requires additional capital investment to provide mobility. Double cables were used most of the time but single cables were used in 46% of the cases. Although birdcage cables have recently become available on the Canadian market, none were found in use at present. One operation had installed birdcage cable bolts in the past but were not using them on a regular basis.

3.6 CABLE ANCHORS

Supporting cable bolts in the hole prior to grouting is an important consideration since the integrity of the anchor must hold the cable until the grouting phase is completed. Cases of anchor failure have been noted at different operations and can have serious consequences. Most operations simply bend back one or two wires on one of the cables, either 135° at the toe of the hole or 45° at the collar. The frictional resistance between the bent wire and the hole surface is sufficient to support the weight of the cable. Pushing cables with bent wires at the toe of the hole is a difficult process due to the resistance of the hole walls, but has the advantage of immediately supporting the cable. As noted in Table 3.1, the bent wire anchor is occasionally placed at the collar of the hole, resulting in an easier installation process, but does not fully support the cable until installation is complete. Pull test results from one operation using 18.3 meter single cables found that a single wire anchor at the toe of the hole required five times the weight of the cable to cause slippage. In the case of a double wire anchor, twelve times the cable weight was required to induce slippage. Four of the mines visited now utilize a ferrule, or end holding device, that is factory mounted at the end of the cable and supplied with spring steel strips that are screwed on underground. The ferrule and spring clips can usually be pushed into the hole easier than a bent wire anchor, but the required strip size varies with hole diameter. Two spring steel strips are usually screwed onto the end of the ferrule perpendicular to each other. Some slippage problems were noted by mine operators and were countered by increasing the number of steel strips. Two other methods of anchoring cables in western Canada included utilizing two lengths of messenger wire and placing a full kink in the cable. The messenger wire was clamped to the cable with packaging strapping and was found to provide successful anchorage at one operation. Kinking of the full cable was used at two operations that employed the use of a fully mechanized cable bolting jumbo. Wedges were used on some occasions to provide secondary support at the hole collar but are not suitable for primary cable anchorage. In terms of installation, it was common practice to anchor a large number of cables in one pass and complete the grouting phase at a later stage.

Mechanical anchors are available for cable bolts but were only used on one occasion (Fraser 1976) in western Canada to tension 25 mm locked coil ropes prior to grouting. The mechanical anchor is similar to the standard mechanical wedge and bail assembly associated with standard rockbolts. Ungrouted tensioned cables provide a soft support system that is dependent on no anchor slippage and good plate contact at the hole collar. Fuller, Dight, and West (1990) suggest that tensioning of cable bolts is not necessary as high loads are built up as a result of rock mass dilation. Tensioning of cable bolts may be useful where support is installed after significant rock mass movement has already occurred. In this situation, the inherent strength of the rock mass is reduced and tensioning is useful in limiting further dilation. Fuller (1981) also suggests that tensioning may be necessary in areas of low horizontal stress where minimal clamping forces are present.

3.7 INSTALLATION PROCEDURE

Fifty percent of the operations visited were found to use the traditional breather tube technique of grouting cables, referred to as the *breather tube method* in this study. This method involves attaching a breather tube to the top of the cable prior to installation and placing this end at the toe of the hole (Figure 2.1). A grout tube is taped to the cable just inside the hole collar and the hole is plugged prior to grouting. Materials used for plugging holes include an expanding polyurethane foam, burlap, rags, shredded cloth and cement grout plugs. The pump discharge hose is then hooked up to the grout tube and the hole is filled from the collar to the toe. Air is forced through the breather tube and can be monitored by bubbling the discharge through water. The grouting phase is complete when grout is observed returning through the breather tube. The grout tube is then disconnected and the tubes are tied off to prevent grout leakage. Forcing the grout to return through the breather tube requires high pump pressures and often results in bursting of hoses (Bourchier, Dib, and O'Flaherty 1992). In addition, the breather tube method is susceptible to leaks, especially near the hole collar where ground may be fractured. Minor leaks can be plugged as they occur but occasionally become too severe to permit successful grouting. In this situation, recommended procedure at most operations suggests leaving the hole for one hour to allow grout to gel within the cracks. The typical breather tube diameter observed was 9.5 mm but was found to range up to 12.7 mm. Cluett (1991) describes the closing of breather tubes near the hole collar when grouting 19.8 meter holes with less than 0.45 water:cement ratios. This was believed to be the result of high hydrostatic pressures due to the long grout columns and was corrected by utilizing high pressure breather tubes.

The second method encountered for grouting upholes is referred to as the *grout tube method*, and involves the use of a thick grout and a single grout tube taped to the cable bolt end placed at the toe of the hole. The grout is pumped through the grout tube and fills the hole from the toe to the collar. It is important that the grout used in this method has sufficient viscosity to remain in the hole and advance as a continuous front. Several authors (Oliver 1992; Reichert, Bawden, and Hyett 1992) have indicated that a 0.35 water:cement ratio grout will remain in an uphole. Some operators plug the hole upon completion, and continue pumping in an attempt to pressurize the grout column and fill any remaining voids. The grout tube is normally left in the hole, but one operation retrieved the grout tube during grouting so that the tube could be reused and installation time reduced. Oliver (1992) observed that it was almost impossible to completely fill a clear plastic tube by retracting the grout tube, and it is believed that this method will only enhance the presence of voids within the grout column. Some mine operators (Cluett 1991; Bourchier, Dib, and O'Flaherty 1992) have observed separation of the grout column while using the grout tube method with low water:cement ratios in holes greater than 15 meters. Grout either appeared at the collar, falsely indicating that the hole was full, or left voids within the grout column. The analysis of grout flow within a cable bolt hole is difficult to observe and would be an interesting area of future research.

When observing the grouting operation during the collection of data for this study, the grout tube method was found to be preferred over the breather tube method. The failure of grout

Table 3.4: Cable Bolt Grouting Procedure (after Cluett 1991)

HOLE LENGTH (m)	HOLE ANGLE (FROM HORIZONTAL)	PUMP	BREATHER TUBE	GROUT TUBE	COLLAR PLUG	GROUT
Uphole > 13.7 m	> 45^0	Minepro 3	3.1 MPa (450 psi)	1.7 MPa (250 psi)	Cement	0.33-0.40
Uphole > 13.7 m	< 45^0	Spedel 6000	3.1 MPa (450 psi)	0.7 or 1.7 MPa (100 or 250 psi)	Cement	0.40-0.45
Uphole 9.1 - 13.7 m	> 45^0	Minepro 3	None	0.7 or 1.7 MPa (100 or 250 psi)	None	0.30-0.33
Uphole < 9.1 m	all	Minepro 3	None	0.5 MPa (75 psi)	None	0.30-0.33
Uphole < 9.1 m	all	Spedel 6000	0.5 MPa (75 psi)	0.5 MPa (75 psi)	Cement	0.40
Downhole > 9.1 m	all	Minepro 3	None	0.7 or 1.72 MPa (100 or 250 psi)	None	0.30-0.33
Downhole > 9.1 m	all	Spedel 6000	None	0.7 or 1.72 MPa (100 or 250 psi)	None	0.40
Downhole < 9.1 m	all	Minepro 3	None	0.5 MPa (75 psi)	None	0.30-0.33
Downhole < 9.1 m	all	Spedel 6000	None	0.5 MPa (75 psi)	None	0.40

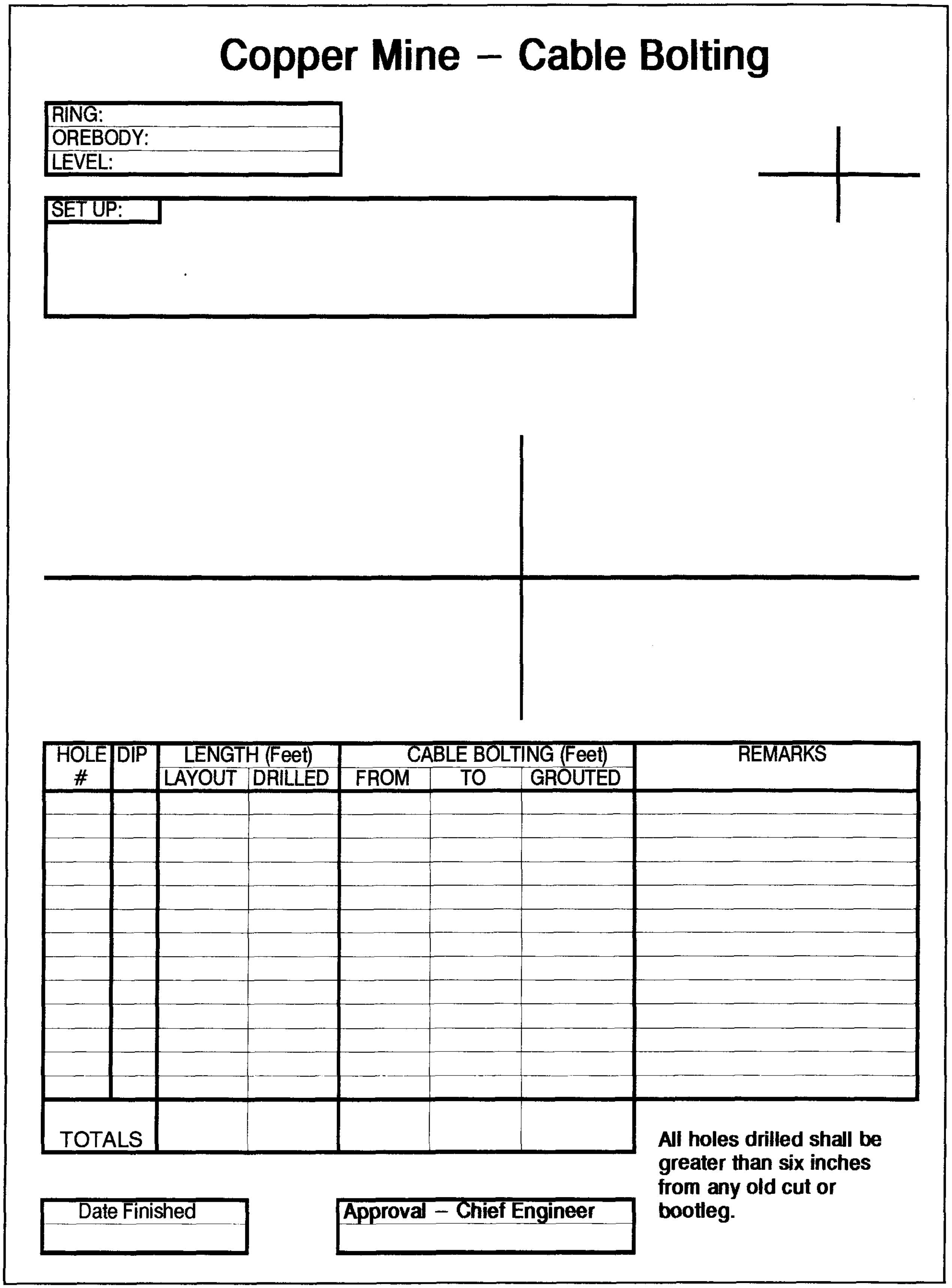
to flow out of the breather tube was commonly observed, especially at lower water:cement ratios. This frequently lead to the addition of water to the grout mixture and a drastic increase in the water:cement ratio. Cluett (1991) describes a grouting procedure (Table 3.4) adapted to a mining operation in Manitoba that tackles some of the problems discussed in this section. This procedure reflects operational experience and limits the use of the Spedel pump to upholes either less than 13.7 meters or less than 45° from horizontal. For holes that are greater than 13.7 m and 45° from horizontal, the Minepro 3 pump is used with the breather tube method and a water:cement ratio of 0.33 to 0.40. The use of the grout tube method is preferred for shorter holes along with a lower water:cement ratio. Recent operational practice has favoured the use of the Minepro 3 pump for all conditions. Table 3.4 is representative of the ideal practice observed in this study and is recommended for consideration when establishing operational procedure. Observed cable bolt installation procedures usually required the flushing of the breather tube with water prior to grouting, to ensure that the tube is not internally blocked. It was also found beneficial to cut breather and grout tube at 45° in order to minimize the size of the cable/tube configuration inserted to the hole toe. Downholes are grouted using the grout tube method but are not restricted to the use of grout below a water:cement ratio of 0.35. Breakthrough holes are to be avoided with the use of downholes since they entail some method of blocking the hole toe. Upon completion of drilling, downholes should be blown clean prior to retrieving drill rods and blasthole plugs should be placed in the hole collars.

3.8 DESIGN LAYOUTS

It was quite common for design layouts of cable bolt holes to be issued as a standard drawing to be adapted by the driller to various stope widths. This method leads to some organizational problems especially if the drill and cable bolt crews are different. If drill hole quality is a problem, the use of issued engineering layouts would be recommended. A sample cable bolting layout sheet is shown in Figure 3.1.

Hole diameter varied from 51 to 64 mm for both single and double cables. No installation problems were encountered with this range of hole size, although procedures often required that larger diameter holes be drilled in poor ground. Schmuck (1979) suggests that drillhole diameter be designed to provide 6 to 13 mm between the cable and the hole.

Cable bolt holes are typically drilled with pneumatic percussive drills that are mounted on a mobile rig. Blasthole drilling experience with these types of drill rigs limit uphole lengths to approximately 18.3 meters, and downholes to 25 meters. Fuller (1981) notes that hole deviation can alter the cable pattern at depths in the range of 20 meters and Hunt and Askew (1977) suggest a maximum length of 19.5 meters for 65 mm holes. Drilling and cable installation are frequently completed by different crews and it is important for both to understand the purpose and importance of cable bolts. The layout sheet in Figure 3.1 can be used by both crews and provides the installation crew with an idea of problems encountered during the drilling phase. This information is important where breakthrough holes or cracked ground can create grouting problems. Cable lengths are usually designed to cover the full length of the drillhole but countersinking is used where bolts are not required for the full hole length. Countersinking involves recessing the cable beyond the hole collar, utilizing aluminium drill rods or segmented loading sticks. The practice of countersinking produces lower insertion productivity but minimizes the unnecessary use of cable. The layout sheet in Figure 1 can be used to indicate recess depths to the cable crew.



Copper Mine – Cable Bolting

RING:
OREBODY:
LEVEL:

SET UP:

HOLE #	DIP	LENGTH (Feet)		CABLE BOLTING (Feet)			REMARKS
		LAYOUT	DRILLED	FROM	TO	GROUTED	
TOTALS							

All holes drilled shall be greater than six inches from any old cut or bootleg.

Date Finished

Approval – Chief Engineer

Figure 3.1: Sample cable bolt layout sheet

3.9 CABLE PATTERNS

Some typical examples of open stope patterns encountered are presented in Figures 3.2 to 3.4. It should be noted that only one case study involved development that was specifically designed for the installation of cable bolts. In all other cases, the support pattern was designed around existing development. Table 3.1 describes each case in terms of the support pattern and a summary of the most common patterns is presented in Table 3.5.

Table 3.5: Summary of Cable Patterns in Western Canadian Practice

SUPPORT PATTERN	NUMBER OF CASES	PERCENTAGE
Fan Back	4	9%
Square Back	20	45%
Point Anchor Back	4	9%
Even HW	2	4%
HW Drift Fan	1	2%
Point Anchor HW	13	29%
Quasi-Mandolin HW	1	2%

3.9.1 Back Support

Case studies of back support were made up largely of blasthole and vertical crater retreat open stoping situations, but some cut-and-fill, room and pillar, and drift cases are also included in the database. Figure 3.2a shows typical cable installations of square back support patterns encountered in cut and fill mining. Long cables up to 18.3 m are normally installed in upholes to cover three or more mining lifts. Extra cables are often installed into the hangingwall. Footwall rolls within the ore zone require the placement of additional bolts on certain lifts.

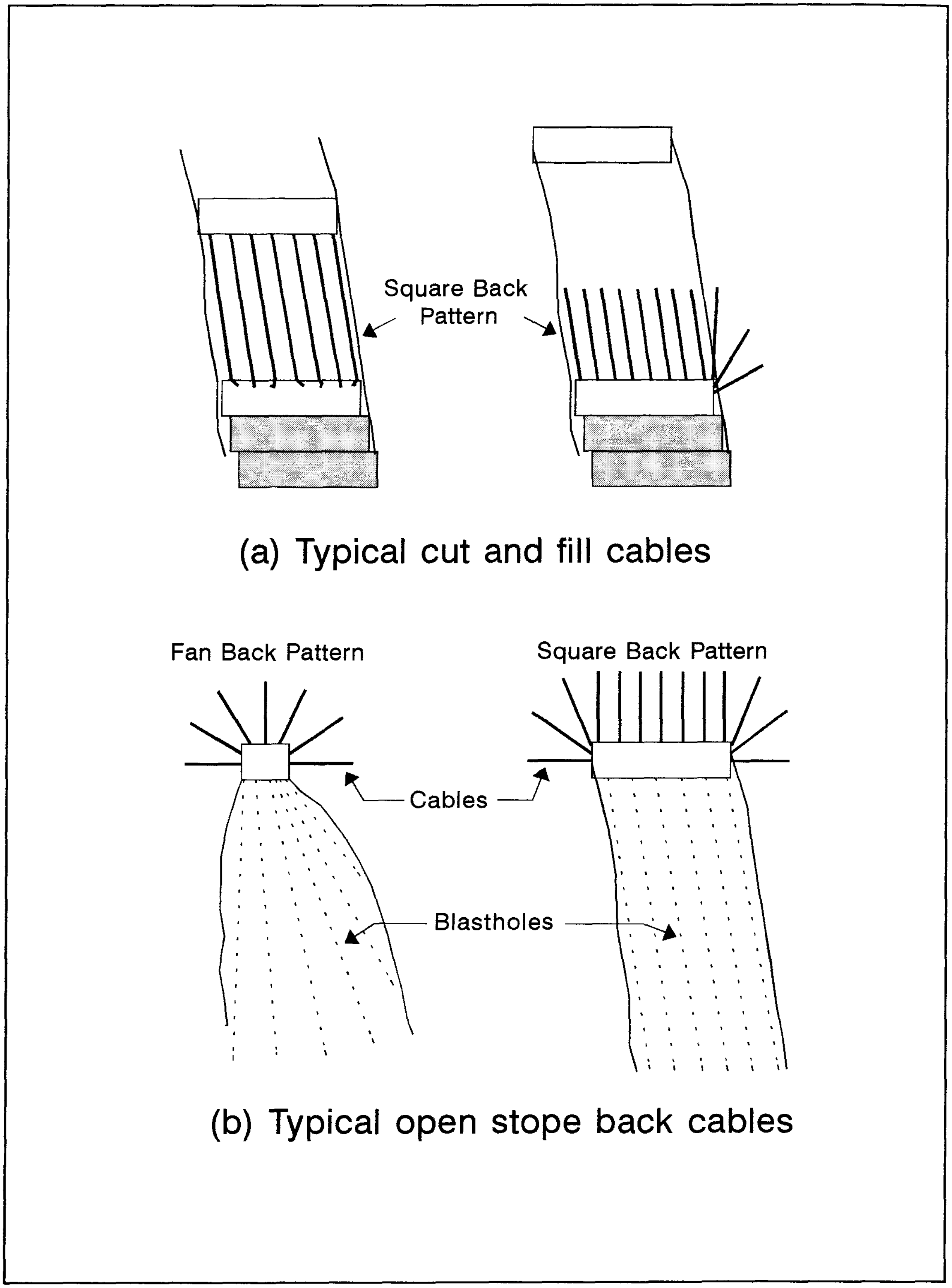


Figure 3.2: Typical back support for open stope and cut and fill mining

Cables can also be installed from an overcut but are restricted by the length of holes that can be accurately drilled. In narrow open stopes, cable bolts are installed in a fan pattern (Fan Back) from footwall to hangingwall (Figure 3.2b). Where the development is large enough, cable bolts are installed on a regular square pattern (Square Back) and sometimes angled into the hangingwall and footwall. Excessive dilution from the hangingwall or footwall can undercut this type of back support and induce failure. Poor distribution of cables into a stope back often occurs as a result of limited access, or where the drill drifts are not slashed to the full width of the orebody. In this case a regular square pattern is not possible and the point anchor approach (Point Anchor Back) is often adopted, as illustrated in Figure 3.4b.

3.9.2 Hangingwall Support

Hangingwall cable bolting was largely installed from a sublevel drill drift to act as a point anchor (Point Anchor HW) as illustrated in Figure 3.3c. The bolt densities encountered on the sublevels varied from two to seven bolts installed on rings spaced 2.4 m along strike. The design strategy in some cases was not to stabilize the whole hangingwall but to limit the effect of undercutting as mining advanced to the next lift. A number of case histories of the point anchor approach to cable support have been assembled in order to develop design strategies for this type of bolt pattern. Further discussion on point anchor design will be pursued in Chapter 7. It is believed that the localized bolt density is not as important as the distance between each point anchor. Hangingwall cables can be evenly distributed over the supported surface by drilling holes from a hangingwall drill drift (HW Drift Fan) or countersinking the bolts through the back of a sublevel drift (Even HW) as illustrated in Figures 3.3a and 3.3b. Cables installed from a separate hangingwall drift were encountered on only one occasion due to the high cost associated with the additional development required.

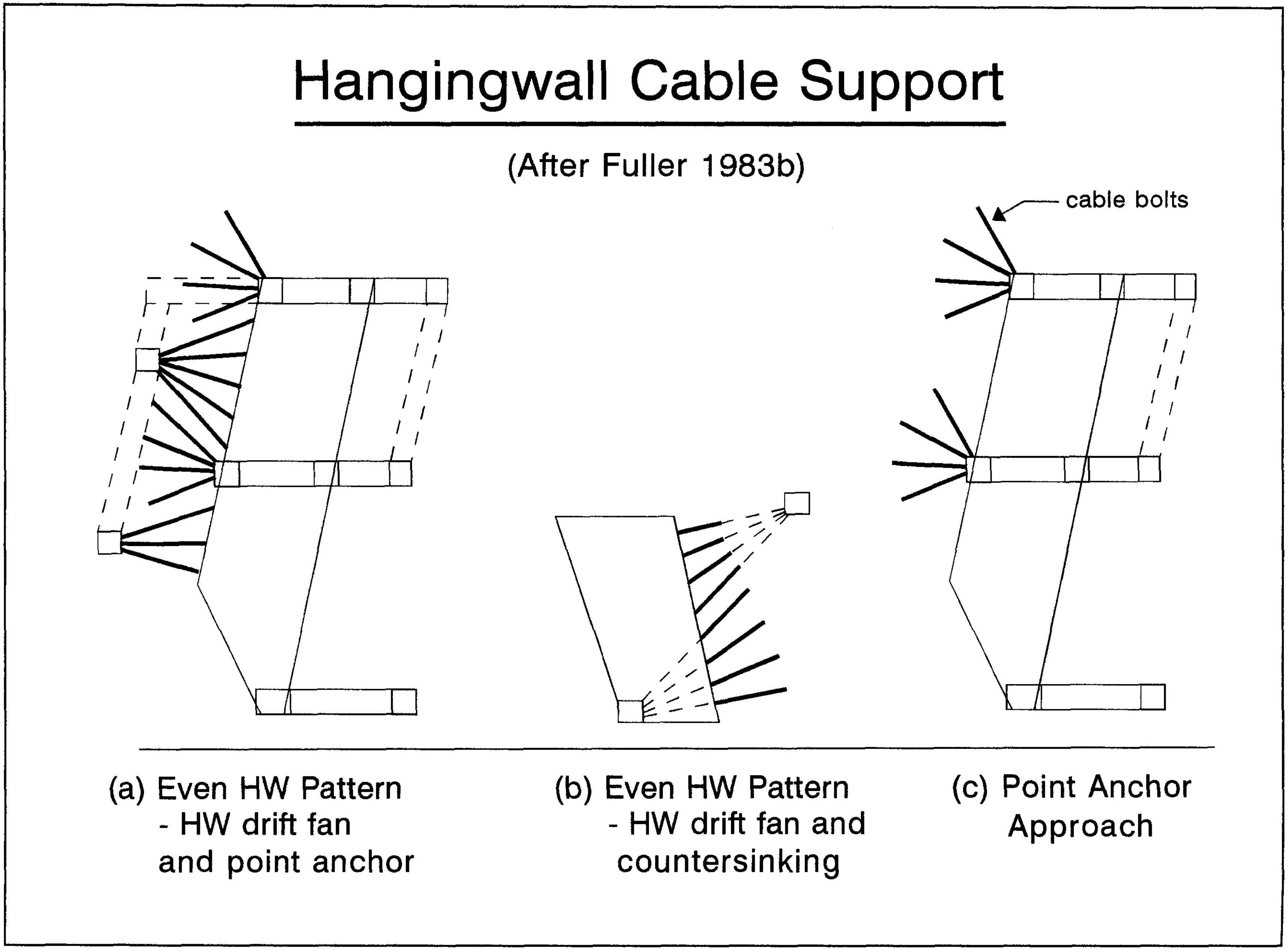


Figure 3.3: Hangingwall cable support patterns (After Fuller 1983b)

3.9.3 Other Support Patterns

Cable slings were encountered in isolated cases of crown pillar recovery, bulkhead support and pillar reinforcement. In the case of crown pillar recovery, cable slings were used to support a log mat below slag or tailings fill as mining advanced forward (Figure 3.4a). Slings were also used to reinforce the back and walls of conventional sublevel development for an open stope slot. Although cable slings were not the main focus of this study, a similar support mechanism was encountered in two design applications. Figure 3.4b illustrates an application of the sling type approach applied to back support of an open stope. This type of support may be useful where there is a poor distribution of cables 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 were discussed at a design level. Mandolin bolting is another method of cable support that was encountered at the design stage. Cable bolts are installed parallel to the stope hangingwall and attached to a second set of angled cables installed above the sublevel drill drift (Figure 3.4c). The cables parallel to the hangingwall are angled less than the dip of the surface in order to place the end of the cable into good quality rock. The sublevel drill drift may be shotcreted to protect the exposed portion of the cables.

3.10 HEALTH AND SAFETY ASPECTS OF CABLE BOLTING

3.10.1 Cable Pushing

A single strand 15.9 mm cable typically has a nominal weight of approximately 1.1 kg/m. In a 9 meter vertical hole, a double strand bolt would generate an effective weight of 20 kg. Manually pushing cables into a drillhole is the most common method of installation. In addition

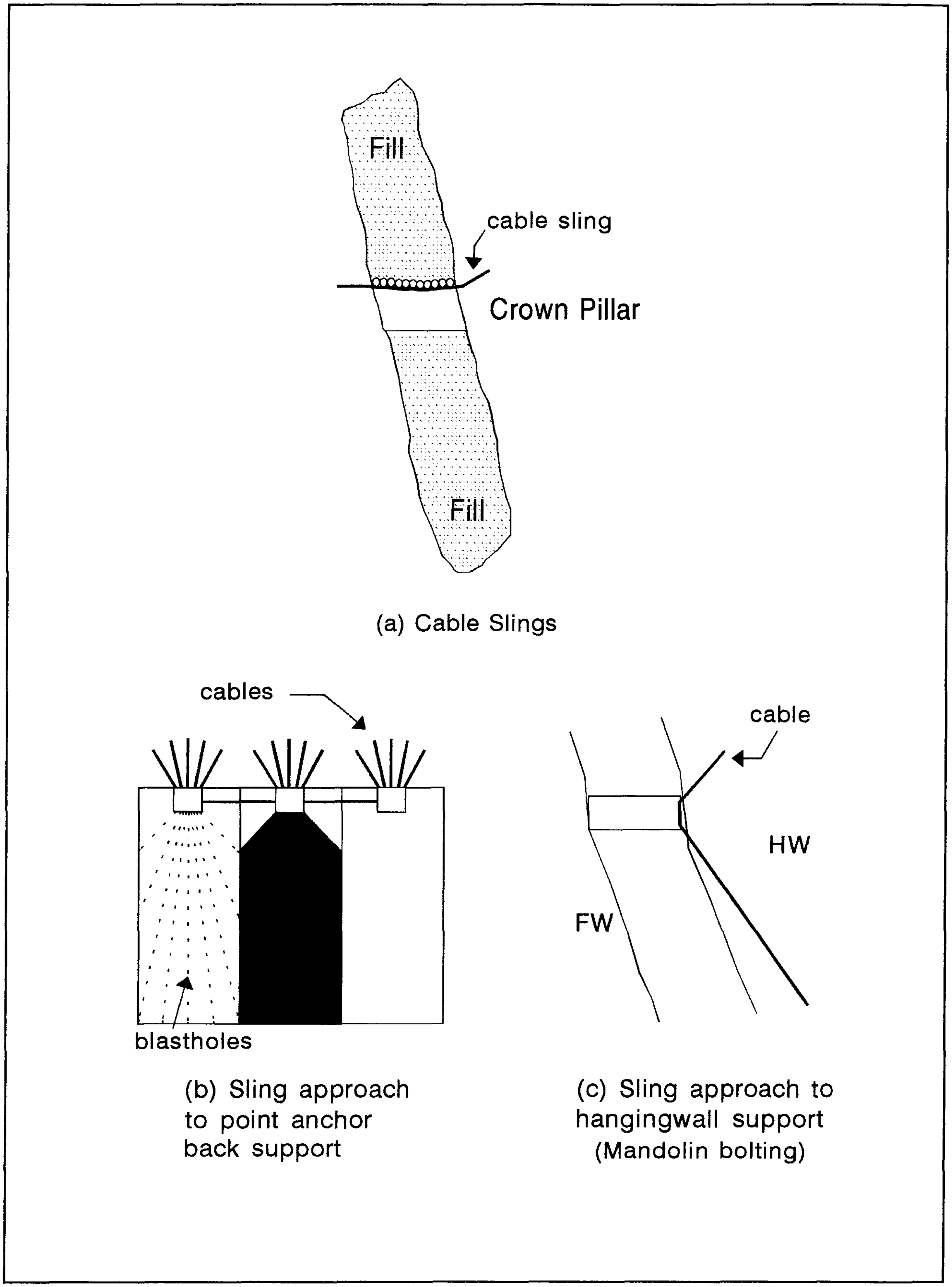


Figure 3.4: Sling approach to open stope support

to the cable weight, the operator has to contend with grout tube, breather tube and an additional force to overcome a cable holding device. The use of bent wire and spring steel anchors have been discussed earlier in this chapter. The pushing of cables into a drillhole is an awkward process and can result in back injury, especially in long upholes. The cable pushing crew should be provided with some means of reaching the hole collar in order to assume a effective stance to manually push cables. A scissorlift is commonly used at many operations and should be dedicated to the cable bolt crew. Cables and other equipment can easily be moved from area to area, and a suitable installation platform is readily available. Specialized mobile cable bolting equipment incorporating a grout pump, storage area and an installation platform is available on the market, or can be custom designed.

Several mechanized methods of pushing cables have been developed in an attempt to reduce the manual effort required and increase productivity. An air powered inserter was commonly encountered among western Canadian mines, but was reluctantly used by cable bolt crews due to cable slippage and lower installation productivity. The opposite is suggested by Pagel (1983), who reports on the potential of doubling cable insertion productivity with the use of a mechanized inserter. The inserter operates similar to an old fashioned roller dryer by feeding cables through two rubber rollers into the drillhole. The deck of a scissor lift was used at one operation to push the final segment of cable incorporating a bent wire anchor at the collar. The resistance provided by cable weight and the frictional component of the anchor was too high to allow for manual insertion. The exposed cable end was inserted into a hollow drill rod and the scissor deck was slowly raised to force the cable into the hole. This was found to be an effective method but the drill rod must be kept aligned with the hole and secure footing ensured. Continual raising and lowering of the scissor deck during this process is bound to increase maintenance costs. Hunt and Askew (1977) describe a procedure of mechanical cable insertion utilizing the feed motor of a longhole drill rig and a cable grab attached to the shank adaptor. A similar method was encountered at one operation in western Canada.

3.10.2 Cable Anchoring

Typical installation practice in western Canada involves the insertion of a large number of cables in one pass with grouting completed at a later stage. If a cable slips from an uphole prior to grouting, it presents a danger to the bolting crew and to other personnel in the area. The cable weight is sufficient in most cases to cause serious injury and the possible whip can cover a large area. It is good practice to push and grout cables with as little delay as possible, and to restrict access during this phase of the operation. Slippage has been reported at several operations in western Canada utilizing both the bent wire and spring steel methods of anchoring. In an effort to reduce cable slippage, procedural changes have been incorporated to increase the number of spring steel strips, or utilize a double instead of a single wire anchor. A double wire anchor at the hole toe is difficult to manually install, and prompted one operation to return to the single wire anchor with the adoption of a protective covering for the cable end. This covering is a plastic mushroom shaped fitting that is pushed onto the exposed cable. In the event that a cable does release from a hole, the protective cover provides a blunt face that reduces the chance of a penetrating injury. Hunt and Askew (1977) suggest that the radius of the bent wire must be kept small, 75 mm for 65 mm hole diameters, in order to maximize the frictional resistance of the anchor. A longer length provides less resistance to slippage since the wire tends to bend farther as the cable is pushed up the hole.

3.10.3 Effect of Working with Cement

Batch mixing of grout is typically done with a mechanical mixer, but the water and cement are added manually. Pouring cement into the mixer produces an excessive amount of dust in the vicinity and the use of masks are recommended. Cable bolting is frequently located in sublevel stope development that requires adequate auxiliary ventilation. Hunt and Askew (1977) note that 80% of the injuries during cable installation at one operation were the result of cement burns from the grout. Schmuck (1979) also indicates that most injuries are a result of cement burns and recommends the use of 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. The use of protective overalls and waterproof suits were commonly used by operators in western Canada. Water is normally on hand during the grouting phase and is recommended for use in the immediate washing of any areas of skin/grout contact.

3.10.4 Handling Cable Bolts

Pre-cut bolts are cut to specific lengths in the factory and shipped in 1.2 meter diameter coils. Each wrap of the cable is secured by strapping and must be cut by the bolting crew prior to cable installation. This is a dangerous process as the cable tends to whip as each strap is cut. The recommended procedure is to cut the straps in sequence while standing in the middle of the coil in order to avoid the cable whip. The cables are strapped in sequence in the factory and must be cut in the reverse sequence to limit the amount of cable released with each strap. It is important to use a cutting device that is quick and effective. Cutting with an air operated disk cutter is preferred over the use of a hacksaw. The use of leather gloves, safety glasses and a cutter protective guard are an important part of this operation. Personnel should be cleared from the area where pre-cut cable coils are being opened. The cutting of cable underground is usually not as clean as in the factory and can result in sharp ends that require special care when handling. The use of the protective cover described in Section 3.10.2 is recommended where cable must be cut underground. Utility knives are frequently used to cut grout or breather tube and open cement bags. High pressure tubing is not easily pierced with a utility knife and the use of a hacksaw is recommended as a much more effective and safer cutting method.

3.10.5 Grout Pressure

Bourchier, Dib, and O'Flaherty (1992) describe bursting of the hoses due to back pressure at the pump hose-grout tube connection. Bursting of hoses and connections have been observed and frequently reported in western Canadian mines. The direct coupling of the pump outlet hose to a grouting assembly has been suggested (Bourchier, Dib, and O'Flaherty 1992) to eliminate the safety hazard of bursting connections. The grout tube method can be used to reduce grout pressure, and some operators have adopted the use of high pressure (1.7 MPa) tubing to reduce the occurrence of hose bursting. High pressures at the hole collar generated with the breather tube method can induce forces in fractured ground that encourage the release of loose rock. Proper scaling practice at the start of each shift will minimize the occurrence of ground falls due to grouting. Plate tensioning and the insertion of bent wire anchors can also induce ground falls. After grouting is completed, the line pressure will remain high and care must be taken when disconnecting the pump hose from the grout tube.

3.10.6 Manpower

Cable bolting is typically a high turnover job since it is not often viewed as desirable work, and is usually associated with a nominal wage and bonus. Personnel are generally not veteran mine employees and have accumulated minimal training. An installation manual should be developed and combined with a practical training program. Cable bolting requires a high degree of quality control, and it is beneficial to retain experienced cable bolting personnel as long as possible. Management could consider incorporating the drilling function into the job description of the cable bolt crew as a way of providing additional drilling responsibility, and perhaps, a source of improved bonus potential. Bonus payments for cable bolting must carefully weigh productivity against performance. Cable bolt quality control is difficult to check and high productivity can often lead to a reduction in installation quality.

3.11 COSTS

A detailed review of cable bolt costs will not be pursued in this thesis but some ideas of the range in component prices is given in Table 3.6. It is recommended that a cable supplier be contacted to provide cost estimates for planning purposes. Cable bolt costs are usually included in a general ground support account. Consideration should be given to establishing a charge code to which cable bolt labour, materials, and maintenance may be assigned. A detailed financial record is then available for future cost analysis. Cable bolt costs in 1991 Canadian dollars ranged from \$19.00/m to \$36.00/m for the mines visited in this study. The average cost per meter of installed cable was approximately \$27, but different accounting structures and cable components make it difficult to compare costs directly.

3.12 DISCUSSION

This chapter has been designed to briefly review the major components that should be considered in the development of a cable support program. Observation of cable performance can be used as a guide to modify current practice. MacSporran, Bawden, and Hyett (1992) relate visual cable observations after failure to an estimate of the maximum load. Undisturbed cables

Table 3.6: Summary of Cable Component Costs (1992 Canadian Dollars)

COMPONENT	COST
Standard 15.9 mm Cable	$1.61 to $1.84 per meter
Birdcage Cable	$2.95 to 3.05 per meter
Barrel & Wedge Anchor	$4.25 to $7.50 each
Plates (152 x 152 x 9.5 mm)	$1.65 to $2.45 each
End Holding Device (ferrule installed)	$2.00 to $3.30 each
Expansion Shell Anchor	$20.00 to $40.00 each
Cable Button (installed)	$1.50 each
Grout Tube	$0.49 to $1.34 per meter
Breather Tube	$0.30 to $0.52 per meter
Spedel 6000 Pump and B3100 Mixer	$9000 each
Minepro 3 Grout Pump: Air/Hydraulic skid mounted Electric/Hydraulic skid mounted Host Hydraulic	 $29000 each $31000 each $25000 each

left hanging after a failure are indications of load carrying capacities in the range of 0 to 5 tonnes. Where unravelling of the lay occurs, the load is estimated at 5 to 15 tonnes. Loads between 15 and 25 tonnes are suggested where cable ends are pigtailed. *Pigtailing* has been coined as a descriptive term applied to failed cables that show evidence of having attained significant load carrying capacity prior to failure. The term is analogous to the effect of curling a strip of paper by pulling it through ones fingers. Pigtailed cables are typically unravelled and the individual wires are curled. This can be a useful measure of cable load, but observations in this study have occasionally indicated the presence of pigtailing with no apparent rock failure. This is typically encountered in hangingwalls where the action of blasted ore moving towards the drawpoint can result in pigtailed cables. The only true measure of attaining the maximum load carrying capacity is observation of steel failure. Frequent observation of undisturbed cables after a failure situation is evidence of the need to revise current practice. Design applications will be introduced in Chapter 4 and should be complemented with observations of cable bolt performance.

CHAPTER 4

CABLE BOLT DESIGN METHODS

4.1 INTRODUCTION

This chapter will review cable bolt design methods with particular reference to western Canadian practice. Discussions with different mine personnel have indicated that there is no standard cable bolt design procedure used in western Canadian mines, but there is a strong desire for some consistent criteria. The flowchart in Figure 4.1 proposes a methodology of cable design that incorporates current practice and will be used later in this thesis to propose revisions. The flowchart is split between discrete and collective analysis, reflecting a distinction between associated design techniques. Discrete analysis is applied to cases of isolated blocks or structure that require support. The discrete design method has been well defined by several authors (Hoek and Brown 1980, 246-248; Stillborg 1986, 58-62) and will be briefly reviewed in this chapter. The collective analysis segment of the flowchart deals with the in-situ rock mass and incorporates design methods that reflect this approach. One of the most important considerations on both sides of the flowchart is access. In this study, only 2% of the supported case studies involved development that was specifically designed for the implementation of support. With this in mind, it is important to consider cable design in relation to the available access and the associated patterns that can result. The collective analysis approach incorporates a number of design techniques reflecting the rock mass as a continuum. These techniques are frequently applied in close collaboration with each other to obtain a final design. An economic analysis is normally performed and may indicate a review of the entire process prior to implementation. A successful design and economic analysis leads to implementation of the support system. The support system

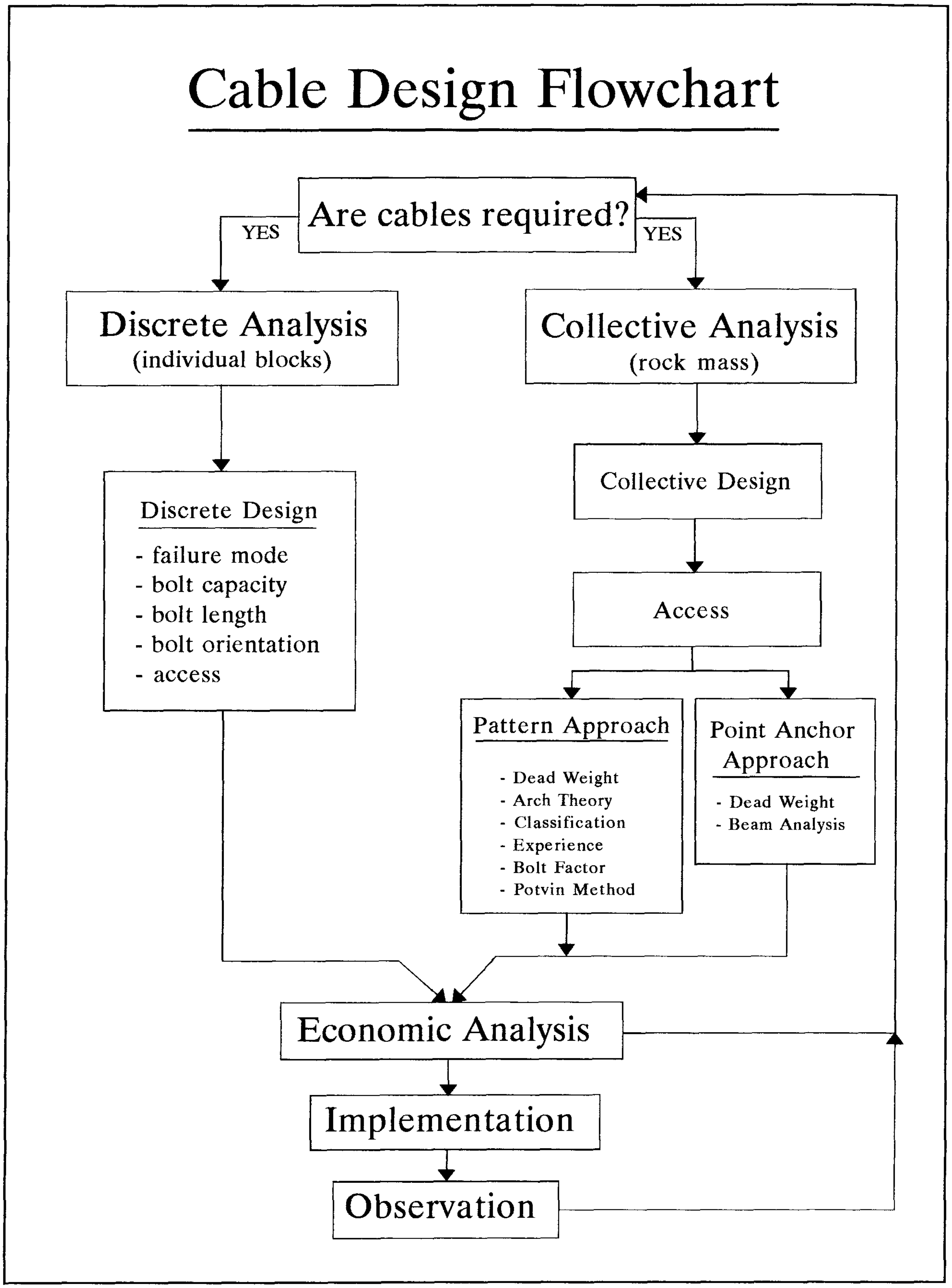


Figure 4.1: Cable design flowchart

performance is then observed and serves as input into subsequent design.

4.2 DISCRETE ANALYSIS

Discrete analysis is reviewed by Hoek and Brown (1980, 246-248) in relation to a specific block or wedge that is either free to fall or slide. The bolt load required to support a block that is free to fall is given by

$$N = \frac{W \times F}{T} \tag{4.1}$$

where N = number of bolts

W = weight of wedge

T = bolt load

and F = factor of safety.

For cases where the block is free to slide, as shown in Figure 4.2, the frictional resistance of the sliding surface must be considered using the following relation,

$$F = \frac{cA + (W\cos\psi + T\cos\theta)\tan\phi}{W\sin\psi - T\sin\theta} \tag{4.2}$$

where c = sliding surface cohesive strength

A = surface area of the sliding surface

ψ = dip of the sliding surface

θ = angle between bolt and normal to the sliding surface

and ϕ = friction angle of the sliding surface.

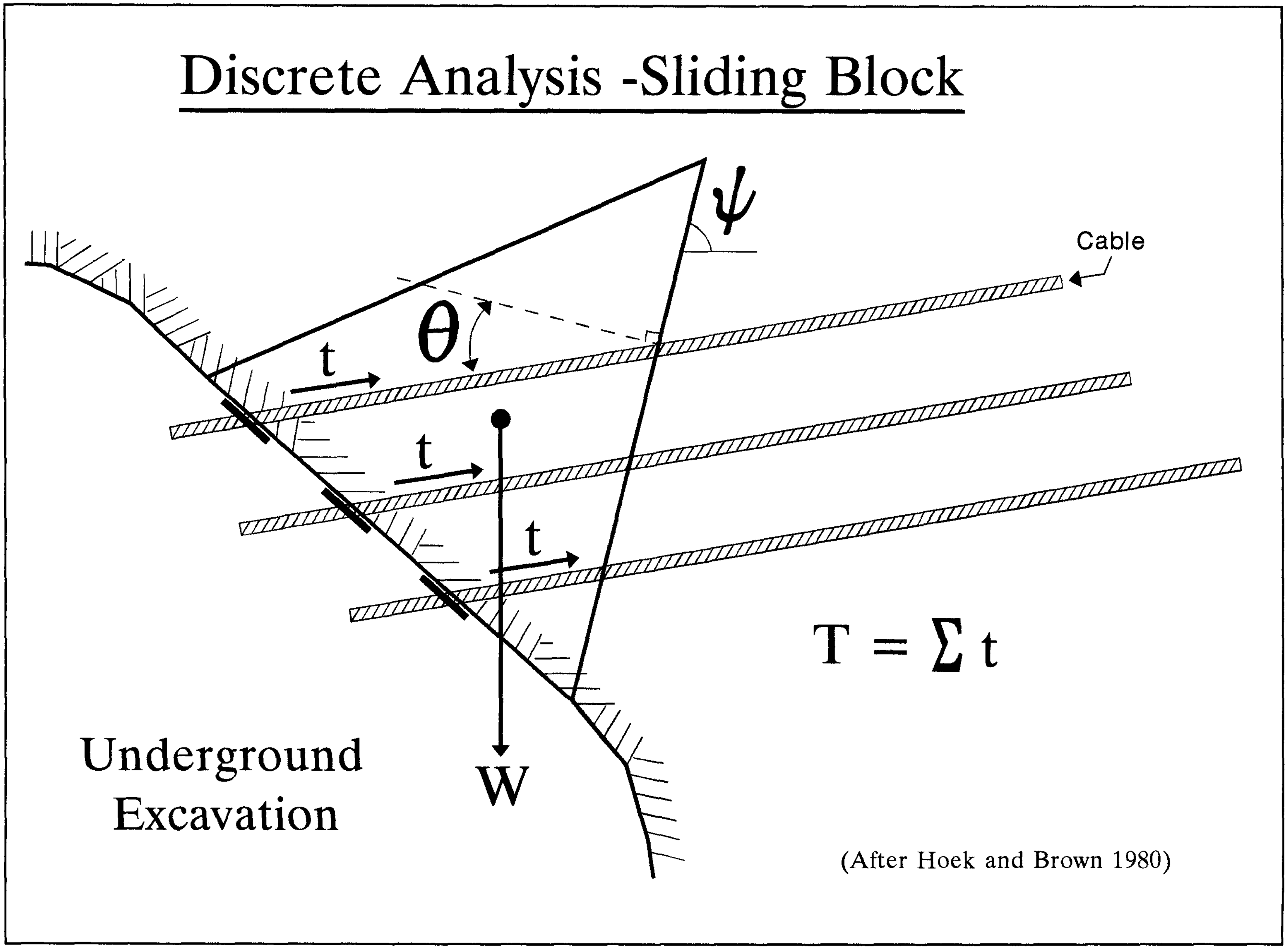


Figure 4.2: Sliding block discrete analysis (After Hoek and Brown 1980)

Equation 4.2 can be rearranged to determine the required bolt load as shown in Equation 4.3.

$$T = \frac{W(Fsin\psi - \cos\psi\tan\phi) - cA}{\cos\theta\tan\phi + Fsin\theta} \tag{4.3}$$

Hoek and Brown (1980, 247) recommend that a factor of safety of 1.5 be used where grouted bolts or cables are used, and bolt length be based on suitable anchorage beyond the block boundaries. Where sliding occurs along two planes, Hoek and Brown (1980, 248) suggest that the dip of the line of intersection be used in the support analysis. Discrete analysis is dependent on good structural definition of the block boundaries in order to estimate the block weight. The bolt load used in equation 4.2 to determine a factor of safety may not relate to the cable breaking strength unless the critical embedment length is realized, or plates are incorporated at the hole collar. Individual cable capacity used in equation 4.2 should reflect the embedment length and grout water:cement ratio. Miller (1984) has shown that the most efficient shear resistance is offered by cables oriented parallel to the shear direction and inclined from 17° to 27°.

4.3 COLLECTIVE ANALYSIS

Helping the rock mass to support itself is described by Hoek and Brown (1980, 244) as the principal objective of underground support. Lang (1961) referred to rockbolting as "the designed use of rock bolts to reinforce and develop 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. Fuller (1981) describes the concept of pre-reinforcement that was successfully adapted to initial cable bolting applications. Experience with rockbolting had demonstrated that pattern support was most successful where installation occurred immediately after mining. With cable bolts, it was possible to install support prior to mining and take advantage of the inherent rock mass strength by limiting joint dilation. Most cable design methods have been developed on the basis of an even distribution of bolts on a regular square pattern. Where access is restricted, an even pattern may not be possible, and bolts are frequently installed in a high density fan to act as a point anchor. The point anchor approach differs from pattern bolting as large areas of the supported surface are left unsupported. Discrete analysis has been described in terms of isolated structure, and is specific to a particular situation. This thesis is concerned with a collective design approach that has the ability to reflect characteristics of the entire rock mass. This section will review pattern and point anchor collective design methods that were encountered in practice

4.3.1 Dead Weight Design

Dead weight design was encountered when cables were required to support the dead weight of a rock mass. This method involves the estimation of the tonnage of rock to be supported, and the use of a factor of safety to determine the number of bolts required, as reviewed in Section 4.2. The number of bolts are typically distributed evenly over the area involved, and extend 1 to 4 meters beyond the projected failure plane. Where the thickness of the supported area is variable, the bolt pattern should be adjusted to reflect higher loads in thicker areas. A typical factor of safety of 1.2 was incorporated in the final design and occasionally the bolt tensile strength was reduced if the cable orientation was not vertical. All of the designs utilizing a dead weight analysis in this study, assumed that the load carrying capacity of the cable was equivalent to the steel tensile strength.

The 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 a area of the rock mass. Stillborg (1986, 66-68) describes the formation of a natural arch above an opening due to stress redistribution as the opening is created. A zone of 'loose' rock below the arch is subject to instability, and can be used to define the dead weight load that cables must support. If the arch area is assumed to approximate a triangular shape, the bolt pattern can then be calculated using the following relationship,

$$S^2 = \frac{2T}{F \times H \times \gamma} \tag{4.4}$$

where T = bolt load in tonnes

S^2 = bolt square pattern (S x S) in meters

H = height of relaxed zone below arch

and γ = unit weight of the rock mass in tonnes/meter^3.

This method assumes that the full bolt tensile strength is utilized, and there is competent ground above the relaxed zone. The height of the natural arch formed, H, can be determined through numerical modelling or through the use of empirical formulations that relate the bolt length to the opening span. Bolt length is typically determined by designing for at least 4 meters of anchorage into the natural arch. Stheeman (1982) describes a procedure used at the Tsumeb Mine to mathematically approximate the curvature of a natural arch above a cut and fill stope. The curved shape of fracture planes forming the walls of a cavity after a rock fall, indicated the limits of the self-supporting arch. Fracture plane angles were noted at different positions and used to derive an elliptical relationship that supported the observations. The 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 natural 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 natural arch, the bond length required to support the weight below the arch, and an allowance for grout bleed.

4.3.2 Rock Classification

Several methods of rock mass classification have been developed to assess the quality of a rock mass and estimate stand-up time or support requirements. The Q-system of rock mass classification (Barton, Lien, and Lunde 1974) describes the rock mass in terms of six parameters as follows,

$$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF} \tag{4.5}$$

where Q = rock mass quality

RQD = rock quality designation

Jn = joint set number

Jr = joint roughness number

Ja = joint alteration number

Jw = joint water reduction factor

and SRF = stress reduction factor.

A full description of each parameter is given in Table 4.1, but a review of the original text (Barton, Lien,and Lunde 1974) is recommended. The rock quality designation, or RQD, is defined as the percentage of core recovered in intact lengths greater than 100 mm (Hoek and Brown 1980, 18). The rock mass quality, Q, increases on a logarithmic scale from 0.001 to 1000 with increasing rock quality. In the initial Q-system proposal, Barton, Lien, and Lunde (1974)

Table 4.1: Description of Q-system Parameters (Barton, Lien, and Lunde 1974)

1.	ROCK QUALITY DESIGNATION	(RQD)	
A.	Very Poor	0-25	Note: (i) Where RQD is reported or measured as ≤ 10 (including 0) a nominal value of 10 is used to evaluate Q. (ii) RQD intervals of 5, i.e. 100, 95, 90, etc. are sufficiently accurate
B.	Poor	25-50	
C.	Fair	50-75	
D.	Good	75-90	
E.	Excellent	90-100	
2.	JOINT SET NUMBER	(Jn)	
A.	Massive, no or few joints	0.5-1.0	Note: (i) For intersections use (3.0 x Jn) (ii) For portals use (2.0 x Jn)
B.	One joint set	2	
C.	One joint set plus random	3	
D.	Two joint sets	4	
E.	Two joint sets plus random	6	
F.	Three joint sets	9	
G.	Three joint sets plus random	12	
H.	Four or more joint sets, random, heavily jointed, "sugar cube like",etc.	15	
J.	Crushed rock, earthlike	20	
3.	JOINT ROUGHNESS NUMBER	(Jr)	
	(a) Rock wall contact and (b) Rock wall contact before 10 cms shear		
A.	Discontinuous joints	4	Note: (i) Add 1.0 if the mean spacing of the relevant joint set is greater than 3 m (ii) Jr=0.5 can be used for planar slickensided joints having lineations, provided the lineations are favourably orientated
B.	Rough or irregular, undulating	3	
C.	Smooth, undulating	2	
D.	Slickensided, undulating	1.5	
E.	Rough or irregular, planar	1.5	
F.	Smooth, planar	1.0	
G.	Slickensided, planar	0.5	
	(c) No rock wall contact when sheared		
H.	Zone containing clay minerals thick enough to prevent rock wall contact	1.0 (nominal)	
J.	Sandy, gravelly or crushed zone thick enough to prevent rock wall contact	1.0 (nominal)	

Table 4.1 (con't): Description of Q-system Parameters (Barton, Lien, and Lunde 1974)

4.	JOINT ALTERATION NUMBER	(Ja)	φ_r (approx.)	
	(a) Rock wall contact			
A.	Tightly healed, hard, non-softening, impermeable filling i.e. quartz or epidote	0.75	(-)	Note: (i) Values of $(\varphi)_r$, the residual friction angle, are intended as an approximate guide to the mineralogical properties of the alteration products, if present Note: (i) For intersections use (3.0 x Jn) (ii) For portals use (2.0 x Jn)
B.	Unaltered joint walls, surface staining only	1.0	$(25^0\text{-}35^0)$	
C.	Slightly altered joint walls. Non-softening mineral coatings, sandy particles, clay-free disintegrated rock etc.	2.0	$(25^0\text{-}30^0)$	
D.	Silty-, or sandy-clay coatings, small clay-fraction (non-softening)	3.0	$(20^0\text{-}25^0)$	
E.	Softening or low friction clay mineral coatings, i.e. kaolinite, mica. Also chlorite, talc, gypsum and graphite etc., and small quantities of swelling clays. (Discontinuous coatings, 1-2 mm or less in thickness)	4.0	$(8^0\text{-}16^0)$	
	(b) Rock wall contact before 10 cms shear			
F.	Sandy particles, clay-free disintegrated rock etc.	4.0	$(25^0\text{-}30^0)$	
G.	Strongly over-consolidated, non-softening, clay mineral fillings. (Continuous, < 5 mm in thickness)	6.0	$(16^0\text{-}24^0)$	
H.	Medium, or low over-consolidation, softening, clay mineral fillings. (Continuous, < 5 mm in thickness)	8.0	$(12^0\text{-}16^0)$	
J.	Swelling clay fillings, i.e. montmorillonite (Continuous, <5 mm in thickness). Value of Ja depends on percent of swelling clay-size particles, and access to water etc.	8.0-12.0	$(6^0\text{-}12^0)$	
	(c) No rock wall contact when sheared			
K,L, M.	Zones or bands of disintegrated or crushed rock and clay (see G, H, J for description of clay condition)	6.0, 8.0 or 8.0-12.0	$(6^0\text{-}24^0)$	
N.	Zones or bands of silty- or sandy-clay, small clay fraction (non-softening)	5.0		
O,P, R.	Thick, continuous zones or bands of clay (see G,H,J for description of clay condition)	10.0, 13.0 or 13.0-20.0	$(6^0\text{-}24^0)$	
5.	JOINT WATER REDUCTION FACTOR	(Jw)	Approx. water pressure (kg/cm^2)	
A.	Dry excavations or minor inflow, i.e. , 5 l/min. locally	1.0	<1	Note: (i) Factors C to F are crude estimates. Increase Jw if drainage measures are installed. (ii) Special problems caused by ice formation are not considered
B.	Medium inflow or pressure, occasional outwash of joint fillings	0.66	1.0-2.5	
C.	Large inflow or high pressure in competent rock with unfilled joints	0.5	2.5-10.0	
D.	Large inflow or high pressure, considerable outwash of joint fillings	0.33	2.5-10.0	
E.	Exceptionally high inflow or water pressure at blasting, decaying with time	0.2-0.1	>10.0	
F.	Exceptionally high inflow or water pressure continuing without noticeable delay	0.1-0.05	>10.0	

Table 4.1 (con't): Description of Q-system Parameters (Barton, Lien, and Lunde 1974)

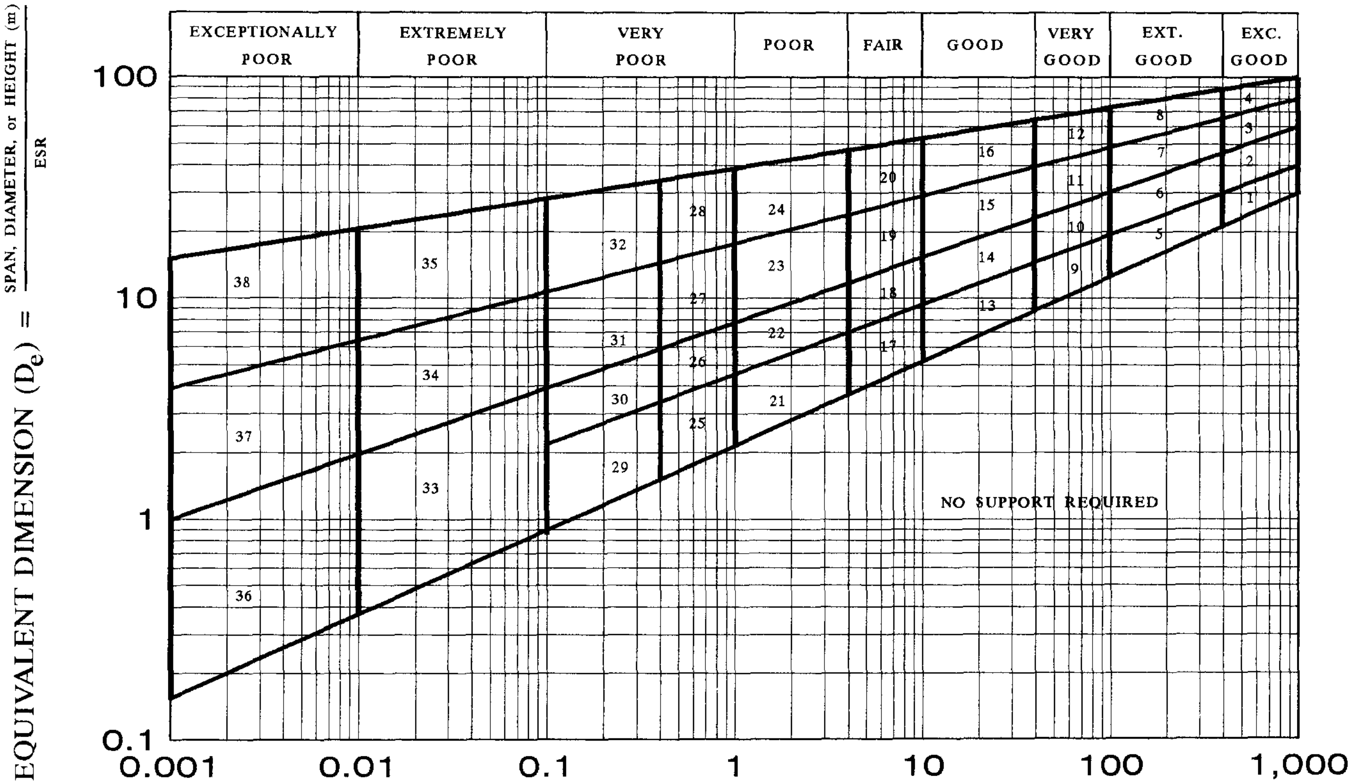
6.	STRESS REDUCTION FACTOR			(SRF)	
	(a) Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated				Note: (i) Reduce these values of SRF by 25-50% if the relevant shear zones only influence but do not intersect the excavation
A.	Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth)			10.0	
B.	Single weakness zones containing clay, or chemically disintegrated rock (depth of excavation ≤ 50 m)			5.0	
C.	Single weakness zones containing clay, or chemically disintegrated rock (depth of excavation > 50 m)			2.5	
D.	Multiple shear zones in competent rock (clay free), loose surrounding rock (any depth)			7.5	
E.	Single shear zones in competent rock (clay free) (depth of excavation ≤ 50 m)			5.0	
F.	Single shear zones in competent rock (clay free) (depth of excavation > 50 m)			2.5	
G.	Loose open joints, heavily jointed or "sugar cube" etc. (any depth)			5.0	
	(b) Competent rock, rock stress problems	σ_c/σ_1	σ_t/σ_1		
H.	Low stress near surface	>200	>13	2.5	
J.	Medium stress	200-10	13-0.66	1.0	
K.	High stress, very tight structure (Usually favourable to stability, may be unfavourable to wall stability)	10-5	0.66-0.33	0.5-2.0	(ii) For strongly anisotropic stress field (if measured): when $5 \leq \sigma_1/\sigma_3 \leq 10$, reduce σ_c and σ_t to $0.8\sigma_c$ and $0.8\sigma_t$; when $\sigma_1/\sigma_3 > 10$, reduce σ_c and σ_t to $0.6\sigma_c$ and $0.6\sigma_t$ where: σ_c = unconfined compressive strength, σ_t = tensile strength (point load), σ_1 and σ_3 = major and minor principal stresses (iii) Few case records available where depth of crown below surface is less than span width. Suggest SRF increase from 2.5 to 5 for such cases (see H)
L.	Mild rock burst (massive rock)	5-2.5	0.33-0.16	5-10	
M.	Heavy rock burst (massive rock)	<2.5	<0.16	10-20	
	(c) Squeezing rock; plastic flow of incompetent rock under the influence of high rock pressures				
N.	Mild squeezing rock pressure			5-10	
O.	Heavy squeezing rock pressure			10-20	
	(d) Swelling rock; chemical swelling activity depending on presence of water				
P.	Mild swelling rock pressure			5-10	
R.	Heavy swelling rock pressure			10-15	

recommend different support ranges as illustrated in Figure 4.3. In this chart, the equivalent dimension, D_e, is related to the rock mass quality, Q. The equivalent dimension relates the size of an excavation to its purpose, and is defined as the span, diameter, or height of the opening divided by the excavation support ratio, ESR. The span or diameter is used in relation to back support, and height is related to wall support. The excavation support ratio is higher for mine openings than for underground civil chambers, and reflects the increased tolerance for instability in underground mines. Barton, Lien, and Lunde (1974) suggest an ESR of 3 to 5 for temporary mine openings, 1.6 for permanent mine openings, and 1.0 for major road or railway tunnels. Each numbered region in Figure 4.3 represents a different level of support, and generally reflect increasing support requirements with higher numbers. Several regions in Figure 4.3 recommend untensioned, grouted pattern support, which has been related to a cable bolt pattern design. The design length is typically based on extending past some critical feature with suitable anchorage. It is important to note that the support recommendations from this method are based on a large database, but only two cases are noted for temporary mine openings. The majority of the case studies were collected from permanent mine or civil excavations, and this method can only be recommended for an initial pass at cable bolt design for mining blocks.

The Geomechanics Classification of Jointed Rock Masses (Bieniawski 1976) is a method of classifying rock for engineering purposes through the estimation of a rock mass rating, or RMR. The RMR is a rating between 0 and 100 that is based on the intact rock strength, drill core RQD, joint spacing, joint condition, and the presence of groundwater. The classification system is described in detail in Hoek and Brown (1980, 22-27) and has been related to rock support by Bieniawski (1976). It has not been directly applied in this thesis since existing methods of cable design are associated with the Q-system of rock classification. Bieniawski (1976) proposed the following relationship between Q and RMR based on the regression analysis

Tunnel Support Chart

(After Barton, Lien, and Lunde 1974)



ROCK MASS QUALITY $Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF}$

Figure 4.3: Tunnel Support Chart (After Barton, Lien, and Lunde 1974)

of a series of case histories,

$$RMR = 9\ln Q + 44. \tag{4.6}$$

Equation 4.6 can be used to convert between Q and RMR but the large data scatter (Bieniawski 1976) around this line should be noted.

4.3.3 Beam Theory

Beam theory has been applied to cable bolt design where structure is parallel to the surface to be supported. This is most commonly applied in a layered rock mass, where cables are used to tie a number of layers together into a stable beam. The design method is similar to the arch concept discussed in Section 4.3.1, and is based on using the dead weight of the beam to determine cable spacing. The volume of rock is approximated as a rectangular area and equation 4.4 becomes,

$$S^2 = \frac{T}{F \times L \times \gamma} \tag{4.7}$$

where L = the beam thickness.

Fuller (1983b) describes a method of cable design for a point anchor cable pattern that is based on beam theory. A hangingwall in layered rock was assumed to behave like a beam, as illustrated in Figure 4.4. Cable fans installed into the hangingwall at two intermediate sublevels are simulated by reactions, R_c. The stope abutments were fixed, but the analysis allowed for some beam deflection at the intermediate points to represent the true action of cables taking load. In this analysis, Fuller (1983b) assumed that the beam experienced uniformly distributed loading

Beam Approach to Hangingwall Design

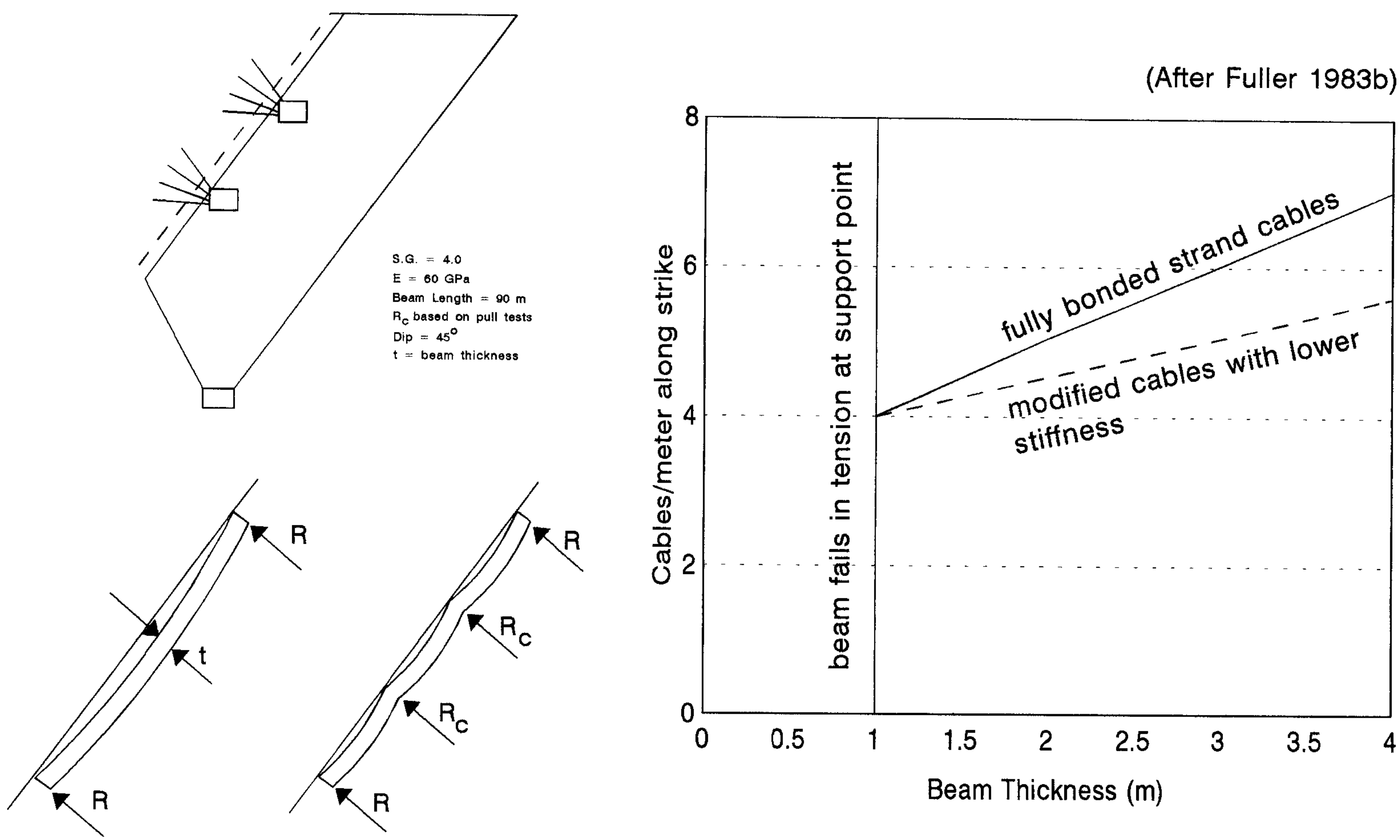


Figure 4.4: Beam approach to hangingwall design (After Fuller 1983b)

based on a rock specific gravity of 4.0, and related the cable reactions to laboratory pull test curves for different bond lengths of 15.2 mm diameter double cables. The results of this analysis relate the number of cables required along strike to the beam thickness, as illustrated in Figure 4.4. A minimum bolt density of 4 cables for each meter of strike length was determined for this particular stope. Beams less than one meter in thickness were found to fail in tension. Cases of hangingwall support collected from western Canadian experience typically reflect major structure parallel to the surface. In most cases however, there is usually a second minor joint set that encourages failure between sublevels, and limits the success of point anchor bolting.

4.3.4 Past Experience

Past experience was the most common design method encountered in Western Canada. This method relies on an initial trial and error process with subsequent adjustments based on performance. In many cases, the initial design is based on typical patterns that are used at other operations. Observation of support performance establishes a database of internal case histories, that can then be used to modify cable design. One operation used past failure situations to determine the opening span at which cable support was required. Larger spans dictated the use of a tighter cable pattern. This method of design was most noticeable where cut and fill mining was in progress. The cables would be installed at lengths of up to 18 meters to cover several mining lifts. In these cases, the performance of a cable pattern can be evaluated fairly easily, since cut and fill lifts are mined rapidly relative to open stoping situations. This approach was also encountered to design open stope back support, although the initial pattern selection was much more critical. Modifications to the installation after mining had begun were difficult if not impossible. Cables were required in the back if the span exceeded an amount determined by experience. This critical span was reduced if the quality of the rock mass deteriorated. In some cases, the cable bolt spacing was determined by the blasthole ring burden, so that bolt holes could be drilled from existing set up points.

Instrumentation can be used to record support performance and develop design criteria. Fuller (1981) describes the monitoring of cables in the immediate back of a cut and fill stope. The strain distribution immediately after a blast, indicated a tensile load in the first 5 meters of cable above the back. Approximately one hour after the blast, the tensile load range decreased to 2 meters from the stope back, and conformed the opinion that cable support helps the rock mass to support itself. Similar observations resulted from an instrumented 2 m x 2 m cable grid, but the ground became self-supporting within minutes of the blast. Based on this experience, Fuller (1983) recommended a 2 m x 2 m pattern for design with a minimum cable overlap of 2 meters. Choquet and Miller (1988) describe the development of a tension measuring device specifically designed for cable bolts. The technique involves the measurement of resistance variation in a thin wire that is wound along a cable strand, and isolated from the grout. Measurements using this device can provide an estimate of cable strain and actual load carrying capacity. It is however highly dependent on location with respect to dilation within the rock mass. Since the wire in the tension measuring device is free along its entire length, cable strain may be significantly higher than the measuring device might indicate.

Hunt and Askew (1977) describe the use of empirical design guidelines developed by the Snowy Mountains Authority for use in the design of cable patterns for wide permanent openings. In this situation the bolt length, L, was determined by the minimum of Equation 4.8,

$$L = 1.83 + 0.013D^2 \tag{4.8}$$

or three times the spacing of the smallest joint set, where D represents the opening span in meters. Stillborg (1986, 69) describes empirical design recommendations formulated by the U.S Corps of Engineers for use in rockbolt design. These are occasionally adapted for use in cable design and suggest that the minimum bolt length should the greatest of:

a) 2 x the bolt spacing

b) 3 x the thickness of critical and potentially unstable blocks

c) For spans less then 6 m: 0.5 x span
For spans between 18 and 30 m: 0.25 x span
For spans between 6 and 18 m: interpolate between 3 and 4.5 meter lengths.

The spacing is based on half the bolt length or 1.5 times the width of critical and potentially unstable blocks, up to a maximum of 2 meters. There are many similar empirical design recommendations, but most are specifically based on rockbolting experience in tunnelling applications. The applicability of these design criteria to cable bolting is limited, but they are useful as initial guidelines.

4.3.5 The Mathews Method and Bolt Factor

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

$$N = Q' \times A \times B \times C \tag{4.9}$$

where Q' is the Q-system rock mass rating with the stress reduction factor set to one

A is the stress factor,

B is the rock defect orientation factor,

and C is the design surface orientation factor.

The shape factor is also called the hydraulic radius, and is determined by dividing the surface area by the perimeter. It was originally used in mining applications by Laubscher (1976) to relate the undercut area of a stoping block to cavability. 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 *length* refers to the maximum dimension. A traditional longitudinal stoping sequence relates length to the distance along strike, and span to the orebody width. Hydraulic radius considers both the span and length of a particular surface. For square openings, hydraulic radius is one fourth of the span, but as the ratio of span to length decreases, the hydraulic radius converges to half the span. Mathews et al. (1981) proposed a design chart (Figure 4.5) that relates the stability number to the shape factor, and defines zones of stability, potential instability, and potential cave. These zones are further described as follows,

(i) Stable - the excavation will stand unsupported with occasional localized ground support to control slabbing.

(ii) Unstable - the excavation will experience localized caving but tend to form a stable arch. Cable bolts and modification of the extraction sequence are suggested as ways to make open stoping feasible in this region.

(iii) Caved - the excavation will not stabilize until the void is full.

The factors A, B, and C are presented graphically on a series of charts in Figure 4.6. The rock stress factor, A, replaces the SRF in the Q-system and is related to σ_c/σ_i, 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 the 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. Structure perpendicular to the surface reflects the most favourable orientation, and is given the highest rating. The design surface orientation factor, C, is based on the assumption that

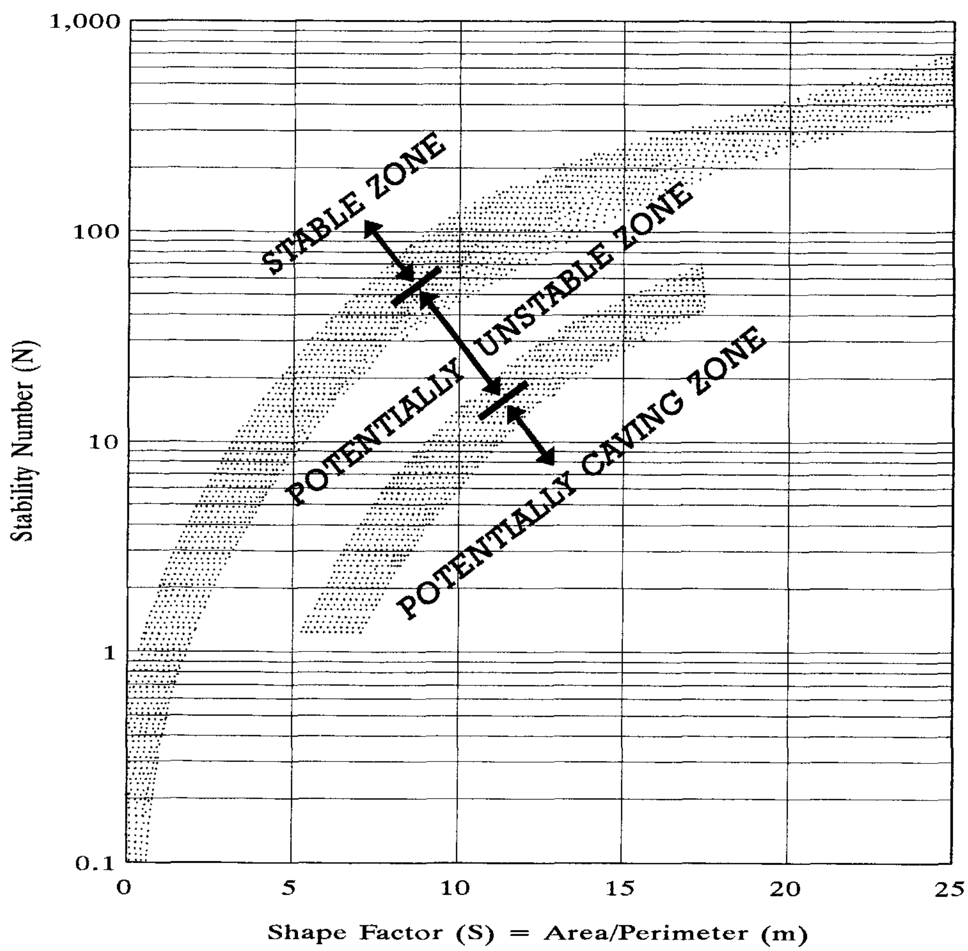


Figure 4.5: Stability graph proposed by Mathews et al. (1981)

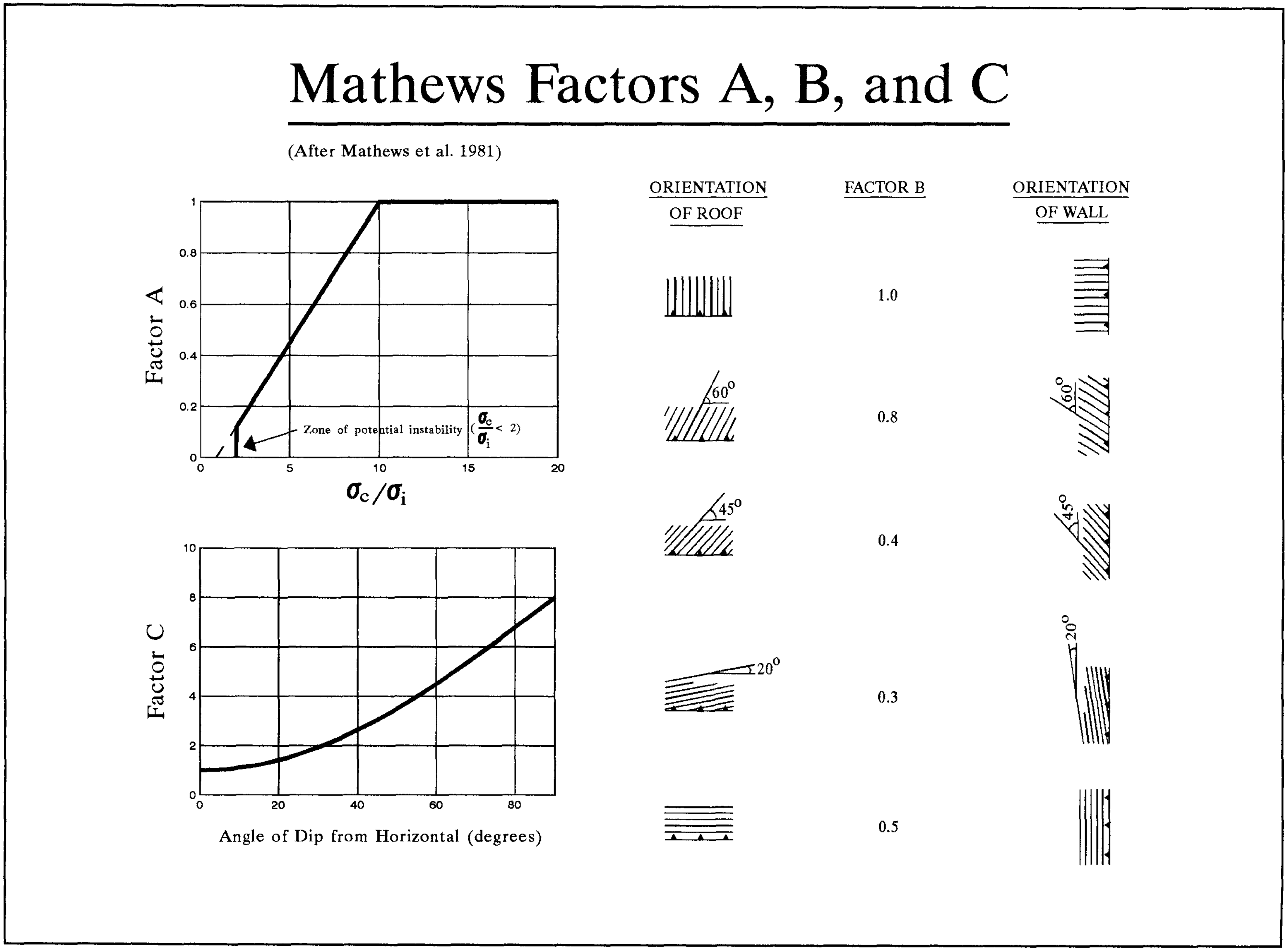


Figure 4.6: Factors A, B, and C proposed by Mathews et al. (1981)

a vertical wall is eight times as stable as a horizontal surface under the effects of gravity.

Bawden et al. (1989) present an attempt to modify rock classification results to reflect the installation of cable support. The bolt factor, BF, is determined by dividing the total length of installed cables by the area of the supported surface. The bolt factor is then related to unsupported and supported Q' values as shown in Figure 4.7. It is important to note that the chart in Figure 4.7 represents a proposed design method that has received little practical calibration, and therefore is not recommended for design. However, the idea of increasing rock mass quality to account for support is a valuable concept. The initial phase of the bolt factor approach, involves completing a stability analysis for the stope surface using the technique developed by Mathews et al. (1981). If the surface plots within the stable zone, cable support is not required. If potential instability or caving are indicated by the stability analysis, a bolt factor is selected to estimate a supported Q' value using Figure 4.7. The selection of the bolt factor is based on establishing a supported Q' that can be related to the prediction of a stable surface in the original stability analysis.

4.3.6 Design Based on Rock Mass Stiffness

Fuller, Dight, and West (1990) describe a design technique that was developed under the coordination of the Australian Mineral Industries Research Association (AMIRA). The method relates the rock mass deformation modulus to cable load-displacement curves obtained from laboratory testing. Section 2.8 discussed an increase in cable load carrying capacity with increasing rock mass stiffness. The Mathews method is used to evaluate initial stability and the need for additional support. The in situ rock mass modulus is estimated from a relationship with the Q classification system described by Hoek and Brown (1980). The size and displacement of a potential failure zone is estimated from numerical modelling or past experience. The specific

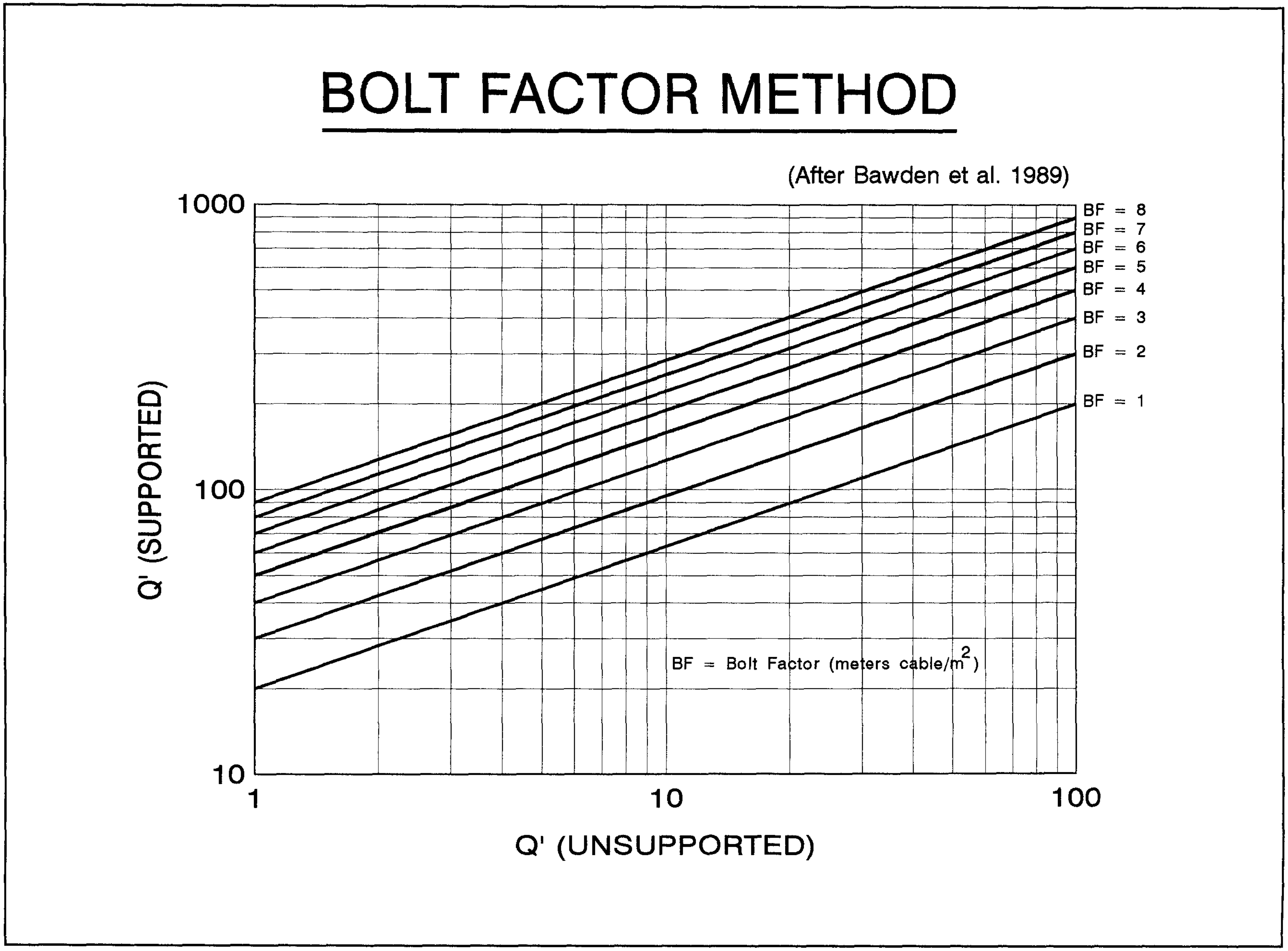


Figure 4.7: Bolt factor design chart (After Bawden et al. 1989)

load-displacement curve for a particular rock modulus provides an estimate of the cable load carrying capacity that can be used for design. The load carrying capacity is then related to the size of the failure zone to determine the number of cables required. A factor of safety of between 1.2 and 2.0 is suggested and cable length is based on extending 3 meters beyond the deepest point of any potential failure. This method has not been widely used in Canadian practice due to the difficulty of estimating the size and allowable displacement of potential failure zones.

Reichert, Bawden and Hyett (1992) suggest a design method that is based on the relationship between embedment length and radial stiffness illustrated in Figure 4.8. The solid line in Figure 4.8 is based on a linear extrapolation of the relationships between load and embedment length in Figure 2.14. The extrapolated values of embedment length are taken at 24 tonnes to reflect an optimum load carrying capacity. This relationship is based on laboratory testing at a 0.30 water:cement ratio and additional curves were estimated for higher ratios. In terms of design, an estimate of radial stiffness and embedment length will provide an indication of expected cable performance.

4.3.7 The Potvin Method

The *Modified Stability Graph* (Potvin 1988) was developed empirically from the analysis of 175 case histories of open stoping situations in Canada. Based on these case studies and the concepts behind the Mathews method, a stable and a caving zone were identified by relating a modified stability number, N', to the hydraulic radius, HR, of the surface. The value of the stability number is determined in a similar procedure to the Mathews method (Mathews et al. 1981), but the term N' is used to signify the use of slightly different A, B, and C factors. Potvin (1988) suggested that the first stage of the cable bolt design process was to complete a stability analysis to determine the expected stability condition of the surface. A zone within the caving

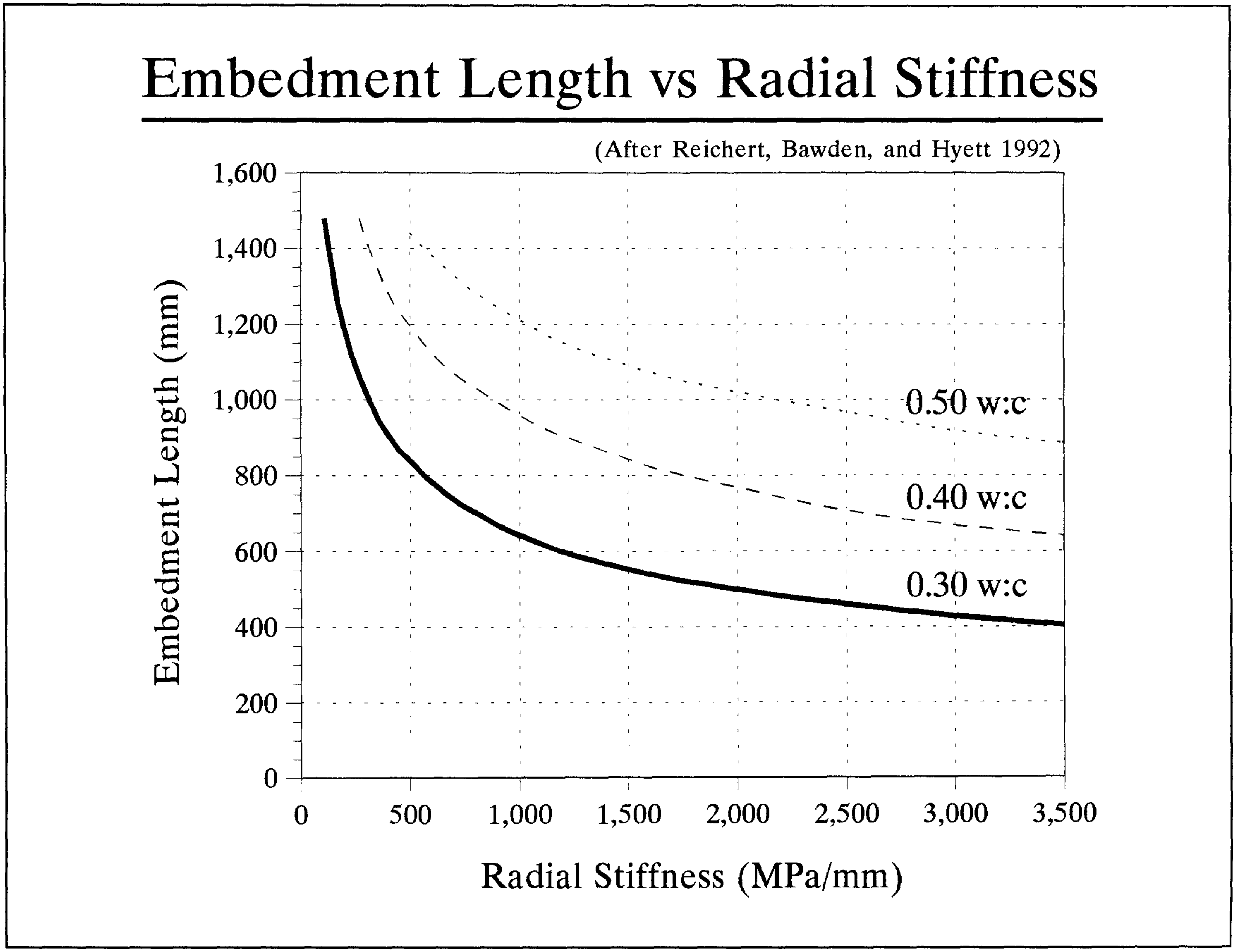


Figure 4.8: Relationship between radial stiffness, embedment length, and water:cement ratio required to mobilize 24 tonnes of cable load carrying capacity (After Reichert, Bawden, and Hyett 1992).

region was identified on the *Modified Stability Graph* to be stable with the addition of cable support. Assuming that cable support was required, the design density could be determined from the *Design Chart for Cable Bolt Density* (Potvin 1988). This chart relates the cable density to a relative block size factor, RQD/Jn/HR, and was developed based on the analysis of 66 case histories of stope backs that were supported with cable bolts. The *Modified Stability Graph* was found to be used at about half of the operations visited. The *Design Chart for Cable Bolt Density* was rarely used for final design, but was consulted in some cases prior to determining the desired pattern. Chapter 5 will discuss the Potvin Method in more detail.

Table 4.2: Design Methods in Western Canadian Practice

Dead Weight	9%
Rock Classification	4%
Beam Theory	4%
Past Experience	71%
Bolt Factor	5%
Potvin Method	7%

4.4 WESTERN CANADIAN PRACTICE

Fifty-six design applications were reviewed during the mine visits conducted in this study. A summary of the different design methods encountered is given in Table 4.2. Experience was found to be the most frequent design method and reflects the lack of acceptable guidelines for cable support. A combination of several design methods is often used to develop a cable support proposal.

CHAPTER 5

THE POTVIN METHOD

5.1 INTRODUCTION

This chapter will review the method of stope design that was proposed by Yves Potvin in his PhD thesis at the University of British Columbia (Potvin 1988). Forty mines were visited in 1986 and 1987, with the objective of back analyzing the stability condition of open stopes to create a database, and subsequently develop a model for stope design. The model database, assembled from 34 mines, consisted of 176 case histories of unsupported stope surfaces and 66 supported surfaces. Five stages of development were involved in creating the model. First, geotechnical parameters were assembled from mine visits. These parameters were subsequently expressed in terms of factors to form a geotechnical model. The Mathews method (Mathews et al. 1981) was chosen as the best method of design analysis since it is based on case histories of open stoping situations. Rock mass performance was related to the stability condition of the surface, and the final stage was to calibrate the effect of each factor by back analysis.

5.2 THE *MODIFIED STABILITY GRAPH*

The Potvin (1988) case histories were divided into a main and complementary database. The main database contained 84 case histories collected during visits to different mining operations. The complementary database contained 92 case histories that were collected from literature or involved some uncertainty in one or more parameters. A summary of the main and complementary Potvin database can be found in Tables 5.1 and 5.2 respectively.

Table 5.1: Summary of Potvin Unsupported Main Database

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability
1	HW	5.0	54.0	1.0	0.65	6.5	228	Stable
2	Wall	8.9	6.0	0.2	0.25	2.5	0.7	Unstable
3	Wall	7.7	6.0	0.1	0.2	2.5	0.3	Caved
4	HW	7.1	10.5	1.0	0.2	3.7	7.8	Unstable
5	HW	14.0	40.0	1.0	1.0	8.0	320	Stable
6	HW	11.0	40.0	1.0	1.0	8.0	320	Stable
7	HW	5.2	40.0	1.0	1.0	6.5	260	Stable
8	HW	8.5	9.0	1.0	0.4	5.0	18	Stable
10	End	4.7	3.2	0.3	0.2	3.5	0.7	Unstable
12	HW	9.1	4.2	1.0	0.2	6.5	5.5	Unstable
13	HW	8.3	30.0	1.0	0.2	7.0	42	Stable
16	Back	5.8	6.25	0.1	0.85	2.0	1.1	Caved
17	Back	4.2	6.25	0.1	0.85	2.0	1.1	Stable
18	HW	8.8	30.0	1.0	0.6	8.0	144	Stable
19	Back	3.5	30.0	0.1	0.4	2.0	2.4	Unstable
20	Back	1.8	16.5	1.0	0.2	2.0	6.6	Stable
21	HW	4.7	16.5	1.0	0.2	4.5	15	Stable
22	HW	8.8	16.5	1.0	0.2	4.5	15	Stable
23	Back	2.1	16.5	1.0	0.2	2.0	6.6	Stable
24	Back	10.5	34.0	1.0	0.2	2.0	14	Caved
25	Back	11.3	34.0	1.0	0.2	2.0	14	Caved
26	Back	12.2	34.0	1.0	0.2	2.0	14	Caved
27	Back	4.1	34.0	1.0	0.2	2.0	14	Stable
28	Wall	7.6	12.0	1.0	0.3	2.0	6.9	Stable
29	Wall	7.6	34.0	1.0	0.2	3.0	20	Stable
30	HW	9.0	34.0	1.0	0.2	5.0	34	Stable
31	HW	16.6	90.0	1.0	1.0	8.0	720	Stable
32	Back	4.0	90.0	0.1	1.0	2.0	18	Stable
33	HW	23.0	90.0	1.0	1.0	8.0	720	Stable
34	Back	10.7	90.0	0.4	1.0	2.0	72	Stable
35	Back	10.5	9.0	0.6	0.3	2.3	3.9	Caved
36	HW	9.0	9.0	0.9	0.3	5.0	13	Stable
53	Back	2.4	43.5	0.5	0.2	2.0	8.8	Stable
54	Back	6.8	43.5	0.5	0.2	2.0	8.8	Caved
55	Back	8.0	43.5	0.5	0.2	2.0	8.8	Caved
56	Wall	19.0	2.0	1.0	0.3	8.0	5.2	Caved
57	Back	3.7	43.5	0.2	0.2	2.0	3.5	Stable
58	Wall	8.4	43.5	1.0	1.0	8.0	352	Stable
59	Wall	4.5	2.0	1.0	0.3	8.0	5.2	Stable
61	HW	7.5	25.5	1.0	0.3	6.0	45	Stable
62	FW	7.5	25.5	1.0	0.3	4.0	30	Stable
132	HW	5.6	6.0	1.0	0.2	8.0	10	Stable
133	HW	6.7	6.0	1.0	0.2	8.0	9.4	Stable

Table 5.1: Summary of Potvin Unsupported Main Database (con't)

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability
134	Back	1.9	5.0	0.1	0.2	2.0	0.2	Stable
135	Back	2.1	26.0	0.6	0.6	2.0	19	Stable
136	Back	2.4	26.0	0.5	0.6	2.0	16	Stable
137	Back	2.9	26.0	0.4	0.6	2.0	13	Stable
138	Back	3.1	26.0	0.4	0.6	2.0	13	Stable
139	Back	3.0	26.0	0.3	0.6	2.0	10	Stable
140	HW	7.5	8.0	1.0	0.3	6.0	15	Stable
141	HW	8.1	8.0	1.0	0.3	6.0	15	Unstable
142	HW	5.3	8.0	1.0	0.2	5.5	9.2	Stable
143	HW	5.7	8.0	1.0	0.2	5.5	9.2	Stable
144	Back	1.9	8.0	0.1	0.2	2.0	0.3	Stable
145	Back	1.8	8.0	0.3	0.2	2.0	1.0	Stable
146	Back	2.1	8.0	0.1	0.2	2.0	0.3	Unstable
147	Back	2.3	8.0	0.1	0.2	2.0	0.3	Unstable
148	Back	5.0	22.0	0.7	0.2	2.0	5.9	Stable
149	HW	9.0	0.5	1.0	0.2	6.0	0.8	Caved
150	HW	11.3	0.5	1.0	0.2	6.0	0.8	Caved
151	Back	10.0	30.0	0.4	0.2	2.0	4.8	Caved
152	Back	6.7	30.0	1.0	0.2	2.0	12	Stable
153	Wall	18.0	30.0	1.0	0.5	8.0	120	Stable
155	HW	9.7	32.0	1.0	0.2	3.0	19	Stable
156	End	5.6	32.0	0.1	1.0	3.0	10	Stable
157	FW	8.4	16.2	1.0	0.2	8.0	26	Stable
158	Back	3.4	25.0	0.1	0.2	2.0	1.0	Stable
159	Back	7.6	16.0	0.1	0.2	2.0	0.6	Caved
161	Wall	20.0	3.0	1.0	0.2	8.0	4.8	Caved
164	Back	8.6	21.0	0.1	0.8	2.0	3.3	Caved
165	HW	9.9	21.0	1.0	0.2	8.0	33	Stable
166	FW	9.9	13.5	1.0	0.2	3.0	8.3	Unstable
170	Back	12.5	27.0	1.0	0.8	2.8	60	Caved
171	Back	15.0	27.0	1.0	0.8	2.8	60	Caved
172	Back	15.9	27.0	1.0	0.8	2.8	60	Caved
173	Back	7.7	27.0	1.0	0.8	2.8	60	Stable
174	Back	5.4	27.0	1.0	0.8	2.8	60	Stable
175	Wall	11.6	27.0	0.5	0.3	8.0	32	Unstable
176	Back	7.3	27.0	0.5	0.85	2.5	29	Stable
177	Back	9.9	27.0	0.5	0.85	2.5	29	Stable
178	Back	11.1	27.0	0.5	0.85	2.5	29	Unstable
180	HW	6.9	6.0	1.0	0.3	6.0	10	Unstable
183	Wall	4.9	24.0	0.1	0.3	8.0	5.8	Stable
184	HW	6.7	6.0	1.0	0.3	7.0	12	Stable

Table 5.2: Summary of Potvin Unsupported Complementary Database

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability
64	HW	6	6.0	1.0	0.3	5.5	10	Stable
65	HW	12	6.0	1.0	0.3	5.5	10	Caved
66	HW	3	2.4	1.0	0.3	7.0	5.0	Stable
67	HW	9	2.4	1.0	0.3	7.0	5.0	Unstable
68	HW	12	2.4	1.0	0.3	7.0	5.0	Caved
69	HW	16	54.0	1.0	0.3	4.5	73	Unstable
70	HW	5	4.8	1.0	0.3	8.0	12	Unstable
71	HW	8	4.8	1.0	0.3	8.0	12	Caved
72	HW	16	0.25	1.0	0.3	6.5	0.5	Caved
73	HW	7	48.0	1.0	0.3	8.0	115	Stable
74	HW	2	12.0	1.0	0.3	4.5	16	Stable
75	HW	11	12.0	1.0	0.3	4.5	16	Stable
76	HW	5	54.0	1.0	0.3	5.0	81	Stable
77	HW	14	0.75	1.0	0.3	8.0	1.8	Caved
78	HW	6	0.75	1.0	0.3	7.0	1.6	Caved
79	HW	10	0.75	1.0	0.3	7.0	1.6	Caved
80	HW	11	0.25	1.0	0.3	6.5	0.5	Caved
81	HW	9	54.0	1.0	0.3	5.0	81	Stable
82	HW	6	2.4	1.0	0.3	5.5	4.0	Unstable
83	HW	13	2.25	1.0	0.3	5.0	3.4	Caved
84	HW	10	54.0	1.0	0.3	4.5	73	Stable
85	HW	4	21.0	1.0	0.3	5.5	35	Stable
86	HW	1	60.0	1.0	0.3	4.5	81	Stable
87	FW	12	60.0	1.0	0.3	4.5	81	Unstable
88	HW	4	16.0	1.0	0.3	8.0	38	Stable
89	HW	11	16.0	1.0	0.3	8.0	38	Stable
90	HW	3	0.75	1.0	0.3	4.0	0.9	Stable
91	HW	11	0.75	1.0	0.3	4.0	0.9	Caved
92	HW	2	0.75	1.0	0.3	5.5	1.2	Stable
93	HW	7	0.75	1.0	0.3	5.5	1.2	Caved
94	HW	9	0.75	1.0	0.3	5.5	1.2	Caved
95	HW	16	0.75	1.0	0.3	5.5	1.2	Caved
96	HW	8	0.25	1.0	0.3	7.0	0.5	Caved
97	HW	3	0.25	1.0	0.3	8.0	0.6	Unstable
98	HW	5	0.25	1.0	0.3	8.0	0.6	Caved
99	HW	3	16.0	1.0	0.3	5.5	26	Stable
100	HW	3	3.0	1.0	0.3	5.0	4.5	Stable
101	HW	6	3.0	1.0	0.3	5.0	4.5	Unstable
102	HW	14	3.0	1.0	0.3	5.0	4.5	Caved
103	HW	3	1.5	1.0	0.3	5.5	2.5	Stable
104	HW	8	1.5	1.0	0.3	5.5	2.5	Unstable
105	HW	13	1.5	1.0	0.3	5.5	2.5	Caved
106	HW	10	30.0	1.0	0.3	6.0	54	Stable
107	HW	4	1.6	1.0	0.3	7.0	3.4	Caved
108	HW	10	1.6	1.0	0.3	7.0	3.4	Caved
109	HW	6	3.0	1.0	0.3	5.0	4.5	Unstable
110	HW	12	3.0	1.0	0.3	5.0	4.5	Caved

Table 5.2: Summary of Potvin Unsupported Complementary Database (con't)

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability
111	HW	3	1.0	1.0	0.3	6.0	1.8	Stable
112	HW	8	1.0	1.0	0.3	6.0	1.8	Unstable
113	HW	14	1.0	1.0	0.3	6.0	1.8	Caved
114	HW	2	2.4	1.0	0.3	5.5	4.0	Stable
115	HW	8	2.4	1.0	0.3	5.5	4.0	Unstable
116	HW	10	2.4	1.0	0.3	5.5	4.0	Unstable
117	HW	10	6.0	1.0	0.3	5.5	10	Stable
118	HW	6	0.25	1.0	0.3	6.5	0.5	Unstable
119	HW	9	0.25	1.0	0.3	6.5	0.5	Caved
120	HW	1	0.25	1.0	0.3	5.0	0.4	Stable
121	HW	2	0.25	1.0	0.3	5.0	0.4	Unstable
122	HW	13	0.25	1.0	0.3	5.0	0.4	Caved
123	HW	6	0.25	1.0	0.3	5.5	0.4	Unstable
124	HW	10	0.25	1.0	0.3	5.5	0.4	Caved
125	HW	1	0.25	1.0	0.3	6.0	0.5	Stable
126	HW	2	0.25	1.0	0.3	6.0	0.5	Unstable
127	HW	13	0.25	1.0	0.3	6.0	0.5	Caved
128	HW	7	9.6	1.0	0.3	4.5	13	Stable
129	HW	12	1.5	1.0	0.3	5.5	2.5	Unstable
130	HW	4	0.25	1.0	0.3	5.0	0.4	Unstable
131	HW	3	0.25	1.0	0.3	5.5	0.4	Unstable
9	Wall	4.7	24.0	0.3	0.2	8.0	12	Stable
11	HW	7.9	3.0	1.0	0.2	7.0	4.2	Stable
14	HW	8.8	4.5	1.0	0.2	6.0	5.4	Caved
15	HW	8.8	4.5	1.0	0.3	7.0	9.5	Caved
154	Back	5.2	32.0	0.1	0.3	2.0	1.9	Unstable
167	HW	7.8	13.5	1.0	0.2	8.0	22	Stable
168	HW	6	22.5	1.0	0.2	8.0	36	Stable
169	Back	5	22.5	0.3	0.3	2.0	4.1	Stable
179	Back	4.1	22.5	0.1	0.85	2.0	3.8	Stable
181	Back	4	22.5	0.1	0.85	2.0	3.8	Stable
182	HW	4.9	22.5	1.0	0.3	8.0	54	Stable
37	Back	2.7	121.5	0.4	1.0	2.0	97	Stable
38	Back	6.1	121.5	0.4	1.0	2.0	97	Unstable
39	Back	7.6	121.5	0.6	1.0	2.0	146	Unstable
40	Back	8.8	39.0	0.6	1.0	2.0	47	Unstable
41	Back	13.4	39.0	0.6	1.0	2.0	47	Unstable
42	Back	6.1	18.2	0.5	0.3	2.0	5.5	Unstable
43	Back	15.2	18.2	0.5	0.3	2.0	5.5	Caved
44	Back	6.4	18.2	0.3	0.3	2.0	3.3	Unstable
46	HW	13.1	39.0	1.0	0.3	8.0	94	Stable
47	Back	7.3	18.0	0.3	1.0	2.0	11	Caved
48	Back	5	18.0	0.1	1.0	2.0	3.6	Unstable
49	Back	9.9	18.0	1.0	1.0	2.0	36	Caved
50	Back	6.8	18.0	0.4	1.0	2.0	14	Caved

The Mathews method, as briefly reviewed in Chapter 4, was selected as the starting point for the model. This method involves the determination of a stability number, N, from a modified Q value and three factors as follows:

$$N = Q' \times A \times B \times C. \tag{5.1}$$

Potvin (1988) starts with the same idea, but expresses the factors A, B and C slightly differently. The Potvin approach described the stability number in terms of a block size factor, a stress factor, a joint orientation factor and a gravity factor. The Potvin stability number is documented as N' since modifications have been made to the original Mathews stability number.

5.2.1 The Block Size Factor

The block size factor is expressed as RQD/Jn and remains unchanged from the Q system of rock classification. Barton, Lien, and Lunde (1974) describe the quotient RQD/Jn as a crude measure of the relative block size within a rock mass. The ratio can differ by a factor of 400 since it ranges from a minimum value of 0.5 to a maximum of 200. RQD can be determined from an examination of drill core, but is subject to the effects of drill hole orientation. Potvin (1988) recommended that RQD be determined from direct underground mapping and the utilization of methods proposed by Palmstrom (1985) and Priest and Hudson (1976). Priest and Hudson (1976) related RQD to the average number of discontinuities per meter using the following relationship:

$$RQD = 100e^{-0.1\lambda}(0.1\lambda + 1) \tag{5.2}$$

where λ represents the average number of discontinuities per meter obtained from a scanline mapping survey. Palmstrom (1985) describes the volumetric joint count, J_v, as the number of joints intersecting one unit volume of rock mass. The value of J_v in joints/m^3 can be evaluated by adding the average number of joints per meter for each joint set in a rock mass. Palmstrom (1976) related RQD to the volumetric joint count with the following relationship:

$$RQD = 115 - 3.3J_v \tag{5.3}$$

where RQD is set to 100% when J_v is less than 4.5 joints/m^3. Potvin (1988) suggests that the ratio of RQD/Jn ranges from 1 to 90 for Canadian open stope situations.

5.2.2 The Stress Factor

The stress factor is related to the uniaxial compressive strength to induced stress ratio, as shown in Figure 5.1a. Mathews et al. (1981) suggested that rock would fail if the ratio of uniaxial compressive strength to induced stress was 2.0 or lower. Potvin (1988) proposed that the lower bound for the stress factor be kept at 0.1 based on the observation of several highly stressed backs that remained stable due to their small size. The uniaxial compressive strength can be taken from the results of laboratory testing and an estimate of induced stress determined from numerical modelling. Since access to numerical modelling software was limited at many minesites, Potvin (1988) developed a series of induced stress curves to aid in the evaluation of the stress factor. The use of two and three dimensional numerical modelling software at mines is increasingly evident and provides a better approach to stress estimation for complex geometries. The stress factor can vary from 0.1 for highly stressed backs to 1.0 for surfaces that are in a state of relaxation.

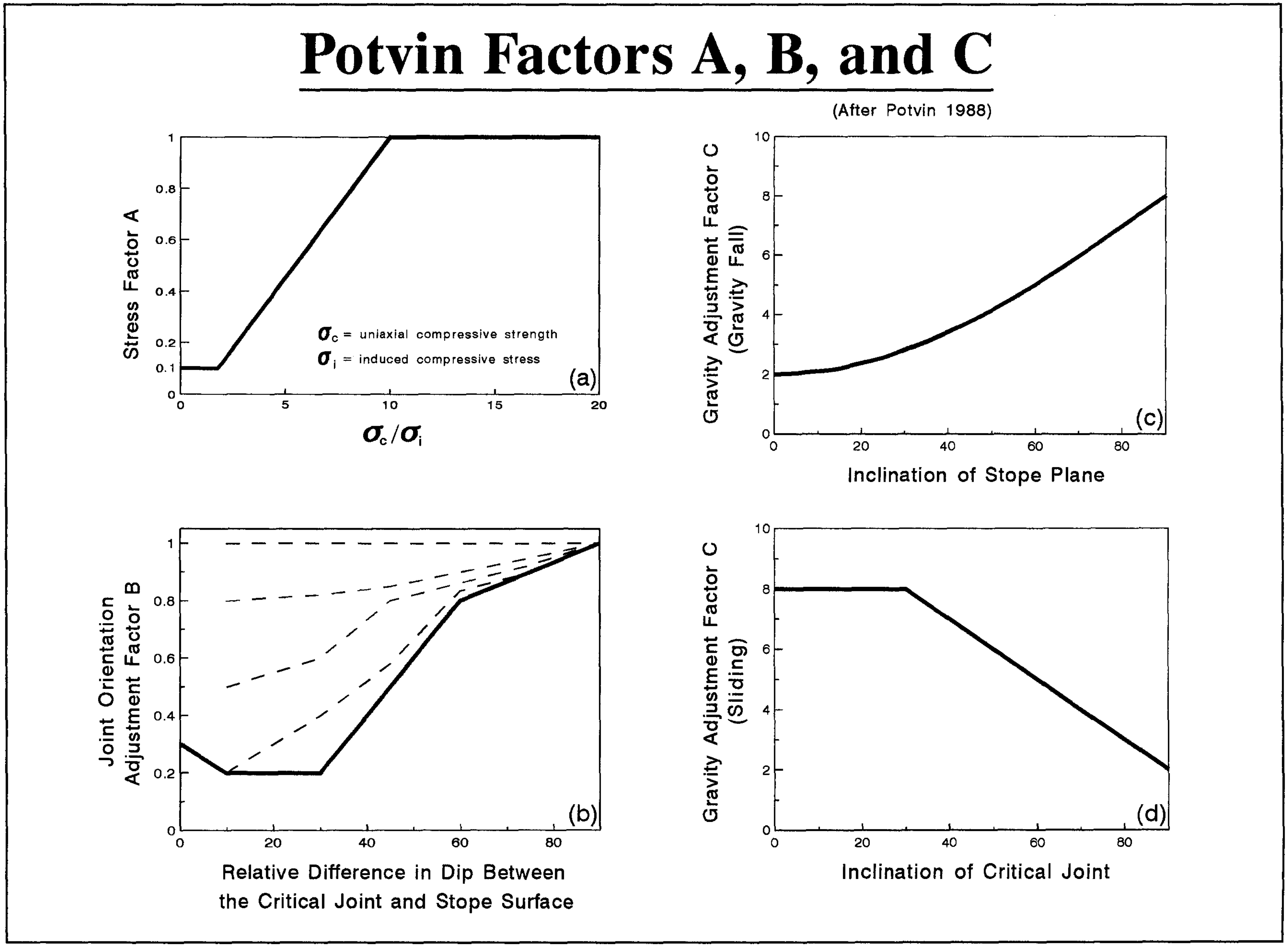


Figure 5.1: Factors A, B, and C proposed by Potvin (1988)

5.2.3 The Joint Orientation Factor

Potvin found that most structurally controlled failures occurred where joints formed a shallow angle with the stope surface. This introduced the concept of the critical joint, as illustrated in Figure 5.2. The critical joint represents the joint set that is most likely to detract from the stability of a particular surface. The least favourable case is where the critical joint dip, θ, is found to be ten to thirty degrees from the dip of the stope surface. The distance, d, is small enough in this case to enhance the chance of failure. The orientation of the critical joint with respect to the stope surface is related to a joint orientation adjustment factor, B, as illustrated in Figure 5.1b. The joint orientation adjustment factor was set to 0.2 for the worst case, and improves to a maximum of 1.0 when the dip difference approaches 90 degrees. A slight improvement in stability is found when the critical joint parallels the stope surface, and the adjustment factor increases marginally to 0.3. A difference in strike increases the difference in true dip and is reflected in the dashed lines in Figure 5.1b. The second parameter that is included in Potvin's Joint Orientation Factor is the shear strength of the critical joint. This value is represented by the Jr/Ja term of the Q system, and is estimated by underground geotechnical mapping. Barton, Lien, and Lunde (1974) suggested that Jr/Ja is representative of the shear strength that might result from various combinations of joint roughness and alteration. Potvin (1988) suggests that Jr/Ja can range from 0.05 to 3 for Canadian open stope practice.

5.2.4 The Gravity Factor

Gravity is an important consideration in the kinematic analysis of rock structures. Potvin identified five modes of failure encountered in open stoping situations, gravity fall, slabbing, buckling, sliding and shearing. Buckling and shearing are similar mechanisms to slabbing and sliding modes of failure, except that the principal driving force is stress rather than gravity. The

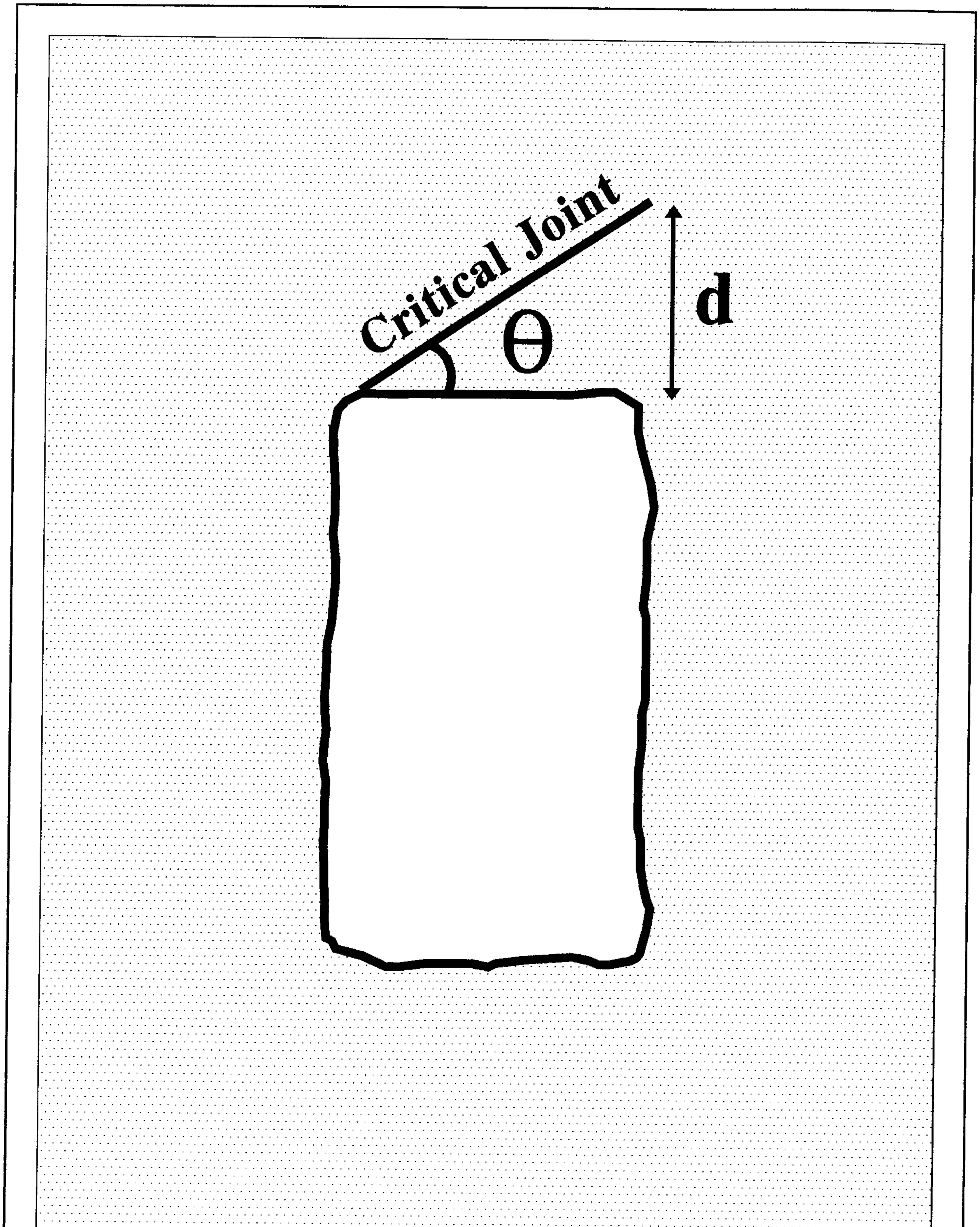


Figure 5.2: Critical joint concept (After Potvin 1988)

effects of stress have already been incorporated in the stress factor. The remaining gravity induced failure modes can be summarized into three mechanisms, namely gravity fall, slabbing and sliding. Gravity fall and slabbing modes of failure are dependent on gravity, and have been related to the inclination of the stope surface as shown in Figure 5.1c. This factor is based on the idea that a vertical stope wall is four times as stable as a stope back. The sliding mode of failure is dependent on the inclination of the critical joint and is derived from Figure 5.1d. The adjustment factor is a maximum where the inclination of the critical joint is less than thirty degrees, since most rocks have an angle of friction in this range. As the inclination of the critical joint increases, the probability of a sliding failure rises, and the adjustment factor decreases. The gravity factor for all failure modes ranges from a value of 2 to 8.

5.2.5 The Unsupported *Modified Stability Graph*

Potvin proposed that all the above factors be combined to determine a modified stability number where:

$$N' = \frac{RQD}{Jn} \; x \; \begin{matrix}Compressive\\ Stress\\ Factor\end{matrix} \; x \; \begin{matrix}Joint\\ Orientation\\ Adjustment\\ Factor\end{matrix} \; x \; \frac{Jr}{Ja} \; x \; \begin{matrix}Gravity\\ Adjustment\\ Factor\end{matrix} \tag{5.4}$$

To avoid confusion, Equation 5.4 can be expressed in terms of the original Mathews terminology where:

$$N' = Q' \times A \times B \times C \tag{5.5}$$

and the factors A, B and C represent the stress factor, joint orientation adjustment factor and gravity adjustment factor proposed by Potvin (1988). The modified stability number is related to the hydraulic radius on the *Modified Stability Graph* (Figure 5.3). The hydraulic radius is adapted directly from the Shape Factor used by Mathews et al. (1981), and is calculated by dividing the area of the surface by its perimeter. Potvin proposed that a stable and a caved zone could be identified and used for stope design. The main database was plotted on the *Modified Stability Graph* and showed good correlation with the proposed design regions. A *transition zone* was indicated within the shaded region, where many of the unstable cases were found to concentrate. The complementary database was added to the chart and used to confirm the original hypothesis. Potvin (1988) recommended that the *Modified Stability Graph* be used in conjunction with economic, scheduling and mining constraints for the design of underground openings. It was suggested that critical openings, where absolute stability is required, be designed to plot within the stable zone. Designs that involve multiple openings in the same area should also plot within the stable zone. It was noted that non-entry methods of mining may still be viable when plotting below the transition zone, since there is greater tolerance for instability. Potvin found that openings plotting within the transition zone were sensitive to the effects of blasting and time. Dilution is expected to increase as a point moves farther into the caved zone, but has not yet been calibrated with the *Modified Stability Graph*.

5.2.6 The Supported *Modified Stability Graph*

Potvin (1988) noted that surfaces plotting below the *transition zone* required support in many cases to remain stable. To investigate the effects of support, 66 case studies of supported surfaces were assembled and plotted on the *Modified Stability Graph*. The parameters for the supported case histories are summarized in Table 5.3. A region below the *transition zone* was

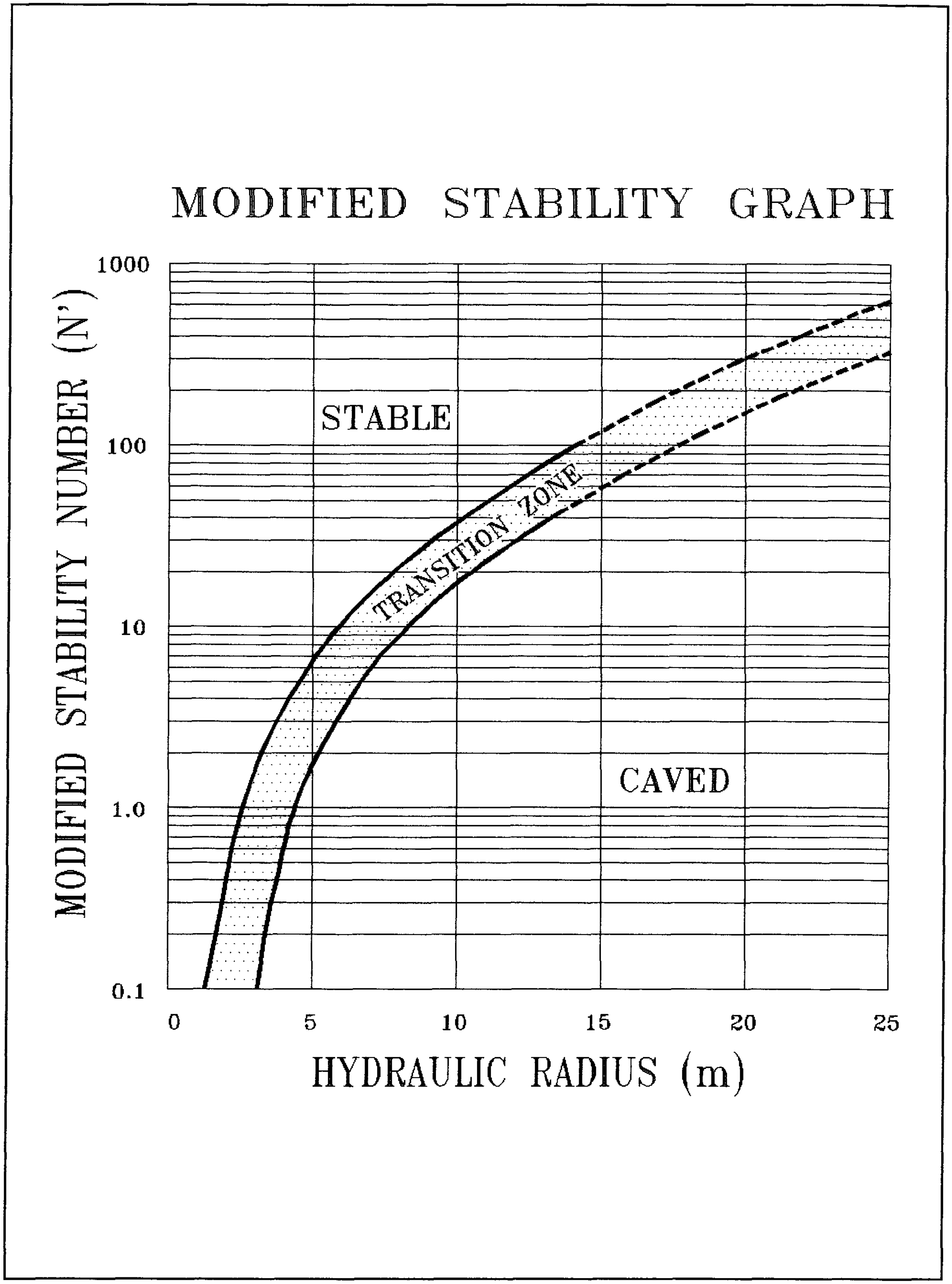


Figure 5.3: Unsupported *Modified Stability Graph* (After Potvin 1988)

Table 5.3: Summary of Potvin Supported Database

(* Cables with rebar, ** Rebar only)

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability	RQD/Jn	RQD/Jn/HR	Cable Bolt Density (Bolts/Sq. m)	Cable Bolt Length (m)
251	*Back	8.4	18.75	0.25	0.2	2.0	1.9	Caved	25	2.98	0.17	21.0
252	*Back	8.4	18.75	0.5	0.2	2.0	3.8	Caved	25	2.98	0.17	21.0
253	*Back	5.3	18.75	0.25	0.2	2.0	1.9	Stable	25	4.72	0.17	21.0
254	*Back	6.4	54.0	0.1	0.2	2.0	2.2	Stable	18	2.81	0.17	21.0
255	HW	5.0	18.75	1.0	0.2	8.0	30	Stable	25	5.00		
256	*Back	5.9	54.0	0.1	0.2	2.0	2.2	Stable	18	3.05	0.16	9.0
257	*Back	6.7	54.0	0.1	0.2	2.0	2.2	Stable	18	2.69	0.16	9.0
258	*Back	7.1	54.0	0.1	0.2	2.0	2.2	Stable	18	2.54	0.16	9.0
259	*Back	4.6	54.0	0.1	0.2	2.0	2.2	Stable	18	3.91	0.16	9.0
260	*Back	5.0	54.0	0.1	0.2	2.0	2.2	Stable	18	3.60	0.16	9.0
261	*Back	13.9	7.0	0.1	0.4	2.6	0.7	Stable	14	1.01	0.17	6.0
											0.17	24.0
262	*Back	16.0	7.0	0.1	0.4	2.6	0.7	Caved	14	0.88	0.17	6.0
											0.17	24.0
263	Back	7.3	4.2	0.36	0.2	2.4	0.7	Stable	6	0.82	0.17	3.0
											0.17	18.0
264	Back	6.0	0.8	0.1	0.4	2.0	0.1	Stable	4	0.67	0.17	3.0
											0.17	18.0
265	Back	8.0	4.2	0.35	0.2	2.4	0.7	Stable	6	0.75	0.17	3.0
											0.17	18.0
											0.05	30.0
266	Back	14.8	4.2	0.77	0.2	2.4	1.6	Caved	6	0.41	0.17	3.0
											0.17	18.0
267	Back	7.8	0.8	0.1	0.85	2.0	0.1	Caved	4	0.51	0.23	18.0
268	Wall	8.9	6.0	1.0	0.2	3.5	4.2	Stable	6	0.67	0.04	10.0
269	Back	4.4	6.0	0.2	0.2	2.0	0.5	Stable	6	1.36	0.20	3.0
270	Back	5.3	6.0	0.1	0.2	2.0	0.2	Stable	6	1.13	0.20	6.0
271	Back	5.3	10.5	0.1	0.5	2.0	1.1	Stable	7	1.32	0.30	9.0
272	Back	6.2	40.0	0.1	1.0	2.0	8.0	Stable	40	6.45	0.70	3.0
273	*Back	2.6	9.0	0.1	0.2	2.0	0.4	Stable	6	2.31	0.24	6.0
274	*Back	4.2	9.0	0.1	0.2	2.0	0.4	Stable	6	1.43	0.24	6.0
275	End	4.7	9.0	0.1	0.4	5.0	1.8	Stable	6	1.28	0.06	9.0
276	End	6.1	9.0	0.23	0.25	4.6	2.4	Stable	6	0.98	0.06	9.0
277	Back	5.2	9.0	0.1	0.2	2.0	0.4	Stable	6	1.15	0.23	9.0
278	**Back	2.5	3.2	0.1	0.2	2.0	0.1	Stable	4	1.60		
279	**Back	7.5	3.2	0.2	0.2	2.0	0.3	Caved	4	0.53		
280	**Back	2.7	30.0	0.6	0.2	2.0	7.2	Stable	15	5.56		

Table 5.3: Summary of Potvin Supported Database (con't)

(* Cables with rebar, ** Rebar only)

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability	RQD/Jn	RQD/Jn/HR	Cable Bolt Density (Bolts/Sq. m)	Cable Bolt Length (m)
281	**Back	3.6	30.0	0.8	0.2	2.0	9.6	Stable	15	4.17		
282	Back	4.1	6.25	0.1	0.9	2.0	1.1	Stable	25	6.10	0.10	15.0
284	HW	7.5	4.5	0.45	0.3	6.0	3.6	Stable	9	1.20	0.03	15.0
285	**Back	6.3	6.0	0.5	0.2	2.0	1.2	Unstable	8	1.27		
286	**HW	10.0	6.0	1.0	0.3	6.0	11	Stable	8	0.80		
287	HW	19.7	30.0	1.0	0.3	8.0	72	Stable	30	1.52	0.07	21.0
289	Back	6.2	34.0	1.0	0.2	2.0	14	Stable	17	2.74	0.07	11.0
290	Back	11.4	9.0	0.6	0.3	2.4	3.9	Caved	6	0.53	0.10	20.0
291	Back	8.0	9.0	1.0	0.3	2.4	6.5	Stable	6	0.75	0.10	20.0
292	Back	20.8	9.0	1.0	0.3	2.4	6.5	Caved	6	0.29	0.10	20.0
293	Back	9.2	9.0	1.0	0.3	2.4	6.5	Stable	6	0.65	0.10	20.0
294	Wall	19.0	2.0	1.0	0.3	8.0	4.8	Caved	4	0.21	0.02	6.0
295	Back	3.7	43.5	0.2	0.3	2.0	5.2	Stable	29	7.84	0.15	6.0
296	Back	5.3	25.5	0.1	0.2	2.0	1.0	Stable	17	3.21	0.23	9.0
297	Back	9.0	50.0	0.3	0.2	2.0	6.0	Stable	25	2.78	0.27	10.0
298	Back	3.9	50.0	0.1	0.2	2.0	2.0	Stable	25	6.41	0.20	5.0
299	Back	8.0	25.5	0.3	0.9	2.0	11	Stable	17	2.13	0.21	10.0
300	Back	4.7	16.2	0.1	0.2	2.0	0.6	Stable	9	1.91	0.22	12.0
301	Back	7.7	16.2	0.1	0.2	2.0	0.6	Caved	9	1.17	0.22	12.0
302	Back	5.6	2.0	1.0	0.2	2.0	0.8	Stable	2	0.36	0.33	10.0
303	Back	4.3	10.0	1.0	0.2	2.0	4.0	Stable	10	2.33	0.33	7.5
304	Back	2.7	5.0	1.0	0.9	2.0	9.0	Stable	5	1.85	0.37	7.5
305	Back	14.0	2.0	1.0	0.2	2.0	0.8	Caved	2	0.14	0.22	10.0
306	Back	9.3	5.0	1.0	0.2	2.0	2.0	Unstable	5	0.54	0.19	10.0
307	Back	12.7	1.0	1.0	0.2	2.0	0.4	Caved	1	0.08	0.28	10.0
308	Back	14.6	8.0	1.0	0.2	2.0	3.2	Stable	8	0.55	0.20	10.0
309	Back	7.1	15.0	0.7	0.2	2.0	4.2	Stable	15	2.11	0.16	15.0
310	Back	8.0	25.0	0.7	0.2	2.0	7.0	Stable	25	3.13	0.16	10.0
311	Back	7.4	20.0	0.7	0.2	2.0	5.6	Stable	20	2.70	0.16	25.0
312	Back	13.7	5.0	0.7	0.2	2.0	1.4	Caved	5	0.36	0.25	10.0
313	Back	10.0	10.0	0.7	0.2	2.0	2.8	Stable	10	1.00	0.16	18.0
314	Back	5.3	20.0	0.5	0.2	2.0	4.0	Stable	20	3.77	0.16	15.0
315	Back	6.9	20.0	0.5	0.2	2.0	4.0	Stable	20	2.90	0.13	25.0
316	HW	8.4	16.2	1.0	0.2	5.5	18	Stable	9	1.07	0.07	12.0
317	Back	8.6	21.0	0.1	0.8	2.0	3.4	Unstable	14	1.63	0.11	15.0
318	Back	5.9	23.4	0.1	0.9	2.0	4.2	Stable	13	2.20	0.31	6.0

identified where supported surfaces seemed to remain stable. Cable bolts were the most dominant type of support within the database of supported case histories, and Potvin proposed that the area below the *transition zone* illustrated in Figure 5.4 be used for the design of cable bolt support. If a surface plots within this supportable region, the use of cable support is recommended.

5.3 *DESIGN CHART FOR CABLE BOLT DENSITY*

If cable bolt support is required, Potvin (1988) proposed that the *Design Chart For Cable Bolt Density* (Figure 5.5) be used to estimate the bolt density required for stope backs. The cable density was related to the ratio of the block size factor to the surface hydraulic radius, RQD/Jn/HR. This term provides a relative estimate of the block size with respect to the surface hydraulic radius, and is referred to as the relative block size factor. A smaller block size, or a larger hydraulic radius, decreases the relative block size and suggests a higher bolt density. The band of three diagonal lines represents the recommended region for cable bolt design. The center line was suggested as a conservative design guideline for cable density. Only one case of cable support was encountered below a density of 0.10 bolts per square meter, and the region below this value is not recommended for design. A success rate of 20% was indicated to the left of the vertical dashed line on the design chart. Potvin (1988) suggested that cable bolts will have limited effectiveness when the relative block size factor is below 0.75. A revised interpretation (Potvin and Milne 1992) of the *Design Chart for Cable Bolt Density* is illustrated in Figure 5.6. The hatched region in Figure 5.6 represents a caving zone where the cable density or relative block size factor is too low. The area between line #1 and #2 is suggested for the design of non-entry open stope backs. A conservative design density for non-entry mining and a realistic design

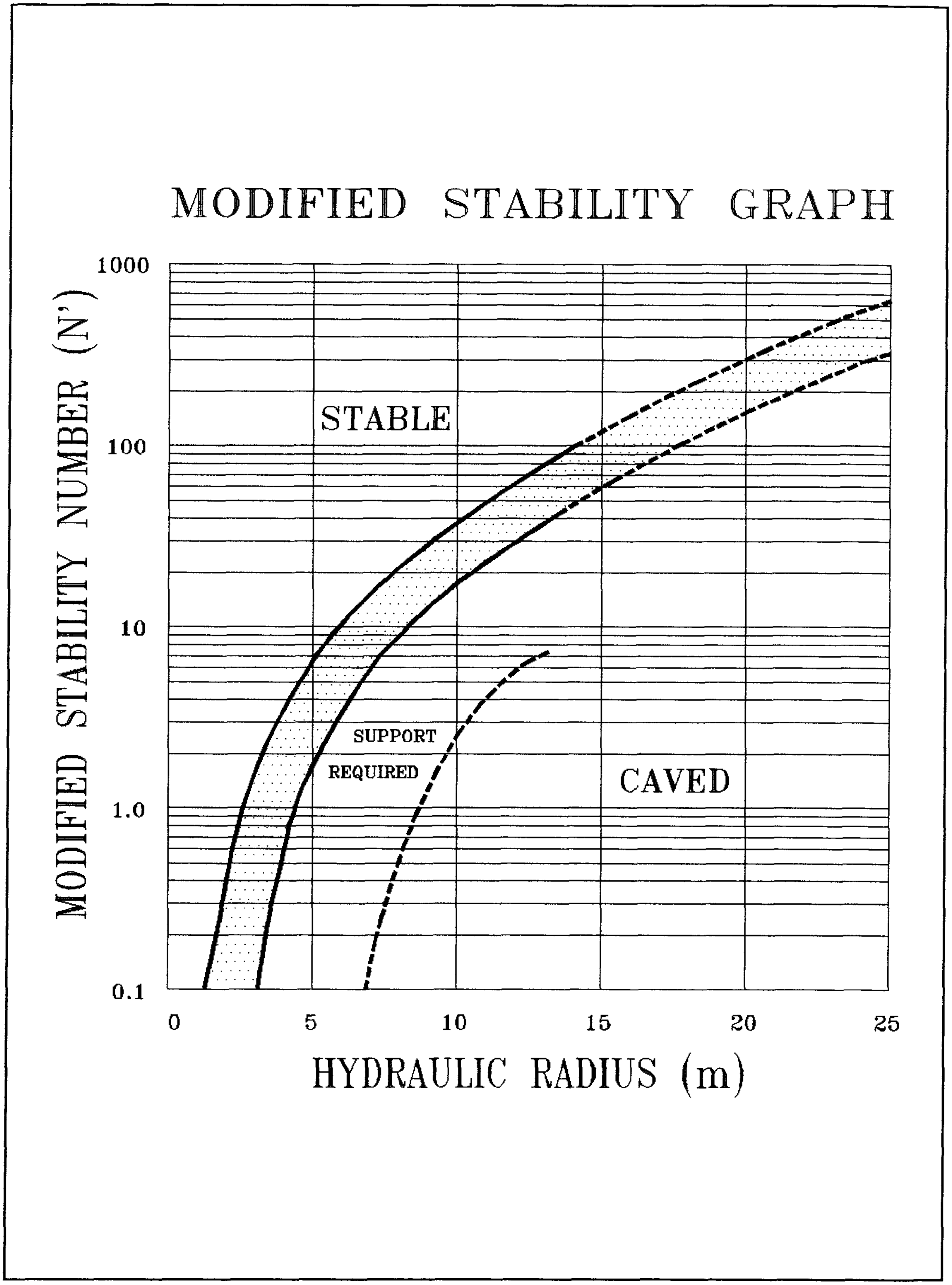


Figure 5.4: Supported *Modified Stability Graph* (After Potvin 1988)

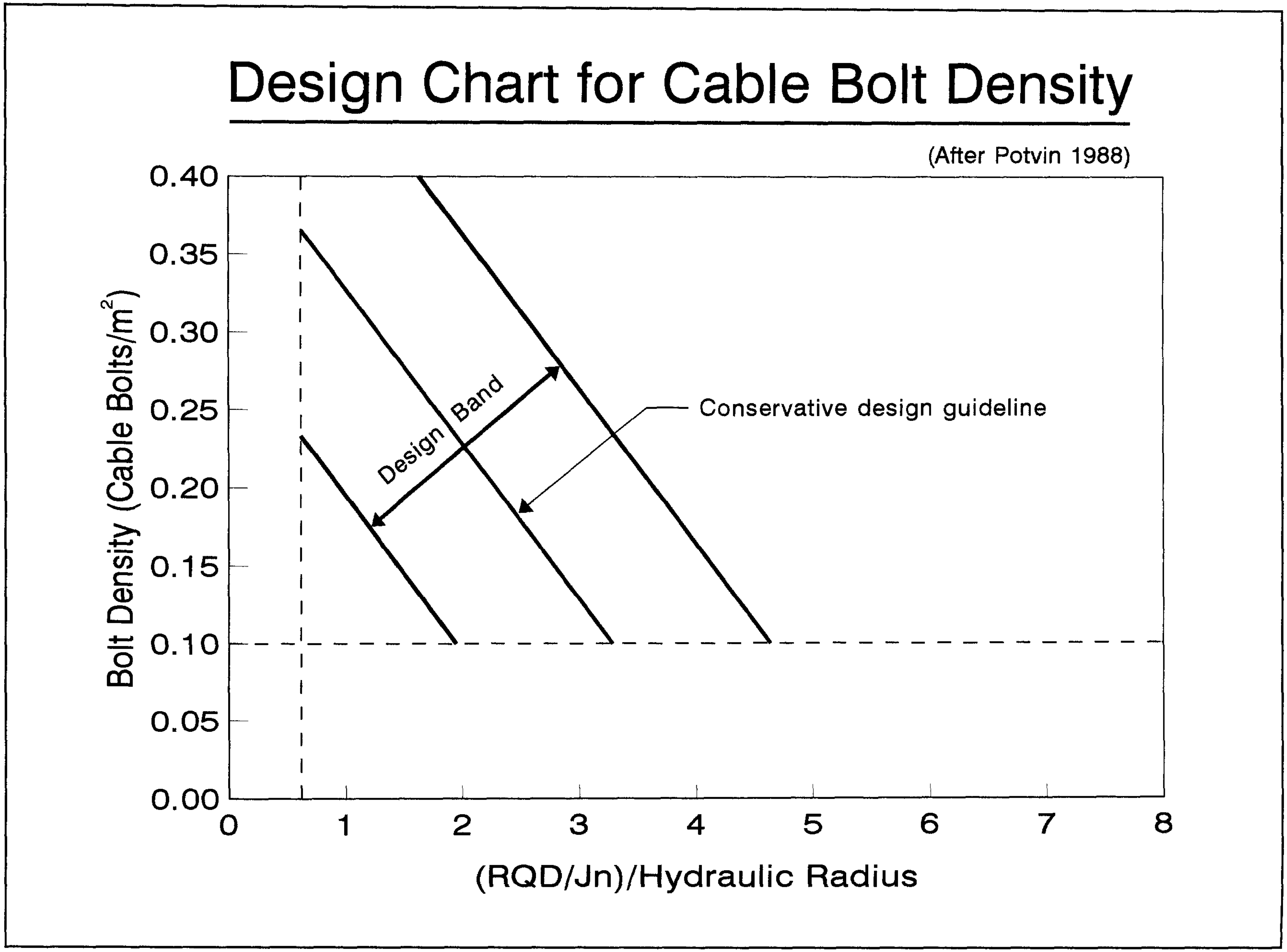


Figure 5.5: *Design Chart for Cable Bolt Density* (After Potvin 1988)

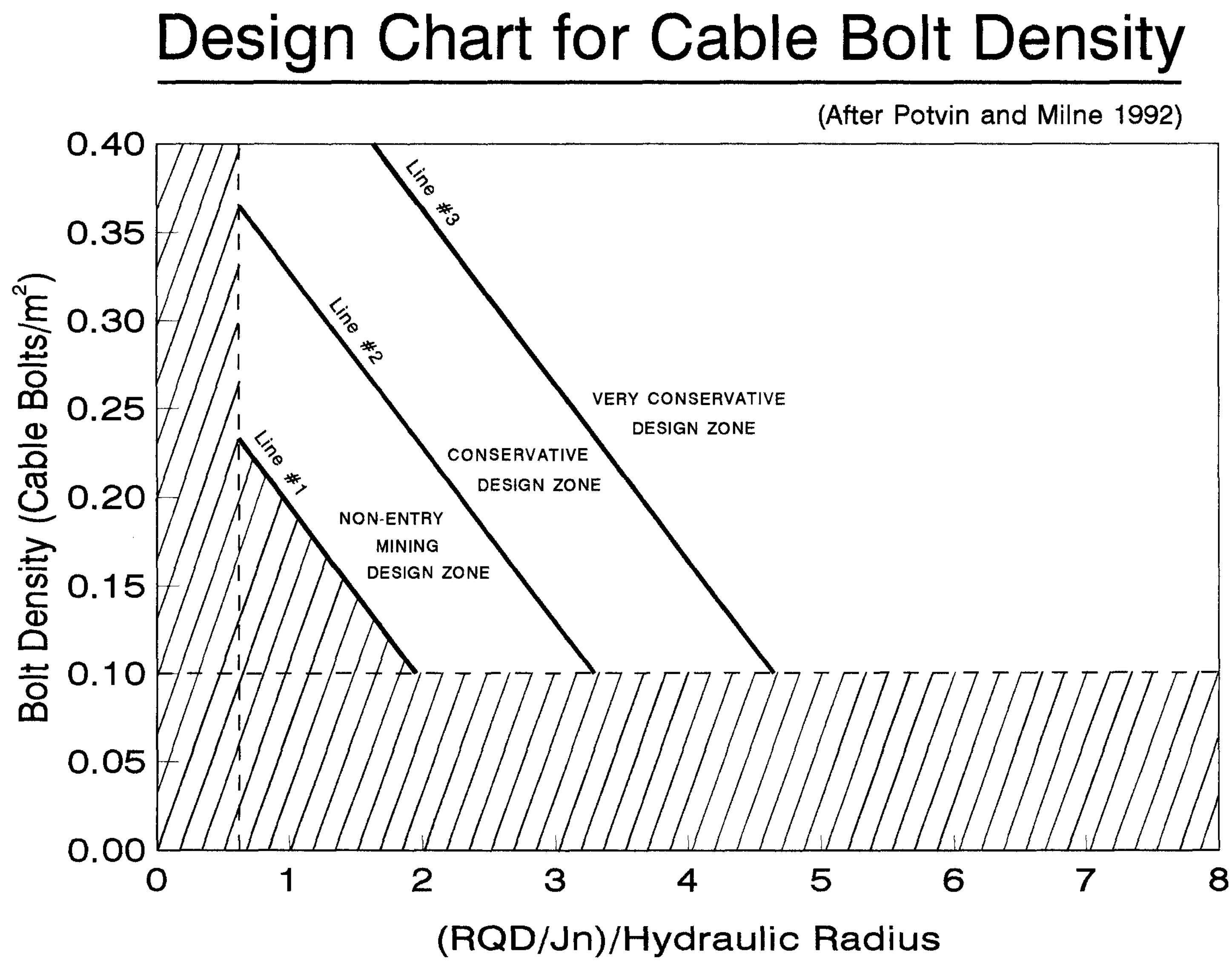


Figure 5.6: Revised *Design Chart for Cable Bolt Density* (After Potvin and Milne 1992)

for entry mining methods is bounded by line #2 and #3. The region above line #3 is associated with very conservative design.

5.4 *DESIGN CHART FOR CABLE BOLT LENGTH*

The line, L, on Figure 5.7 was suggested as an approximate and conservative guideline for the determination of cable bolt length (Potvin, Hudyma, and Miller 1989). This line indicates the point at which the cable bolt length is approximately equivalent to stope span for a range of hydraulic radii. This is based on the concept that hydraulic radius converges to half the span at low span to length ratios. The hydraulic radius encountered by Potvin (1988) varied from 3 m to 21 m, but limited success was found with values exceeding 10m.

5.5 DISCUSSION

The design techniques developed by Potvin (1988) are based on Canadian open stoping experience. The application of the *Modified Stability Graph* for stope design was widely encountered in western Canadian practice, although it was not used extensively for the design of cable support. The *Design Chart for Cable Bolt Density* was rarely consulted due to the uncertainty in locating particular design ranges. The data scatter was noted by Potvin (1988) and revised design bands have been suggested (Potvin and Milne 1992) to improve the cable density selection process. The revised guidelines proposed by Potvin and Milne (1992) offer more descriptive design criteria, but still require further calibration. The applicability of the *Design Chart for Cable Bolt Density* is restricted to an even distribution of cables over the design

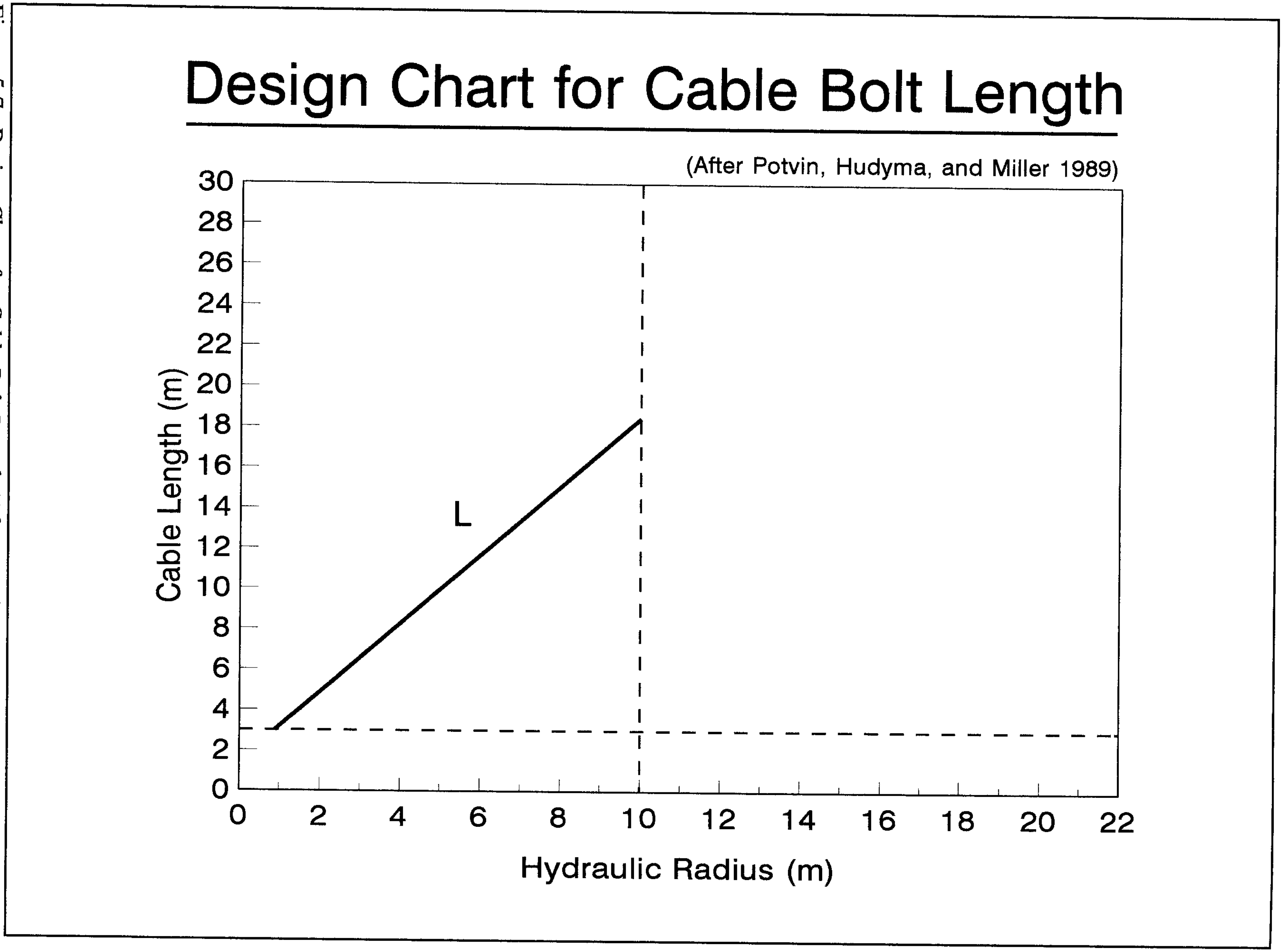


Figure 5.7: *Design Chart for Cable Bolt Length* (After Potvin, Hudyma, and Miller 1989)

surface, and is only intended for use with stope backs that plot in the supportable zone of the *Modified Stability Graph*. The chart was not intended for use with cases of hangingwall support or point anchor back support. Potvin and Milne (1992) also suggest that inclusions of weak material, slot raise or brow development, and poor quality grout will limit the application of the density chart.

The ideas proposed by Potvin (1988) are valuable resources to the mining engineer since they relate the characteristics of a rock mass to the design of underground openings. The additional database collected in this study will be used to evaluate and improve upon existing design guidelines. Limited statistical analysis of the database was completed by Potvin (1988), but Chapter 6 will explore this topic in more detail.

CHAPTER 6

DATABASE

6.1 INTRODUCTION

The database assembled for this study was obtained during visits to twelve mines in Western Canada and the northwestern United States in the spring, summer and fall of 1991. One overseas mining operation was visited to review cable bolting procedures but did not contribute to the case history database. Eleven of the twelve mines in North America used cable bolts for the support of underground openings.

Design regions for the *Modified Stability Graph* and the *Design Chart for Cable Bolt Density* (Potvin 1988) were visually calibrated through the empirical analysis of 242 case histories of supported and unsupported open stope surfaces. In this chapter, a new database will be evaluated in relation to the design methods proposed by Potvin (1988). Section 6.4 will review statistical methods that were applied to aid in the identification of design trends. Although statistical methods have traditionally not been widely applied in the field of empirical rock mechanics, any collection of data warrants a numerical review to aid in the identification of an implied trend. The following quote provides some insight into the goals of this statistical analysis.

> *There is no magic about numerical methods, and many ways in which they can break down. They are a valuable aid to the interpretation of data, not sausage machines automatically transforming bodies of numbers into packets of scientific fact (Marriott 1974).*

6.2 DATA COLLECTION

The collection of data was done during visits arranged to different mining operations. There is some variability in the methods of data collection due to the variation in time allocated to each particular mine. Some operations had the flexibility to allow the mine visit to extend beyond a week, but most were limited in the amount of time that could be allocated to the purposes of this study. Where time permitted, detailed mapping and underground investigation were used to evaluate the parameters involved. On some occasions this was not possible, and the data was collected through discussions with mine personnel and a review of underground plans. Given the limited time constraints, this type of study requires the ability to quickly gain an understanding of the general mine environment and subsequently extract relevant information. The geological department at most operations is a tremendous resource in this regard and often have mapping and classification data logged on site.

6.3 DATABASE

The new database is made up of 13 unsupported and 46 supported case histories. A supported case history refers to a stope surface that contains cable bolts as a means of reinforcing the rock mass. One isolated supported case applies to a surface reinforced with long extension bolts. An unsupported case history is defined as a surface that does not contain cable bolts. Supported and unsupported surfaces may incorporate short pattern bolting with point anchor or fully bonded bolts. In an effort to distinguish between the effect of long bolts and short pattern bolting, surfaces with cable bolt lengths less than

3.7 m were not considered in this database.

The parameters collected for this study are identical to those assembled in the Potvin (1988) database. A stability number, N', and hydraulic radius, HR, were determined for each case history to compare the collected database to the design ranges on the *Modified Stability Graph* proposed by Potvin (1988). The cable bolt density and relative block size factor, RQD/Jn/HR, were also evaluated for the supported case histories to compare results to the *Design Chart for Cable Bolt Density* (Potvin 1988). The unsupported database is summarized in Table 6.1 and the supported database in Table 6.2. A more detailed summary of each case history is presented in Appendix A.

The Potvin (1988) unsupported database is made up of stable, unstable and caved case histories. A stable stope surface contributes low amounts of dilution, while an unstable surface is characterized by operational problems due to dilution or ground falls. A caved surface represents case histories with severe ground control problems. In this study, unsupported surfaces are described in a similar fashion. The caved classification was applied to those surfaces that were observed or reported to be well beyond the designed excavation limits, due to ground fall or excessive dilution. Several cases of ground instability that lead to stope closure were classified as caved. The stable classification was applied to surfaces that were observed or reported to be within the designed excavation limits. Cases that did not fit clearly into either the stable or caved categories were classified as unstable.

The supported case histories assembled by Potvin were classified as stable, unstable or caved in relation to the support system. The caved classification was applied to cases where the support system failed. Cases involving ravelling of the rock mass between the cables were classified as unstable. The stable classification was applied to

Table 6.1: Summary of Unsupported Database

Case #	Surface	Hydraulic Radius (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	HW	10.3	10.0	1.0	0.3	5.0	15.0	Caved
31	HW	16.4	5.9	1.0	0.2	5.5	6.5	Caved
35	HW	7.0	13.1	1.0	0.2	8.0	21.0	Stable
38	HW	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	HW	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	HW	6.1	21.5	1.0	0.2	5.0	21.5	Unstable
44	HW	5.9	7.2	1.0	0.2	5.0	7.2	Caved
51	HW	10.4	8.3	1.0	0.3	5.0	12.5	Stable
58	HW	4.9	3.1	1.0	0.3	6.0	5.6	Unstable

Table 6.2: Summary of Supported Database

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability	RQD/Jn	RQD/Jn/HR	Cable Bolt Density (Bolts/Sq. m)	Cable Bolt Length (m)
1	HW	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	HW	11.7	2.5	1.0	0.3	7.5	5.6	Stable	6.7	0.57	0.021	6.1
4	HW	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	HW	17.1	27.3	1.0	0.3	7.5	61.4	Caved	13.7	0.80	0.022	6.1
7	HW	10.9	2.5	1.0	0.2	7.5	3.8	Unstable	6.7	0.61	0.011	6.1
8	HW	12.7	2.5	1.0	0.3	7.5	5.6	Unstable	6.7	0.53	0.020	6.1
9	HW	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	HW	10.8	30.0	1.0	0.3	7.5	67.5	Stable	15	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	Back	3.6	11.7	1.0	0.2	2.0	4.7	Stable	11.7	3.25	0.116	6.1
14	Back	4.3	0.6	1.0	0.4	2.0	0.48	Stable	1.67	0.39	0.180	14.0
15	Back	7.6	0.5	1.0	0.3	3.0	0.45	Stable	2.5	0.33	0.280	8 &15
16	Back	11.2	0.5	1.0	0.3	3.0	0.45	Caved	2.5	0.22	0.180	8 &12
17	Back	8.6	0.6	1.0	0.2	2.0	0.24	Caved	1.67	0.19	0.140	12 &15
18	Back	4.2	13.3	0.1	0.3	2.0	0.80	Stable	13.3	3.17	0.290	9.8
19	Back	5.2	13.3	0.1	0.3	2.0	0.80	Unstable	13.3	2.56	0.270	9.8
20	HW	12.4	15.8	1.0	0.2	5.0	15.8	Caved	15.8	1.27	0.035	14.6
21	HW	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	HW	11.5	15.8	1.0	0.3	7.0	33.2	Stable	15.8	1.37	0.025	14.6
27	HW	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	HW	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	*Rockbolts	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	HW	15.5	9.9	1.0	0.3	4.7	14.0	Stable	13.1	0.85	0.160	9.1
50	HW	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	HW	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	HW	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

cases where the support system was effective in maintaining the integrity of the surface, and therefore minimizing operational problems and dilution. A similar classification system was used for the supported database collected in this study. Dilution was used in some cases to gauge the effectiveness of a support system. Pakalnis et al. (1987) describe the empirical prediction of dilution at one operation, and relate the results to the design ranges proposed on the Mathews (1981) stability graph. Dilution values for stopes plotting in the *potentially caving zone* ranged from 9% to 25%, but seem to concentrate on the 20% to 30% range. The relationship between dilution and the stability condition of a surface has not yet been well defined and would be an interesting area of future research. In this study, the caved classification was generally applied to stopes that experienced greater than 30% dilution, but it must be noted that the effect of dilution will vary depending on mineralized content and individual mine financial structure. The operational philosophy at a particular minesite also has to be considered when discussing the effect of dilution. Smaller stopes tend to have longer stand-up times, and as a result, may be mucked longer and generate high dilution values. Larger stopes may be more susceptible to caving and a reduction in the planned production period, which in turn can produce low dilution.

A summary of average cable densities and lengths is presented in Table 6.3 for the database collected in this study. The results vary with the type of cable pattern and are presented in relation to the pattern descriptions discussed in Chapter 3. Mandolin and hangingwall drift fan patterns are not included in Table 6.3 since only one case of each was collected in this database. The average density for stable square back patterns is 0.25 cable bolts/m^2, or 2 x 2 meters. Single cables were found to reflect a higher installed density than double cables. The average density for fan back patterns is in the region of

Table 6.3: Summary of Average Cable Density and Length

a) Square Back Pattern

	All Cases	Stable Cases	Stable Cases (single cables)	Stable Cases (double cables)	Non-entry Cases	Non-entry Stable Cases
Number of cases	20	17	8	9	11	8
Average cable density (bolts/sq. meter)	0.24	0.25	0.30	0.21	0.23	0.24
Average square pattern (m x m)	2.04	2.00	1.83	2.18	2.09	2.04
Average length (m)	11.5	11.7	13.0	10.5	9.3	8.8

b) Fan Back Pattern

	All Cases	Stable Cases	Stable Cases (single cables)	Stable Cases (double cables)	Non-entry Cases	Non-entry Stable Cases
Number of cases	4	4	3	1	3	3
Average cable density (bolts/sq. meter)	0.49	0.49	0.58	0.45	0.45	0.45
Average square pattern (m x m)	1.43	1.43	1.31	1.49	1.49	1.49
Average length (m)	8.0	8.0	8.1	7.6	8.1	8.1

c) Even Hangingwall Pattern

	All Cases	Stable Cases	Stable Cases (single cables)	Stable Cases (double cables)	Non-entry Cases	Non-entry Stable Cases
Number of cases	2	2	2	0	2	2
Average cable density (bolts/sq. meter)	0.13	0.13	0.13	N/A	0.13	0.13
Average square pattern (m x m)	2.77	2.77	2.77	N/A	2.77	2.77
Average length (m)	7.1	7.1	7.1	N/A	7.1	7.1

d) Point Anchor Hangingwall Pattern

	All Cases	Stable Cases
Number of cases	13	7
Average cable density (bolts/sq. meter)	0.047	0.046
Average ring spacing (m)	2.43	2.34
Average number of cables/ring	5.4	6.1
Average length (m)	9.5	10.7

e) Point Anchor Back Pattern

	All Cases	Stable Cases
Number of cases	4	2
Average cable density (bolts/sq. meter)	0.20	0.16
Average ring spacing (m)	1.98	1.75
Average number of cables/ring	5.3	5.5
Average length (m)	9.0	9.5

0.49 bolts/m^2, reflecting a high concentration of bolts within a small area. Point anchor hangingwall patterns exhibit an average 2.4 meter ring spacing along strike with 5 bolts installed on each ring. The average point anchor density for hangingwalls is 0.047 bolts/m^2, approximately one fifth of the average density found for square back patterns. The low densities are a direct result of limited access and larger hydraulic radii associated with hangingwall support. The point anchor approach applied to backs is associated with smaller surface areas than hangingwalls, and exhibits densities above 0.10 bolts/m^2. The average cable lengths for square back support is expressed in terms of non-entry cases to reflect the open stope situation. Some cut and fill cases are included in this database and typically involve cables up to 18.3 meters that are designed to cover a number of mining lifts. The average length for open stope square back patterns is 9 meters. Average lengths for other patterns range from 7 to 11 meters.

6.3.1 Comparison with the *Modified Stability Graph* Design Ranges

The unsupported and supported data points are plotted on the *Modified Stability Graph* in Figures 6.1 and 6.2, respectively. Eighty-five percent of the collected case histories were found to agree with the design ranges proposed by Potvin. The unsupported database is limited in size since the main interest of this study was to examine supported case histories. In terms of the unsupported database, eighty-three percent of the stable surfaces plot above the *transition zone* while eighty percent of the caved surfaces plot below the *transition zone*. Figure 6.2 illustrates that twenty percent of the supported cases plot above the *transition zone*. This region is classified as stable without support and indicates a conservative approach to cable support design by some operators. Within the

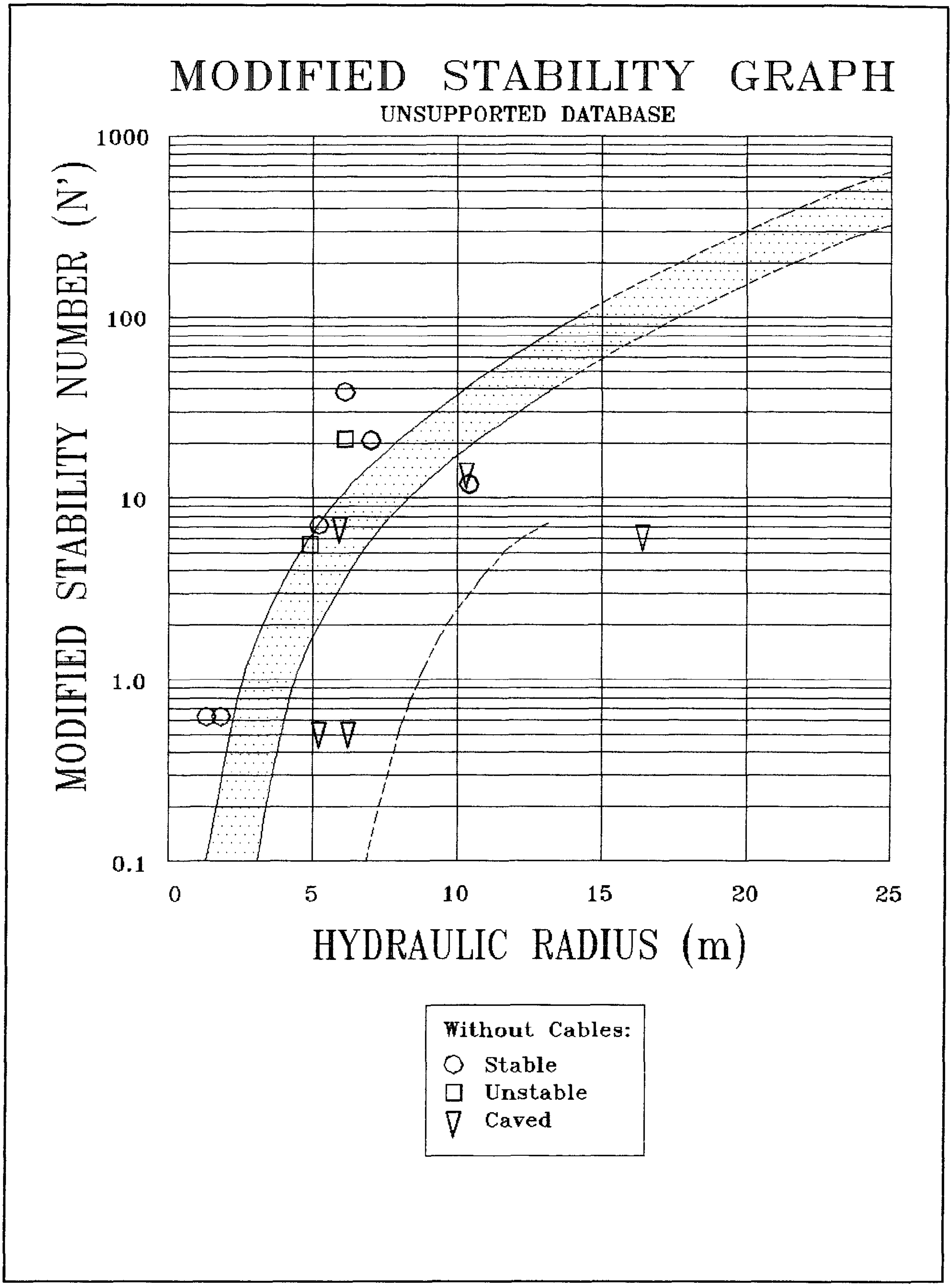


Figure 6.1: Unsupported case histories compared to the design ranges proposed by Potvin (1988)

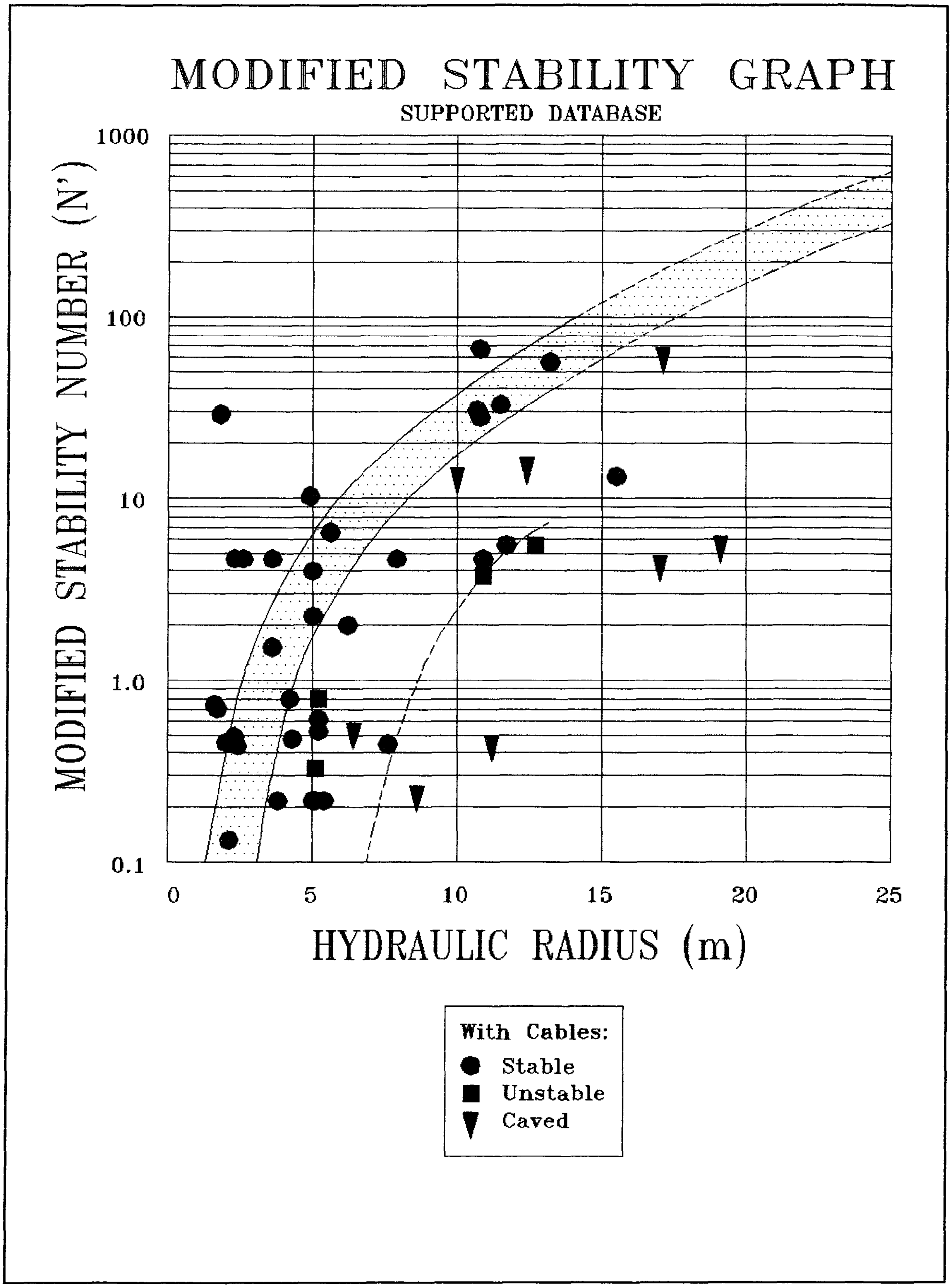


Figure 6.2: Supported case histories compared to the design ranges proposed by Potvin (1988)

Table 6.4: Hydraulic Radius Comparison		
DATABASE	NICKSON	POTVIN (1988)
Back Support Cases	29 (63%)	57 (86%)
Hangingwall Support Cases	17 (37%)	5 (8%)
Average HR (m) Backs	4.5 Range 1.6 - 11.2	7.5 Range 2.5 - 20.8
Average HR (m) Hangingwalls	12.2 Range 4.9 - 19.1	10.1 Range 5.0 - 19.7
Average HR (m) Stable Backs	3.9 Range 1.6 - 7.6	6.1 Range 2.5 - 14.6
Average HR (m) Stable Hangingwalls	10.8 Range 4.9 - 15.5	10.1 Range 5.0 - 19.7

supportable region below the *transition zone*, seventy-one percent of the cases were found to be stable. The *transition zone* was identified by Potvin (1988) as a region sensitive to the effects of blasting and time. Twenty-four percent of the supported cases were found to plot within the *transition zone*. A high degree of confidence is indicated by designing in or above the *transition zone* with support, since no cases of instability were found in these regions. The Potvin supported database has few successful cases with a hydraulic radius greater than 10 meters, and reflects the large percentage (86%) of back support cases. Table 6.4 shows that the average hydraulic radius of stope backs is significantly smaller than hangingwalls. In terms of design, Figure 6.2 shows that the supportable zone above a hydraulic radius of 10m is not as reliable as the region below 10m. This suggests that there is a limit to the hydraulic radius that can be effectively supported and the supportable region defined by Potvin (1988) on the *Modified Stability Graph* may not necessarily parallel the *transition zone*.

6.3.2 Comparison with the *Design Chart for Cable Bolt Density* Design Ranges

The collected case histories of back support are plotted on the *Design Chart for Cable Bolt Density* in Figure 6.3. Seventy-nine percent of the cases were found to be in agreement with the design ranges proposed by Potvin (1988). Only one case of instability was found when plotting in the *Conservative* and *Very Conservative* design zones. The database supports Potvin's observation that bolt densities of less than 0.10 bolts/m^2 are not used by mine operators. Unfortunately, the availability of caved cases was limited and the majority of the case studies obtained for stope backs were found to be stable.

6.4 STATISTICAL METHODOLOGY

Two statistical methods will be used in this chapter to analyze the database assembled for this study. The methods involve linear regression analysis and a form of discriminant analysis that will be used as a tool to develop design proposals for cable support. A brief review of both of these techniques is presented in this section.

6.4.1 Linear Regression Analysis

The determination of a relationship between two variables is often of interest in the statistical evaluation of a database. Regression defines the process of estimating a dependent variable y, from an independent variable x. The scatter diagram in Figure 6.4a represents a plot of two variables x and y, where (x_1,y_1), (x_2,y_2), ..., (x_n,y_n) represent n

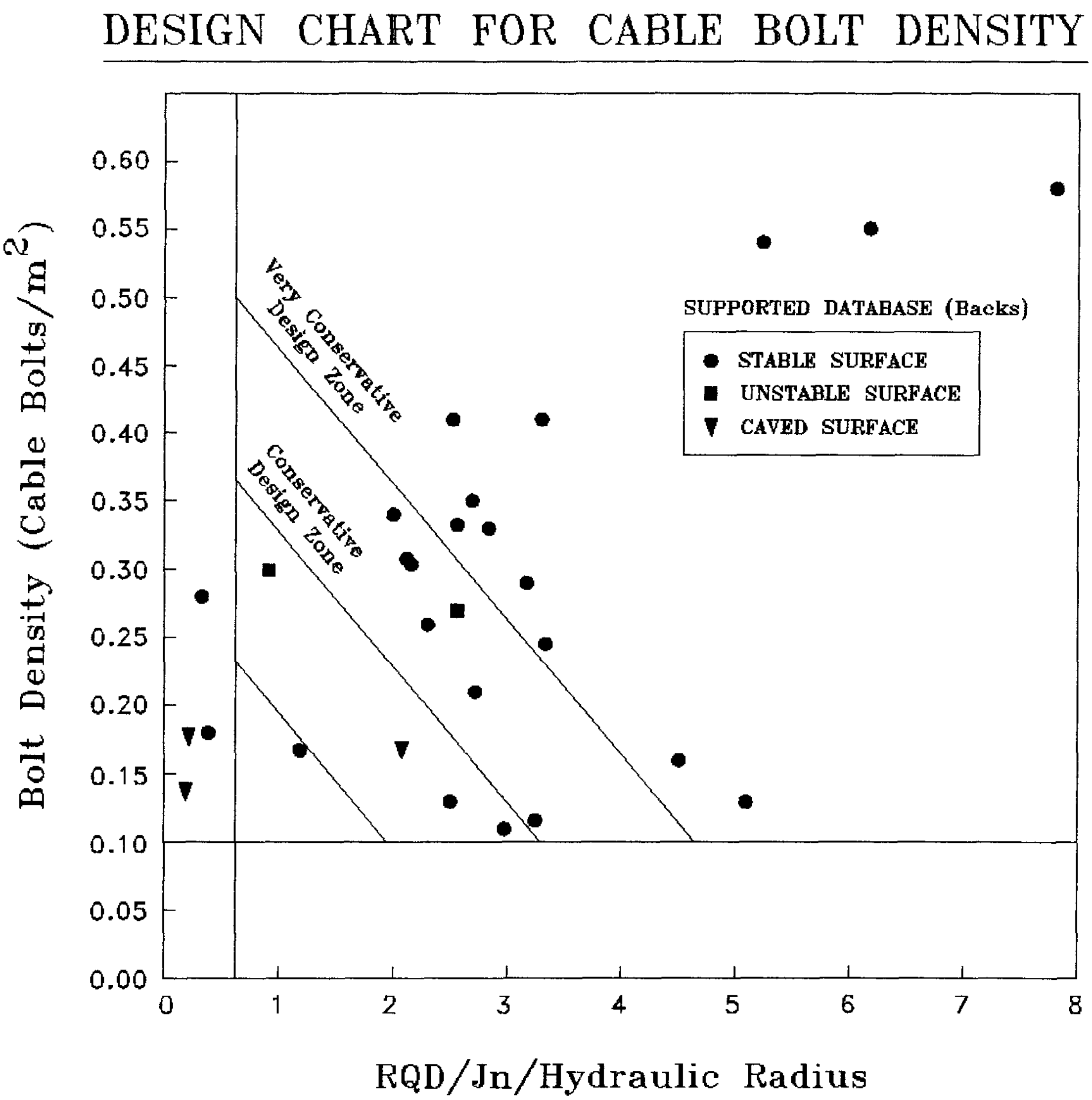


Figure 6.3: Cable bolted backs compared to the design ranges proposed by Potvin (1988)

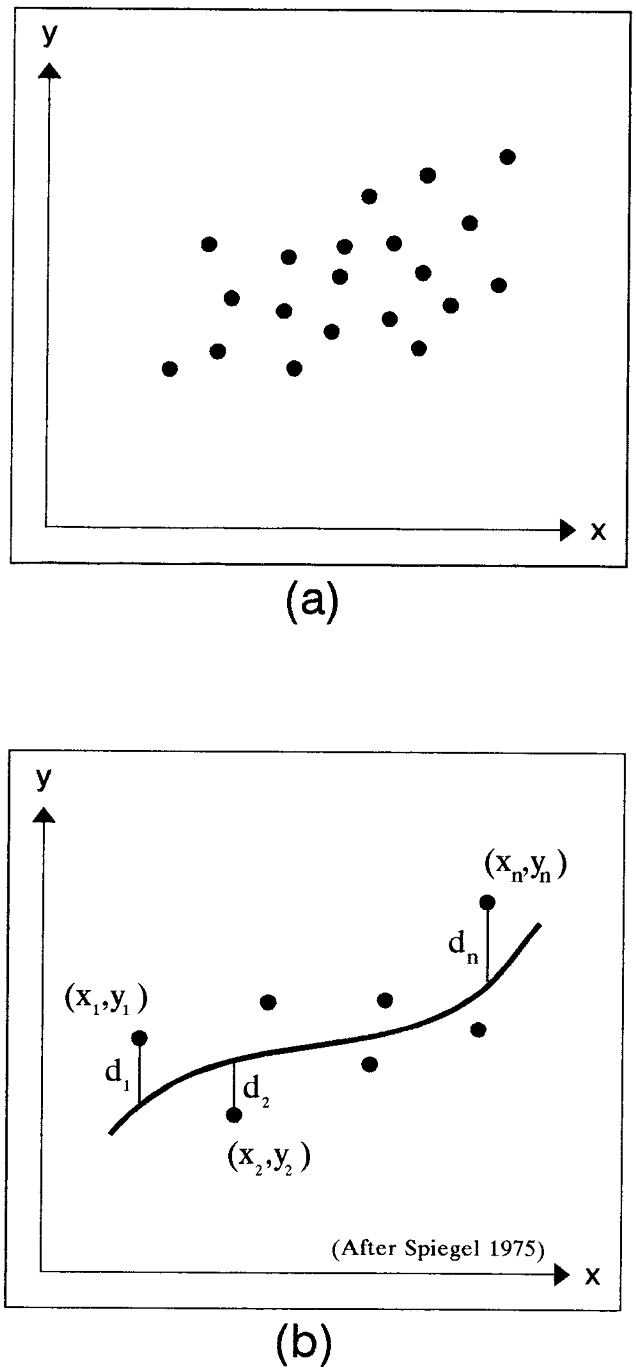


Figure 6.4: Concepts of linear regression analysis

individual sample points. A visual relationship may be apparent from the scatter diagram, but the regression technique provides a method of evaluating a mathematical relationship. Figure 6.4b illustrates the application of the method of least squares to estimate the "best-fitting curve" through a series of data points. The "best-fitting curve", or least squares regression curve, is defined by minimizing the sum of the residual values $d_1^2, d_2^2, \ldots, d_n^2$. Where a linear relationship is desired, the "line of best fit" can be derived (Spiegel 1975, 258) using the method of least squares, and is defined by

$$y = ax + b \tag{6.1}$$

where a and b are constants determined by simultaneously solving the equations

$$\sum_{i=1}^{n} y_i = an + b\sum_{i=1}^{n} x_i \tag{6.2}$$

$$\sum_{i=1}^{n} x_i y_i = a\sum_{i=1}^{n} x_i + b\sum_{i=1}^{n} x_i^2. \tag{6.3}$$

The standard error of estimate (Spiegel 1975, 262) of y on x is a measure of the scatter around the regression line and can be calculated by

$$s_{yx} = \sqrt{\frac{\sum_{i=1}^{n} (y_i - y_{i_{est}})^2}{n-2}} \tag{6.4}$$

where y_{est} is the estimated value of y for a given value of x, as obtained from the regression line. The value $s_{y.x}$ has similar properties to standard deviation and can be used to construct lines parallel to the regression line. Lines constructed at vertical intervals of $s_{y.x}$, $2s_{y.x}$ and $3s_{y.x}$, would respectively bound 68%, 95% and 99.7% of the sample points, providing the sample size is large (n > 30) and approximates a normal distribution.

Correlation analysis can be used to measure the strength of a linear relationship between two variables. The correlation coefficient, *r*, is related (Johnson 1976, 100-107) to the sample variances and covariance of *x* and *y* where

(6.5*a*) $$\text{Sample variance of } x = s_x^2 = \frac{\sum_{i=1}^{n}(x_i-\bar{x})^2}{n-1}$$

(6.5*b*) $$\text{Sample variance of } y = s_y^2 = \frac{\sum_{i=1}^{n}(y_i-\bar{y})^2}{n-1}$$

(6.5*c*) $$\text{Sample covariance} = s_{xy} = \frac{\sum_{i=1}^{n}(x_i-\bar{x})(y_i-\bar{y})}{n-1}$$

and $\bar{x}$, $\bar{y}$ represent the sample mean of x and y.

The sample correlation coefficient, *r*, is defined by

(6.6) $$r = \frac{s_{xy}}{s_x\, s_y}$$

and has a value between -1 and +1. A positive correlation reflects a simultaneous increase in both variables, while a negative correlation indicates that one variable decreases as the other increases. When the absolute value of the correlation coefficient is high, a close linear relationship is said to exist between two variables. In particular, perfect linear correlation results in a correlation coefficient of ± 1, and arises when all points fall directly on a straight line. A value of zero is assigned to r if there is no correlation between the variables. Between $r = 0$ and $r = |\pm 1|$, there is a statistical decision point related to the sample size, that separates a zone of correlation from no correlation. The values of x and y discussed in this section represent a random sample that is taken from a population. Since every member of the population cannot be collected in a typical database, a sample is used to represent that population. The final analysis is to determine if the sample correlation coefficient, r, indicates that there is a dependency between the variables for the population from which the sample was taken. If ρ represents the population linear correlation coefficient, it is possible to propose that $\rho = 0$, which would indicate that the two variables are linearly unrelated. The proposal that $\rho = 0$ is referred to as the null hypothesis. Rejection of the null hypothesis implies that there is linear dependency between the variables, and the sample correlation coefficient is statistically significant (Johnson 1976, 480-483). Table 6.5 contains critical values for the sample correlation coefficient that can be used to consider the null hypothesis at different levels of significance, α (Johnson 1976, A47). The sample correlation coefficient is used as the test statistic and related to the number of degrees of freedom, df, where df = n - 2. The null hypothesis is rejected for a certain level of significance if the test statistic, r, is greater than the critical value found in Table 6.5 for a particular df and α. Spiegel (1975, 211-223) suggests that results found to be significant at $\alpha = 0.01$ are **highly significant**, results significant at $\alpha = 0.05$ but not at $\alpha = 0.01$ are **probably**

significant, and results found to be significant at levels larger than 0.05 are **not significant**. If the sample correlation coefficient is significant at α=0.05, then there is a 5% chance that the null hypothesis should have been accepted. The 5% level of significance is generally standard in hypothesis testing.

Table 6.5: Values of *r* for Different Levels of Significance

df	α			df	α		
	0.10	0.05	0.01		0.10	0.050	0.01
1	0.988	0.997	1.00	16	0.400	0.468	0.590
2	0.900	0.950	0.990	17	0.389	0.455	0.575
3	0.805	0.878	0.959	18	0.378	0.444	0.561
4	0.729	0.811	0.917	20	0.360	0.423	0.537
5	0.669	0.754	0.874	25	0.323	0.381	0.487
6	0.621	0.707	0.834	30	0.296	0.349	0.449
7	0.582	0.666	0.798	35	0.275	0.325	0.418
8	0.549	0.632	0.765	40	0.257	0.304	0.393
9	0.521	0.602	0.735	45	0.243	0.287	0.372
10	0.497	0.576	0.708	50	0.231	0.273	0.354
11	0.476	0.553	0.683	60	0.211	0.250	0.325
12	0.457	0.532	0.661	70	0.195	0.232	0.302
13	0.441	0.514	0.641	80	0.183	0.217	0.283
14	0.426	0.497	0.623	90	0.173	0.205	0.267
15	0.412	0.482	0.605	100	0.164	0.195	0.254

df = the number of degrees of freedom

6.4.2 Discriminant Analysis

In this study a series of case histories have been collected and described in terms of three variables. In statistical terms the case histories form a three dimensional multivariate database. In all cases, two variables, for example (HR, N') or (RQD/Jn/HR, Cable Density), are quantified and a third is determined from the stability condition of the surface. The stability condition is a qualitative variable, and is expressed in terms of being either stable, unstable or caved. For design purposes, it is necessary to separate the database into zones of stability. Discussions with the Department of Statistics at the University of British Columbia were initiated with the goal of developing a method of statistical analysis that could be used to separate an empirical database based on stability. Discriminant analysis is a multivariate technique that is concerned with either separating observations into different groups, or assigning a new observation to existing groups (Johnson and Wichern 1988, 470). The method that was selected for use in this study, was a form of discriminant analysis that has the ability of separating three dimensional multivariate data into different groups, using a statistical measure of distance called the Mahalanobis distance (Seber 1984, 10). This form of discriminant analysis is designed to derive a linear function that will separate a database into two or more select groups.

If a random sample of two dimensional stable, unstable and caved points are represented by the following variables:

$$x_1 \,\ldots\ldots\ldots\, x_{n_1} \ - \ \textit{class of stable points where } x_1 = \begin{bmatrix} HR_1 \\ N_1 \end{bmatrix} \textit{ etc.}$$

$$y_1 \,\ldots\ldots\ldots\, y_{n_1} \ - \ \textit{class of unstable points}$$

z_1 .... z_{n_3} – *class of caved points,*

then the sample mean vector for each class is given by Equation 6.7a, 6.7b and 6.7c (Johnson and Wichern 1988, 221).

(6.7*a*) $$\bar{x} = \frac{1}{n_1}\sum_{i=1}^{n_1} x_i$$

(6.7*b*) $$\bar{y} = \frac{1}{n_2}\sum_{i=1}^{n_2} y_i$$

(6.7*c*) $$\bar{z} = \frac{1}{n_3}\sum_{i=1}^{n_3} z_i \ .$$

The sample covariance matrices can be evaluated for each class using the following relationships:

(6.8*a*) *Stable Class:* $$S_1 = \frac{1}{n_1-1}\sum_{i=1}^{n_1} (x_i-\bar{x})(x_i-\bar{x})^T$$

(6.8*b*) *Unstable Class:* $$S_2 = \frac{1}{n_2-1}\sum_{i=1}^{n_2} (y_i-\bar{y})(y_i-\bar{y})^T$$

(6.8*c*) *Caved Class:* $$S_3 = \frac{1}{n_3-1}\sum_{i=1}^{n_3} (z_i-\bar{z})(z_i-\bar{z})^T.$$

If each class can be considered as multivariate normal distributions with the same variance, then a pooled sample covariance matrix, $\mathbf{S_P}$, (Johnson and Wichern 1988, 222) can be formed where:

$$S_P = \frac{(n_1-1)S_1+(n_2-1)S_2+(n_3-1)S_3}{n_1+n_2+n_3-3} \tag{6.9}$$

Figure 6.5a contains a representation of two classes of data, say stable and unstable cases, plotted on some fictional axis. If a new point, $\boldsymbol{l} = (l_1,l_2)$ is added to the database, into which group should it be placed? One technique that can be used to answer this question is the Mahalanobis squared distance technique, where D_i^2 is the Mahalanobis squared distance from $\boldsymbol{l}$ to each class, determined using the following relationships (Seber 1984, 10):

$$D_1^2 = (\boldsymbol{l}-\bar{\boldsymbol{x}})^T S_P^{-1}(\boldsymbol{l}-\bar{\boldsymbol{x}}) \textit{ for the stable class} \tag{6.10a}$$

$$D_2^2 = (\boldsymbol{l}-\bar{\boldsymbol{y}})^T S_P^{-1}(\boldsymbol{l}-\bar{\boldsymbol{y}}) \textit{ for the unstable class.} \tag{6.10b}$$

The Mahalanobis distance differs from the straight Euclidean distance (Manly 1976, 47-52) in that it considers correlations between variables. The minimum Mahalanobis squared distance will define the class to which the point, $\boldsymbol{l}$, should be assigned. This technique would have to be repeated for every possible point within a design population and has been simplified to deal more efficiently with the data available in this study. Instead of assigning a point to a particular class, as illustrated in Figure 6.5a, the procedure has been revised to develop a linear boundary that separates two classes. A line exists

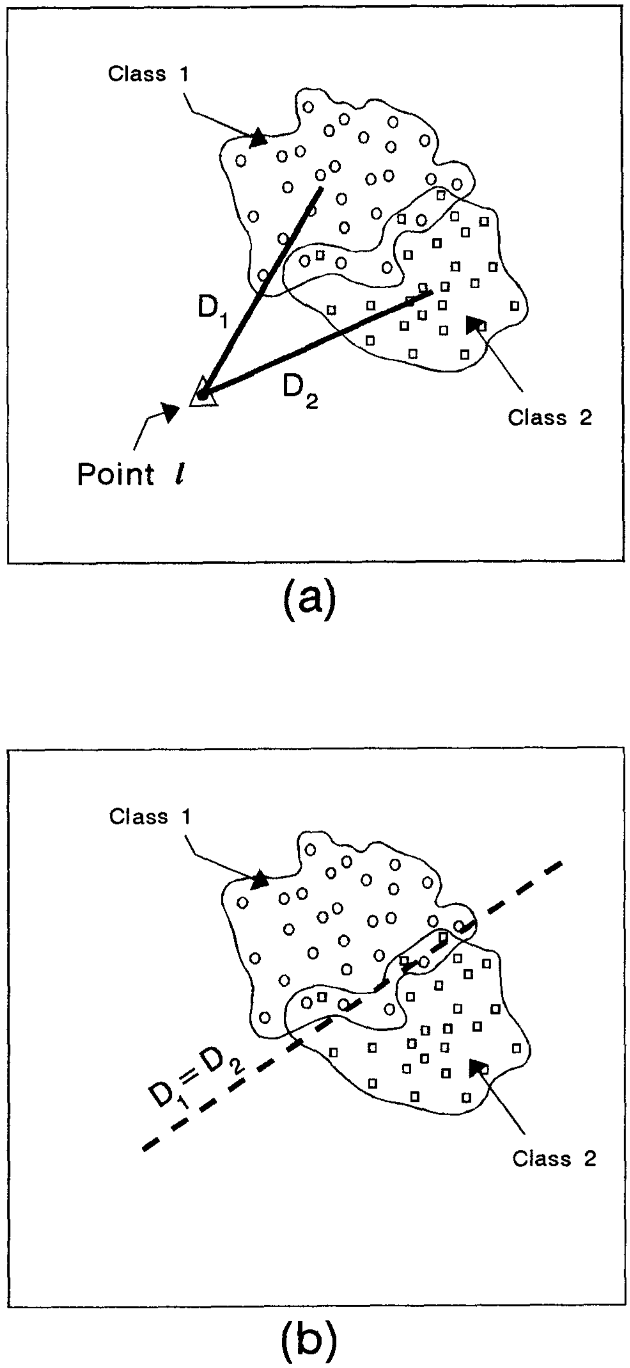


Figure 6.5: Methods of separating two classes of data

between two classes where the Mahalanobis squared distance to both classes is equivalent for any point on that line (Figure 6.5b). The case where $D_1^2 = D_2^2$ could represent a statistically derived division between a stable and unstable data class. Seber (1984, 10) notes that the Mahalanobis distance can be interpreted as a probabilistic distance where equal distances imply equal probability. Points that plot on the line defined by $D_1^2 = D_2^2$ can therefore be interpreted as having an equal likelihood of being in either class. A linear relationship can be determined for this case by first equating the squared distances for each class as in Equation 6.11.

$$(6.11) \qquad (\boldsymbol{l}-\bar{\boldsymbol{x}})^T \boldsymbol{S}_P^{-1} (\boldsymbol{l}-\bar{\boldsymbol{x}}) = (\boldsymbol{l}-\bar{\boldsymbol{y}})^T \boldsymbol{S}_P^{-1} (\boldsymbol{l}-\bar{\boldsymbol{y}})$$

Upon expanding this expression:

$$(6.12) \qquad \boldsymbol{l}^T \boldsymbol{S}_P^{-1} \boldsymbol{l} - \bar{\boldsymbol{x}}^T \boldsymbol{S}_P^{-1} \boldsymbol{l} - \boldsymbol{l}^T \boldsymbol{S}_P^{-1} \bar{\boldsymbol{x}} + \bar{\boldsymbol{x}}^T \boldsymbol{S}_P^{-1} \bar{\boldsymbol{x}} = \boldsymbol{l}^T \boldsymbol{S}_P^{-1} \boldsymbol{l} - \bar{\boldsymbol{y}}^T \boldsymbol{S}_P^{-1} \boldsymbol{l} - \boldsymbol{l}^T \boldsymbol{S}_P^{-1} \bar{\boldsymbol{y}} + \bar{\boldsymbol{y}}^T \boldsymbol{S}_P^{-1} \bar{\boldsymbol{y}}$$

and cancelling like terms, the following relation is obtained:

$$(6.13) \qquad (\bar{\boldsymbol{y}}^T - \bar{\boldsymbol{x}}^T) \boldsymbol{S}_P^{-1} \boldsymbol{l} + \boldsymbol{l}^T \boldsymbol{S}_P^{-1} (\bar{\boldsymbol{y}} - \bar{\boldsymbol{x}}) + \bar{\boldsymbol{x}}^T \boldsymbol{S}_P^{-1} \bar{\boldsymbol{x}} - \bar{\boldsymbol{y}}^T \boldsymbol{S}_P^{-1} \bar{\boldsymbol{y}} = 0$$

Since $\mathbf{y}^T\mathbf{x} = \mathbf{x}^T\mathbf{y}$ then Equation 6.13 can be simplified to:

$$(6.14) \qquad \boldsymbol{l}^T [2 \boldsymbol{S}_P^{-1} (\bar{\boldsymbol{y}} - \bar{\boldsymbol{x}})] + C = 0$$

Equation 6.14 represents the equation of a straight line where, (6.15) $$a_1 l_1 + a_2 l_2 + C = 0 \ ,$$

(6.16) $$\begin{bmatrix} a_1 \\ a_2 \end{bmatrix} = \underline{a} = 2S_P^{-1}(\bar{y}-\bar{x}) \ ,$$

and

(6.17) $$C = \bar{x}^T S_P^{-1} \bar{x} - \bar{y}^T S_P^{-1} \bar{y}$$

If (l_1,l_2) = (HR,N'), then an evaluation of a_1, a_2, and C can generate a linear division between two different classes, where each point is a three dimensional variable defined by the hydraulic radius, stability number, and stability condition. The method presented in this section can be used to separate more than two classes of points if desired.

6.5 STATISTICAL ANALYSIS

A series of three statistical tests were completed using the format outlined in Section 6.4.2 and a spreadsheet software package. The result of each test was a linear separation between stable and caved points that will be used to review the design ranges proposed by Potvin (1988). The three tests are summarized as follows:

Case #1: Derive a linear separation between stable and caved unsupported case histories and compare the result to the *transition zone* proposed by Potvin (1988) on the *Modified Stability Graph*.

Case #2: Derive a linear separation between stable and caved supported case histories and compare the result to the supportable region proposed by Potvin

(1988).

Case #3: Derive a linear separation between stable and caved cases of back support and compare the result to the stable and caved regions on the *Design Chart for Cable Bolt Density*.

The method of multivariate analysis described in the previous section requires that the sample be randomly collected, satisfy the condition of multivariate normality, and have similar variances. The database obtained in this study was combined with the Potvin (1988) database to improve upon the quality of analysis. This was also required to meet the conditions of multivariate normality and similar variance. Unstable cases were removed from the database, since their definition makes it difficult to consider a separate unstable class. In addition, the unstable case histories are limited, and an excessive variance was noted for Case #2 and #3 data, when compared to stable and caved classes. The results of this analysis therefore predicts a separation between stable and caved case histories that will enable the definition of possible design regions. The case histories were randomly selected in the sense that they were collected on the basis of availability, with no bias to the addition of one case or another. Unfortunately, the availability of case histories is limited due to access and time constraints, so there is not a large database from which to select a true random sample. The conditions of multivariate normality and similar variance were tested using dot plots and probability plots obtained using Systat (Wilkinson 1990b), a statistical software package. A dot plot is similar to a histogram (Wilkinson 1990a, 182) and graphically compares the variance of different classes of points. In order to have similar variances, which is a condition of the pooled sample covariance matrix, the lines for each class on a dot plot should have a similar measure of dispersion. Figure 6.6 illustrates the dot plots obtained for the combined unsupported

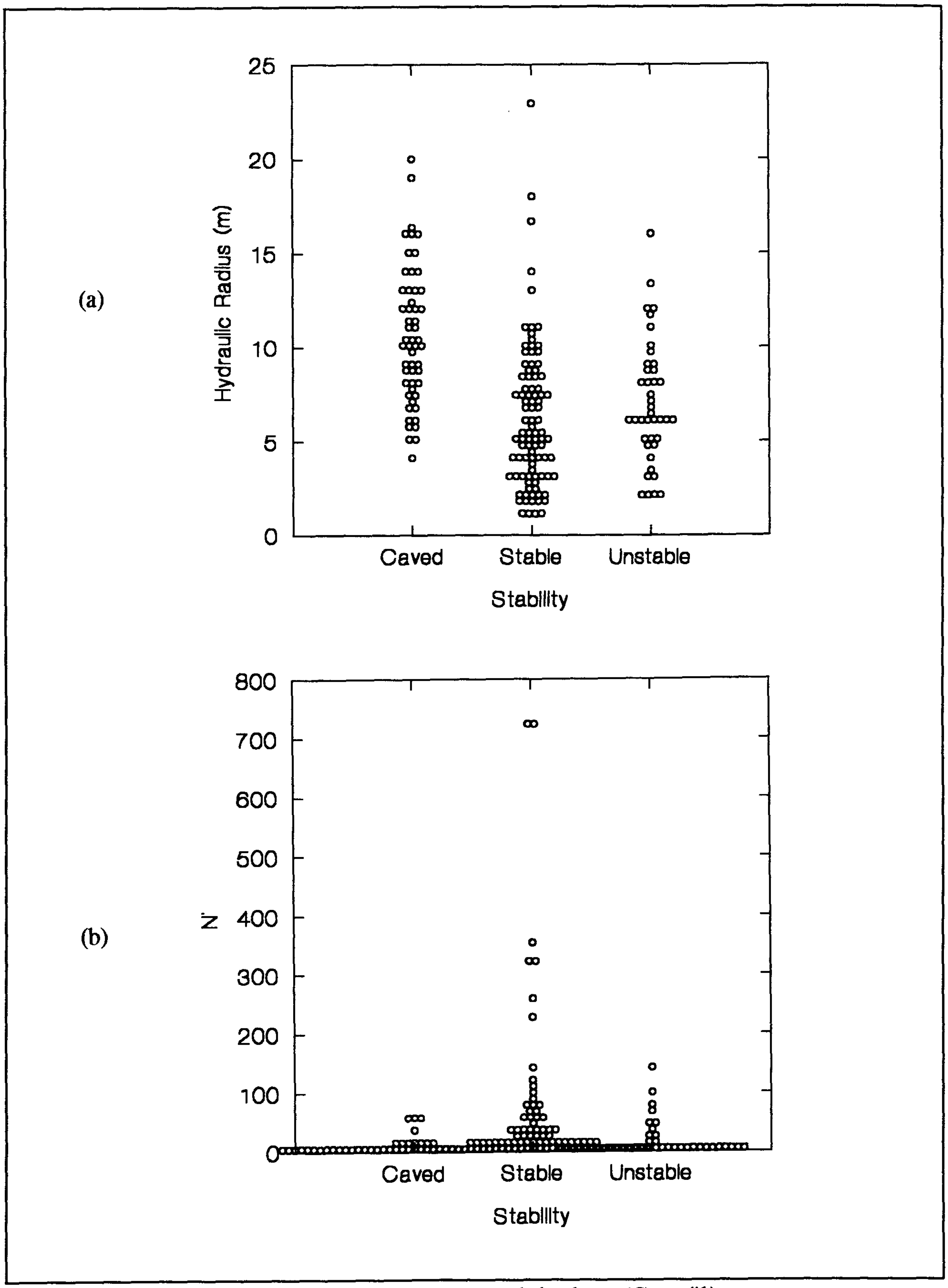


Figure 6.6: Dot plots for the combined unsupported database (Case #1)

database (Case #1). The stability number dot plot was interpreted as having excessive variance, and resulted in the application of a logarithmic transformation. Although not ideal, the dot plots obtained for the transformed data (Figure 6.7) were thought to have similar variances. The assumption of multivariate normality was checked through the use of normal probability plots. A normal probability plot compares the sorted database to the corresponding values of a mathematical normal distribution (Wilkinson 1990a, 345). The data should plot along a straight line if it is normally distributed. If the individual variables are normally distributed, then it can be assumed that the joint distribution is multivariate normal (Manly 1986, 15). The probability plots obtained from the unsupported database are shown in Figures 6.8 and 6.9. Both stable and caved classes cannot be considered normally distributed unless a logarithmic transformation is applied. Similar results were found for Case #2 and Case #3 data. The outcome of the analysis indicated that a logarithmic transformation had to be applied to meet the condition of normality. The transformation occasionally detracted from the condition of similar variance, but significantly improved the normality characteristics.

Prior to proceeding with the statistical analysis, a final review of the database was undertaken to determine if any case studies should not be considered. No cases from the combined unsupported database were eliminated and it is fully represented by the data in Tables 5.1, 5.2 and 6.1. Several cases of rebar support were removed from the Potvin supported database presented in Table 5.3, namely case number 278, 279, 280, 281, 285, and 286. It was felt that the statistical analysis should be restricted to cable support. Case number 43 in Table 6.2 was supported with grouted extension bolts and was removed for the same reason. Two other cases, number 1 and 25 were removed from the data in Table 6.2 prior to the statistical analysis. Case number 1 involves the use of cables installed

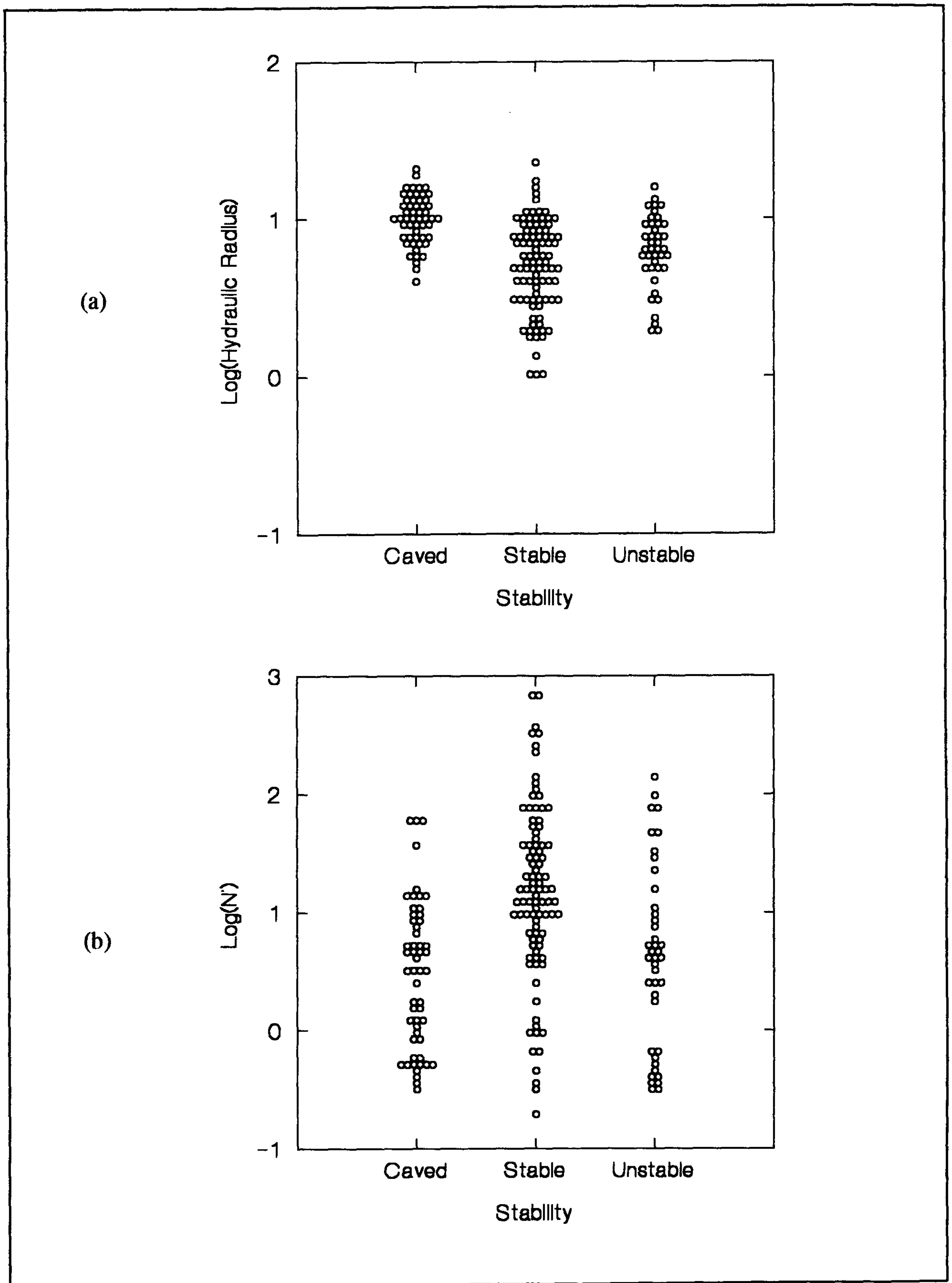


Figure 6.7: Dot plots for the combined unsupported database (Case #1 - transformed data)

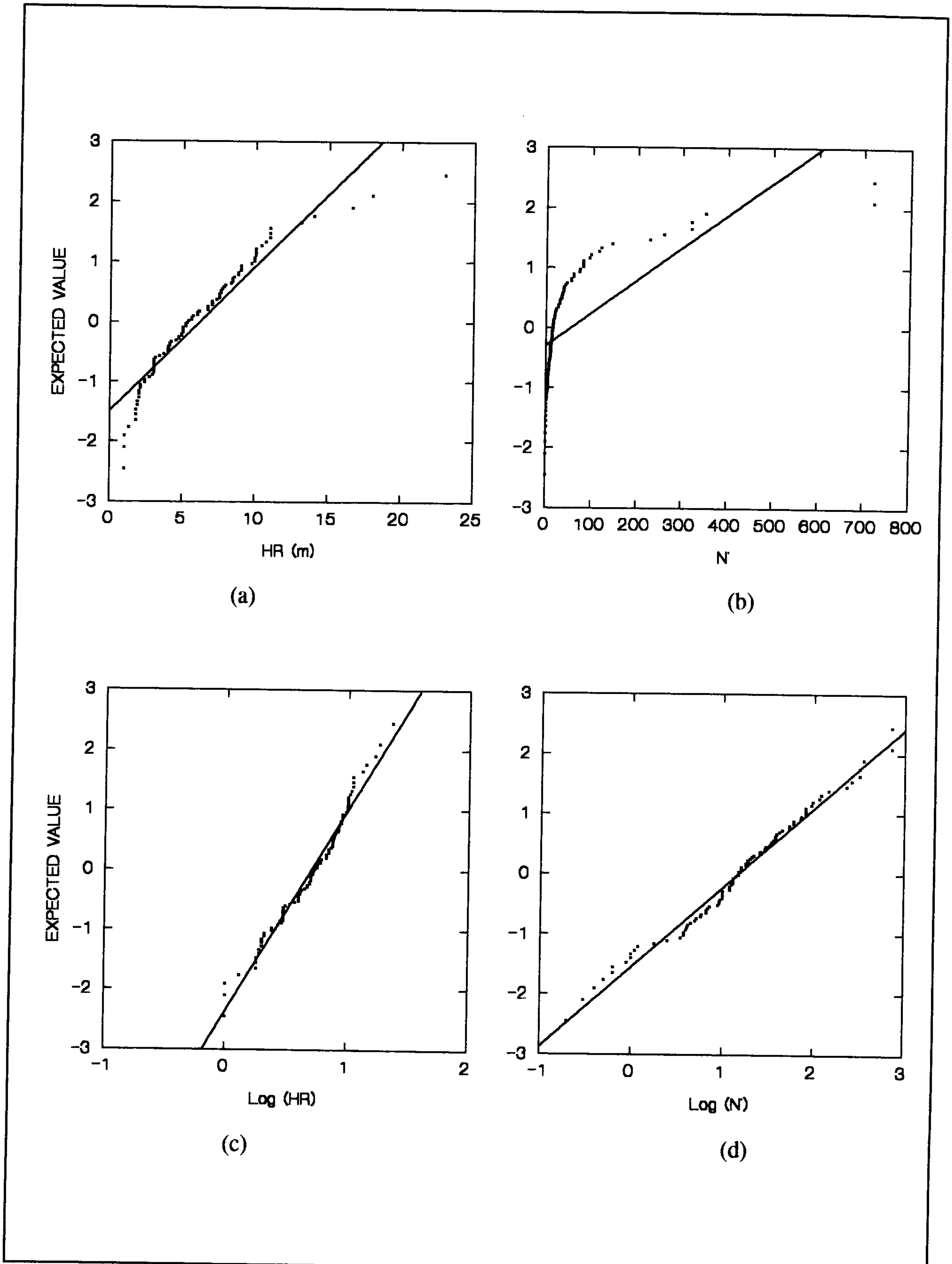


Figure 6.8: Probability plots for stable cases from the combined unsupported database (Case #1)

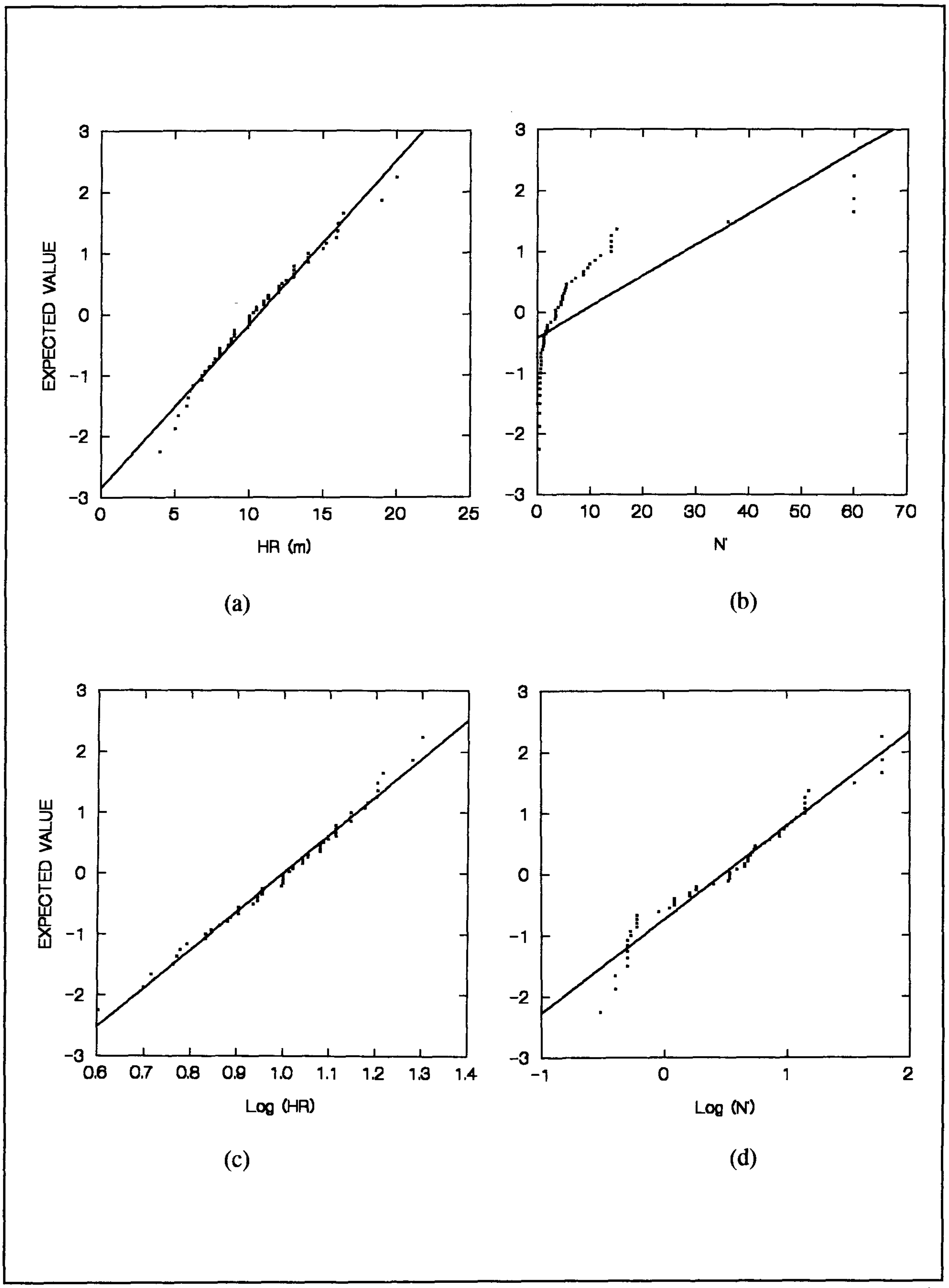


Figure 6.9: Probability plots for caved cases from the combined unsupported database (Case #1)

from a hangingwall drill drift, but the cable pattern was not designed to cover the complete hangingwall. In addition, a limited free face was available at the time of the final blast in this stope, and was thought to significantly contribute to the ensuing failure. Case number 25 involved two cable bolted stopes that were originally separated by a small rib pillar. The pillar between these stopes partially collapsed, and the case study was based on the new exposed stope back. Blasting practice was again thought to affect the stability of the surface and was combined with an uneven distribution of cables in the back. The remaining 103 cases in Tables 5.3 and 6.2 comprise the combined supported database used in the statistical analysis.

6.5.1 Unsupported Database (Case #1)

The results of the statistical analysis performed on the combined unsupported database is shown in Figure 6.10, and is mathematically represented in Equation 6.18.

$$HR\ (meters) = 10^{(0.573\ +\ 0.338 \log N')} \qquad (6.18)$$

This line was found to compare well with the *transition zone* proposed by Potvin (1988), except for a possible increase in the size of the stable region at a higher hydraulic radius. The extremities of the statistical line however are prone to the most error, since the data points are sparsely distributed in these regions. The location of many unstable surfaces within the *transition zone* proposed by Potvin (1988), suggests that this zone has already been well defined, and no further adjustment is required at this stage. The Potvin (1988) *transition zone* is recommended for the design of unsupported stope surfaces, and will be subsequently referred to as the *unsupported transition zone*.

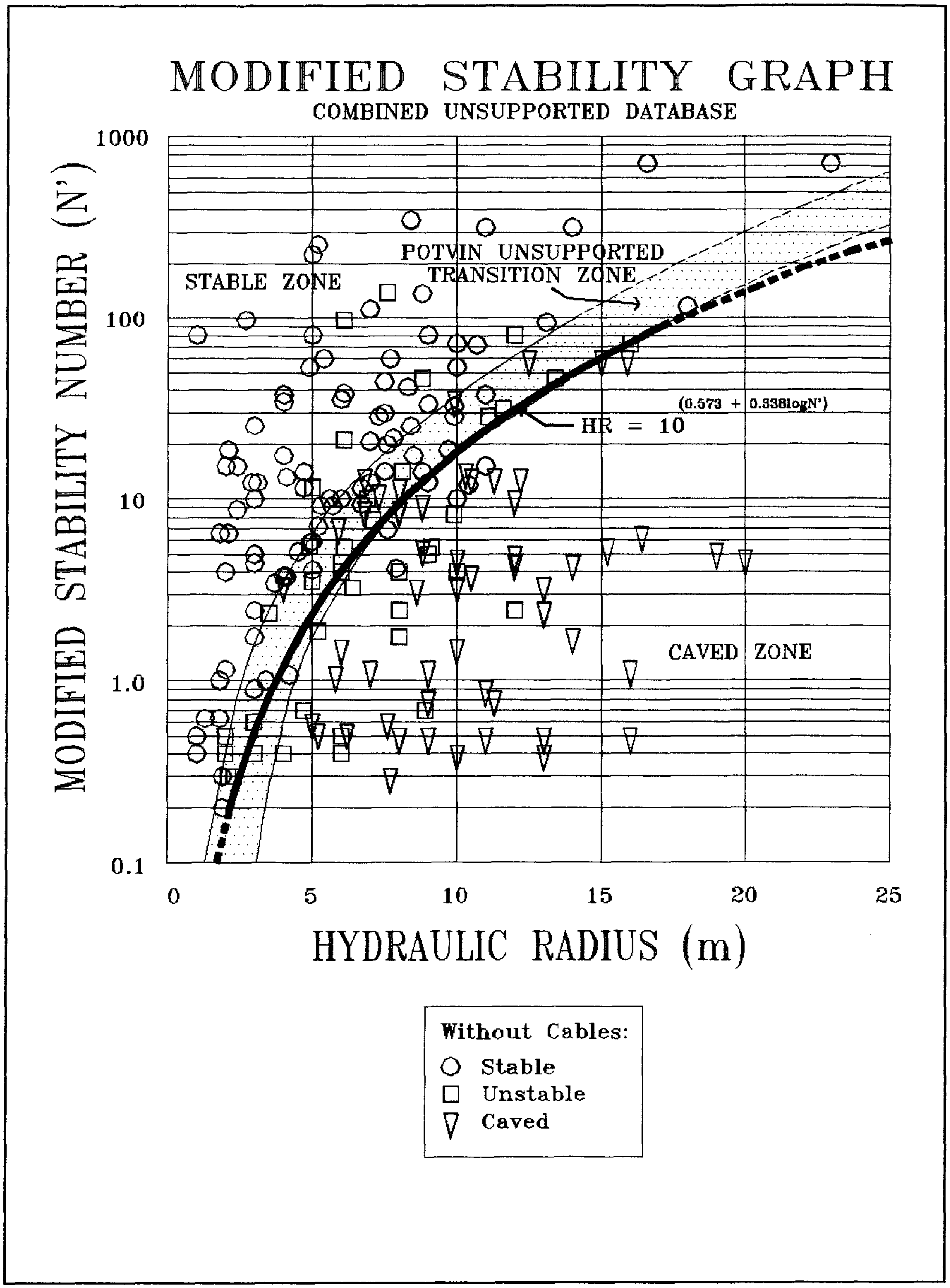


Figure 6.10: Statistical analysis of stable and caved combined unsupported database (Case #1)

6.5.2 Supported Database (Case #2)

The results of the statistical analysis for the combined supported database is shown in Figure 6.11, and is mathematically represented in Equation 6.19.

$$HR\ (meters) = 10^{(0.872\ +\ 0.171 \log N')} \tag{6.19}$$

The line separating stable and caved supported surfaces intersects the *unsupported transition zone* at a hydraulic radius of approximately 15 m. This suggests that the supportable region may be smaller than originally proposed by Potvin (1988) and does not necessarily parallel the *unsupported transition zone*.

In terms of design confidence, 100% of the supported case histories that plot in or above the *unsupported transition zone* are stable. Eighty-five percent of the cases plotting between the *unsupported transition zone* and the derived statistical line (Zone A on Figure 6.11) are stable. The stability of the points plotting between the derived statistical line and the Potvin (1988) lower supportable line (Zone B) is much more variable with only 58% of the cases being stable. This analysis suggests there is a reduction in design confidence as points plot farther into the caved zone. A modification to the supported *Modified Stability Graph* (Potvin 1988) is proposed in Figure 6.12. The statistical analysis revealed that the supportable region does not parallel the *unsupported transition zone* and is therefore reflected in the proposed modification. This concept is an indication of the limitations of cable bolt support in underground mining applications. It suggests that the addition of cable support beyond a stability number of approximately 80 and a hydraulic radius of 16 m will not change the stability condition of the surface. The block size at this

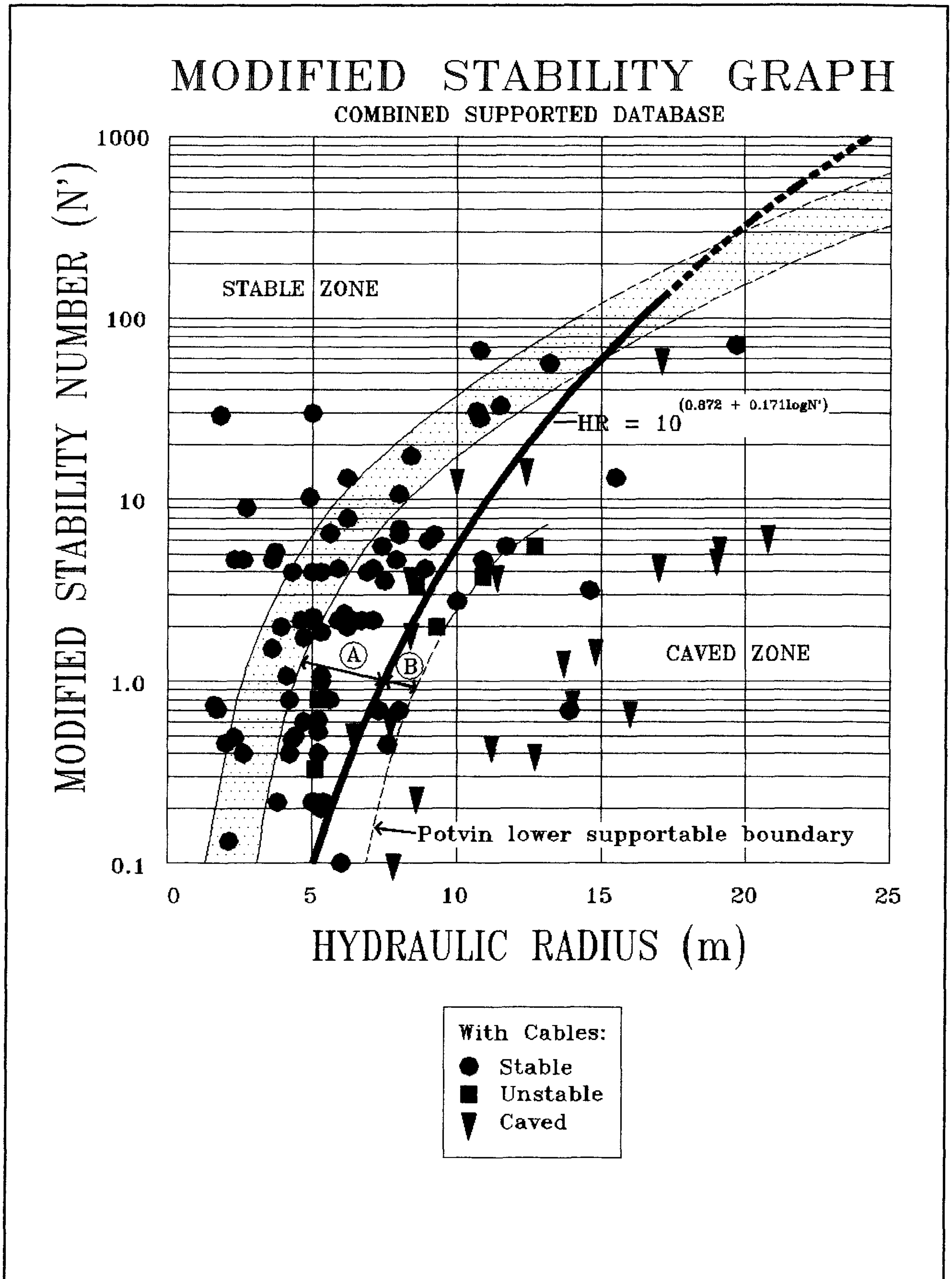


Figure 6.11: Statistical analysis of stable and caved combined supported database (Case #2)

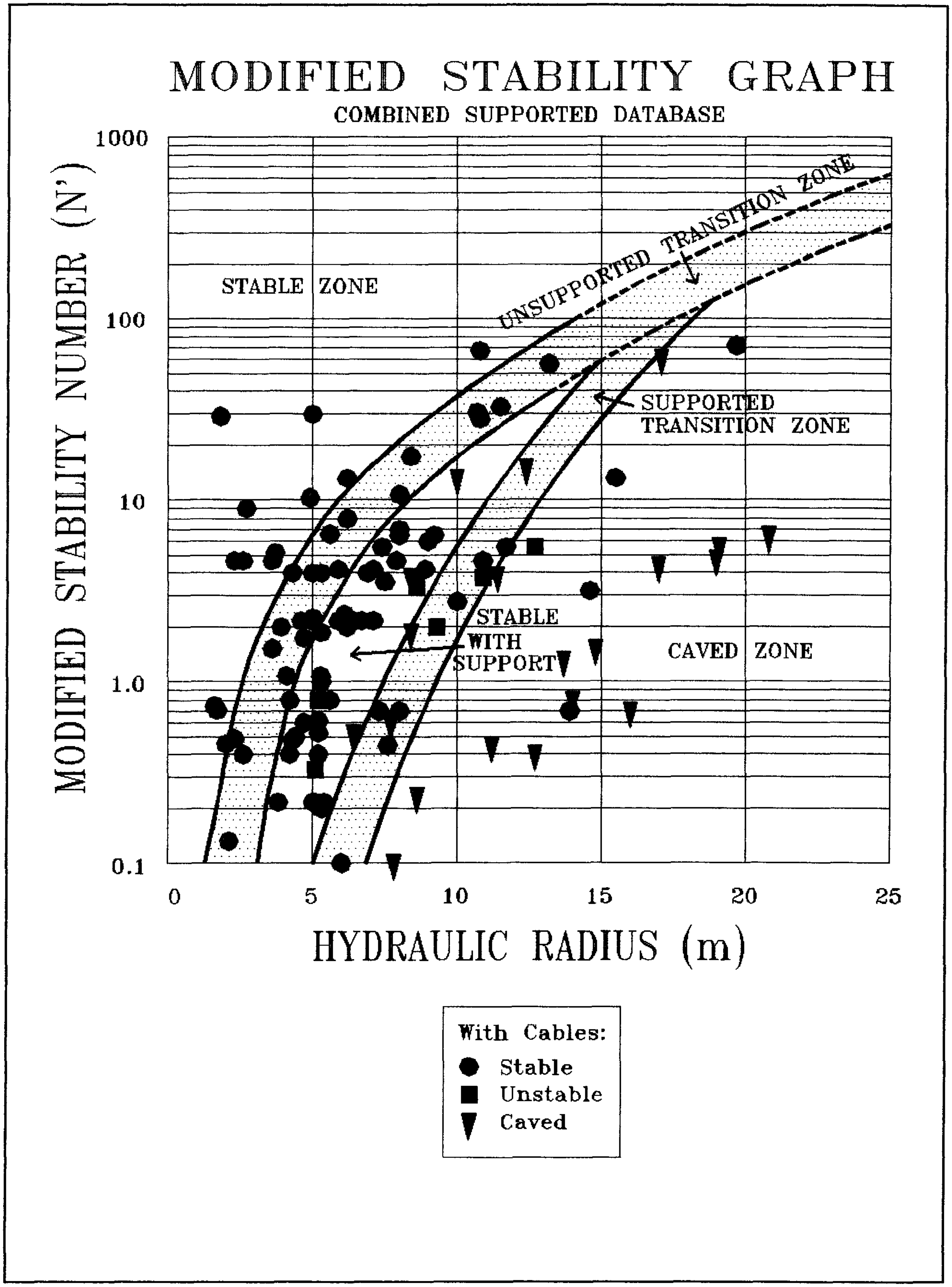


Figure 6.12: Proposed modifications to the *Modified Stability Graph*

cutoff point is likely too large to be effectively supported given economic constraints pertaining to cable length. In terms of scale, it has been shown in Table 6.4 that there is limited success with a hydraulic radius greater than 10 m. A *supported transition zone* has been incorporated in Figure 6.12 to reflect the reduction in design confidence farther into the caved zone of the graph. The location of the *supported transition zone* is based on the statistical line that was derived in this analysis and the edge of the supportable region proposed by Potvin (1988). Since a transition zone was found to exist for the unsupported database, it is unlikely that a single line can adequately separate stable and caved supported cases. The region between the two proposed transition zones is called the *stable with support zone* and reflects the high degree of design confidence suggested by the database.

6.5.3 Back Cable Support Database (Case #3)

The method of discriminant analysis discussed in Section 6.4.2 was used to divide the stable and caved cases of back support on the *Design Chart for Cable Bolt Density* (Potvin 1988). The statistical line is illustrated on Figure 6.13 and is mathematically reflected in Equation 6.20.

$$\text{(6.20)} \qquad \textit{Cable Density}\ (bolts/m^2) = 10^{(-0.697\ -\ 4.207\log(RQD/Jn/HR))}$$

The line proposed by Potvin (1988) traces a line that joins points A, B, C, and D on Figure 6.13. The statistical interpretation follows the same trend as Potvins' line, but suggests that the limiting relative block size factor may be closer to 1.0. A possible

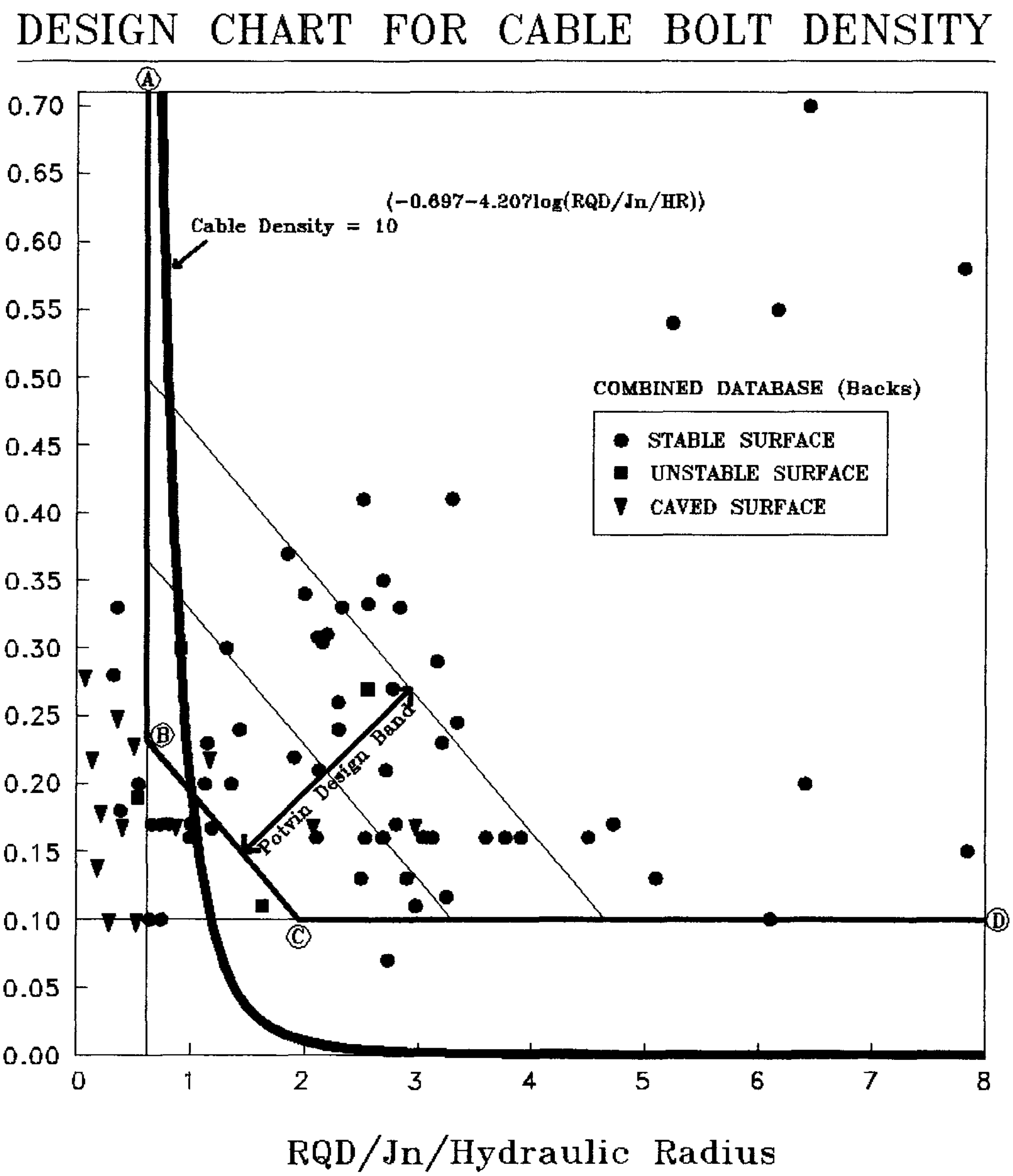


Figure 6.13: Statistical analysis of combined cable bolted back database (Case #3)

transition zone, as illustrated on Figure 6.14, could be considered between a relative block size factor of 0.75 and 1.0. The design band proposed by Potvin (1988) suggests that cable bolt density should increase with a decreasing relative block size factor. The combined database illustrated in Figure 6.13 does not seem to correlate with this particular trend. However, it should be noted that the design ranges proposed by Potvin (1988) and the statistical analysis illustrated on Figure 6.13, involved the use of the entire back support database. Figure 6.15 isolates cases of back support that plot within the *unsupported transition zone*, the *stable with support zone*, and the *supported transition zone* proposed in Figure 6.12. A design line that reflects a minimum cable bolt density is incorporated within each chart, based on maintaining 100% stability above the line. This method of analysis produces increasing cable bolt density farther into the caving zone of the *Modified Stability Graph*, but shows no correlation with the relative block size factor. The relative block size factor may not represent the ideal relationship with cable bolt density for back support. This thesis will examine other possible relationships to improve upon the definition of design zones for back cable support.

The techniques of linear regression analysis discussed in Section 6.4.1 were used to review statistically relevant design trends related to stable cases of back support. Table 6.6 summarizes the cases of back support that plot within the *stable with support zone* and the *supportable transition zone*. With cable bolt density as the dependent variable, a series of linear regression relationships were evaluated for different independent variables, utilizing the stable cases from Table 6.6 and the Systat (Wilkinson 1990b) software package. These relationships are illustrated in Figures 6.16 to 6.22, and are summarized in Table 6.7. Figure 6.16 shows the regression line produced for the relationship between cable bolt density and the relative block size factor, RQD/Jn/HR. This is the basis for the

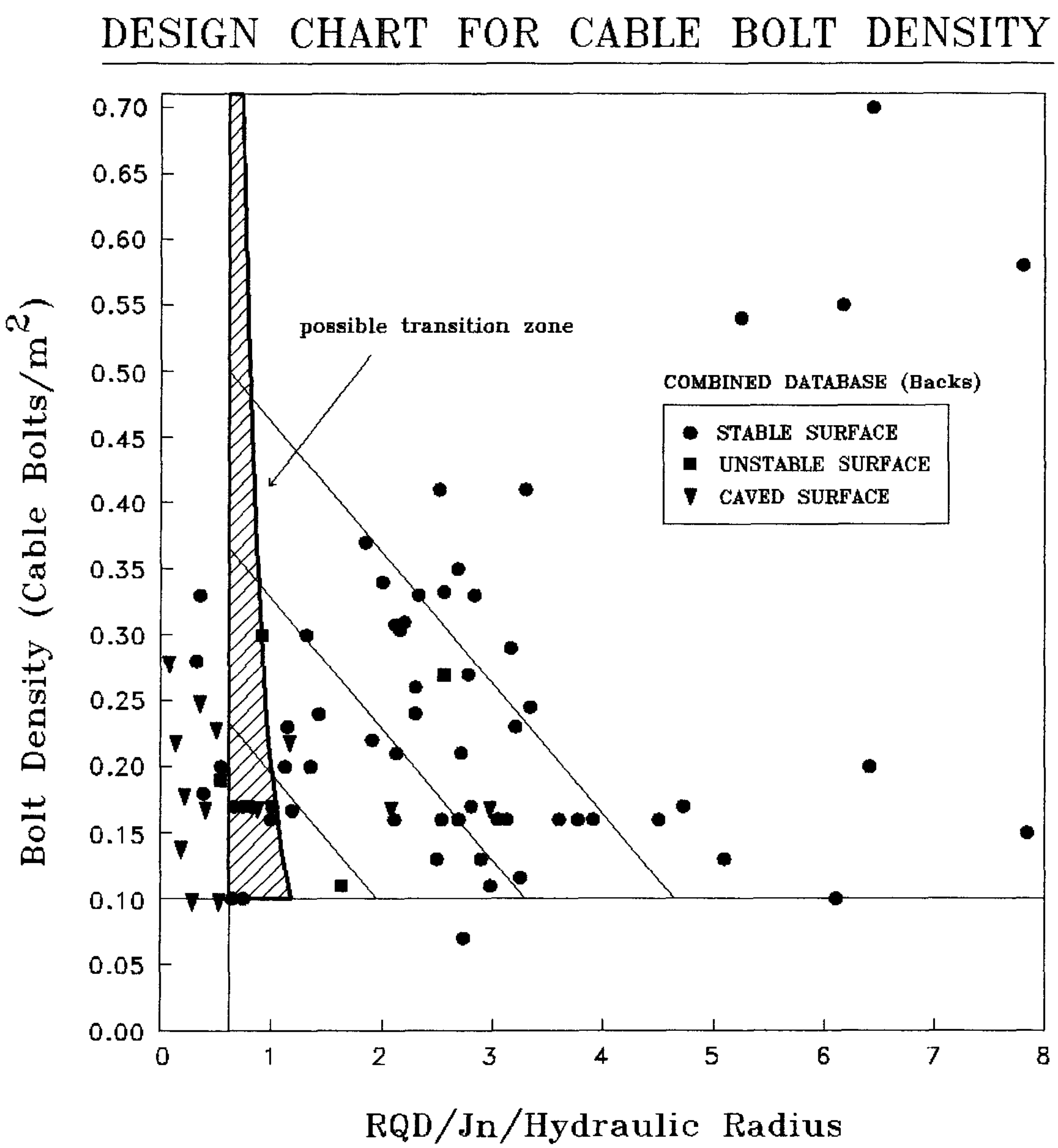
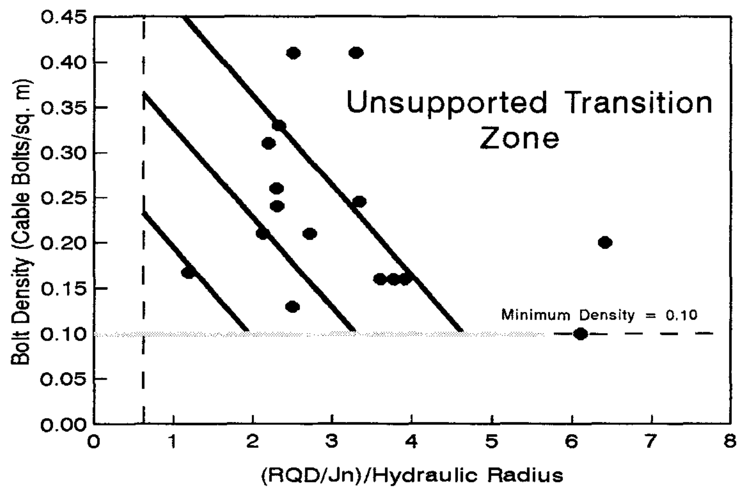
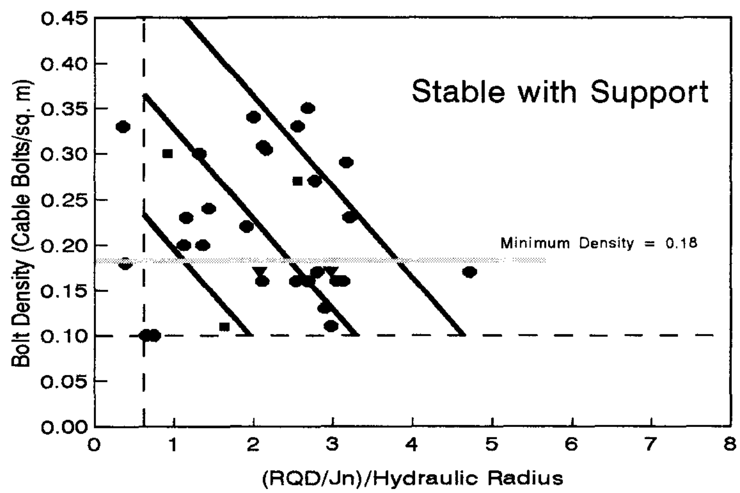


Figure 6.14: Possible transition zone for the *Design Chart for Cable Bolt Density*

(a)



(b)



(c)

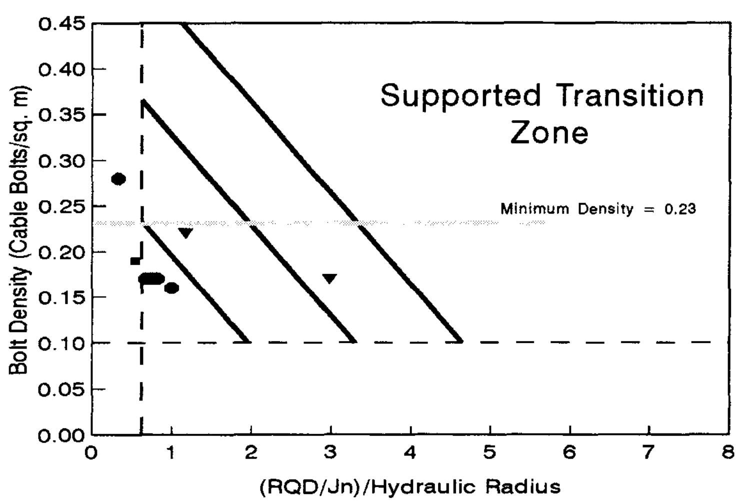


Figure 6.15: Minimum cable density for design regions of the revised *Modified Stability Graph*

Table 6.6: Cases of Back Support in the Supportable Region of the Revised Modified Stability Graph – Combined Database

Case #	Surface	Hydraulic Radius (m)	Q'	A	B	C	N'	Stability	RQD/Jn	RQD/Jn/HR	Cable Bolt Density (Bolts/Sq. m)	Cable Bolt Length (m)
14	Back	4.3	0.6	1.0	0.4	2.0	0.48	Stable	1.67	0.39	0.180	14.0
15	Back	7.6	0.5	1.0	0.3	3.0	0.45	Stable	2.5	0.33	0.280	8 &15
18	Back	4.2	13.3	0.1	0.3	2.0	0.80	Stable	13.3	3.17	0.290	9.8
19	Back	5.2	13.3	0.1	0.3	2.0	0.80	Unstable	13.3	2.56	0.270	9.8
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
34	Back	5.1	8.3	0.1	0.2	2.0	0.33	Unstable	4.7	0.92	0.300	9.1
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
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
55	Back	6.2	25.0	0.2	0.2	2.0	2.0	Stable	16.7	2.69	0.350	5.0
57	Back	5.4	5.4	0.1	0.2	2.0	0.22	Stable	10.8	2.00	0.340	6.4
251	+Back	8.4	18.75	0.25	0.2	2.0	1.9	Caved	25	2.98	0.17	21.0
252	+Back	8.4	18.75	0.5	0.2	2.0	3.8	Caved	25	2.98	0.17	21.0
253	+Back	5.3	18.75	0.25	0.2	2.0	1.9	Stable	25	4.72	0.17	21.0
254	+Back	6.4	54.0	0.1	0.2	2.0	2.2	Stable	18	2.81	0.17	21.0
256	+Back	5.9	54.0	0.1	0.2	2.0	2.2	Stable	18	3.05	0.16	9.0
257	+Back	6.7	54.0	0.1	0.2	2.0	2.2	Stable	18	2.69	0.16	9.0
258	+Back	7.1	54.0	0.1	0.2	2.0	2.2	Stable	18	2.54	0.16	9.0
263	Back	7.3	4.2	0.36	0.2	2.4	0.7	Stable	6	0.82	0.17 0.17	3.0 18.0
264	Back	6.0	0.8	0.1	0.4	2.0	0.1	Stable	4	0.67	0.17 0.17	3.0 18.0
265	Back	8.0	4.2	0.35	0.2	2.4	0.7	Stable	6	0.75	0.17 0.17 0.05	3.0 18.0 30.0
269	Back	4.4	6.0	0.2	0.2	2.0	0.5	Stable	6	1.36	0.20	3.0
270	Back	5.3	6.0	0.1	0.2	2.0	0.2	Stable	6	1.13	0.20	6.0
271	Back	5.3	10.5	0.1	0.5	2.0	1.1	Stable	7	1.32	0.30	9.0
274	+Back	4.2	9.0	0.1	0.2	2.0	0.4	Stable	6	1.43	0.24	6.0
277	Back	5.2	9.0	0.1	0.2	2.0	0.4	Stable	6	1.15	0.23	9.0
291	Back	8.0	9.0	1.0	0.3	2.4	6.5	Stable	6	0.75	0.10	20.0
293	Back	9.2	9.0	1.0	0.3	2.4	6.5	Stable	6	0.65	0.10	20.0
296	Back	5.3	25.5	0.1	0.2	2.0	1.0	Stable	17	3.21	0.23	9.0
297	Back	9.0	50.0	0.3	0.2	2.0	6.0	Stable	25	2.78	0.27	10.0
300	Back	4.7	16.2	0.1	0.2	2.0	0.6	Stable	9	1.91	0.22	12.0
301	Back	7.7	16.2	0.1	0.2	2.0	0.6	Caved	9	1.17	0.22	12.0
302	Back	5.6	2.0	1.0	0.2	2.0	0.8	Stable	2	0.36	0.33	10.0
306	Back	9.3	5.0	1.0	0.2	2.0	2.0	Unstable	5	0.54	0.19	10.0
309	Back	7.1	15.0	0.7	0.2	2.0	4.2	Stable	15	2.11	0.16	15.0
310	Back	8.0	25.0	0.7	0.2	2.0	7.0	Stable	25	3.13	0.16	10.0
311	Back	7.4	20.0	0.7	0.2	2.0	5.6	Stable	20	2.70	0.16	25.0
313	Back	10.0	10.0	0.7	0.2	2.0	2.8	Stable	10	1.00	0.16	18.0
315	Back	6.9	20.0	0.5	0.2	2.0	4.0	Stable	20	2.90	0.13	25.0
317	Back	8.6	21.0	0.1	0.8	2.0	3.4	Unstable	14	1.63	0.11	15.0

(+ Cables with Rebar)

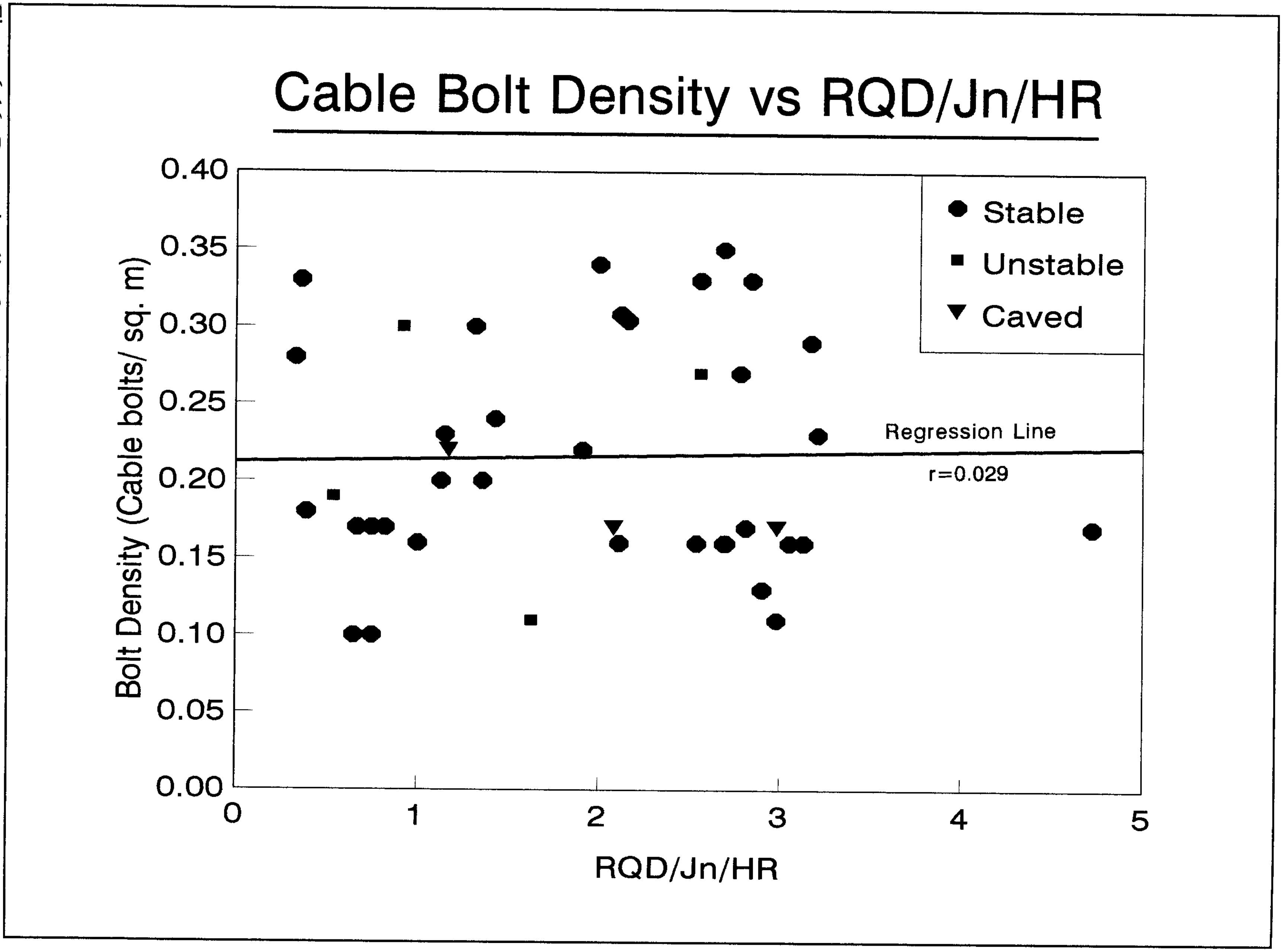


Figure 6.16: Regression line for cable bolt density versus RQD/Jn/HR

Table 6.7: Possible Linear Relationships for Cable Bolt Density

TEST SERIES	INDEPENDENT VARIABLE (x)	EQUATION	r	SIGNIFICANCE
1	RQD/Jn/HR	y = 0.212 + 0.002x	+0.029	NOT SIGNIFICANT
2	A x RQD/Jn/HR	y = 0.243 - 0.053x	-0.369	PROBABLY SIG.
	A_1 x RQD/Jn/HR	y = 0.228 - 0.013x	-0.203	NOT SIGNIFICANT
	A_2 x RQD/Jn/HR	y = 0.235 - 0.027x	-0.314	NOT SIGNIFICANT
	Log(A x RQD/Jn/HR)	y = 0.184 - 0.072x	-0.344	PROBABLY SIG.
	Log(A_1 x RQD/Jn/HR)	y = 0.203 - 0.041x	-0.268	NOT SIGNIFICANT
	Log(A_2 x RQD/Jn/HR)	y = 0.193 - 0.061x	-0.351	PROBABLY SIG.
3	Q'	y = 0.234 - 0.001x	-0.231	NOT SIGNIFICANT
	Q'/HR	y = 0.231 - 0.006x	-0.183	NOT SIGNIFICANT
	A x Q'/HR	y = 0.250 - 0.057x	-0.432	PROBABLY SIG.
	A_1 x Q'/HR	y = 0.223 - 0.007x	-0.126	NOT SIGNIFICANT
	A_2 x Q'/HR	y = 0.230 - 0.016x	-0.248	NOT SIGNIFICANT
	Log(Q'/HR)	y = 0.220 - 0.017x	-0.113	NOT SIGNIFICANT
	Log(A x Q'/HR)	y = 0.189 - 0.061x	-0.397	PROBABLY SIG.
	Log(A_1 x Q'/HR)	y = 0.202 - 0.043x	-0.341	PROBABLY SIG.
	Log(A_2 x Q'/HR)	y = 0.195 - 0.056x	-0.409	PROBABLY SIG.
4	N'/HR	y = 0.256 - 0.148x	-0.495	HIGHLY SIGNIFICANT
	Log(N'/HR)	y = 0.160 - 0.072x	-0.443	HIGHLY SIGNIFICANT

Design Chart for Cable Bolt Density (Potvin 1988), but no significant relationship is revealed. It is interesting to note that the regression line is roughly horizontal at a density of between 0.21 and 0.22 bolts/m^2. This corresponds closely with the average density of 0.25 bolts/m^2 presented in Table 6.3 for stable cases of square back support.

Stope backs that are subject to high stress may exhibit stress induced rock failure. The relative block size factor considers the effect of gravity but does not consider the influence of stress. There are two failure modes that must be considered when discussing the effects of stress in relation to cable bolt support. A low uniaxial compressive strength

(σ_c) to induced stress (σ_i) ratio, can induce failure within a rock mass and result in a potential reduction in the inherent block size. Mathews et al. (1981) proposed that failure occurs at a σ_c/σ_i ratio less than two and this is incorporated in the Potvin (1988) stress factor, A, at a value of 0.1. Jaeger and Cook (1979) suggest that rock can be regarded as being strong, massive and competent if its uniaxial strength is three or more times the field stress around an excavation. This suggests that stress will affect a rock mass below a σ_c/σ_i ratio of 3, which is equivalent to an A factor less than 0.3. Stress in stope backs can also enhance stability by clamping blocks together, but depending on the orientation of the critical joint, may induce a sliding failure. To consider the two stress induced failure mechanisms discussed, the stress factor, A, was considered in combination with the relative block size factor, as illustrated in Figure 6.17a. A modification of the A factor was expressed in terms of A_1 and A_2, to consider a reduction in block size as a result of high stress. The A_1 factor had a value of 0.1 or 1.0, and was created to account for an increase in block size due to stress induced fractures. The value of A_1 was set to 1.0 when the modified stability number stress factor was greater than 0.1. All other cases were assigned an A_1 value of 1.0. The A_2 factor considered the comments of Jaeger and Cook (1979), and was assigned a value of 0.1 when the modified stability number stress factor was less than 0.3. The A_2 factor for all other cases again went to 1.0. The relative block size factor is combined with A_1 and A_2 in Figures 6.17b and 6.17c, and the regression results are shown in Table 6.7. The best correlation resulted with the use of the modified stability number stress factor (A), and suggests that all ranges of stress should be considered in an empirical relationship with cable bolt density. A logarithmic transformation was applied, as illustrated in Figure 6.18, and displayed slightly higher correlations.

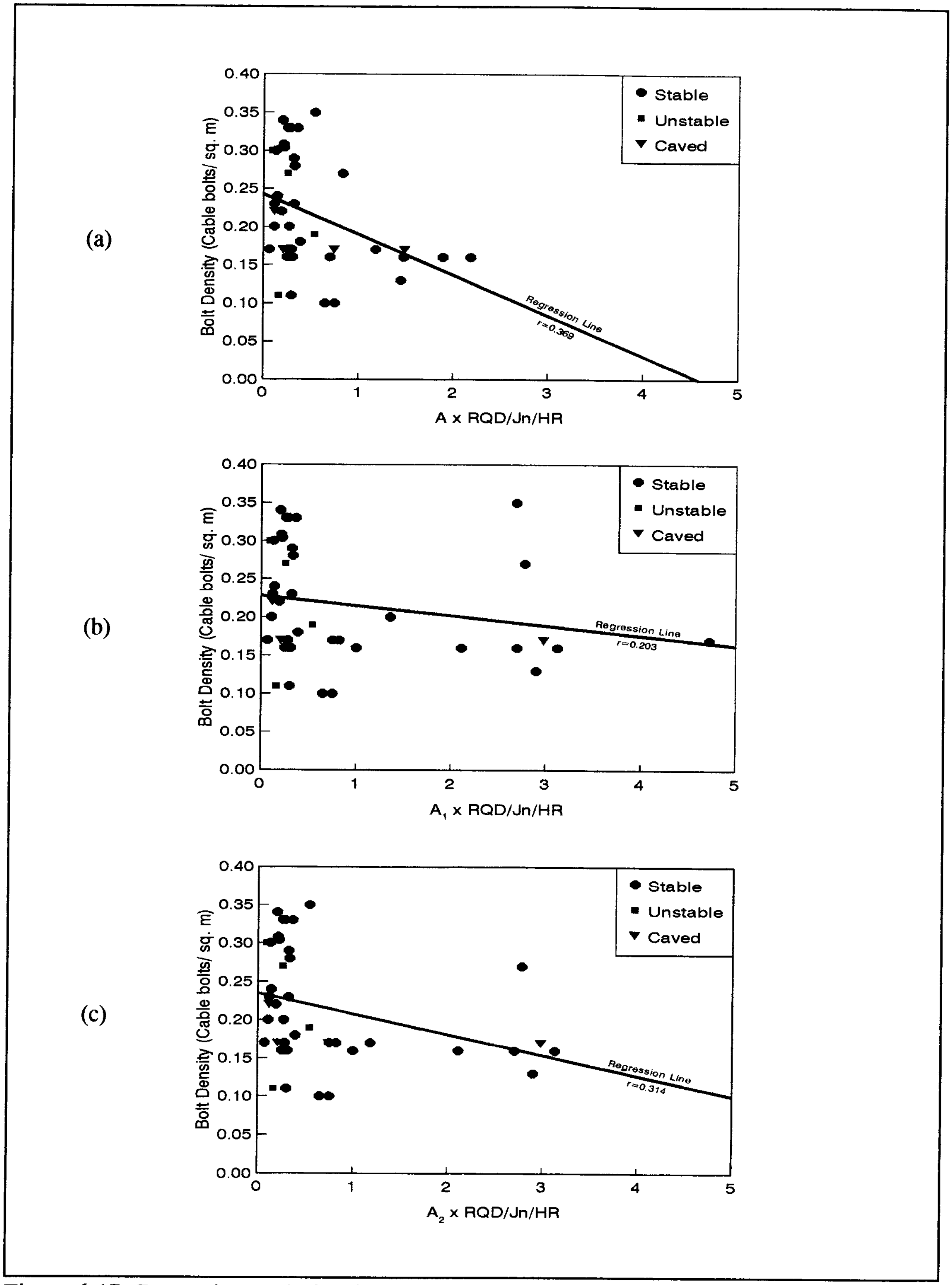


Figure 6.17: Regression analysis with the relative block size factor and the stress factor

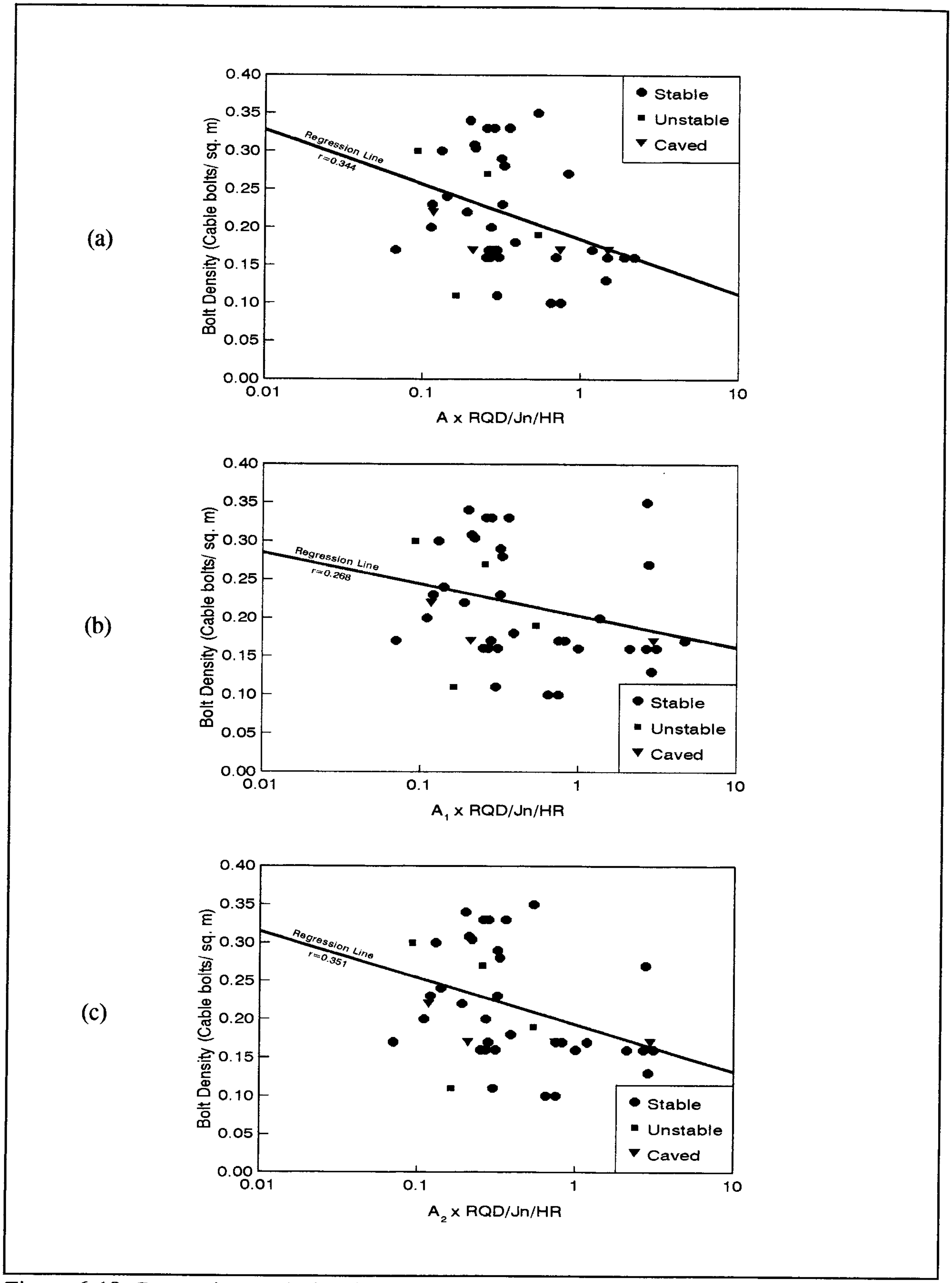


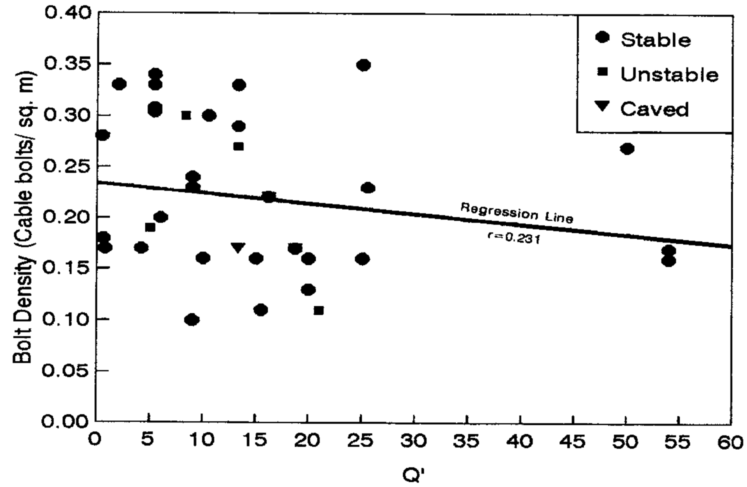
Figure 6.18: Regression analysis with the relative block size factor and the stress factor

Figures 6.19 to 6.21 incorporate the use of Q' within the independent variable. Table 6.7 shows that correlations are generally highest when Q' is combined with the stress factor, A. Based on this result, it was decided to incorporate the entire modified stability number within the independent variable. The results, as illustrated in Figure 6.22, revealed highly significant correlations for both N'/HR and Log (N'/HR), with respect to cable bolt density. The best correlation was obtained for the plot of bolt density versus N'/HR. The regression line shown in Figure 6.22a represents the line of best fit for the stable cases of back support, and is proposed as a guideline for the determination of cable bolt density for stope backs. The 68% confidence interval has been plotted on Figure 6.22 for both cases of regression analysis with N'/HR. Relating cable density to the modified stability number allows for the consideration of the state of stress in a supported back. Although stress is often thought of as a clamping force in relation to stope backs, it can also act as a driving force in a sliding failure defined by joint structure intersecting the design surface. The relative block size factor does not consider the effect of stress or joint orientation. The ratio N'/HR is directly related to the revised *Modified Stability Graph* and a range of selected ratios are plotted in Figure 6.23. Design cable bolt densities can be directly related from Figure 6.22a for each region on the revised *Modified Stability Graph*. Chapter 7 will pursue this concept further in the context of design.

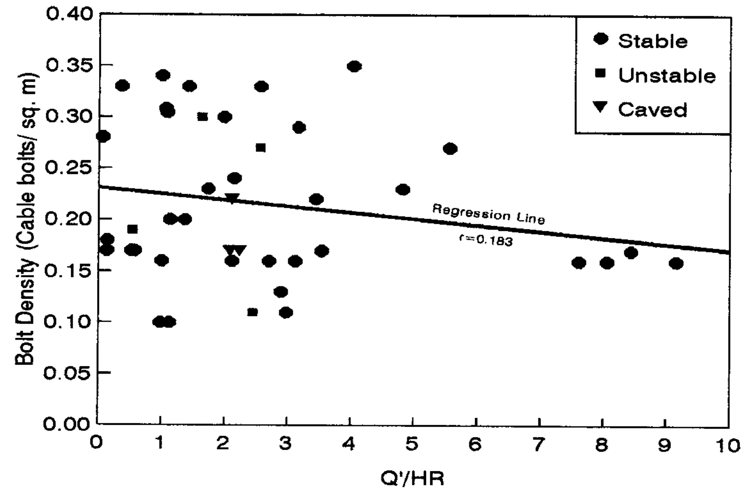
6.6 CONCLUSIONS

The unsupported *Modified Stability Graph* (Potvin 1988) has been statistically validated and is recommended for the design of stope surfaces. The supportable region

(a)



(b)



(c)

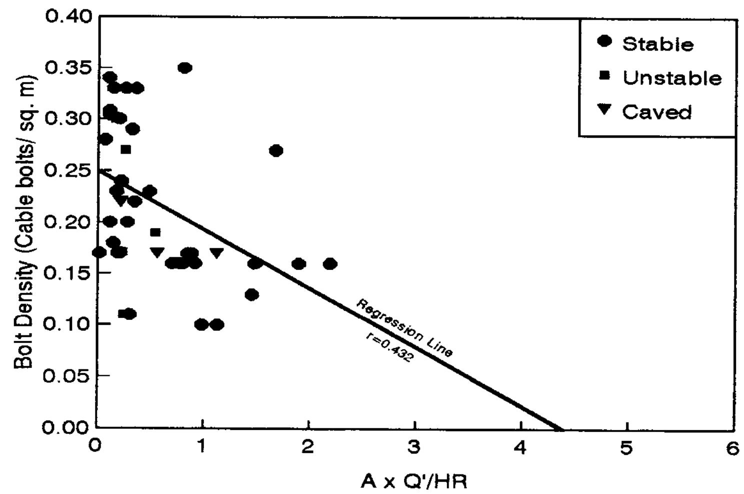


Figure 6.19: Regression analysis with Q'

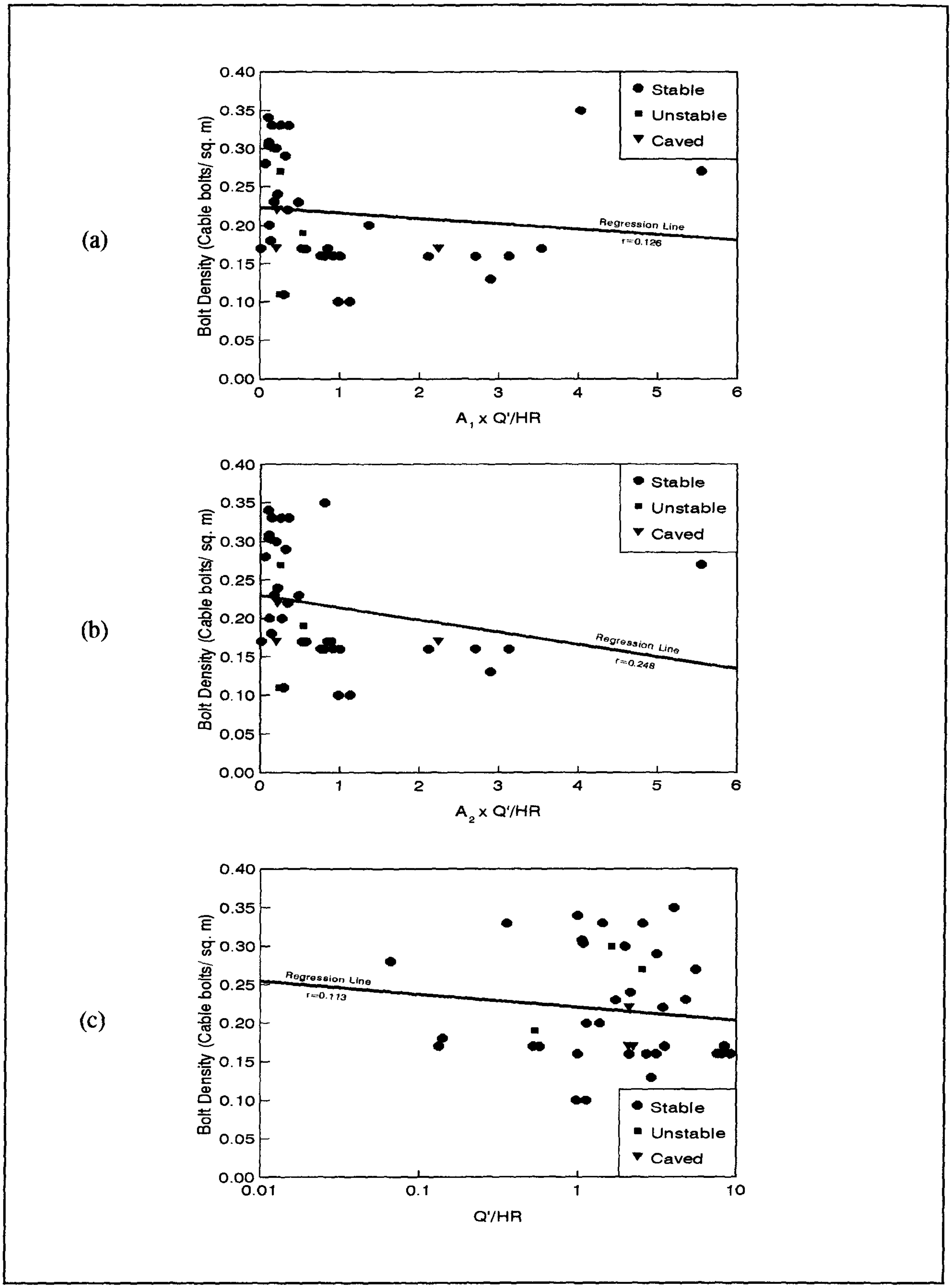


Figure 6.20: Regression analysis with Q’

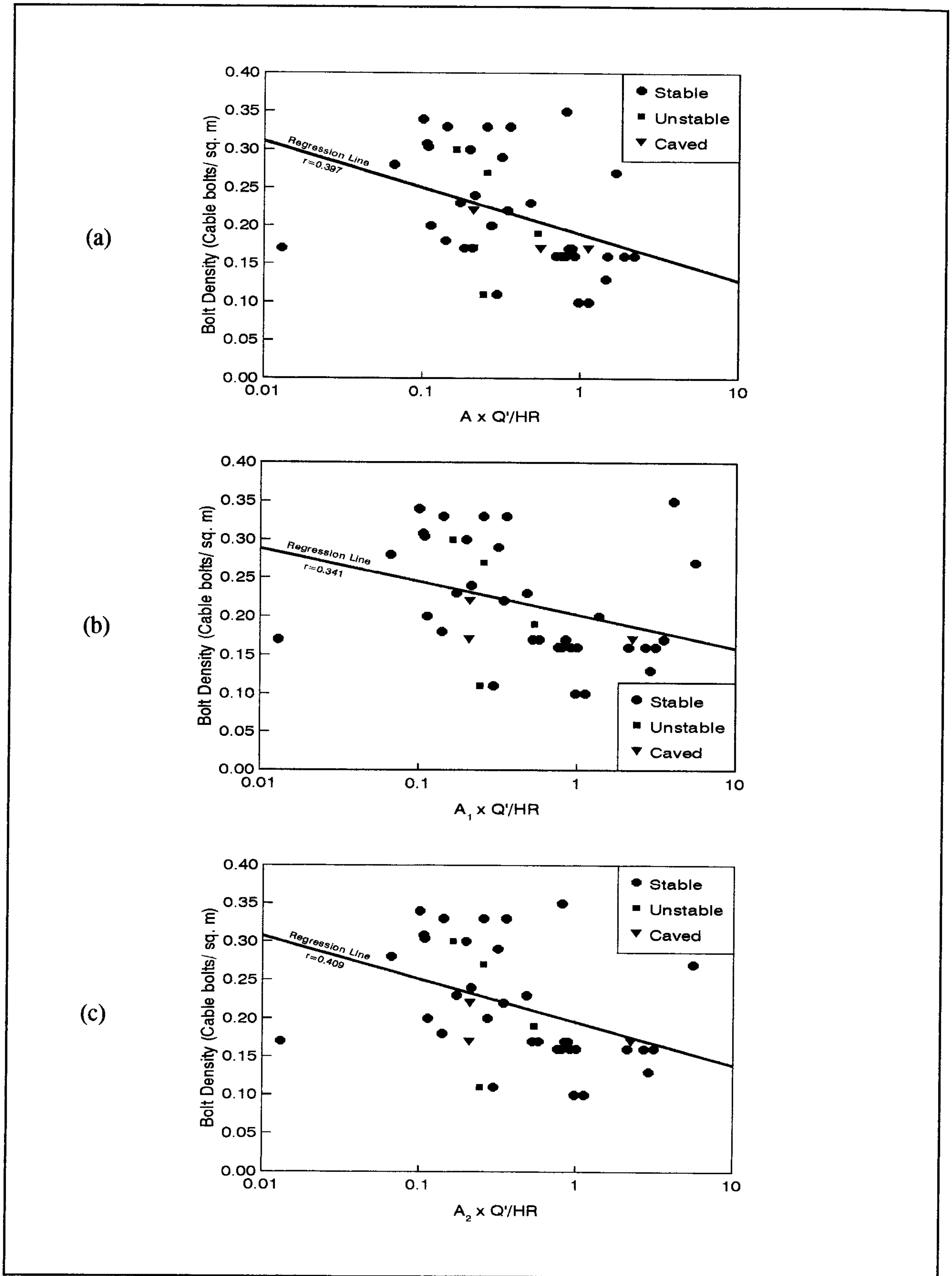


Figure 6.21: Regression analysis with Q'

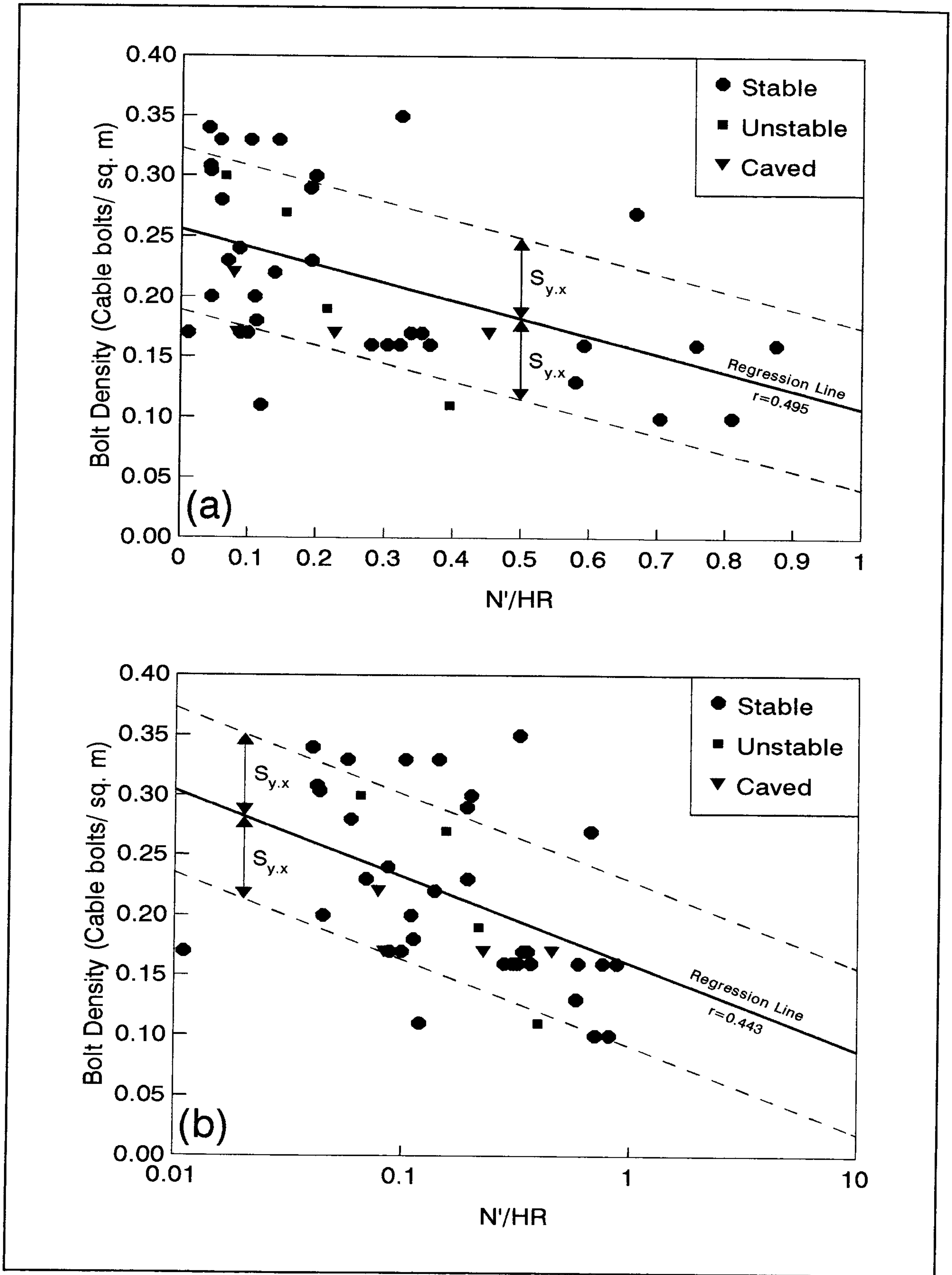


Figure 6.22: Regression analysis with N'

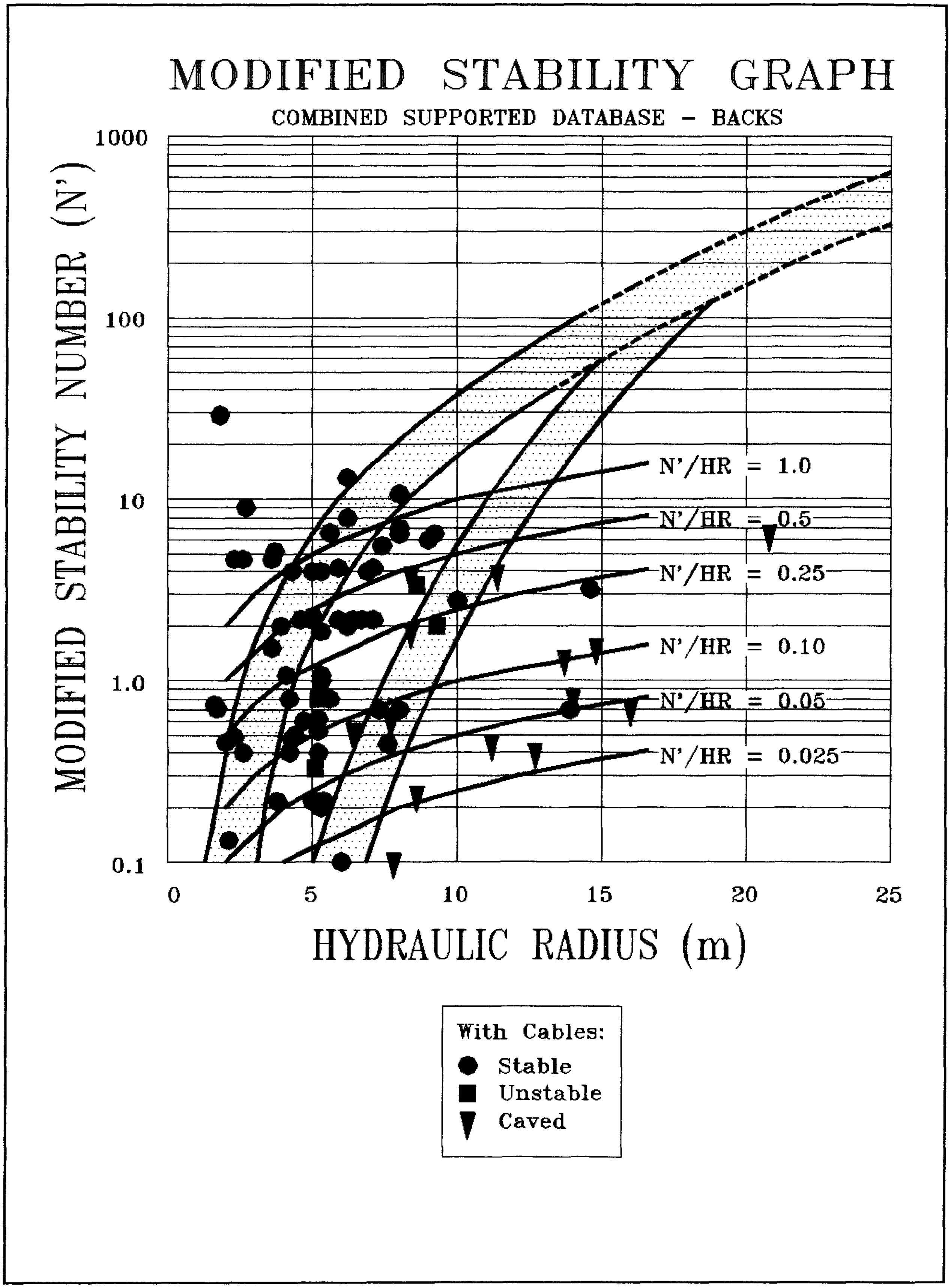


Figure 6.23: Cable support design ranges for the revised *Modified Stability Graph*

proposed by Potvin (1988) has been revised to incorporate a *supported transition zone* and a *stable with support zone* for use in the design of cable support. The *supported transition zone* was found to intersect the *unsupported transition zone*, and suggests that there is a limit to the effect of cable support on the stability of large competent stope surfaces. The combined database suggests that the *supported transition zone* reflects a lower design confidence than the *stable with support zone*. The failure regions identified by Potvin for the *Design Chart for Cable Bolt Density* are in close agreement with the statistical division between stable and caved cases. No significant correlation however, was found to exist between cable bolt density and the relative block size factor, RQD/Jn/HR. A statistically significant relationship between cable bolt density and N'/HR is proposed for future design purposes. The proposed relationship can be directly related to design ranges on the *Modified Stability Graph*.

CHAPTER 7

CABLE BOLT SUPPORT GUIDELINES

7.1 INTRODUCTION

The concepts of current cable bolt practice and design have been reviewed in the first section of this thesis. A statistical analysis was introduced in Chapter 6 to develop revised design guidelines for cable bolt support. This chapter will propose a methodology for cable design and discuss recommended design procedures.

7.2 CABLE BOLT DESIGN METHODOLOGY

A proposed design methodology is presented in Figure 7.1 based on the flowchart discussed in Section 4.1. The first priority in cable design is to assess the need for cable support. In the case of discrete design, this is related to the feature to be supported. The advantages of length and bolt capacity are often considered as justification for the use of cables, but due consideration should be given to other support mechanisms. Prior exposure to cable bolt practice is helpful, but not required to successfully implement a cable bolting proposal. Grouting equipment can be obtained on the rental market and cable bolt materials are readily available from several suppliers. The concepts of discrete analysis have been discussed in Section 4.2, and should be applied to isolated blocks or structure that require support. In the case of collective analysis, a revised version of the *Modified Stability Graph* (Potvin 1988) is proposed in Figure 7.2 for use in assessing the need of cable support. The supportable region originally proposed by Potvin (1988) has been divided into a *stable with support zone* and a *supported transition*

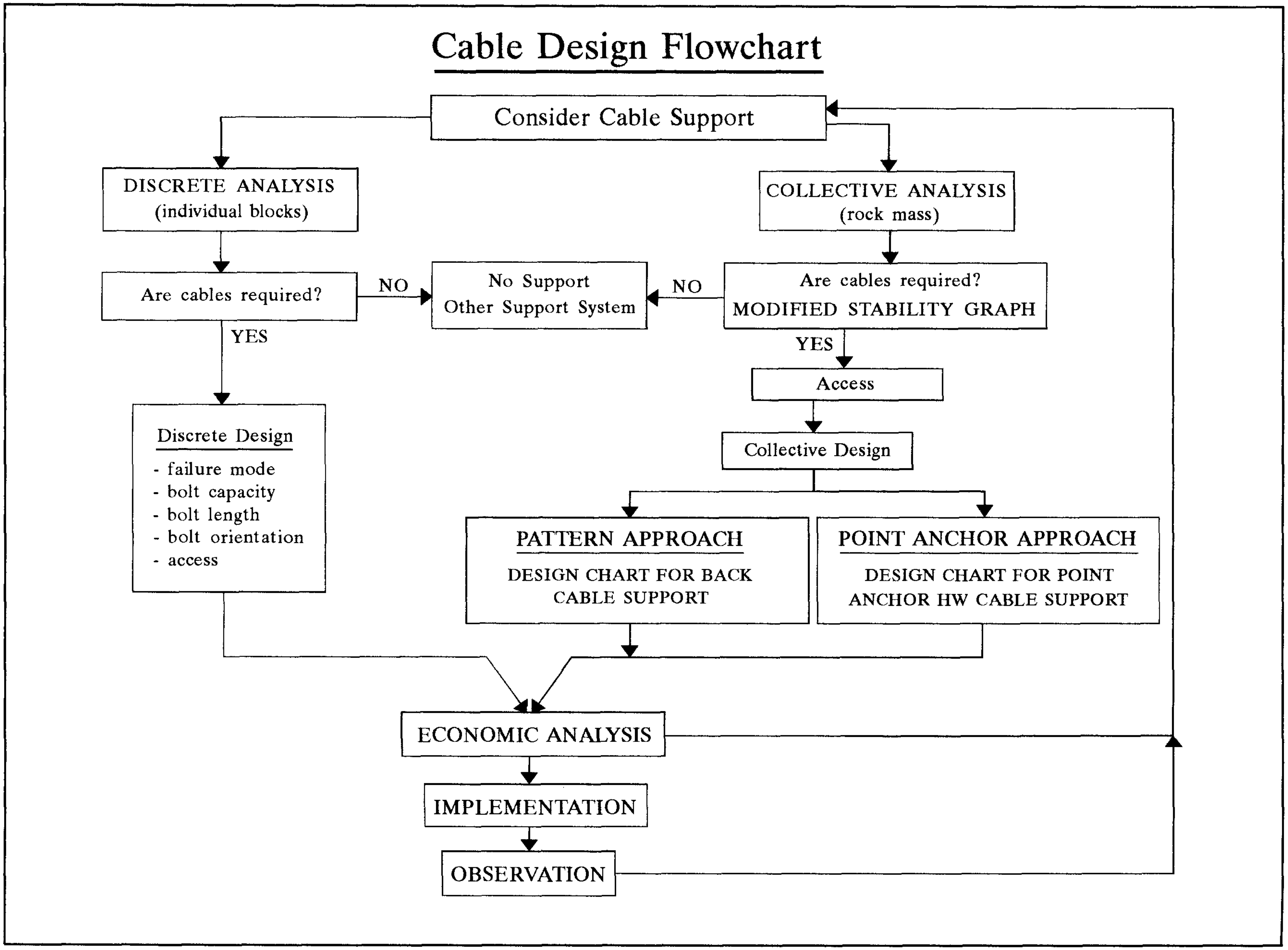


Figure 7.1: Proposed design methodology for cable bolt support

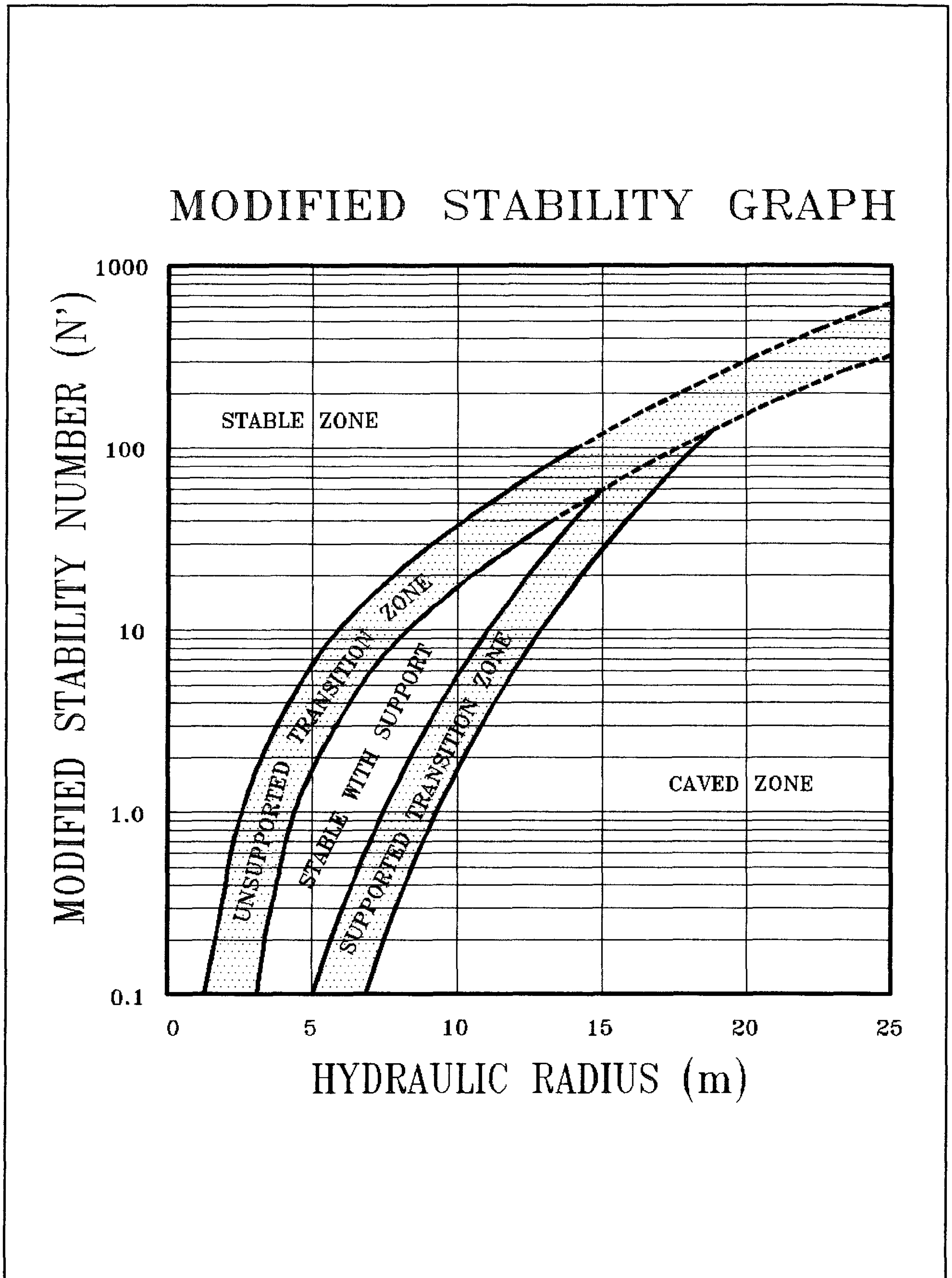


Figure 7.2: Proposed revisions to the *Modified Stability Graph* design regions

Table 7.1: Percentage of Supported Stable Cases in the Revised *Modified Stability Graph* Design Zones	
DESIGN ZONE	% STABLE
Stable Zone	100%
Unsupported Transition Zone	100%
Stable with Support Zone	85%
Supported Transition Zone	58%
Caved Zone	25%

zone, based on the statistical analysis discussed in Chapter 6. The supportable region includes both of the above zones, but based on the combined database, increased design confidence is suggested in the *stable with support* zone, as summarized in Table 7.1. There is a notable decrease in the percentage of stable cases plotting in the *supported transition zone*. All of the cases plotting in the *stable zone* or the *unsupported transition zone* are stable, and indicate a high degree of design confidence. Cable support should be considered when a design surface plots within the supportable region of the *Modified Stability Graph* in Figure 7.2. The revised supportable region proposed for cable design is similar to the original Potvin (1988) proposal. The trend of the *supported transition zone* however, reflects a slight difference as it does not parallel the *unsupported transition zone*. It has been suggested in Section 6.5.2, that this may reflect economic and technological constraints of supporting large competent surfaces. Future improvements in the application of cable support should move the *supported transition zone* farther into the *caved zone*. If the revised *Modified Stability Graph* suggests the use of cables, collective analysis then considers the available access and potential design geometries. Cable support design has been classified into two approaches related to the potential pattern. Section

7.3 will discuss a design method to determine cable density for back support, where cables are evenly distributed over the surface. A proposed method for the design of point anchor hangingwall support is reviewed in Section 7.4. Prior to implementation, an economic analysis is a necessary constituent of both discrete and collective design proposals. Observation and monitoring comprise the final and perhaps most important stage of the design process. An internal database and foundation for future design modifications can be developed through an assessment of support performance.

7.3 PATTERN APPROACH TO CABLE DESIGN

The pattern approach to cable design applies to situations where cables are distributed evenly over the supported surface. Square and fan back patterns are the best application of this type of support and comprise most of the cases in the combined database where an even distribution of cables exist. Bolt density has been discussed in terms of the number of bolts per square meter, and will be included in the pattern approach to cable design. A bolt density conversion chart has been included in Appendix B, for use in converting bolt density from bolts/m^2 to a square pattern equivalent in metric or imperial units.

7.3.1 Cable Bolt Density for Back Support

A relationship between cable bolt density and the ratio N'/HR is proposed for the design of back support, where cables are evenly distributed over the supported surface. This relationship is reflected in the *Design Chart for Back Cable Support*, illustrated in Figure 7.3. This chart is recommended for use with *square* and *fan back* cable patterns. The design line indicated in Figure 7.3 represents the regression line obtained in Section 6.5.3. The regression analysis was

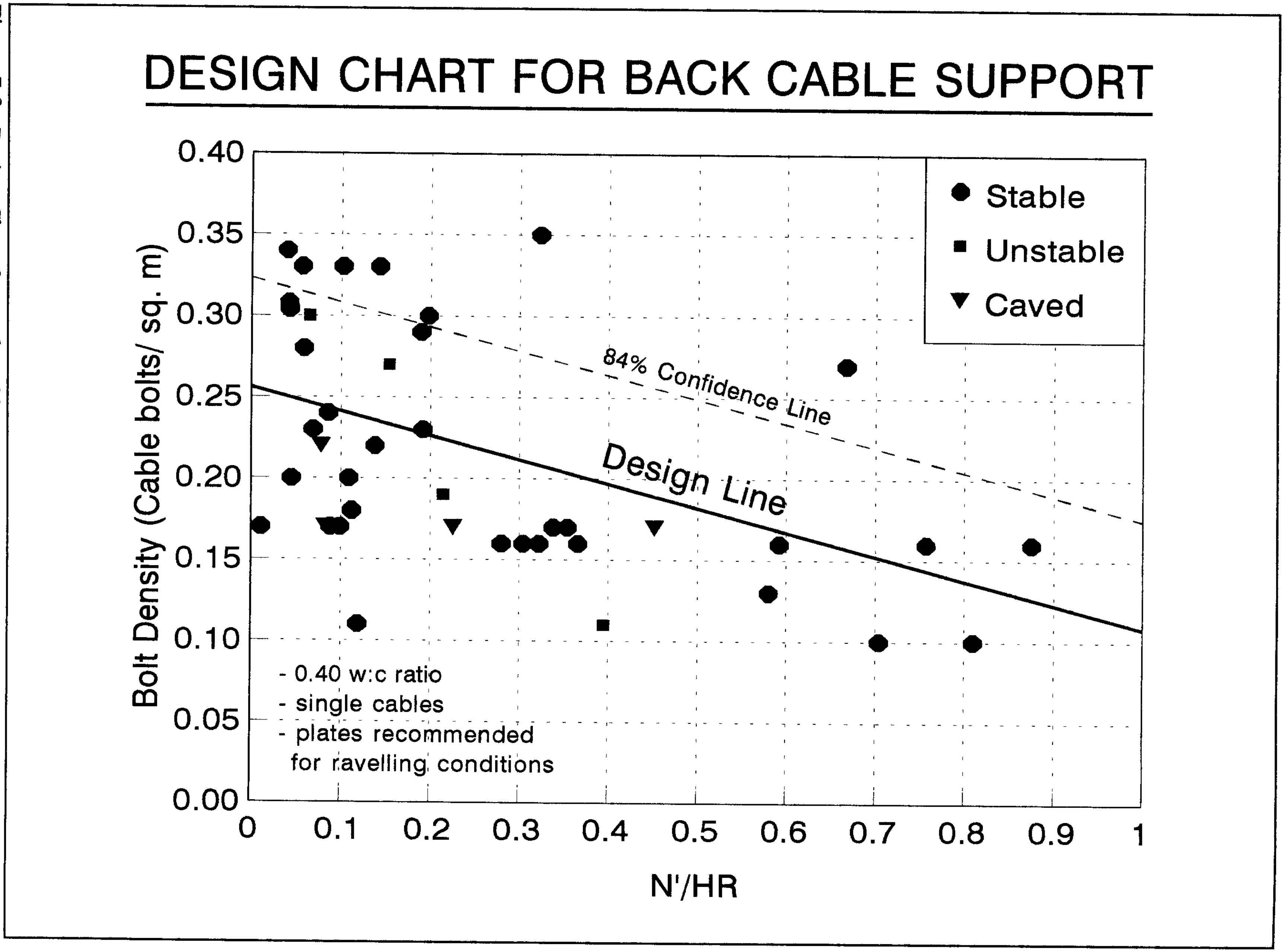


Figure 7.3: *Design Chart for Back Cable Support*

based on stable cases of back support within the combined database that fall in the supportable region of the revised *Modified Stability Graph*. This design line reflects a minimum cable density that is higher than 59% of the stable cases in the combined database, and above the region that contains caved case histories. It is recommended that cable densities be located above the design line for cases that plot in the supportable region of the revised *Modified Stability Graph*. A reduction in design confidence within the *supported transition zone* suggests that a higher bolt density should be considered for cases plotting within this zone. The upper limit of the 68% confidence band, or the 84% confidence line in Figure 7.3, is suggested as a minimum design density for the *supported transition zone*. This line represents a design level that is greater than 84% of the cases within the combined stable database. No unstable or caved cases are located in the region above the 84% confidence line. Further research is required to calibrate regions of higher design confidence on the *Design Chart for Back Cable Support*. A proposal to relate N'/HR directly to the revised *Modified Stability Graph* is illustrated in Figure 7.4. The supportable region has been divided into design ranges determined by different ratios of N'/HR. Minimum design densities have been determined from the *Design Chart for Back Cable Support* for each range, as illustrated in Figure 7.5. The 84% confidence line has been used for the ranges that plot within the *supported transition zone*. The *Design Chart for Back Cable Support* is based on average conditions within a database of stable case histories. The majority of the case histories within this database incorporate single cables and limited use of plates. The average water:cement ratio is approximately 0.40. Improvements in design confidence can be expected with the use of alternate cable geometries, water:cement ratios less than 0.40, and plates installed at the hole collar. Plates are recommended if ravelling of the rock mass is a possibility due to small block size or high critical bond lengths.

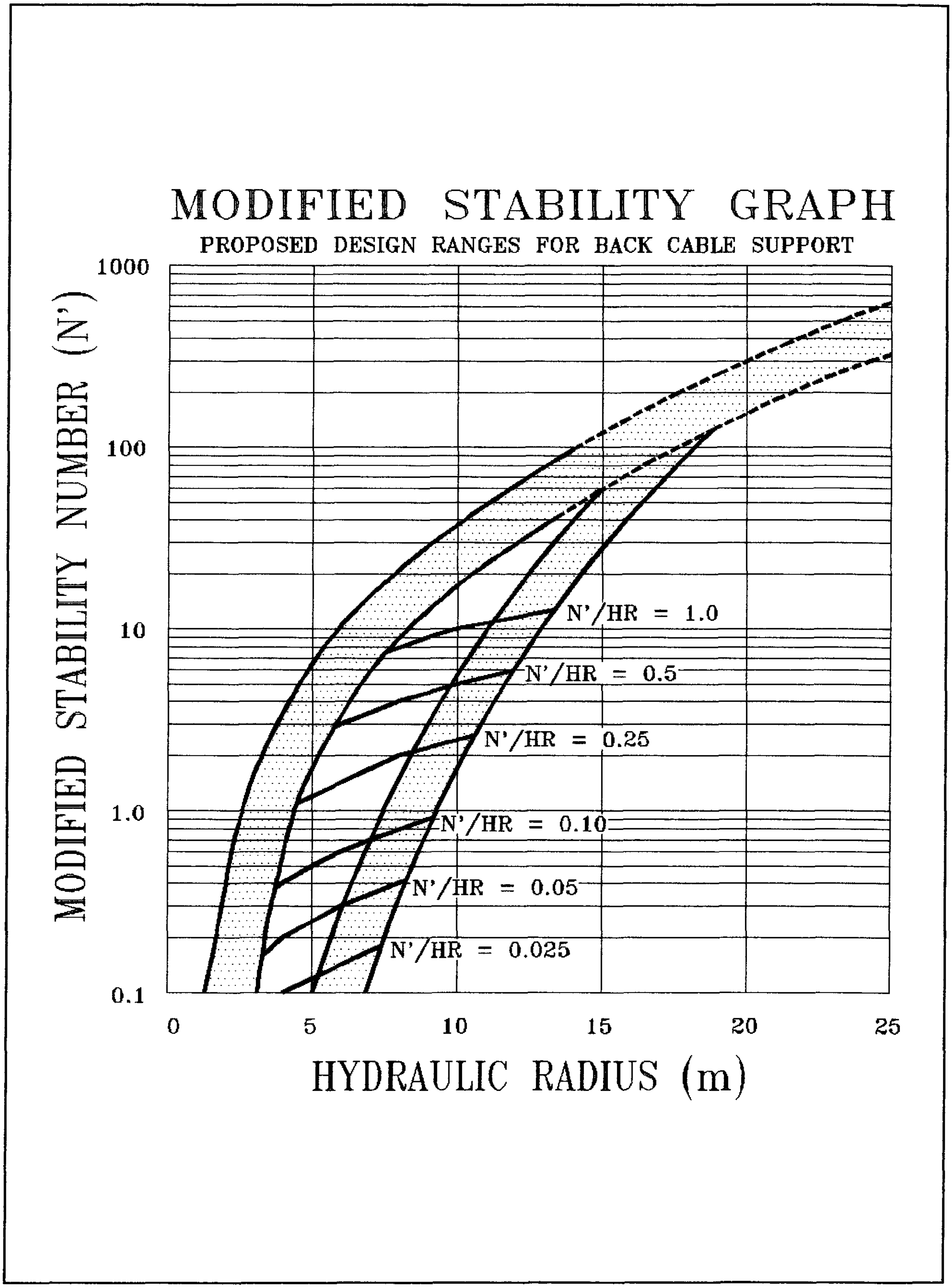
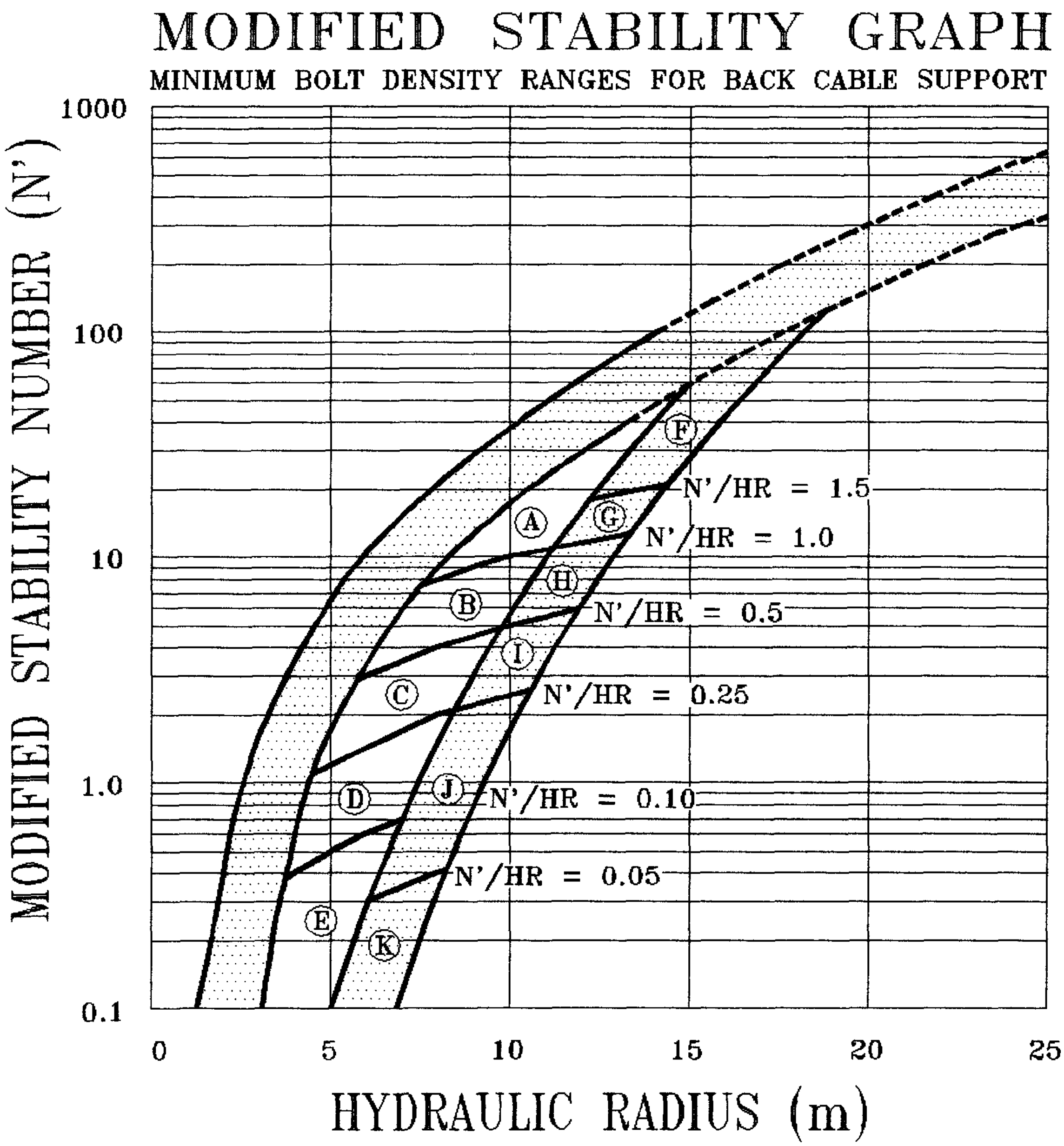


Figure 7.4: Proposed back cable support design ranges on the revised *Modified Stability Graph*



MINIMUM BOLT DENSITIES (Bolts/m^2)

A: 0.10	E: 0.25	I: 0.29
B: 0.18	F: 0.10	J: 0.31
C: 0.22	G: 0.18	K: 0.32
D: 0.24	H: 0.25	

Figure 7.5: Proposed minimum bolt density design ranges for back cable support

7.3.2 Cable Bolt Length for Back Support

Potvin, Hudyma, and Miller (1989) related cable bolt length to the hydraulic radius of the supported surface for stope backs. Figure 7.6 illustrates this approach for cases of open stope back support in the combined database that plot within the supportable zone of the revised *Modified Stability Graph*. Regression analysis was used to define a linear relationship between cable length and hydraulic radius based on the stable cases within this database. A highly significant (r=0.495) relationship was revealed and is defined by

$$Cable\ Length = 1.30 + 1.84HR. \tag{7.1}$$

Figure 7.6 shows that the regression line is very similar to the design line proposed by Potvin, Hudyma, and Miller (1989), and it is suggested as a guideline for the determination of cable length for back support. The suggested ratio of cable length to hydraulic radius ranges from 3.1 to 2.0, and can be related to span as shown in Table 7.2.

Table 7.2: Relationship Between Cable Length and Span for Back Support

STOPE DIMENSIONS		CABLE LENGTH
LENGTH/SPAN	HR/SPAN	
1:1	0.25	(0.5 to 0.8) x SPAN
2:1	0.33	(0.7 to 1.0) x SPAN
4:1	0.40	(0.8 to 1.2) x SPAN
9:1	0.45	(0.9 to 1.4) x SPAN

Design methods related to arch theory and discrete analysis presented in Chapter 4, often suggest lengths that permit cable anchorage in competent rock beyond a potential failure zone,

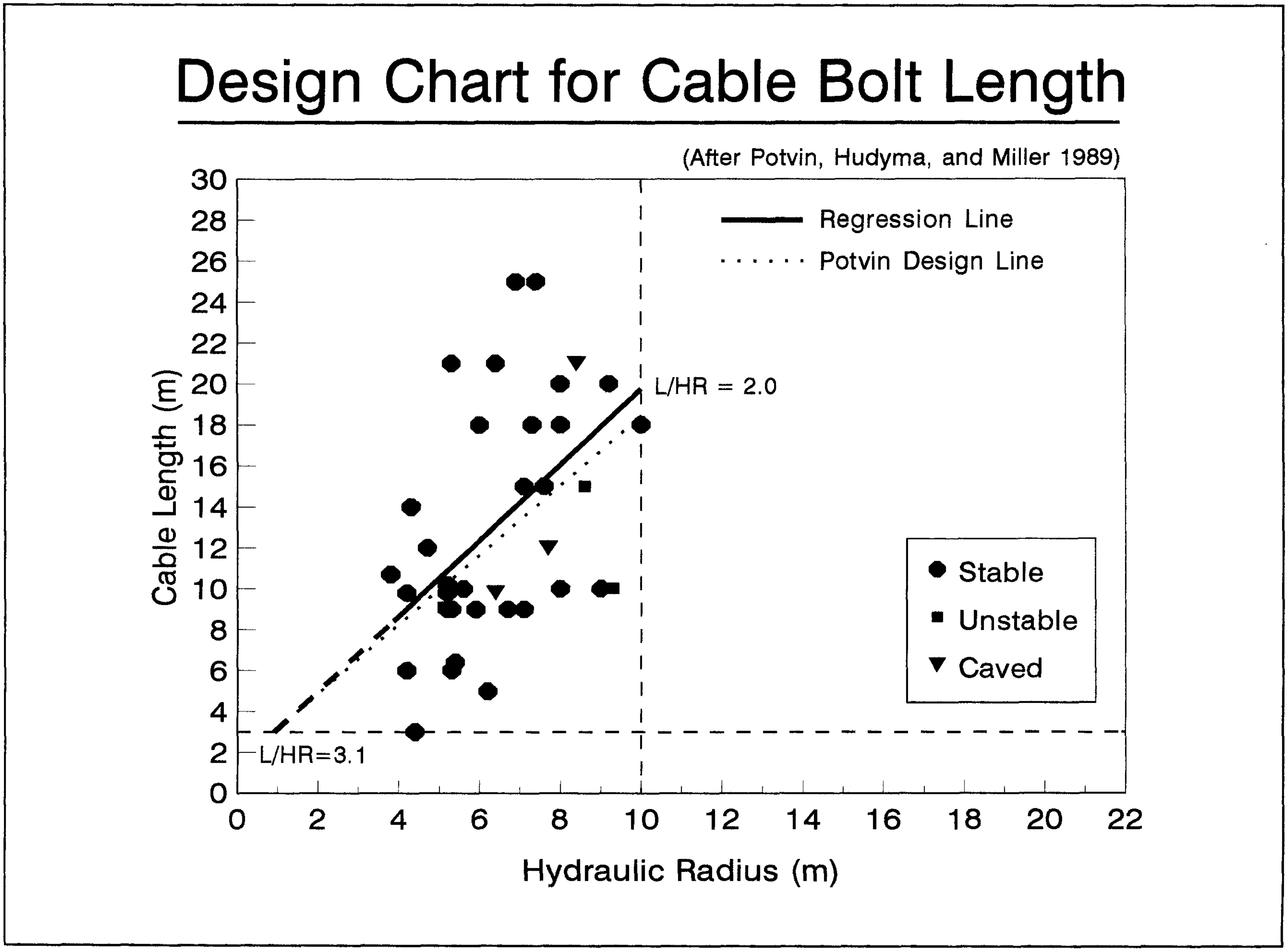


Figure 7.6: Cable bolt length for back support

as illustrated in Figure 7.7a. It is important in cases of discrete analysis to anchor into competent material. Economic conditions or large stope surfaces may produce a situation where cables cannot be anchored beyond a potential failure zone, as illustrated in Figure 7.7b. In terms of collective analysis, cable bolts act to reinforce the rock mass and develop the inherent strength by limiting relative block movement. A reduction in stability however is expected where cables cannot penetrate beyond a potential failure zone, and the use of plates and strapping is recommended to prevent ravelling of the rock mass.

7.3.3 Other Support Patterns

The design guidelines proposed in this section have been discussed in terms of an even distribution of cables over the supported surface, and include *square back* and *fan back* cable patterns. Other patterns that were encountered in practice include *point anchor backs*, *point anchor hangingwalls*, *even hangingwalls*, and *hangingwall drift fans*. It is suggested that the proposed design guidelines for cable length and density discussed in this section are appropriate for use with all of the above, except *point anchor hangingwalls*. *Even hangingwall* patterns and *hangingwall drift fan* support reflect an even distribution of bolts over the supported surface. Point anchor hangingwalls and backs reflect a concentration of cables at particular points along the surface, that depend upon the available development. The average density of point anchor back support from Table 6.3 is 0.16 bolts/m^2 for stable cases compared to 0.046 bolts/m^2 for stable point anchor hangingwalls. The higher densities are a result of smaller hydraulic radii combined with improved accessability, and suggest that point anchor back support is similar to an even distribution of cables. The cases of point anchor back support in this database fit best with the design guidelines proposed in this section, but require further calibration. The bolt

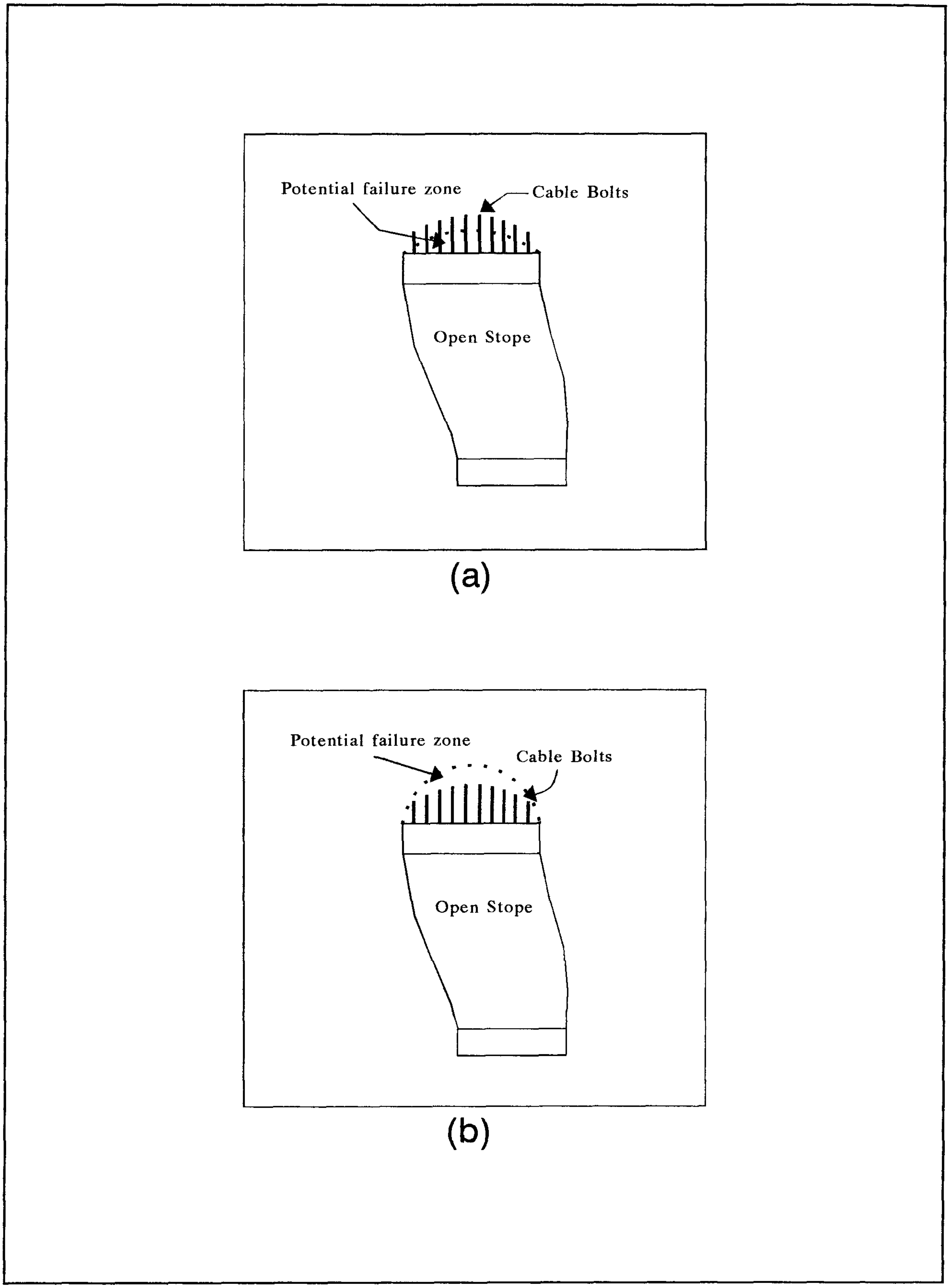


Figure 7.7: Relation between potential failure zone and cable length

density of point anchor backs in this study was determined based on the number of installed bolts, divided by the surface area. Occasionally, cable back patterns are fanned into the footwall and hangingwall, making it difficult to calculate a true back cable density, especially in the case of small openings. In this study, it was assumed that bolts fanned more than 45° from vertical were not included in the density calculation for back support. Point anchor hangingwalls exhibit densities that are well below patterns encountered for even cable distributions, and will be discussed separately in Section 7.4.

7.4 POINT ANCHOR APPROACH TO CABLE DESIGN

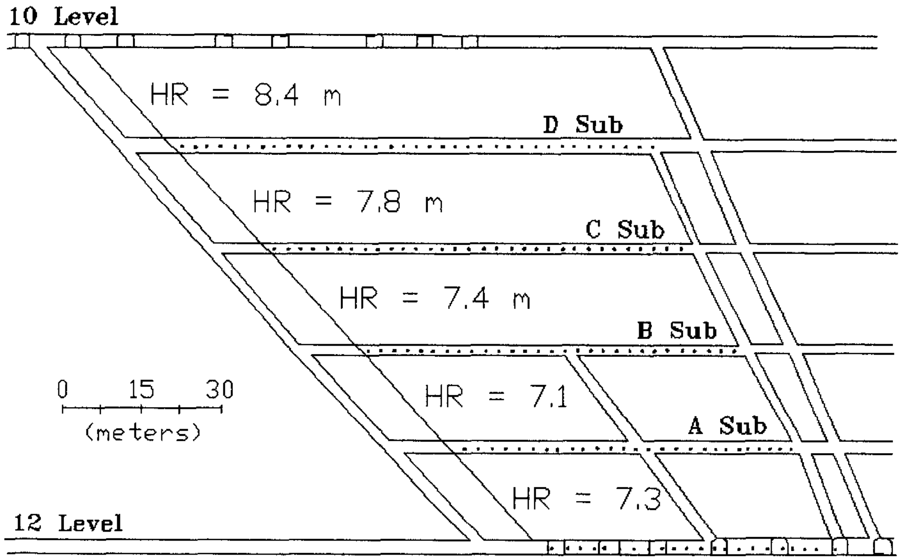
Fuller (1983b) made the distinction between a localized and uniform cable distribution in open stope hangingwalls. The purpose of the localized, or point anchor approach to hangingwall support, is described as dividing the hangingwall into smaller unsupported stable spans. The location of cable support is usually determined by sublevel development, and *span* is therefore related to the distance between sublevels. Fabjanczyk (1982) suggests that this type of support simulates a series of reinforced beams along the hangingwall. It has evolved as a method of installing cable bolts where access is typically restricted. Beer, Meek, and Cowling (1983) have described the formation of individual plates parallel to the hangingwall, due to deformation into the open stope. The stability of the hangingwall is then related to beam thickness, frequency of cross jointing, surface dimensions, and ground support. Hangingwall structure reviewed in this study was predominantly parallel to the surface, and stability seemed to be related to the unsupported span between sublevels and the size of blocks created by associated cross jointing. This section will review applications of point anchor support and propose a design method that relates factors controlling surface stability to the unsupported span.

7.4.1 Hangingwall Cable Support

An interesting case of point anchor hangingwall support (Marshall 1963) was applied at the Wilroy Mine in the early sixties. This particular installation was designed to reduce hangingwall spalling and control dilution in the No. 3 orebody. The hangingwall and footwall rock was described as a fairly massive grey gneiss with occasional biotitic bedding planes and little alteration. The orebody was mined using sublevel blasthole benching with gradually advancing stoping blocks down dip. Hangingwall dilution was a concern during mining of the first three stopes. Efforts to control dilution included the separation of stoping blocks by sill pillars, leaving random pillars within the stope limits to reduce the unsupported wall dimensions, and leaving a skin of ore to absorb blast damage. Dilution however continued to remain high, and planning for the third stope included an increase in production rate to reduce exposure time, along with an attempt to pre-shear the hangingwall. In addition, 2.4 m rockbolts were installed with strapping at intervals of 1.2 m along each sublevel. The support system was not successful as rockbolt anchor slippage was noticed, and failed waste slabs that contained undisturbed bolts indicated that the support design length was insufficient. Breakage of some bolts also suggested that the support tensile strength was too low. The 1210 stope between the 10th and 12th level was the fourth stope to be mined, and included a point anchor approach to hangingwall support using 6.7 m (22') steel bars with a diameter of 28.6 mm (1 1/8"). The bolts were grouted along the hangingwall of each sublevel at a 2.4 m spacing, as illustrated in Figure 7.8. Bolts were grouted with a mixture of water and High Early Strength cement in 76.2 mm (3") diameter holes that were inclined downwards at 1°. A 63.5 mm (2.5") anchor nut and washer was installed at the end of the bolt, and the bar was greased prior to installation. A bearing plate was placed at the hole collar on a grout pad, and the bar was tensioned three days after grouting to approximately 45 tonnes. The grouted bolt length was actually 6.1 m (20') since the collar end

WILROY 1210 STOPE LONGITUDINAL

(After Marshall 1963)



Dip = 75^{o}

Supported HR = 19.3 m

RQD = 100

Jn = 4

RQD/Jn/Supp. HR = 1.3

Max. Unsupp. HR = 8.4 m

.... Bolt Location

Support: 6.7 m steel bar (28.6mm diameter)
2.4 m spacing

Figure 7.8: Wilroy case history of the point anchor approach to hangingwall support

protruded to allow for plate attachment and tensioning. The sublevels were slashed to full width and eliminated problems of handling 6.7 m bolts. In cases of narrow ore width however, the use of coupled bars were required. At an estimated hydraulic radius of 19.3 m, this stope was mined successfully using the point anchor approach to hangingwall support. Sublevel spacings varied from 19 to 21 meters along the hangingwall.

Greenelsh (1985) describes the use of the point anchor approach to cable support in the design of the N663 stope experiment at Mount Isa Mine. The N663 stope was located at a depth of approximately 1000 meters, and was designed to test the feasibility of open stope mining in place of mechanized cut and fill. The structure of the N663 stope hangingwall is characterized by shale bedding planes that are described as being fissile, frequently smooth, continuous and graphitic. Joints and faults frequently intersect the bedding planes. A typical section through the N663 stope from the 19C to 16E sublevel is shown in Figure 7.9. The stope was 170 meters in height from the 19C to 16B sublevel, and was mined in two separate lifts. The lower section of the stope from the 19C sublevel to the limit of the upholes above 17L sublevel was mined first. The stope dimensions were 95 meters high, 15 meters wide, 20 meters along strike, and the hangingwall was dipping at 65°. These dimensions translate to a hangingwall hydraulic radius of approximately 8.4 meters. Four double cable bolts were installed on rings spaced 2 meters apart on level 17 and 18B sublevel. Two meters of the cable bolts at the hole collar were debonded to reduce the support stiffness. Steel straps were used at the hole collar in conjunction with barrel and wedge anchors, and the debonded section was tensioned to a 2 tonne load. Blastholes were 70 mm in diameter and rings were fired singly or in pairs. Instrumentation was installed on the level 17 cables and rod extensometers were used to monitor hangingwall movement. The N663 stope was mined successfully with an estimated hangingwall dilution of just over 3%. Cable instrumentation indicated that the debonded section of cable at the hole

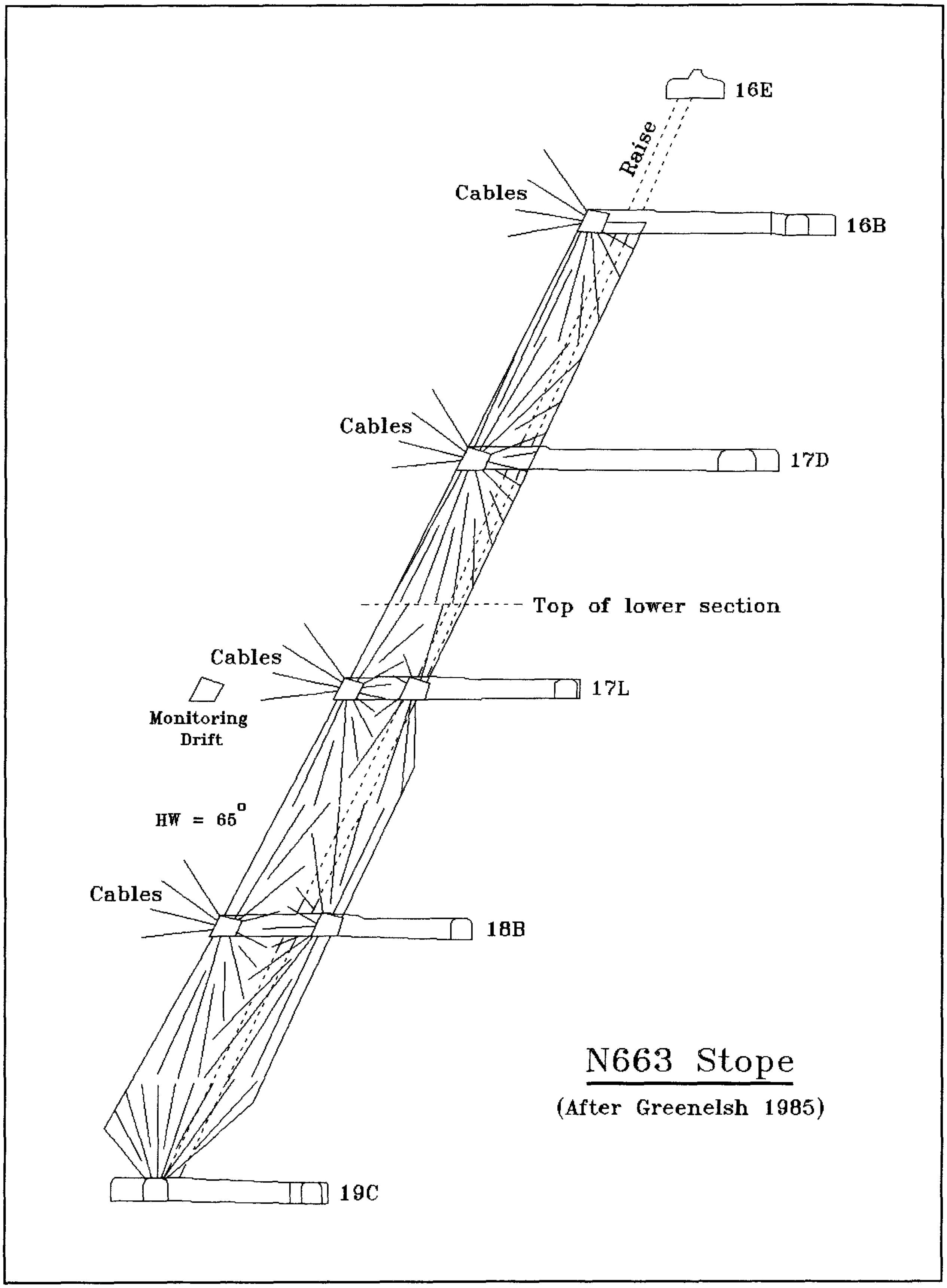


Figure 7.9: N663 stope case history of point anchor hangingwall cable support

collars exhibited loads close to the cable yield strength. Debonding is a useful design consideration in hangingwall support, as large amounts of movement are expected near the surface. The cable instrumentation revealed minimal loading at the toe of the holes and indicated that 9 to 12 meters was adequate for cable length.

7.4.2 *Design Chart for Point Anchor Hangingwall Cable Support*

Beer, Meek, and Cowling (1983) proposed that hangingwall behaviour in bedded rock is based on joint frequency and spacing, joint frictional and cohesive properties, excavation geometry, the virgin stress field, and ground support. Thirteen cases of point anchor hangingwall support were assembled from the database and are summarized in Table 7.3. Supported and unsupported spans have been defined as illustrated in Figure 7.10. Unsupported spans for a particular case were variable, and the maximum unsupported span was used in this analysis. It is proposed that the success of the point anchor approach to cable support is related to the distance between sublevels and the rock mass block size. The *Design Chart for Point Anchor Hangingwall Cable Support* is illustrated in Figure 7.11. The chart relates the maximum unsupported hydraulic radius to the relative block size factor expressed in terms of the supported hydraulic radius. The point anchor database was used to derive a support line for design, based on the method of discriminant analysis discussed in Section 6.4.2. The conditions of multivariate normality and similar variance are not ideal due to the limited size of the database, but the support line does approximate a visual division between stable and caved points. The derived statistical line favoured the caved cases and was shifted vertically down in order to place all the caved cases above the line. For design purposes, underground mapping and stope planning will give an indication of the relative block size factor. An acceptable design is indicated by projecting vertically up from the horizontal axis to the design line, and reading a recommended

Table 7.3: Point Anchor Hangingwall Supported Database

Case	Stability	Supported HR (m)	Average Dip	Strike (m)	Supported Span (m)	# Spans	Unsupported Span (m)			Unsupported HR (m)			RQD/Jn	RQD/JN/HR	N'	Length (L) (meters)	L/Max. Unsupported Span
							Min.	Max.	Avg.	Min.	Max.	Avg.					
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	33.2	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	1.39	67.5	6.1	0.23
20	Caved	12.4	62	33.3	103	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	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	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	44	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	62	111	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

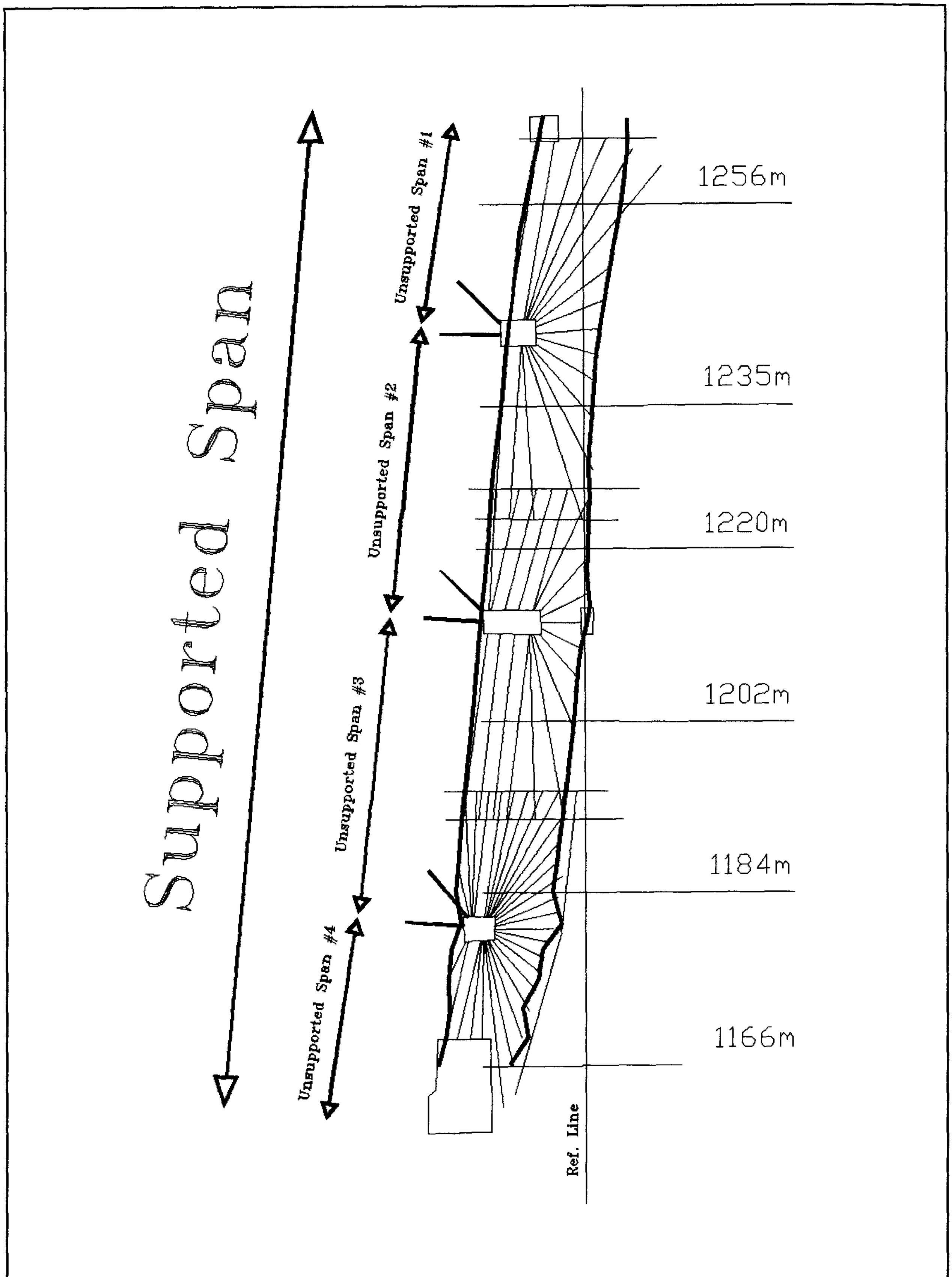


Figure 7.10: Description of geometry for point anchor approach to cable support design

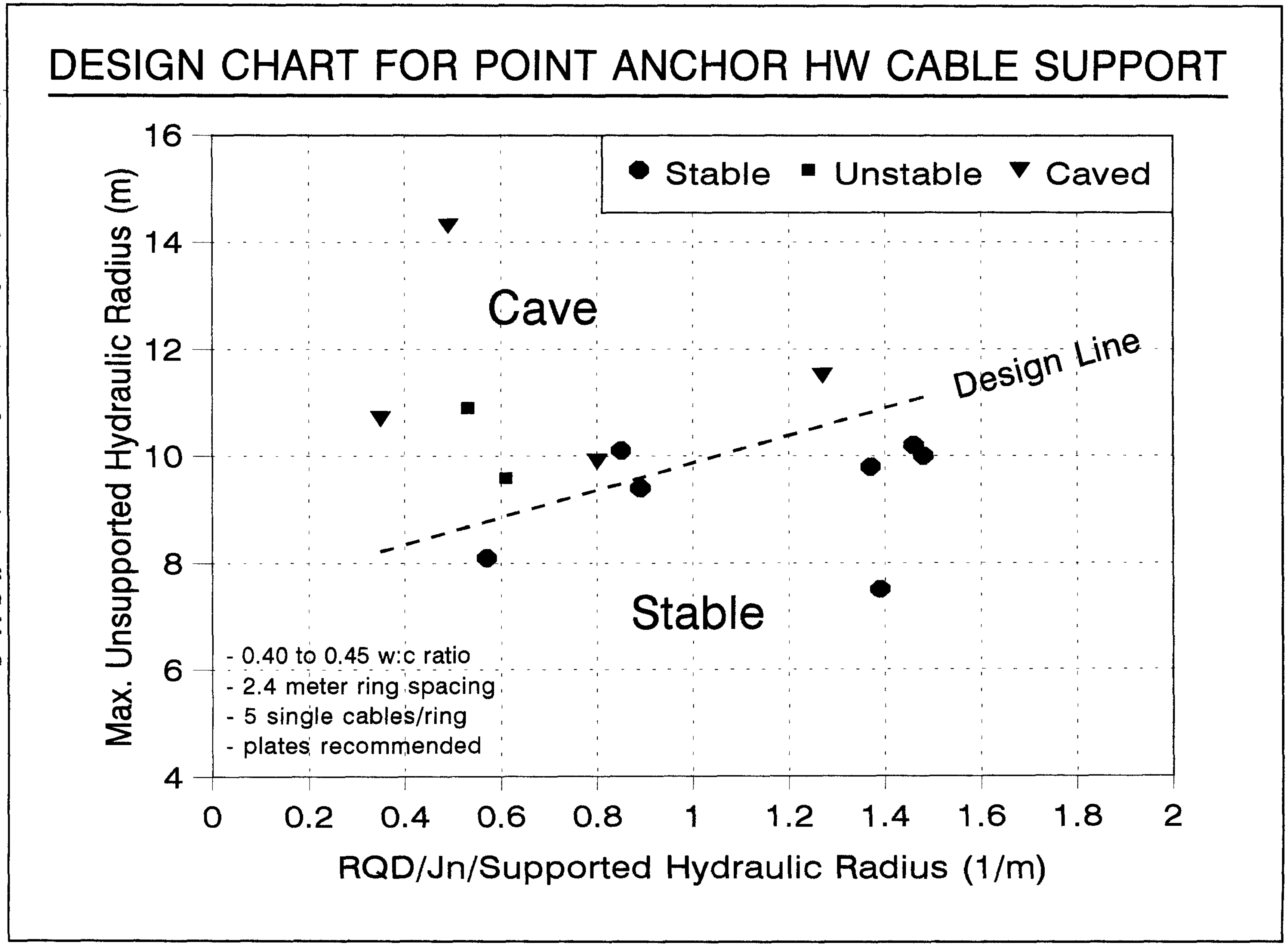


Figure 7.11: *Design Chart for Point Anchor Hangingwall Cable Support*

unsupported hydraulic radius on the vertical axis. The unsupported hydraulic radius 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 strike length to be excavated. The state of stress and joint properties discussed by Beer, Meek, and Cowling (1983) are not directly included in this analysis. Due to their geometry, hangingwalls are typically destressed and a zone of relaxation subject to the effects of gravity, is created. Pakalnis (1991) suggests that hangingwall dilution is generally due to slough within the relaxed zone. All of the hangingwalls in this study were found to be in a state of relaxation, and therefore the state of stress was not directly included in this analysis. The effect of joint properties and surface orientation were addressed by attempting to relate N'/HR, the modified stability number to supported hydraulic radius ratio, directly to the unsupported hydraulic radius. No apparent design criteria resulted from this analysis. Since the relaxed zone is subject to the effects of gravity, it is suspected that surface orientation should be included in hangingwall point anchor design. This has been accomplished to some degree by determining span and hydraulic radius based on the dip of the surface. The use of the gravity adjustment factor (C) from the modified stability number calculation, was related to the relative block size factor, but no design relationship was suggested. It is recommended that further research into point anchor cable support consider the effects of surface orientation and joint properties. The size of this database is limited, and additional case histories are required to improve the reliability of this design method.

The *Design Chart for Point Anchor Hangingwall Cable Support* is based on the assumption that the revised *Modified Stability Graph* can be used to determine if cable support is required. Figure 7.12 shows that the case histories of point anchor cable support do not strongly correlate with the supportable region of the revised *Modified Stability Graph*. The collection of additional case histories is required to improve upon this relationship. This design

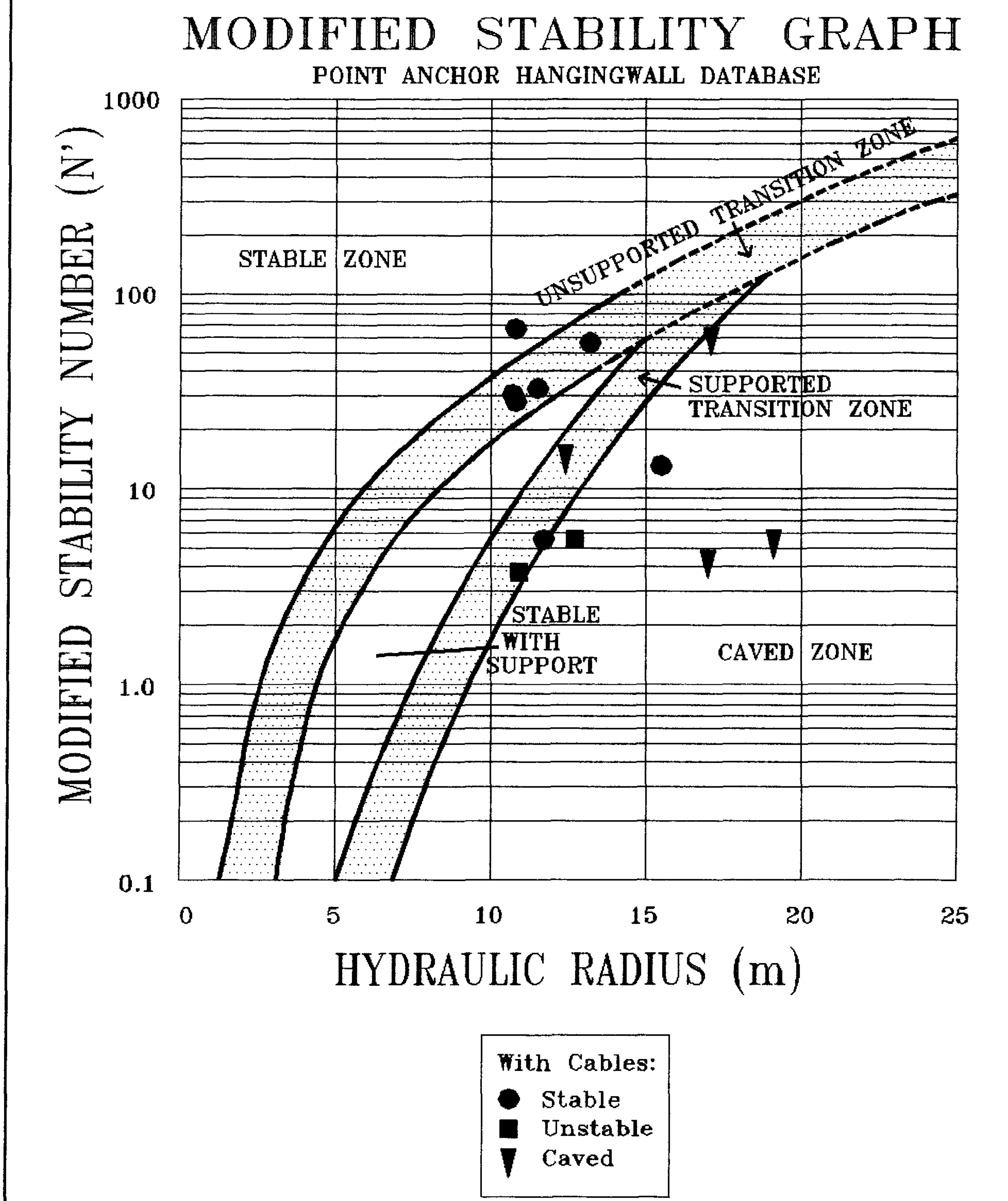


Figure 7.12: Point anchor hangingwall database compared to the revised *Modified Stability Graph*

method also assumes that adequate support is installed at each sublevel, and it is recommended that the use of plates be incorporated. The average water:cement ratio for the point anchor database is in the 0.40-0.45 range. Due to 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 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 4 bolts per ring and 2.4 meters between rings. If double cables are treated as two separate bolts, then an average of slightly over 5 cable strands per ring is reflected. It is suggested that the average values for ring spacing and the number of cable strands per ring, be used as preliminary design guidelines. Further calibration of these parameters is required.

7.4.3 Cable Bolt Length For Point Anchor Hangingwall Support

It is proposed that design bolt length for a point anchor approach to hangingwall cable support is related to distance between sublevels. The ratio of cable length to maximum unsupported span listed in Table 7.3 can be used as a preliminary guideline for the determination of cable length. Table 7.3 indicates that cable length should exceed 25% of the maximum unsupported span.

7.5 DESIGN CASE HISTORIES

The *Design Chart for Point Anchor Hangingwall Cable Support* is based on a minimal number of case histories, and further calibration is required. Two case histories of hangingwall support design will be briefly reviewed as an illustration of the proposed design method.

7.5.1 Detour Lake Mine

Hangingwall cable design for the 560-660 Block at the Detour Lake Mine (Detour Lake Mine 1992) is based on the point anchor approach described in this chapter. The structure of the hangingwall rock was assessed as indicated below:

$Q' = 68$, $A = 1.0$, $B = 0.2$, $C = 5.0$, and $N' = 68$.

RQD was estimated at 85% and Jn was set to 3 based on one major joint set and additional random jointing in the hangingwall rock. A typical section through the proposed stope is shown in Figure 7.13. The supported hydraulic radius is approximately 30 meters, based on a strike length of 150 meters and a supported span of 100 meters. The relative block size factor is 0.9 (RQD/Jn/Supported HR = 85/3/30) and the *Design Chart for Point Anchor HW Cable Support* suggests that the unsupported hydraulic radius should be kept to a maximum of 9.6 meters. Based on the 150 meter strike length, this hydraulic radius can be back calculated to reflect a maximum unsupported span of 20.5m. Cable rings are planned every 3 meters along strike with three bolts/ring. Cable orientation is designed to establish a pattern approximately perpendicular to the hangingwall with cables at 0°, +20°, and +40° from horizontal.

7.5.2 Wilroy Mine

In an effort to compare the Wilroy case history discussed in Section 7.4.1 to the proposed *Design Chart for Point Anchor Hangingwall Cable Support*, an estimate of the parameters involved was taken from information provided in the literature (Marshall 1963). This application relates to the use of steel bar for support, but it is suspected that this could be correlated with a plated fan of cable bolts. The hangingwall is described as a massive rock and the RQD was estimated at 100%. Gneiss is a metamorphic rock that is associated with a banded distribution and may grade to a schist (Kyrine and Judd 1957). Occasional biotitic planes at Wilroy were

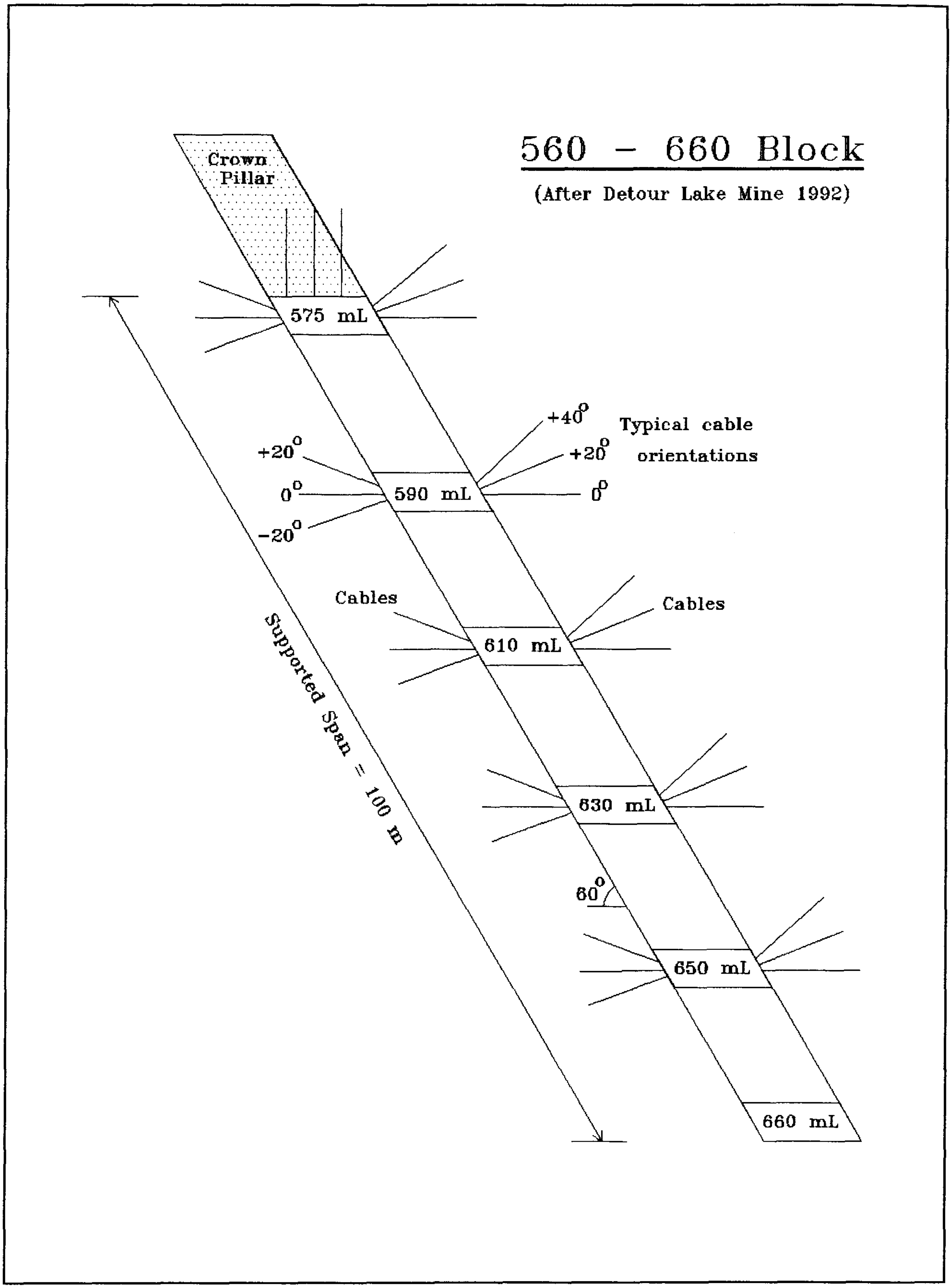


Figure 7.13: Detour Lake Mine case history of point anchor hangingwall cable support

noted to affect the hangingwall stability. Based on a joint set number of 4, a joint roughness number of 2, and a joint alteration number of 4, the Q' value is estimated at 12.5. An approximation of the stability number is outlined as follows:

Q' = 12.5

A = 1.0 since hangingwall in relaxation

B = 0.3 as critical joint appears to be parallel to the hangingwall

C = 6.5 for a 75° wall

N' = 24.4.

The hangingwall hydraulic radius was estimated at 19.2 m from Figure 7.8, and the surface plots within the caved zone of the revised *Modified Stability Graph*. This reflects some of the uncertainty with the relationship between design ranges on the revised *Modified Stability Graph* and point anchor hangingwall cable support. The maximum supported hydraulic radius was estimated at 8.4 m for the lift between D Sublevel and 10 Level (Figure 7.8). With an unsupported relative block size factor (RQD/Jn/Supported HR) of approximately 1.3, the *Design Chart for Point Anchor Hangingwall Support* suggests that the 1210 stope should be stable. The analysis also suggests that the sublevel interval could be increased to reflect a maximum hydraulic radius of 10.5 meters. A back analysis of this hydraulic radius would suggest that consideration could be given to eliminating one sublevel.

7.6 DISCUSSION

The design proposals presented in this chapter have been derived from an empirical database assembled from Canadian hard rock mining experience. The collective design of cable bolt support has been related to a revised version of the *Modified Stability Graph*, that was originally proposed by Potvin (1988). The *unsupported transition zone* has been statistically calibrated with the addition of a new database, and is recommended for the design of unsupported surfaces. The supportable region of the *Modified Stability Graph* (Potvin 1988) has also been calibrated, and a *stable with support zone* and a *supported transition zone* have been proposed for the design of supported surfaces. Cable support is suggested for design surfaces that plot within the supportable region of the revised *Modified Stability Graph*.

No significant relationship was found between the relative block size factor and cable bolt density. Block size will affect a cable bolt system, but the statistical analysis suggests that other factors are also involved. The modified stability number accounts for block size, surface orientation, joint properties, stress, and joint orientation. These factors have been successfully related to stope design by Potvin (1988), and this thesis suggests that they are also related to the determination of cable bolt density. The *Design Chart for Back Cable Support* is recommended for use with fan and square back cable patterns. The application of even hangingwall and point anchor back patterns to this chart is suggested, but requires further analysis. Minimum cable bolt densities have been related directly to the revised *Modified Stability Graph*.

The *Design Chart for Point Anchor Hangingwall Cable Support* is proposed for use in the determination of hangingwall cable patterns. Support design is related to rock mass block size, surface hydraulic radius, and the sublevel interval. It is proposed that the supportable region of the revised *Modified Stability Graph* be used as an indicator of the requirement for cable support. The case histories discussed in Section 7.5 suggest that point anchor design may extend into the caving zone of the revised *Modified Stability Graph*. The hangingwall database is limited due to restricted accessibility, and further case histories are required to calibrate the proposed design criteria.

The design proposals for point anchor and back cable support do not directly distinguish between cable geometry and the application of plates at the hole collar. The database of stable backs collected in this study reflect the limited use of plates. Single cables were used in 60% of the cases. Increasing support stiffness by using double cables or installing plates at the hole collar, can be used as an additional safety factor in cable design. Plates prevent blocks from sliding off the cable and maximize the available load carrying capacity. This is important in hangingwall design, since the surface is typically in a state of relaxation. The recommended number of cables for point anchor support has been based on the number of installed cable strands. It is also recommended that cable distribution be maximized for point anchor support.

There are several external factors that relate to successful cable design. Undercutting of ore contacts can frequently lead to a progressive failure of hangingwall surfaces, and detract from the performance of a cable installation. Drifts driven under surveyed or geological control, can still break the ore contact due to the uncertainty in diamond drilling predictions. A skilled development crew can often follow a predetermined contact with only minor guidance for elevation. High density point anchor fans with plates were used on occasion to control, or stop progressive hangingwall failures initiated from undercutting on previous mining horizons. Hangingwall failures can effect the stability of stope backs by increasing the exposed hydraulic radius. Blasting practice can also affect the performance of cable support. Bywater and Fuller (1983) have suggested that slot raises should be located on the footwall side of the stope, to reduce the effect of initial tight slot blasts on hangingwall stability. The use of decoupled charges along the hangingwall can also assist in this regard. Cable design should consider manpower availability and time constraints in relation to planned stope production. When time is a factor, production requirements often take priority, and support installations may be left uncompleted.

CHAPTER 8

CONCLUSIONS

8.1 INTRODUCTION

The objective of this thesis was to expand upon the existing database of cable support practice and develop revised design criteria for the support of underground openings. An empirical database was collected during an extensive field study that involved visits to operating mines in Western Canada, the United States and Ireland. Guidelines proposed by Potvin (1988) for the design of open stope surfaces were applied to the data collection process. The Potvin (1988) stope design criteria were reviewed and revised in the context of cable support. A statistical analysis was presented as a tool to aid in the interpretation of empirical data. Revised guidelines are proposed in this thesis for use in the design of back and hangingwall cable support.

8.2 CONCLUSIONS

A new empirical database of 46 case histories has been assembled during this study to expand upon the existing knowledge of cable support in underground mining applications. The new data was combined with the Potvin (1988) database to develop design guidelines for cable support. Improvements in blast technology, monitoring, quality control and production methods will inevitably alter the picture described by this empirical database. The process of calibration and revision suggested in this thesis should continue, in order to reflect technology improvements and changes in operational procedure.

A methodology for cable design is proposed based on a distinction between discrete and collective analysis. Discrete analysis should be applied to isolated blocks or structure that require support. Collective analysis applies to the rock mass, rather than distinct features, and provides a method of relating structure and stress to the design of cable support. The Potvin (1988) approach has related characteristics of the rock mass to the design of open stope surfaces, by relating a modified stability number to the surface hydraulic radius on the *Modified Stability Graph*. The unsupported *Modified Stability Graph* proposed by Potvin (1988) has been statistically verified using a method of discriminant analysis, and is recommended for use in the design of open stope surfaces. The supportable region defined on the supported *Modified Stability Graph* (Potvin 1988) has been revised for use in cable design. The revised supportable region incorporates a *stable with support zone* and a *supported transition zone* that are based on different levels of design confidence. The revised supportable region is very similar to that proposed by Potvin (1988), but does not parallel the *unsupported transition zone*. This suggests that there is a limit to the effectiveness of cable support with respect to large competent stope surfaces. Consideration of cable support is recommended for surfaces that plot in the supportable region of the revised *Modified Stability Graph* (Figure 7.2). This thesis advocates the use of on-site calibration as the ideal method of applying empirical design techniques. The mining engineer will find that the best use of time is often spent in observing underground geotechnical activity in relation to production. The revised *Modified Stability Graph* provides a tool for the mining engineer to use in the documentation process, and the development of site specific design guidelines. A rough relationship between the increase in rock mass quality due to the addition of support can also be derived. The *stable zone* for an unsupported surface can be related to the *stable with support zone* for a supported surface. This provides a method of quantifying the effect of support in terms of either an increase in Q', or RMR. This is a valuable concept but requires additional study.

The collective design process uses the revised *Modified Stability Graph* to determine if cable support is warranted for a particular surface. Cable design has been classified into two categories, a pattern approach and a point anchor approach. The pattern approach involves an even distribution of cable support over the design surface, and is the most common method encountered in practice. The point anchor approach is characterised by a large concentration of bolts placed at particular points along a surface. Two design methods that reflect the distinction between the point anchor and pattern approaches, are proposed in this thesis.

The new database was compared with the *Design Chart for Cable Bolt Density* that was proposed by Potvin (1988) for the design of cable support for stope backs. Close agreement was found with the stability conditions proposed by Potvin (1988), but no significant statistical relationship was found between cable density and the relative block size factor. A statistical analysis in this thesis suggests that joint properties, stress, and joint orientation should also be considered in the determination of cable density for stope backs. The modified stability number proposed by Potvin (1988) and the surface hydraulic radius have been statistically related to cable density in the *Design Chart for Back Cable Support*. It is recommended for use in the determination of cable patterns for back support, where cables are evenly distributed over the entire surface. Cable length for back support has been related to the surface hydraulic radius.

Point anchor hangingwall cable design is related to the determination of a maximum stable unsupported span between beams of high density cable fans. The block size and surface dimensions have been related to unsupported span on the *Design Chart for Point Anchor Hangingwall Cable Support*. Attempts to include the modified stability number and surface orientation into the design proposals were unsuccessful. Cable support is based on average conditions. It is recommended that cable rings be spaced 2.4 meters along strike and five plated cables be incorporated within each ring. The point anchor design proposals are based on a limited database and require additional calibration. Cable length for point anchor cable support is related to the distance between sublevels.

8.3 FUTURE WORK

Pakalnis et al. (1987) indicate that dilution is used as a measure of stope design quality, but is not necessarily defined in the same manner by open stope operators. In this study, design dilution estimates were difficult to obtain, but actual dilution values were frequently recorded. This indicates that the relationship between design and actual dilution merits further investigation. Non-entry mining methods typically exhibit a tolerance for instability that is difficult to describe, but can be related to the degree of dilution. It is recommended that dilution be considered in future expansion of the empirical database, and related to design ranges on the revised *Modified Stability Graph*. The influence of economic and operational parameters should also be considered.

A review of design practice has indicated that cable length is often related to a zone of instability. Pakalnis (1991) suggests that hangingwall dilution is usually a result of slough within the zone of relaxation. A parametric modelling study (Pakalnis 1991) showed that the zone of relaxation for hangingwalls, predicted by traditional two dimensional modelling, can be 300% higher than three dimensional modelling predictions. Further examination of the relationship between the zone of relaxation obtained from three dimensional modelling, and cable length, is suggested.

The characteristics of grout flow in a cable bolt hole are difficult to observe and are frequently related to deficiencies in cable bolt support. Chapter 2 notes that improvements in grout quality can significantly improve cable load carrying capacity. Further research in this area is recommended to consider the variables involved in grout flow. The analysis should consider pumping equipment, installation methods, and properties of grout mixtures. Steel pipes have been used to simulate underground cable installations (Cluett 1991). This type of laboratory testing would be useful in the evaluation of grout flow under operating conditions.

The *Design Chart for Back Cable Support* has been proposed for use with all forms of cable support that feature an even distribution of cables over the design surface. Most of the database used in the determination of this chart was based on case histories of square back and fan back cable patterns. Additional case histories of point anchor back and even hangingwalls are required to verify that they agree with the proposed design ranges.

The *Design Chart for Point Anchor Hangingwall Cable Support* is based on a limited database. The chart requires the collection of additional case histories to calibrate the proposed design ranges. It is recommended that future research consider point anchor hangingwall cable design in terms of surface orientation and the modified stability number. Mandolin bolting was also encountered on a limited basis for hangingwall support. This methods simulates a cable sling, and may merit further investigation for small hangingwalls.

Discrimination based on the Mahalanobis distance, has been introduced as a method of separating a multidimensional database based on stability. It produces a division between two classes of points, but since it is a linear technique, it is not sensitive to non-linear division. It is therefore difficult to use this method to extrapolate a division between classes beyond the range of collected data. With this limitation in mind, discrimination based on the Mahalanobis distance is recommended for use with forms of empirical analysis described in this thesis. A logarithmic transformation has been successfully applied to meet the conditions of multivariate normality and similar variance. There are other methods of statistical analysis that deal with data classification or group assignment. Cluster analysis is one such technique that is concerned with the identification of groupings within a database (Manly 1986, 13). The method of discriminant analysis discussed in this thesis, assumes that the database is separated based on the stability condition of the surface. Cluster analysis is a numerical process that determines the number of classes based on the database. It is inherently more complex and may not apply to the type of database discussed in this thesis, but it is recommended for further study. It is not restricted to a linear separation between classes, but instead identifies the nature of true groupings within a database.

8.4 FINAL REMARKS

The design methods proposed in this thesis have been derived from an empirical database based on Canadian hard rock mining experience. They are not intended to suggest rigid guidelines, but instead to provide a wide degree of latitude in the design of cable support systems. The techniques described in this thesis can be applied at operating mines to develop site specific criteria. Expansion of the empirical database and calibration of the proposed design guidelines, is required to account for operational and technological improvements.

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APPENDIX A

DATABASE SUMMARY

TABLE A.1: Case Study Summary (Case 1 to 6)

CASE #	1	2	3	4	5	6
Surface	HW	Back	HW	HW	Back	HW
Cables/Hole	2	2	2	2	2	2
Hole Length (m)	9.1 – 18.3	6.1	6.1	6.1	9.1	6.1
Pattern	HW Drift Fan	Square	Point Anchor	Point Anchor	Square	Point Anchor
Density (bolts/m2)	0.018	0.13	0.021	0.018	0.16	0.022
Equiv. Pattern (ft x ft)	24.5	9.1	22.64	24.46	8.2	22.12
Stability	Caved	Stable	Stable	Caved	Stable	Caved
Strike Length (m)	32	20	66.4	59.7	26.5	67.2
Average Width (m)	4.6	6	7.7	8.1	4.1	6.1
Dip (degrees)	74		80	80	76	82
Vertical Height (m)	69.5	2.7	34	104	33	69
Area (m2)	2063	124	2384	6345	167	4826
Perimeter (m)	207	54	205	332	64	282
Hydraulic Radius (m)	10.0	2.3	11.7	19.1	2.6	17.1
Mining Method	Blasthole	Drift	Blasthole	Blasthole	Blasthole	Blasthole
Dilution (%)	24		<20	42	25	66
Cable Diameter (mm)	15.9	15.9	15.9	15.9	15.9	15.9
Hole Diameter (mm)	51	51	51	51	51	51
Cost ($/m)	29.46	29.46	29.46	29.46	29.46	29.46
Bonus	$40/Shift	$40/Shift	$40/Shift	$40/Shift	$40/Shift	$40/Shift
Supplier	Thiessen	Thiessen	Thiessen	Thiessen	Thiessen	Thiessen
Timing	7–14d	28d+	7–14d	7–14d	14d	7–14d
Plates	No	No	305 x 305 mm	305 x 305 mm	No	No
Straps	No	No	No	No	No	No
Other Support	No	No	No	No	No	No
Grout Pump	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000
Water:Cement Ratio	0.45	0.45	0.45	0.45	0.45	0.45
Grout Tube Dia. (mm)	19	19	19	19	19	19
Breather Tube Dia. (mm)	9.5	9.5	9.5	9.5	9.5	9.5
Rock Type	AA & CA	AA/CA	NS	NS	TS	CA/AA
RMR'	56	59	40	40	59	70
Q'	11.7	11.7	2.5	2.5	11.7	27.3
A	1.0	1.0	1.0	1.0	1.0	1.0
B	0.2	0.2	0.3	0.3	0.2	0.3
C	6.0	2.0	7.5	7.5	2.0	7.5
N'	14.0	4.7	5.6	5.6	4.7	61.4
RQD	70	70	40	40	70	82
Jn	6	6	6	6	6	6
Jr	2	2	1.5	1.5	2	2
Ja	2	2	4	4	2	1
RQD/Jn/H.R.	1.17	5.09	0.57	0.35	4.50	0.80
Blasthole Dia (mm)	51	51	51	51	51	51
Backfill	None	None	None	None	None	C&F Below

TABLE A.2: Case Study Summary (Case 7 to 12)

CASE #	7	8	9	10	11	12
Surface	HW	HW	HW	Back	HW	Back
Cables/Hole	2	2	2	2	2	2
Hole Length (m)	6.1	6.1	6.1	22	6.1	7.6
Pattern	Point Anchor	Point Anchor	Point Anchor	Square C&F	Point Anchor	Fan
Density (bolts/m2)	0.011	0.020	0.018	0.13	0.023	0.58
Equiv. Pattern (ft x ft)	31.28	23.2	24.46	9.10	21.63	4.31
Stability	Unstable	Unstable	Stable	Stable	Stable	Stable
Strike Length (m)	32.6	40.2	68.6	21.0	33.4	21.9
Average Width (m)	9.1	7.0	4.9	15.5	8.3	4.0
Dip (degrees)	85	84	87	85	85	N/A
Vertical Height (m)	66.3	68	44	4.3	63	3.7
Area (m2)	2171	2897	3103	412	2069	52
Perimeter (m)	199	229	235	81.7	192	33
Hydraulic Radius (m)	10.9	12.7	13.2	5.0	10.8	1.6
Mining Method	Blasthole	Blasthole	Blasthole	Cut and Fill	Blasthole	Drift
Dilution (%)	43	30	30	10–15	30	N/A
Cable Diameter (mm)	15.9	15.9	15.9	15.9	15.9	15.9
Hole Diameter (mm)	51	51	51	57	51	51
Cost ($/m)	29.46	29.46	29.46	40.3	29.46	29.46+
Bonus	$40/Shift	$40/Shift	$40/Shift	Contractor	$40/Shift	$40/Shift+
Supplier	Thiessen	Thiessen	Thiessen	Thiessen	Thiessen	Thiessen
Timing	14–28d	7–14d	14–28d	28d+	7–14d	28d+
Plates	305 x 305 mm	305 x 305 mm	No	No	No	305 mm dia.
Straps	No	No	No	No	No	Steel Sets
Other Support	No	No	No	Swellex	No	Swellex/RB
Grout Pump	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000
Water:Cement Ratio	0.45	0.45	0.45	0.45	0.45	0.45
Grout Tube Dia. (mm)	19	19	19	19	19	19
Breather Tube Dia. (mm)	9.5	9.5	9.5	9.5	9.5	9.5
Rock Type	NS	NS	CA	TS	CA	TS
RMR'	40	40	70	75		75
Q'	2.5	2.5	23.7	18.8	30	18.8
A	1.0	1.0	1.0	0.2	1.0	0.1
B	0.2	0.3	0.3	0.3	0.3	0.2
C	7.5	7.5	8.0	2.0	7.5	2.0
N'	3.8	5.6	56.9	2.3	67.5	0.75
RQD	40	40	71	75	90	75
Jn	6	6	6	6	6	6
Jr	1.5	1.5	2	1.5	2	1.5
Ja	4	4	1	1	1	1
RQD/Jn/H.R.	0.61	0.53	0.89	2.50	1.39	7.81
Blasthole Dia (mm)	51	51 & 76	51	38	51	38
Backfill	C&F Below	C&F Below	C&F Below	Sand	None	None

TABLE A.3: Case Study Summary (Case 13 to 18)

CASE #	13	14	15	16	17	18
Surface	Back	Back	Back	Back	Back	Back
Cables/Hole	2	2	2	2	2	2
Hole Length (m)	6.1	14	8 & 15	8 & 12	12 & 15	9.8
Pattern	Square	Point Anchor	Square	Point Anchor	Square	Square
Density (bolts/m2)	0.116	0.18	0.28	0.18	0.14	0.29
Equiv. Pattern (ft x ft)	9.63	7.80	6.20	7.73	8.77	6.09
Stability	Stable	Stable	Stable	Caved	Caved	Stable
Strike Length (m)	24.4	15	20	34	62	28.4
Average Width (m)	7.3	20			27	21
Dip (degrees)	84	15	30–40	30–40		68
Vertical Height (m)	35.4	22			28	95
Area (m2)	231.9	300	1300	1930	1604	358
Perimeter (m)	64.9	70	170	172	187	85
Hydraulic Radius (m)	3.6	4.3	7.6	11.2	8.6	4.2
Mining Method	Blasthole/Sh	Blasthole	Blasthole	Blasthole	Blasthole	Blasthole
Dilution (%)	25					18
Cable Diameter (mm)	15.9	15.9	15.9	15.9	15.9 & 12.7	15.9
Hole Diameter (mm)	51	51 & 57	51 & 57	51 & 57	51 & 57	51
Cost ($/m)	29.46	16.40	19.71	19.71	19.71	28.84
Bonus	$40/Shift	$45–55/MS	$45–55/MS	$45–55/MS	$45–55/MS	
Supplier	Thiessen					Thiessen
Timing	7–14d					28d
Plates	No	No	No	No	No	254 x 254 mm
Straps	No	No	No	No	No	No
Other Support	No	RB				RB
Grout Pump	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000	Spedel 6000
Water:Cement Ratio	0.45	0.55			0.55	0.45
Grout Tube Dia. (mm)	19	19			19	19
Breather Tube Dia. (mm)	9.5	9.5			9.5	9.5
Rock Type	CA/AA	Fault	Fault	Fault	Fault	MS
RMR'	56				28	80
Q'	11.7	0.6	0.5	0.5	0.6	13.3
A	1.0	1	1	1	1.0	0.1
B	0.2	0.4	0.3	0.3	0.2	0.3
C	2.0	2.0	3.0	3.0	2.0	2.0
N'	4.7	0.48	0.45	0.45	0.24	0.80
RQD	70	5	10	10	5	80
Jn	6	3	4	4	3	6
Jr	2	1.5			1.5	1
Ja	2	4			4	1
RQD/Jn/H.R.	3.25	0.39	0.33	0.22	0.19	3.17
Blasthole Dia (mm)	51	51?			51	114
Backfill	None	Yes	Yes	No	Yes (P5 Tight)	None

TABLE A.4: Case Study Summary (Case 19 to 24)

CASE #	19	20	21	22	23	24
Surface	Back	HW	HW	Back	Back	Back
Cables/Hole	2	1	2	N/A	2	N/A
Hole Length (m)	9.8	14.6	14.6	N/A	9.8	14.6
Pattern	Sq & HW Fan	Point Anchor	Point Anchor	N/A	Sq. & HW Fan	HW/FW Fan
Density (bolts/m2)	0.27	0.035	0.031	N/A	0.33	N/A
Equiv. Pattern (ft x ft)	6.31	17.54	18.63	N/A	5.19	N/A
Stability	Unstable	Caved	Stable	Caved	Stable	Caved
Strike Length (m)	61	33.3	28.8	41.5	26	41.2
Average Width (m)	11.9	10	6.2	13.2	15	14
Dip (degrees)	70	62	72	84	69	70
Vertical Height (m)	97	89	90	59	55	56.8–70
Area (m2)	742	3187	2477	644	455	573
Perimeter (m)	144.1	256.4	229.6	103.4	80	109.3
Hydraulic Radius (m)	5.2	12.4	10.8	6.2	5.2	5.2
Mining Method	Blasthole	Blasthole	Blasthole	Blasthole	Blasthole	Blasthole
Dilution (%)	19	30	26.9	58.4	28.4	
Cable Diameter (mm)	15.9	15.9	15.9	N/A	15.9	N/A
Hole Diameter (mm)	51	51	51	N/A	51	N/A
Cost ($/m)	28.84	28.84	28.84	N/A	28.84	N/A
Bonus				N/A		N/A
Supplier	Thiessen	Thiessen	Thiessen	N/A	Thiessen	N/A
Timing	28d	28d	28d	N/A	28d	N/A
Plates	254 x 254 mm	102 x 102 mm	102 x 102 mm	N/A	102 x 102 mm	N/A
Straps	No	No	No	No	No	N/A
Other Support	RB	No	No	RB	RB	RB
Grout Pump	Spedel 6000	Spedel 6000	Spedel 6000	N/A	Spedel 6000	N/A
Water:Cement Ratio	0.45	0.45	0.45	N/A	0.45	N/A
Grout Tube Dia. (mm)	19	19	19	N/A	19	N/A
Breather Tube Dia. (mm)	9.5	9.5	9.5	N/A	9.5	N/A
Rock Type	MS	Acid Sed.	Acid Sed/Qte	MS	MS	MS
RMR'	80	69	64	70–80		
Q'	13.3	15.8	15.8	13.3	13.3	13.3
A	0.1	1.0	1.0	0.1	0.1	0.1
B	0.3	0.2	0.3	0.2	0.2	0.2
C	2.0	5.0	6.0	2.0	2.0	2.0
N'	0.80	15.8	28.4	0.53	.53	0.53
RQD	80	95	95	80	80	80
Jn	6	6	6	6	6	6
Jr	1	1	1	1	1	1
Ja	1	1	1	1	1	1
RQD/Jn/H.R.	2.56	1.27	1.46	2.15	2.56	2.56
Blasthole Dia (mm)	114	114	114	114	114	114
Backfill	None	None	None	None	None	None

TABLE A.5: Case Study Summary (Case 25 to 30)

CASE #	25	26	27	28	29	30
Surface	Back	HW	HW	HW	Back	Back
Cables/Hole	2	1	1	N/A	2	1
Hole Length (m)	9.8	14.6	14.6	N/A	18.3	6.1
Pattern	HW/FW & Sq.	Point Anchor	Point Anchor	N/A	Square C&F	Fan
Density (bolts/m2)	0.17	0.025	0.041	N/A	0.167	0.41
Equiv. Pattern (ft x ft)	7.96	20.75	16.20	N/A	8.0	5.12
Stability	Caved	Stable	Stable	Caved	Stable	Stable
Strike Length (m)	71	27.7	28.6	28.4	4.6	68.1
Average Width (m)	14	9	8.7	11.5	42.6	4.3
Dip (degrees)	70	79	74	59	70	60(30-80)
Vertical Height (m)	55-70	139	85	72	4.0	54
Area (m2)	1105	3854	2402	2297	195	289
Perimeter (m)	172.1	335	225	222	95	144.7
Hydraulic Radius (m)	6.4	11.5	10.7	10.3	2.1	2.0
Mining Method	Blasthole	Blasthole	Blasthole	Blasthole	Cut and Fill	Blasthole
Dilution (%)						
Cable Diameter (mm)	15.9	15.9	15.9	N/A	15.9	15.9
Hole Diameter (mm)	51	51	51	N/A	57	51
Cost ($/m)	28.84	28.84	28.84	N/A	23.79	
Bonus				N/A	$0.42/m	
Supplier	Thiessen	Thiessen	Thiessen	N/A		Thiessen
Timing	28d	28d	28d	N/A	28d	28d
Plates	102 x 102 mm			N/A	No	152 x 152 mm
Straps	No	No	No	No	No	Yes
Other Support	RB	No	No	No	RB	RB/Scr/Ex
Grout Pump	Minepro 3?	Spedel 6000	Spedel 6000	N/A	Moyno	Spedel 6000
Water:Cement Ratio	0.27-0.33?	0.45	0.45	N/A	0.5	0.4-0.45
Grout Tube Dia. (mm)	19	19	19	N/A	19	19
Breather Tube Dia. (mm)	9.5	9.5	9.5	N/A	12.7	N/A
Rock Type	MS	MS/Chl Sch	MS/Chl Sch			MS/Sx/Qte
RMR'		60	68		30	56
Q'	13.3	15.8	15.8	10	0.9	11.6
A	0.1	1.0	1.0	1.0	0.1	0.1
B	0.2	0.3	0.3	0.3	0.8	0.2
C	2.0	7.0	6.5	5.0	2.0	2.0
N'	0.53	33.2	30.8	15.0	0.14	0.46
RQD	80	95	95	90	10	79
Jn	6	6	6	9	4	12
Jr	1	1	1	1	1.5	2.3
Ja	1	1	1	1	4.0	1.3
RQD/Jn/H.R.	2.08	1.37	1.48	0.97	1.19	3.30
Blasthole Dia (mm)	114	114	114	114	38	114
Backfill	None	None	None	None	Tailings	None

TABLE A.6: Case Study Summary (Case 31 to 36)

CASE #	31	32	33	34	35	36
Surface	HW	HW	Back	Back	HW	Back
Cables/Hole	N/A	1	1	1	N/A	1
Hole Length (m)	N/A	12	6.1	9.1	N/A	6.1
Pattern	N/A	Quasi-Mandolin	Fan	Point Anchor	N/A	Square
Density (bolts/m2)	N/A	0.07	0.54	0.30	N/A	0.55
Equiv. Pattern (ft x ft)	N/A	12.4	4.46	5.99	N/A	4.42
Stability	Caved	Stable	Stable	Unstable	Stable	Stable
Strike Length (m)	68.1	23.8	34.1	22.6	22.6	27.1
Average Width (m)	4.3					3.7
Dip (degrees)	60(30-80)	83		90	90	N/A
Vertical Height (m)	54	40.4	42.7	35.1	35.1	3.7
Area (m2)	4290	295	135.4	420	825	169.8
Perimeter (m)	262.2	60	80.6	82	118	92.7
Hydraulic Radius (m)	16.4	4.9	1.7	5.1	7.0	1.8
Mining Method	Blasthole	Blasthole	VCR	Blasthole	Blasthole	Drift
Dilution (%)						
Cable Diameter (mm)	N/A	15.9	15.9	15.9	N/A	15.9
Hole Diameter (mm)	N/A	64	51	51	N/A	51
Cost ($/m)	N/A				N/A	
Bonus	N/A				N/A	
Supplier	N/A	Thiessen	Thiessen	Thiessen	N/A	Thiessen
Timing	N/A	28d	28d	28d	N/A	28d
Plates	N/A	152 x 152 mm	152 x 152 mm	152 x 152 mm	N/A	152 x 152 mm
Straps	Sub1	Yes	No	No		No
Other Support	Sub1	No	RB/Scr	Rebar/Ex		RB/Ex
Grout Pump	N/A	Spedel 6000	Spedel 6000	Spedel 6000	N/A	Spedel 6000
Water:Cement Ratio	N/A	0.4	0.45	0.4	N/A	0.4-0.45
Grout Tube Dia. (mm)	N/A	19	19	19	N/A	19
Breather Tube Dia. (mm)	N/A	None	9.5	9.5	N/A	N/A
Rock Type	Qte	Prdt	M.Sch	MS/Sx/Prdt/Msh	Msch/Sch	Qte
RMR'	55	49	60	48	64	
Q'	5.9	10.4	8.9	8.3	13.1	29.2
A	1.0	1.0	0.1	0.1	1.0	1.0
B	0.2	0.2	0.4	0.2	0.2	0.5
C	5.5	5.0	2.0	2.0	8.0	2.0
N'	6.5	10.4	0.71	0.33	21.0	29.2
RQD	55	83	80	70	82	100
Jn	15	12	9	15	15	9
Jr	2.1	2.4	2	2.5	2.4	2.1
Ja	1.3	1.6	2	1.4	1.0	0.8
RQD/Jn/H.R.	0.22	1.41	5.24	0.92	0.78	6.17
Blasthole Dia (mm)	114	114		114	114	
Backfill	None	C&F Below		Cem R/F	Cem R/F	None

TABLE A.7: Case Study Summary (Case 37 to 42)

CASE #	37	38	39	40	41	42
Surface	Back	HW	Back	HW	Back	HW
Cables/Hole	1	N/A	N/A	N/A	N/A	N/A
Hole Length (m)	12.2	N/A	N/A	N/A	N/A	N/A
Pattern	Fan	N/A	N/A	N/A	N/A	N/A
Density (bolts/m2)	0.41	N/A	N/A	N/A	N/A	N/A
Equiv. Pattern (ft x ft)	5.12	N/A	N/A	N/A	N/A	N/A
Stability	Stable	Stable	Stable	Stable	Stable	Unstable
Strike Length (m)	15.2	15.2	20.1	20.1	19.8	19.8
Average Width (m)	6.4	6.4	3.0	3.0	3.1	3.1
Dip (degrees)	54	54	70	70	59	59
Vertical Height (m)	29	29	30.5	30.5	29	29
Area (m2)	97.3	500	60.1	613	90	628
Perimeter (m)	43.2	96	46.2	101.2	51	103
Hydraulic Radius (m)	2.3	5.2	1.3	6.1	1.8	6.1
Mining Method	VCR	VCR	VCR	VCR	VCR	VCR
Dilution (%)						
Cable Diameter (mm)	15.9	N/A	N/A	N/A	N/A	N/A
Hole Diameter (mm)	51	N/A	N/A	N/A	N/A	N/A
Cost ($/m)			N/A	N/A	N/A	N/A
Bonus			N/A	N/A	N/A	N/A
Supplier	Thiessen	N/A	N/A	N/A	N/A	N/A
Timing	28d+	N/A	N/A	N/A	N/A	N/A
Plates	152 x 152 mm	N/A	N/A	N/A	N/A	N/A
Straps	Yes(Laced)	N/A	No	No	No	No
Other Support	RB/Scr/Ex	RB/Scr	RB/Scr/Ex	RB/Scr	RB/Scr/Ex	RB/Scr/Ex
Grout Pump	Spedel 6000	N/A	N/A	N/A	N/A	N/A
Water:Cement Ratio	0.4–0.45	N/A	N/A	N/A	N/A	N/A
Grout Tube Dia. (mm)	19	N/A	N/A	N/A	N/A	N/A
Breather Tube Dia. (mm)	None	N/A	N/A	N/A	N/A	N/A
Rock Type	Sumx	Prdt	Sumx	Bio Sch	Sumx	Sch
RMR'		46				
Q'	12.3	7.2	15.8	21.5	15.8	21.5
A	0.1	1.0	0.1	1.0	0.1	1.0
B	0.2	0.2	0.2	0.3	0.2	0.2
C	2.0	5.0	2.0	6.0	2.0	5.0
N'	0.49	7.2	0.63	38.7	0.63	21.5
RQD	70	60	90	92	90	92
Jn	12	12	12	9	12	9
Jr	2.1	2.3	2.1	2.1	2.1	2.1
Ja	1	1.6	1	1	1	1
RQD/Jn/H.R.	2.52	0.96	5.77	1.68	4.17	1.68
Blasthole Dia (mm)	114	114	114	114	114	114
Backfill			C&F Below	C&F Below	None	None

TABLE A.8: Case Study Summary (Case 43 to 48)

CASE #	43	44	45	46	47	48
Surface	Back	HW	Back	Back	Back	Back
Cables/Hole	N/A	N/A	1	1	1	1
Hole Length (m)	N/A	N/A	15.8	18.3	18.3	18.3
Pattern	Fan	N/A	Square C&F	Square C&F	Square C&F	Square C&F
Density (bolts/m2)	N/A	N/A	0.21	0.304	0.308	0.245
Equiv. Pattern (ft x ft)	N/A	N/A	7.16	5.99	5.89	6.63
Stability	Stable	Caved	Stable	Stable	Stable	Stable
Strike Length (m)	15.8	15.8	31	76	112	184
Average Width (m)	6.7	6.7	10.1	12	11.6	10.7
Dip (degrees)	56	56	33	65–70	70–80	60–70
Vertical Height (m)	40.5	40.5	5.2	6.1	6.1	6.1
Area (m2)	105.9	746	316	875	1295	2020
Perimeter (m)	45	126	87	175	253	406
Hydraulic Radius (m)	2.4	5.9	3.6	5.0	5.1	5.0
Mining Method	VCR	VCR	Cut and Fill	Cut and Fill	Cut and Fill	Cut and Fill
Dilution (%)						
Cable Diameter (mm)	N/A	N/A	15.9	15.9	15.9	15.9
Hole Diameter (mm)	N/A	N/A	51	51	51	51
Cost ($/m)	N/A	N/A	31.8	19.69	19.69	19.69
Bonus	N/A	N/A	$0.33/m	$0.35/ft – $4/hr	$0.35/ft – $4/hr	$0.35/ft – $4/hr
Supplier	N/A	N/A	Thiessen	Thiessen	Thiessen	Thiessen
Timing	N/A	N/A	28d+	28d	28d	28d
Plates	152 x 152 mm	N/A	No	Yes(#2&3 cut)	Yes(#2&3 cut)	Yes(#2&3 cut)
Straps	Yes & 4.9m Ex	Yes	No	No	No	No
Other Support	RB/Scr/Ex	RB/SS/Scr	Swellex/RB	RB	RB	RB
Grout Pump	N/A	N/A	Minepro 3	Minepro 3	Minepro 3	Minepro 3
Water:Cement Ratio	N/A	N/A	0.35–0.4	0.35–0.40	0.35–0.40	0.35–0.40
Grout Tube Dia. (mm)	N/A	N/A	19	19	19	19
Breather Tube Dia. (mm)	N/A	N/A	9.5	9.5 Hi Press	9.5 Hi Press	9.5 Hi Press
Rock Type	Sumx/Prdt	Prdt	SS	DS/SS	DS/SS	SS
RMR'		46	79	55	55	84
Q'	11.1	7.2	26.1	5.4	5.4	25
A	0.1	1.0	0.1	0.1	0.1	0.4
B	0.2	0.2	0.3	0.2	0.2	0.2
C	2.0	5.0	2.0	2.0	2.0	2.0
N'	0.44	7.2	1.6	0.22	0.22	4.0
RQD	75	60	88	65	65	100
Jn	12	12	9	6	6	6
Jr	2.3	2.3	2.0	2.0	2.0	1.5
Ja	1.3	1.6	0.75	4.0	4.0	1.0
RQD/Jn/H.R.	2.60	0.85	2.72	2.16	2.12	3.34
Blasthole Dia (mm)	114	114	38	38	38	38
Backfill	C&F Below	C&F Below	Tailings	Sand/Hyd	Sand/Hyd	Sand/Hyd

TABLE A.9: Case Study Summary (Case 49 to 54)

CASE #	49	50	51	52	53	54
Surface	HW	HW	HW	Back	Back	HW
Cables/Hole	1	1	N/A	1	1	1
Hole Length (m)	9.1	9.1	N/A	10.2	10.7	9.1
Pattern	Point Anchor	Point Anchor	N/A	Square	Square	Even
Density (bolts/m2)	0.16	0.18	N/A	0.11	0.33	0.13
Equiv. Pattern (ft x ft)	8.20	7.73	N/A	9.89	5.71	9.10
Stability	Stable	Caved	Stable	Stable	Stable	Stable
Strike Length (m)	130	111	76	76	70	70
Average Width (m)	11.5	11.0	7.8	12	17.2	17.2
Dip (degrees)	62	62	60	60	60	60
Vertical Height (m)	41	57.5	25	25	17.5	17.5
Area (m2)	6111	7565	2236	936	654	1378
Perimeter (m)	395	445	215	180	172	175
Hydraulic Radius (m)	15.5	17.0	10.4	5.2	3.8	7.9
Mining Method	Blasthole	Blasthole	Blasthole	Blasthole	Blasthole	Blasthole
Dilution (%)						
Cable Diameter (mm)	15.9	15.9	N/A	15.9	15.9	15.9
Hole Diameter (mm)	51	51	N/A	51	51	51
Cost ($/m)	19.69	19.69	N/A	19.69	19.69	19.69
Bonus	$0.34/ft−$4/hr	$0.34/ft−$4/hr	N/A	$0.34/ft−$4/hr	$0.34/ft−$4/hr	$0.34/ft−$4/hr
Supplier	Thiessen	Thiessen	N/A	Thiessen	Thiessen	Thiessen
Timing	28d	28d	N/A	28d	28d	28d
Plates	No	No	N/A	No	Yes	No
Straps	No	No	No	No	No	No
Other Support			RB/Resin	RB/Resin	RB	No
Grout Pump	Minepro 3	Minepro 3	N/A	Minepro 3	Minepro 3	Minepro 3
Water:Cement Ratio	0.35−0.40	0.35−0.40	N/A	0.35−0.40	0.35−0.40	0.35−0.40
Grout Tube Dia. (mm)	19	19	N/A	19	19	19
Breather Tube Dia. (mm)	9.5 (hi press)	9.5 (hi press)	N/A	9.5 (hi press)	9.5 (hi press)	9.5 (hi press)
Rock Type	SCQP	SCQP	SCQP	SS(80%)/DS	DS	SCQP
RMR'			62		55	59
Q'	9.9	3.1	8.3	15.5	5.4	3.1
A	1.0	1.0	1.0	0.1	0.1	1.0
B	0.3	0.3	0.3	0.2	0.2	0.3
C	4.7	4.7	5.0	2.0	2.0	5.0
N'	14.0	4.4	12.5	0.62	0.22	4.7
RQD	92	75	75	93	65	75
Jn	7	9	6	6	6	9
Jr	1.5	1.5	2	1.6	2.0	1.5
Ja	2	4	3	1.6	4	4
RQD/Jn/H.R.	0.85	0.49	1.20	2.98	2.84	1.05
Blasthole Dia (mm)	76	76	51	51	76	76
Backfill	No	No	Waste/Hyd	Waste/Hyd	No	No

TABLE A.10: Case Study Summary (Case 55 to 59)

CASE #	55	56	57	58	59	
Surface	Back	HW	Back	HW	Back	
Cables/Hole	1	1	1	N/A	2	
Hole Length (m)	5.0	7.6–9.1	6.4	N/A	4.9	
Pattern	Point Anchor	Even	Square	N/A	Square R&P	
Density (bolts/m2)	0.35	0.12	0.34	N/A	0.26	
Equiv. Pattern (ft x ft)	5.55	9.47	5.71	N/A	6.43	
Stability	Stable	Stable	Stable	Unstable	Stable	
Strike Length (m)	141	141	25	25	144	
Average Width (m)	13.7	13.7	18.8	18.8	12.2	
Dip (degrees)	60	60	70	70	20	
Vertical Height (m)	15.7	15.7	15	15	5.2	
Area (m2)	1862	2439	473	400	1757	
Perimeter (m)	302	317	88	82	312	
Hydraulic Radius (m)	6.2	10.9	5.4	4.9	5.6	
Mining Method	Blasthole	Blasthole	Blasthole	Blasthole	Room & Pillar	
Dilution (%)						
Cable Diameter (mm)	15.9	15.9	15.9	N/A	15.9	
Hole Diameter (mm)	51	51	51	N/A	64	
Cost ($/m)	19.69	19.69	19.69	N/A		
Bonus	$0.34/ft–$4/hr	$0.34/ft–$4/hr	$0.34/ft–$4/hr	N/A	$133/shift	
Supplier	Thiessen	Thiessen	Thiessen	N/A	Thiessen	
Timing	28d	28d	28d	N/A	28d	
Plates	No	No	No	No	No	
Straps	No	No	No	No	No	
Other Support	RB	No	RB	RB	Swellex	
Grout Pump	Minepro 3	Minepro 3	Minepro 3	N/A	Minepro 3	
Water:Cement Ratio	0.35–0.40	0.35–0.40	0.35–0.40	N/A	0.35–0.40	
Grout Tube Dia. (mm)	19	19	19	N/A	19 recovered	
Breather Tube Dia. (mm)	9.5 (hi press)	9.5 (hi press)	9.5 (hi press)	N/A	None	
Rock Type	SS	SCQP	DS	SCQP	Sulphide	
RMR'	84	59	55	59	69	
Q'	25	3.1	5.4	3.1	25	
A	0.2	1.0	0.1	1.0	0.4	
B	0.2	0.3	0.2	0.3	0.3	
C	2.0	5.0	2.0	6.0	2.2	
N'	2.0	4.7	0.22	5.6	6.6	
RQD	100	75	65	75	90	
Jn	6	9	6	9	6–9	
Jr	1.5	1.5	2.0	1.5	1.5–2.1	
Ja	1.0	4.0	4.0	4.0	0.75–1	
RQD/Jn/H.R.	2.69	0.76	2.00	1.70	2.30 avg.	
Blasthole Dia (mm)	76	76	51	51	38	
Backfill	No	No	No	No	None	

APPENDIX B

BOLT DENSITY CONVERSION CHART

TABLE B.1: Bolt Density Conversion Chart

BOLTS/SQ. METER	BOLTS/SQ. FOOT	SQUARE PATTERN EQUIVALENT (m x m)	SQUARE PATTERN EQUIVALENT (ft x ft)	BOLTS/SQ. METER	BOLTS/SQ. FOOT	SQUARE PATTERN EQUIVALENT (m x m)	SQUARE PATTERN EQUIVALENT (ft x ft)
0.000	0.0000	0.00	0.00	0.275	0.0255	1.91	6.26
0.005	0.0005	14.14	46.40	0.280	0.0260	1.89	6.20
0.010	0.0009	10.00	32.81	0.285	0.0265	1.87	6.15
0.015	0.0014	8.16	26.79	0.290	0.0269	1.86	6.09
0.020	0.0019	7.07	23.20	0.295	0.0274	1.84	6.04
0.025	0.0023	6.32	20.75	0.300	0.0279	1.83	5.99
0.030	0.0028	5.77	18.94	0.305	0.0283	1.81	5.94
0.035	0.0033	5.35	17.54	0.310	0.0288	1.80	5.89
0.040	0.0037	5.00	16.41	0.315	0.0293	1.78	5.85
0.045	0.0042	4.71	15.47	0.320	0.0297	1.77	5.80
0.050	0.0046	4.47	14.67	0.325	0.0302	1.75	5.76
0.055	0.0051	4.26	13.99	0.330	0.0307	1.74	5.71
0.060	0.0056	4.08	13.39	0.335	0.0311	1.73	5.67
0.065	0.0060	3.92	12.87	0.340	0.0316	1.71	5.63
0.070	0.0065	3.78	12.40	0.345	0.0320	1.70	5.59
0.075	0.0070	3.65	11.98	0.350	0.0325	1.69	5.55
0.080	0.0074	3.54	11.60	0.355	0.0330	1.68	5.51
0.085	0.0079	3.43	11.25	0.360	0.0334	1.67	5.47
0.090	0.0084	3.33	10.94	0.365	0.0339	1.66	5.43
0.095	0.0088	3.24	10.64	0.370	0.0344	1.64	5.39
0.100	0.0093	3.16	10.38	0.375	0.0348	1.63	5.36
0.105	0.0098	3.09	10.13	0.380	0.0353	1.62	5.32
0.110	0.0102	3.02	9.89	0.385	0.0358	1.61	5.29
0.115	0.0107	2.95	9.68	0.390	0.0362	1.60	5.25
0.120	0.0111	2.89	9.47	0.395	0.0367	1.59	5.22
0.125	0.0116	2.83	9.28	0.400	0.0372	1.58	5.19
0.130	0.0121	2.77	9.10	0.405	0.0376	1.57	5.16
0.135	0.0125	2.72	8.93	0.410	0.0381	1.56	5.12
0.140	0.0130	2.67	8.77	0.415	0.0386	1.55	5.09
0.145	0.0135	2.63	8.62	0.420	0.0390	1.54	5.06
0.150	0.0139	2.58	8.47	0.425	0.0395	1.53	5.03
0.155	0.0144	2.54	8.33	0.430	0.0399	1.52	5.00
0.160	0.0149	2.50	8.20	0.435	0.0404	1.52	4.97
0.165	0.0153	2.46	8.08	0.440	0.0409	1.51	4.95
0.170	0.0158	2.43	7.96	0.445	0.0413	1.50	4.92
0.175	0.0163	2.39	7.84	0.450	0.0418	1.49	4.89
0.180	0.0167	2.36	7.73	0.455	0.0423	1.48	4.86
0.185	0.0172	2.32	7.63	0.460	0.0427	1.47	4.84
0.190	0.0176	2.29	7.53	0.465	0.0432	1.47	4.81
0.195	0.0181	2.26	7.43	0.470	0.0437	1.46	4.79
0.200	0.0186	2.24	7.34	0.475	0.0441	1.45	4.76
0.205	0.0190	2.21	7.25	0.480	0.0446	1.44	4.74
0.210	0.0195	2.18	7.16	0.485	0.0451	1.44	4.71
0.215	0.0200	2.16	7.08	0.490	0.0455	1.43	4.69
0.220	0.0204	2.13	7.00	0.495	0.0460	1.42	4.66
0.225	0.0209	2.11	6.92	0.500	0.0464	1.41	4.64
0.230	0.0214	2.09	6.84	0.505	0.0469	1.41	4.62
0.235	0.0218	2.06	6.77	0.510	0.0474	1.40	4.59
0.240	0.0223	2.04	6.70	0.515	0.0478	1.39	4.57
0.245	0.0228	2.02	6.63	0.520	0.0483	1.39	4.55
0.250	0.0232	2.00	6.56	0.525	0.0488	1.38	4.53
0.255	0.0237	1.98	6.50	0.530	0.0492	1.37	4.51
0.260	0.0242	1.96	6.43	0.535	0.0497	1.37	4.49
0.265	0.0246	1.94	6.37	0.540	0.0502	1.36	4.46
0.270	0.0251	1.92	6.31	0.545	0.0506	1.35	4.44

Table B.1: Bolt Density Conversion Chart (con't)

BOLTS/SQ. METER	BOLTS/SQ. FOOT	SQUARE PATTERN EQUIVALENT (m x m)	SQUARE PATTERN EQUIVALENT (ft x ft)	BOLTS/SQ. METER	BOLTS/SQ. FOOT	SQUARE PATTERN EQUIVALENT (m x m)	SQUARE PATTERN EQUIVALENT (ft x ft)
0.550	0.0511	0.00	0.00	0.825	0.0766	1.10	3.61
0.555	0.0516	1.34	4.40	0.830	0.0771	1.10	3.60
0.560	0.0520	1.34	4.38	0.835	0.0776	1.09	3.59
0.565	0.0525	1.33	4.36	0.840	0.0780	1.09	3.58
0.570	0.0529	1.32	4.35	0.845	0.0785	1.09	3.57
0.575	0.0534	1.32	4.33	0.850	0.0790	1.08	3.56
0.580	0.0539	1.31	4.31	0.855	0.0794	1.08	3.55
0.585	0.0543	1.31	4.29	0.860	0.0799	1.08	3.54
0.590	0.0548	1.30	4.27	0.865	0.0804	1.08	3.53
0.595	0.0553	1.30	4.25	0.870	0.0808	1.07	3.52
0.600	0.0557	1.29	4.24	0.875	0.0813	1.07	3.51
0.605	0.0562	1.29	4.22	0.880	0.0817	1.07	3.50
0.610	0.0567	1.28	4.20	0.885	0.0822	1.06	3.49
0.615	0.0571	1.28	4.18	0.890	0.0827	1.06	3.48
0.620	0.0576	1.27	4.17	0.895	0.0831	1.06	3.47
0.625	0.0581	1.26	4.15	0.900	0.0836	1.05	3.46
0.630	0.0585	1.26	4.13	0.905	0.0841	1.05	3.45
0.635	0.0590	1.25	4.12	0.910	0.0845	1.05	3.44
0.640	0.0595	1.25	4.10	0.915	0.0850	1.05	3.43
0.645	0.0599	1.25	4.09	0.920	0.0855	1.04	3.42
0.650	0.0604	1.24	4.07	0.925	0.0859	1.04	3.41
0.655	0.0608	1.24	4.05	0.930	0.0864	1.04	3.40
0.660	0.0613	1.23	4.04	0.935	0.0869	1.03	3.39
0.665	0.0618	1.23	4.02	0.940	0.0873	1.03	3.38
0.670	0.0622	1.22	4.01	0.945	0.0878	1.03	3.38
0.675	0.0627	1.22	3.99	0.950	0.0882	1.03	3.37
0.680	0.0632	1.21	3.98	0.955	0.0887	1.02	3.36
0.685	0.0636	1.21	3.96	0.960	0.0892	1.02	3.35
0.690	0.0641	1.20	3.95	0.965	0.0896	1.02	3.34
0.695	0.0646	1.20	3.94	0.970	0.0901	1.02	3.33
0.700	0.0650	1.20	3.92	0.975	0.0906	1.01	3.32
0.705	0.0655	1.19	3.91	0.980	0.0910	1.01	3.31
0.710	0.0660	1.19	3.89	0.985	0.0915	1.01	3.31
0.715	0.0664	1.18	3.88	0.990	0.0920	1.01	3.30
0.720	0.0669	1.18	3.87	0.995	0.0924	1.00	3.29
0.725	0.0673	1.17	3.85	1.000	0.0929	1.00	3.28
0.730	0.0678	1.17	3.84	1.005	0.0934	1.00	3.27
0.735	0.0683	1.17	3.83	1.010	0.0938	1.00	3.26
0.740	0.0687	1.16	3.81	1.015	0.0943	0.99	3.26
0.745	0.0692	1.16	3.80	1.020	0.0948	0.99	3.25
0.750	0.0697	1.15	3.79	1.025	0.0952	0.99	3.24
0.755	0.0701	1.15	3.78	1.030	0.0957	0.99	3.23
0.760	0.0706	1.15	3.76	1.035	0.0961	0.98	3.23
0.765	0.0711	1.14	3.75	1.040	0.0966	0.98	3.22
0.770	0.0715	1.14	3.74	1.045	0.0971	0.98	3.21
0.775	0.0720	1.14	3.73	1.050	0.0975	0.98	3.20
0.780	0.0725	1.13	3.72	1.055	0.0980	0.97	3.19
0.785	0.0729	1.13	3.70	1.060	0.0985	0.97	3.19
0.790	0.0734	1.13	3.69	1.065	0.0989	0.97	3.18
0.795	0.0739	1.12	3.68	1.070	0.0994	0.97	3.17
0.800	0.0743	1.12	3.67	1.075	0.0999	0.96	3.16
0.805	0.0748	1.11	3.66	1.080	0.1003	0.96	3.16
0.810	0.0752	1.11	3.65	1.085	0.1008	0.96	3.15
0.815	0.0757	1.11	3.63	1.090	0.1013	0.96	3.14
0.820	0.0762	1.10	3.62	1.095	0.1017	0.96	3.14

APPENDIX C

SUMMARY OF INSTALLATION PROCEDURES

CABLE BOLT INSTALLATION PROCEDURES

MINE #1

Spedel 6000 Grout Pump
Double cables
2 man crew

Downholes:
Use 3/4" loading hose to clean all holes prior to inserting cable bolts.
Tape 3/4" grout tube 6" from one end of the cable using enough tube to run the whole length of the hole and attach to the grout pump.
Lower cables into holes to be grouted with taped end of grout tube at toe of hole.
Prepare .35 w:c grout and hook up pump to grout tube.
Pump grout until it begins to come out of the collar.

Upholes:
Tape 3/8" breather tube to end of cable and insert into hole with taped end of breather tube at the toe of hole.
Insert grout tube 2' into collar of hole and leave enough to reach to the pump.
Plug hole collar with rags.
Prepare 0.35 w:c grout and hook up pump to grout tube.
Pump grout until it comes out of the breather tube.
Bend and tie off breather and grout tubes to prevent grout from leaking.

MINE #2

Minepro 3 Pneumatic Grout Pump.
213'/ms (65m/ms) inserted (Double cables, 30-49' bolts (9-15m) ,countersink 0-16' (0-5m))
591 '/ms (180m/ms) grouted (w:c=0.32, 9-19m)
2 man crew

Upholes > 40' (12m):
Install spring steel on end holding device.
Tape 1/2" breather tube to end of cable with end of breather tube cut at 45 degrees.
Insert cable in hole and ensure that breather tube is not pinched.
Connect water hose to breather tube and flush hole.
Place 3/4" grout tube 1m into hole collar with end cut at 45 degrees.
Plug hole with MONOFOAM and allow to cure 24 hours.
Fill hole with 0.375 w:c grout
Ensure breather tube is completely full of grout.

Upholes < 40' (12m):
Install spring steel on end holding device.
Tape 3/4" grout tube to end of cable.
Insert cable into hole.
Connect water hose to breather tube and flush hole.
Fill hole with 0.32 w:c grout
No plug to be used at collar

MINE #3

Spedel 6000 Pump & B3100 Mixer
Grout & Install in 45 minutes.
Double cables @ 33' (10m)
2 man crew

Upholes:
Wash down cable to be installed.
Install spring steel on end holding device.
Attach 3/8" breather tube 6" (15cm) from end of cable with electrical tape.
Wrap with tape at several locations along the cable.
Tape 3/4" grout tube 6.5' (2m) from the hole collar.
Insert cable into the hole.
Seal hole collar with burlap and wooden wedges.
Cut breather & grout tubes 10' (3m) from the collar.
Attach grout tube to grout pump feeder hose and pump 0.4 w:c grout until it discharges from the breather tube.

MINE #4

Moyno 3L3 Pump with Chemgrout CG-550 mini grout plant.
600'/ms installed and grouted (60' upholes)
2 man crew

Upholes:
Bend back one wire if hole < 40' or 2 wires if > 40'
Use cable pusher if greater than 40'
Install breather and grout tube combination
Plug holes with shredded cloth.
Mix grout like pudding so will squeeze through fingers.

MINE #5

Spedel 6000 grout pump.
8 40' holes grouted in three hours - w:c 0.35-0.4
Single cables
2 man crew

<u>Upholes:</u>
Install spring steel on end holding device.
Tape 3/4" grout tube to top end of cable with 2' taped segment.
Insert cable into hole
Cut grout tube to allow end to reach grout pump
Mix 1 bag (40kg) cement with 12 litres water. (w:c = 0.3)
Pump grout until expelled at hole collar.
Plug collar with steel wool (4 rolls)
Pump grout until pump stalls.
Crimp and tape grout tube at collar.

MINE #6

Spedel 6000 grout pump
Minpro 3 Host Hydraulic purchased June 91.

Procedure same as Mine #5

MINE #7

Pneumatic skid mounted Minpro 3 grout pump.
Single cables
2 man crew

<u>Upholes:</u>
Bend wire at top of cable 135 degrees.
Install cable in hole.
Push in grout tube to toe of hole and pull back 6".
Tape grout tube to cable sticking out of collar.
Cut off grout tube with enough to reach the grout pump hose.
Install wedges in hole collar.
Pump grout and bend end of grout tube and tie off.

MINE #9

Minpro 3 Grout pump
2 man crew

Upholes > 45':
Install spring steel on end holding device.
Tape 3/8" (450 psi) breather tube to the cable in two or three places.
Keep the breather tube end within 6" of the end holding device and cut the end at 45 degrees.
Install the cable in the hole.
Push a 3/4" (250 psi) grout tube 1' to 3' into the hole and leave a 3' tail out of the hole.
Wedge the cable in the hole and plug the collar with cotton waste and thick grout.
Let the collar seals set for eight hours.
Mix grout between 0.33 and 0.40 w:c (90 litres water and pump until grout flows out of the breather tube.
Bend end of the grout and breather tube 180 degrees and tie in this position.

Upholes < 45':
Install spring steel on end holding device.
Tape a 3/4" (100 psi) grout tube to the cable in two or three places.
Keep the grout tube end within 6" of the end holding device.
Insert the cable in the hole.
Wedge the cable at the hole collar with a wooden wedge.
Do not pinch the grout tube.
Mix grout to a 0.30 w:c ratio and pump until it squeezes out of the hole collar.
Kink and tie off the grout tube.
Grout w:c ratios > 0.35 will require a grout plug so it is important to maintain a 0.3 w:c ratio.

Downholes:
Tape a 3/4" grout tube (100 psi) within 6" of the cable end.
Insert the cable into the hole and push to the end of the hole.
Mix a 0.3 w:c ratio grout and pump until the grout appears at the collar.
Do not extract the grout tube while pumping as tests have shown that air gaps may result in the grout column.

MINE #10

Pneumatic skid mounted Minpro 3 grout pump.
Double cables, 16'(4.9m)
2 man crew

Bend 9" length of two cable wires back 135 degrees.
Install cables in hole with anchor end towards toe of hole.
Push 3/4" grout tube to toe of hole.
Pump 0.32 w:c ratio grout and retrieve grout tube as grout is pumped into hole.
Grout tube should gently push back as grout is pumped and a slight hand pressure is required to hold the tube in place.
When hole is full push a pilgrim hat plug into the collar.

MINE #11

Tamrock Cabolt Machine

3/4" grout tube is fed into the hole from a reel on the Cabolt rig.
Grout is mixed to a 0.3 w:c ratio and pumped into the hole.
Cable is fed into the grouted hole from a reel underneath the bolting rig and cut with a hydraulic cutting device.
The cable is kinked prior to final installation in order to aid in anchoring the cable.