IMPROVEMENTS TO THE STABILITY OF ROCK WALLS IN OPEN PIT MINES

by

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SYNOPSIS

This research project has investigated the behaviour of untensioned fully grouted rock reinforcing and its application to the stabilization of rock walls in open pit mines. It has involved laboratory and theoretical studies related to understanding the contribution of the passive reinforcing to the shear strength of smooth and rough rock joints. A theory was developed to predict the load-deformation response of passive reinforcing subject to shear forces. Numerical studies investigated the effect of surface reinforcing in stabilizing ravelling rock slopes. A design approach was developed and field studies were performed to determine the effectiveness of improving the stability of rock slopes using fully grouted untensioned reinforcing.

CHAPTER 1

INTRODUCTION

1.1 GENERAL

The stability of excavated rock slopes is an important consideration in both civil engineering and mining. In civil engineering the possible consequences of failure of rock slopes may be so serious as to encourage designers towards very conservative solutions. One option which may be adopted could be realignment of the slope to obtain a more favourable geometry In general, as civil engineering structures are considered to be permanent, this may also govern the final solution.

In mining there has been a tendency to make major investment decisions regarding the method of mining based on the grade of the ore and experience. These decisions have then governed the pit slope angles, depth of pit and subsequent location of haul roads, crushers and waste dumps without undertaking a geotechnical investigation. The stability of the mine is related to the geometry of the major discontinuity sets, which have scant regard for the economics of the ore body. Hence the problems associated with unfavourable joint orientations, major geologic structures and weathered ground are often not recognised until well into the economic life of the pit. The consequences of instability vary enormously, from minor significance requiring periodic cleaning up to the potential loss of ore at depth, loss of haul roads or crushers, or loss of men and machinery working under a failing slope. Instability may involve large through going faults, or contact zones, or may be confined to unravelling of benches which if unchecked, may involve hundreds of tonnes of waste rock to be removed.

Arising from the concern about the stability of pit slopes in several mines, some failures, and proposals for deeper mining in open pits, Barron *et al.* (1970) suggested some extensive and expensive pit stabilization measures. These measures included tensioned anchors, buttresses, meshing and shotcreting. In view of the perceived necessity to improve stability and the acknowledged expense of tensioned anchors, some workers have proposed an alternative method using untensioned fully grouted dowels (or passive reinforcing) (Londe and Bonazzi (1974), Bjurström (1974) and Azuar (1977)). This approach is seen as a cheaper method of stabilizing the face of a pit and is probably more acceptable as a method of support for a temporary excavation. However no adequate theories had been developed to explain the behaviour of fully grouted reinforcing and its success in maintaining stability. This thesis attempts to provide an understanding of the behaviour of passive reinforcing in a discontinuous rock mass.

Fundamental to understanding the behaviour of discontinua is an appreciation of the effects of joint roughness. Work undertaken during the period of the author's candidature in collaboration with a colleague Chiu Hong Keong, has demonstrated a simple procedure to evaluate the effect of joint roughness. Predictions of the shear behaviour of joints using profiles (Dight and Chiu (1981)) and the prediction of the load settlement behaviour of side-resistance-only socketed piles (Chiu and Dight (1982)) have been made. A copy of the first paper is appended to this thesis as frequent reference is made to it.

1.2 SCOPE OF THE PROJECT

This project examines the behaviour of fully grouted rock reinforcing and its application to the stabilizing of rock walls in open pit mines. In order to achieve these aims the behaviour of fully grouted untensioned rock bolts was studied in the laboratory. The laboratory investigation examined the behaviour of smooth and rough joints reinforced with bars at different angles. The bars were strain gauged to provide information on the deformation behaviour. Tests were also performed on deformed bar and strand in shear. A preliminary investigation examined the pullout behaviour of anchored rock bolts.

A theory was developed to explain the results of the laboratory work which was also tested against published data. The theory predicts the load deformation behaviour of fully grouted rock reinforcement subject to shear forces.

A series of numerical experiments was performed using a program based on the finite element method. These experiments investigated the effect of surface reinforcement on stabilizing an imbricated rock mass.

A design procedure was developed to analyse the stability of passively reinforced rock slopes in open pit mines. Based on a design undertaken for a mine, field trials were instigated and preliminary results evaluated.

CHAPTER 2

BACKGROUND

2.1 INTRODUCTION

The need to maximize ore recovery from existing deposits has become of paramount importance with rising establishment and infrastructure costs, fuel costs and interest charges, hence there is a trend to deeper open cut mines (300 m to 1000 m). If unfavourable slope conditions are encountered during the excavation, the consequences of slope failure or loss of ore at depth become much more significant with a deeper mine where a commitment to a higher stripping ratio may result in premature closure of the mine. Some causes of slope instability have been summarized by McMahon and Le Roy (1977) as the incidence of unfavourable joint sets, weathering, throughgoing faults, water pressure, seismic activity, poor excavation techniques, or a combination of these conditions. Unfavourable joint sets may comprise stratification and foliation joints, unconformities, contacts between lithological groups, base of weathering lineation, fracture cleavage or solution channels. Figure 2.1.1 shows some typical mechanisms of failure.

Recognition of the potential mechanisms of instability and consequent slope failures are essential before the appropriate remedial action can be undertaken. However remedial action is usually only considered for stabilizing the final wall (Sage (1977)) unless associated mining facilities (haul roads, crusher ramps, crushers or mills) may be threatened during the life of the mine. Action required for handling potential instabilities will vary according to their impact on the mining operation and may range from a simple warning system, through protection, to a complete stabilization program. These actions are obviously not mutually exclusive. Table 2.1.1 summarizes the hierarchy of activity, adapted from Piteau *et al.* (1979). Early warning systems are generally of the form of electrically operated displacement monitoring systems designed to trigger an alarm or flashing light when "excessive movement has occurred. One such system used at Savage River Mines is shown in Figure 2.1.2. The systems serve to provide sufficient notice for personnel and machinery working in the vicinity of the potential collapse to vacate the area.

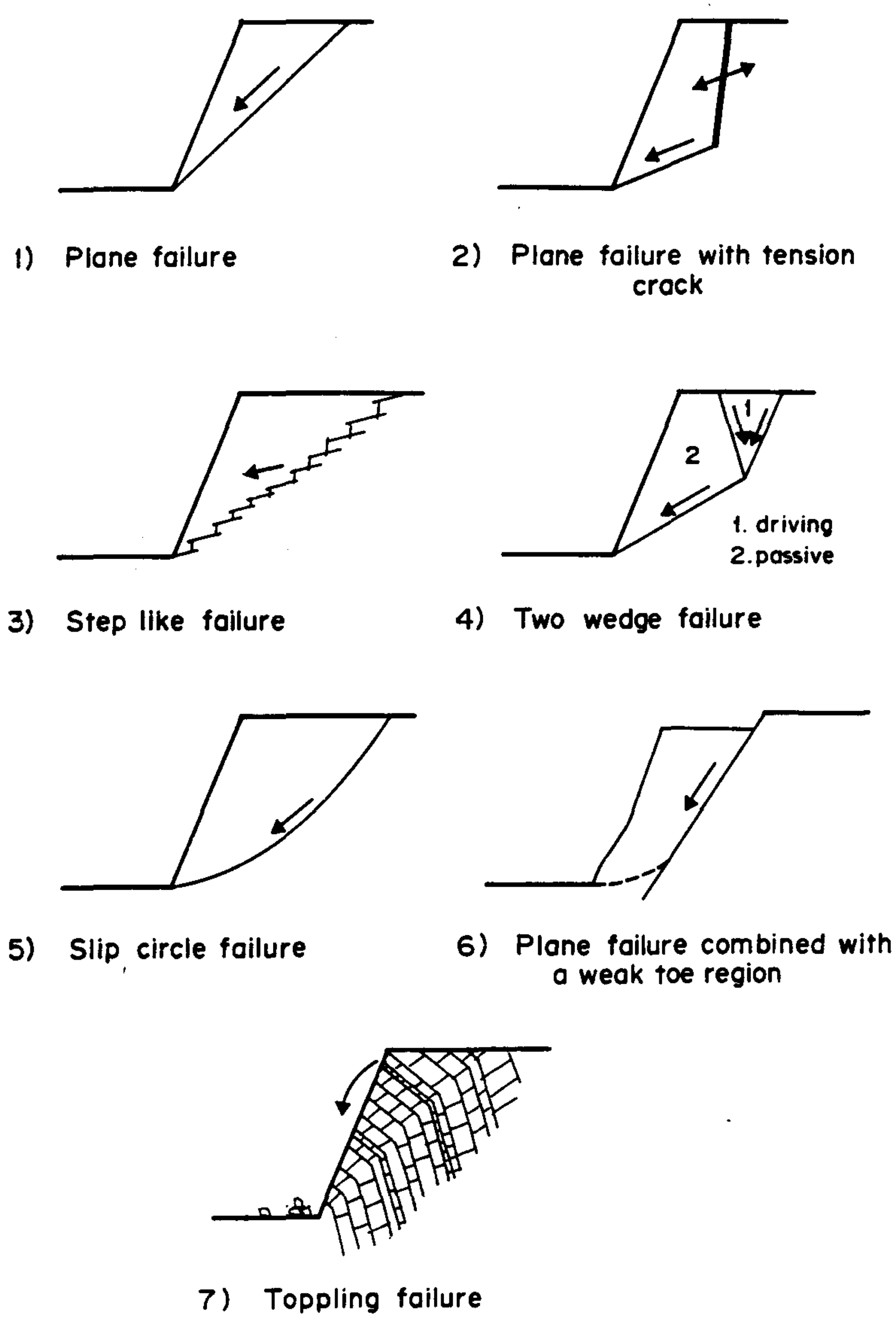


FIGURE 2.1.1. Modes of Rock Slope Failure (from McMahon and LeRoy, 1977).

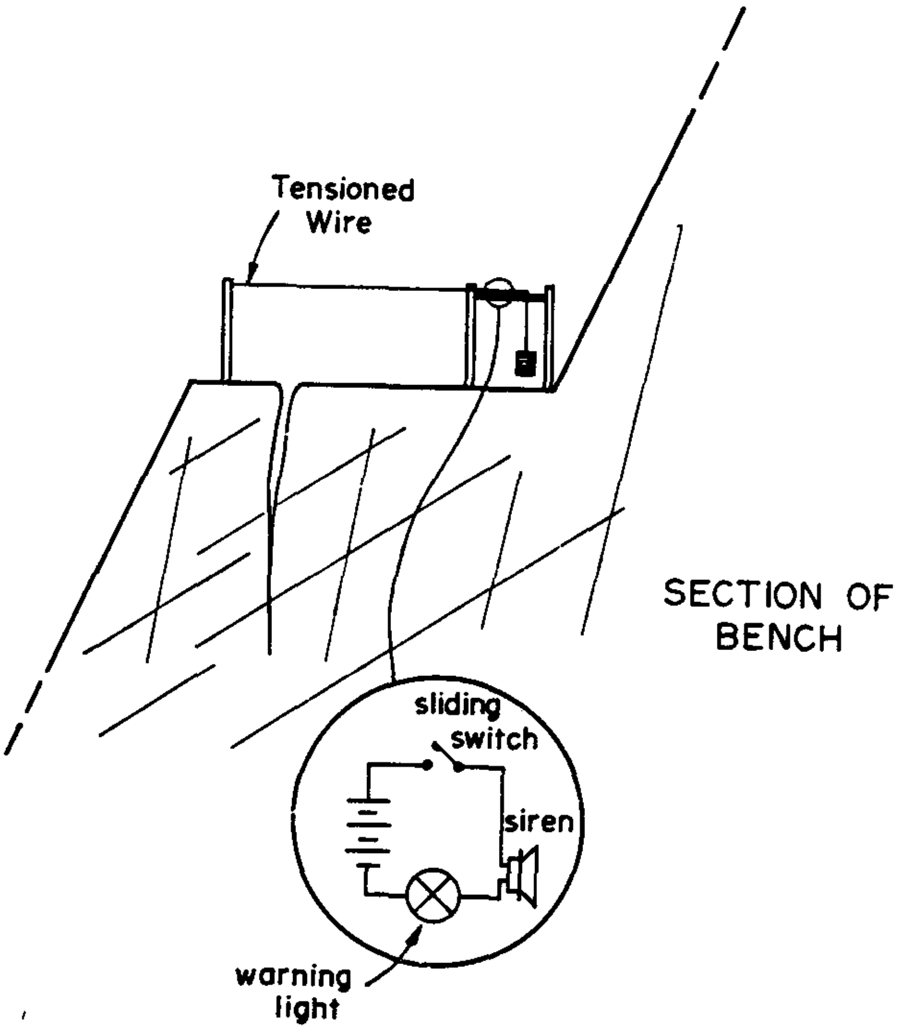


FIGURE 2.1.2. Simple Monitoring System to Warn of Bench Instability.

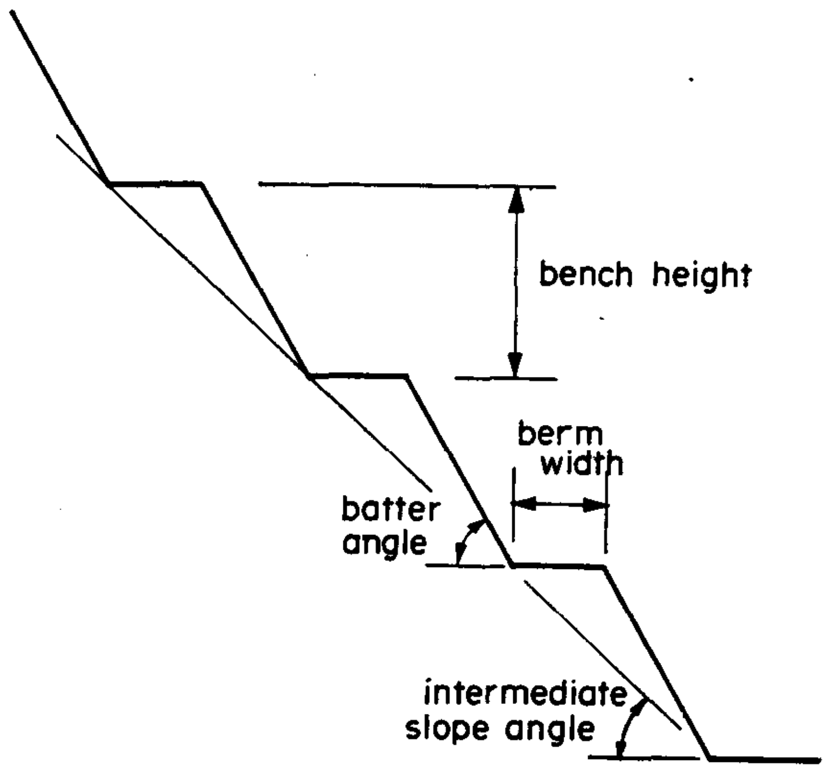
TABLE 2.1.1

HIERARCHY OF ACTIVITY IN SLOPE STABILIZATION
(adapted from Piteau *et al.* (1979))

Monitoring Methods	Protection Methods	Stabilization Methods
extensometers	safety beams, ditches	reduce slope angle
inclinometers	wire mesh covering	reduce slope height
displacement monitor-	catch nets, fences	drainage
ing of tension cracks	and walls	rock bolts, dowels
electric wire		anchors
electric fence		blasting control
		shotcreting
		buttress and bulkheads

Successful use of monitoring to plot the course of slope failures have been reported by Kennedy and Niermeyer (1970) and Brawner and Stacey (1979). Protection is provided in open pit mines by the use of safety berms, as shown in Figure 2.1.3 which are designed to reduce the incidence of loosened rock reaching lower working areas. Other methods used in civil engineering applications involve safety fences, and meshing over the slope face. Barron *et al.* (1970) and Sage (1977) discuss the use of mesh and shotcrete to reduce the incidence of unravelling and weathering of pit slopes. Maintenance of berms is essential to ensure that they remain clean and free of debris in order that they function as designed. Loss of benches due to minor slides and unravelling will restrict access back to the berms, thus reducing their effectiveness in providing protection.

Stabilization of slopes involves positive action. This may be achieved by reducing the destabilizing forces and/or increasing the resistance of the rock in order for it to support itself. Common methods of stabilization involve reducing the slope angle, drainage, improved blasting procedures or rock bolting. Often measures adopted to improve stability may involve a combination of these methods. Table 2.1.2 from Fookes and Sweeney (1976) provides a guide to selecting the appropriate stabilization procedure for the problem at hand and could be adapted to a mining environment. In this thesis it will be shown that fully grouted dowels are as efficient as fully



Benchs are used to minimize rock falls reaching lower levels.

Berm width governed by

— equipment
— nature of rock
— size of potential slide

FIGURE 2.1.3. Safety Berms.

TABLE 2.1.2

ROCK SLOPE FAILURE TYPES AND SOME APPROPRIATE STABILISATION MEASURES.

(from Fookes and Sweeney (1976))

STABILISATION MEASURES

			EXCAVATION			STRUCTURAL SUPPORT									DRAINAGE				ROCKFILL CONTROL				
			FLATTEN SLOPE	BENCH	LOCAL EXCAVATION	GUNITE FACING	PERMEABLE (MASONRY) FACING	LOCAL STRUCTURAL DENTITION	BUTTRESS	ANCHORED WALL	STRAP	DOWEL	BOLT	ANCHOR	DRAINAGE DITCH	SCREEDED (PAVED) SURFACE	SHORT DRAINHOLES	LONG DRAINHOLES/ADITS	MOVE STRUCTURE/ HIGHWAY	ROCK TRAP DITCH	ROCK TRAP FENCE/WALL	NETTING	SCALING OF LOOSE BLOCKS
	PLANE FAILURE	LARGE	1	2	3	3	3	3	1	1	2	3	1	1	1	1	1	2	2	2	2	3	2
		SMALL	2	2	2	2	3	2	1	1	2	3	1	2	1	1	1	3	2	1	1	1	1
	WEDGE FAILURE	LARGE	1	3	3	3	3	3	1	1	3	2	2	1	1	1	3	1	2	3	3	3	3
		SMALL	1	2	2	3	3	2	1	1	1	1	1	3	1	1	2	2	2	2	2	2	2
	TOPPLING	LARGE	1	3	3	3	3	3	2	2	3	3	2	1	1	1	3	1	2	3	3	3	3
		SMALL	2	2	2	3	3	2	2	3	2	2	1	2	1	1	2	2	2	2	2	2	2
	ROCK OR DEBRIS FALL & GENERAL DEGRAD-ATION	LARGE	1	2	2	2	2	2	1	1	2	3	2	3	1	1	1	3	2	2	2	2	1
		SMALL	2	1	1	1	1	1	1	1	2	2	1	3	1	1	1	3	2	1	1	1	1

1. - Use will be beneficial.
2. - Possible use depending on situation.
3. - Use unlikely to be economic or effective.

grouted tensioned rock bolts, which based on work by Haas (1976), implies that they are more efficient than ungrouted tensioned rock bolts.

Typical measures taken in open pit mines to improve stability are summarized in such publications as the CANMET Pit Slope Manual (1977) and Hoek and Bray (1977). They include drainage control, improved blasting procedures, and rock bolting or cable anchors. Reports of the success of these measures, where they have been applied in open pit mining, are scarce however. Problems exist in defining the groundwater flow regime in particular in discontinuous rock slopes (Hoek and Bray (1977), CANMET (1977)). Improved blasting procedures are being used which take acount of fracture orientation and spacing (e.g. Hagan (1980)) however production requirements tend to dominate procedures rather than consideration for the consequences of over blasting with the attendent problems of overbreak and opening of tight joints behind the face. The influences of groundwater and blasting procedures have not been investigated in this thesis.

Barron *et al.* (1970) reported on trial stabilization procedures using rock bolting, mesh and shotcreting as means of surface stabilization while Seegmiller (1976) reported on a large stabilization program designed to halt a massive instability in a mine in New Mexico, U.S.A. These have been based on the use of active rock support using tensioned rock bolts or cable anchors, which have proved to be very expensive. The stabilization of rock slopes using rock bolts and cable anchors has been very successful in civil engineering applications (e.g. Rawlings (1968), Williams and Muir (1972), Littlejohn and Bruce (1975, 1976)). If a simple relatively inexpensive method of rock stabilization was available, then wider use of this technique to stabilize existing walls, and perhaps to increase the overall slope angle may be attractive to mine operators.

Some authors have suggested the use of passive reinforcement (also known as dowels or fully grouted untensioned rock bolts) as an alternative to the use of tensioned rock bolts, in rock slopes (Bjurström (1974), Hoek and Londe (1974), Londe and Bonazzi (1974), Ervin and Black (1976), Azuar (1977)). A French group (Azuar *et al.* (1979)) reported on the success of the use of passive reinforcement in numerous civil engineering applications. In addition, Hoek and Londe (1974) suggested that surface reinforcement may lead to increased slope stability, a method successfully employed by Chappell and Maurice (1981) using fully grouted tensioned rock bolts.

In mines where bench instability means the loss of primary protection as shown in Figure 2.1.4, the concept of a relatively inexpensive meaus of stabilization such as passive reinforcement could prolong the life of the mine

The next section reviews the current state of the art with respect to passive reinforcement.

2.2 "PASSIVE" REINFORCING

2.2.1 Introduction

Rock bolting can be divided into two broad categories - active and passive, which Lang (1972) compared to the behaviour of prestressed and reinforced concrete respectively. However, Hoek and Londe (1974) argue that this analogy is unsatisfactory for two reasons:

(a) The rock mass is a discontinuous medium with a mechanical behaviour drastically different from that of concrete.

(b) The steel cannot be installed in rock masses either at optimal location or at optimal time.

Lang (1961) developed the classical treatment of active reinforcement with particular reference to underground excavations. The method is attractive in an analysis because of a clear conception of the force vectors. Each bar or cable is equivalent to a given and well known applied load, and hence it is easy for it to be introduced into any mathematical or physical model. The applied load will reduce, by vectorial addition, the effect of applied loads which are detrimental to stability. By increasing the normal stress across the joint the bars will enhance the frictional characteristics of the joint. It is possible to produce sufficient stress to avoid joint opening (i.e. reduce tensile cracking). The loads that can be applied, however, are very small compared to the forces present in the rock mass, particularly in rock slopes. Problems have also been identified at anchor zones where stress concentrations can approach the crushing strength of the host rock. Tensile splitting of the rock can occur in the vicinity of the buried anchor, a problem which has been recognised (Cox (1974)) in underground mines. While failure by stress corrosion (Littlejohn and Bruce (1975, 1976)) is also a threat to the long term behaviour of anchors, it has not been is presented in Chapter 6 to take acount of the increase in joint strength cause by the fully grouted reinforcing. In Chapter 5, the effect of changing joint strength in a numerical experiment is also investigated.

An alternative approach to the design of fully grouted reinforcing was reported by Ervin and Black (1976), who investigated the stabilization of batters adjacent to major carriageways. The method of analysis was based on force equilibrium methods commonly used to analyse tensioned rock bolts but neglecting to tension the bolts on installation. Barton (personal communication (1979)) commented that this approach was also a popular practice in Norway.

The procedure however needs some logical basis before being adopted as a satisfactory design method.

For the purposes of this thesis the rock has been assumed to be elastic-brittle plastic, while in general the joint behaviour has been assumed to be elastic-plastic; it is from these assumptions that the subsequent theories have been developed. Raffoux and Dejean (1980), however, draw attention to the element of time in the effectiveness of support, and in particular the creep behaviour of the host rock. They emphasise the importance of monitoring to determine whether the support is achieving the desired aim.

2.3 CONCLUSIONS

Two actions appear to be in operation which dominate the behaviour of the bar and they appear to be related to the different placement angles of the bar with regard to the direction of shearing. They are:

1. Tensile behaviour of the bar when placed at acute angles to the direction of shearing.
2. Dowel or bearing capacity behaviour for bars placed at angles greater than or equal to 90^{o} to the direction of shearing.

Insufficient information has been published to account fully for the effects of rock strength and stiffness, grout strength and stiffness, bar diameter, hole diameter, angle of bar to direction of shearing or the effect of rough rock joints on the contribution that the bar makes to the shear strength of the joint.

Chapter 3 presents an experimental program which investigated all these parameters. In Chapter 4, a theory is developed to take account of all the parameters. In Chapter 5, the effect of surface reinforcing on the stability of a rock slope is investigated numerically while in Chapter 6 a design and evaluation of a field trial for stabilizing rock slopes using fully grouted bars is presented.

CHAPTER 3

LABORATORY MODELLING OF FULLY GROUTED UNTENSIONED ROCK BOLTS

3.1 INTRODUCTION

Many methods have been used to simulate rock discontinuity behaviour, in order to compare measured response with limit equilibrium equations (Patton (1966), Ladanyi and Archambault (1970), Barton (1971), Schneider (1976), Dight and Chiu (1981)). In general the important parameters required to describe rock discontinuity behaviour, as defined by ISRM (1978), have been identified as residual or base friction angle (ϕ_b), angle of dilation (i), uniaxial compression strength (q_u) and uniaxial tensile strength (T_s) of the rock, normal stress (σ_n) and type of test (either constant normal load test or constant stiffness test).

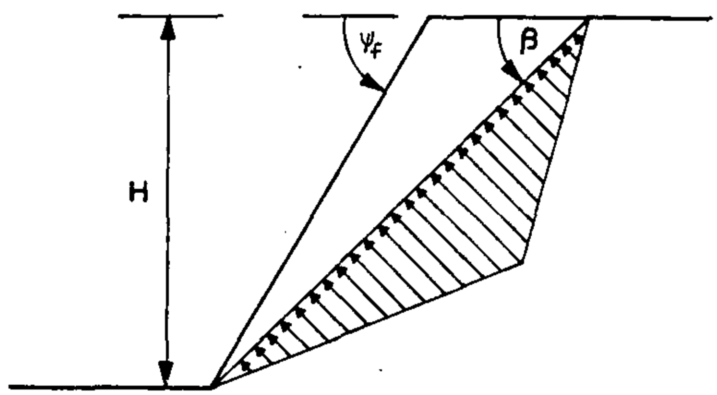
Variable dilation formulae (Ladanyi and Archambault (1970) and Barton (1973)) have gained acceptance over the less sophisticated formula suggested by Patton (1966). The use of a variable dilation formula becomes necessary when the normal stress is less than the transition pressure which is often taken as the uniaxial compression strength of the rock (e.g. Goodman (1976)). If the normal stress is very much less than the uniaxial compressive stress of the rock however, Patton's equation is still representative of the behaviour of the rock joint in shear. Barton (1973) showed that for rock slopes the normal stresses are relatively low, except in areas that may be subject to residual ground stress. Typical values for normal stresses to be found in rock slopes are given in Table 3.1.1.

Difficulties arise in determining *a priori*, the initial dilation angle, particularly when random discontinuity profiles exist. Work by Dight and Chiu (1981) has provided a simple model for the simulation of random profiles. The method has been used to determine the load-settlement curves for side-resistance-only socketed piles (Chiu and Dight (1982)). The techniques developed have been used to profile rough rock joints (Section 3.6) and for a preliminary investigation into the behaviour of rock anchors. (Appendix B).

Before investigating the behaviour of untensioned fully grouted bars on random rock joints, it was decided to simulate dilation in the traditional manner using fixed angles and plaster as a modelling material. The relevant properties of the plaster model material have been summarized in Table 3.1.2, where they can be compared with some typical rock properties published by Kulhawy (1975).

TABLE 3.1.1

TYPICAL STRESSES IN A ROCK SLOPE. (after Barton (1973)).



$$\sigma_{max} = \gamma H(\cot\beta - \cot\psi_f)\sin\beta\cos\beta$$

let $\psi_f = 65^o$, $\beta = 40^o$, $\gamma = 27.0\ kN/m^3$, H varies.

H	σ_{max}
10m	96.4 kPa
20	192.7
30	289.1
40	385.5
50	481.8
100	963.7

TABLE 3.1.2

SOME TYPICAL ROCK PROPERTIES

MATERIAL	DENSITY ρ t/m^3	POROSITY n %	MODULUS OF ELASTICITY E(GPa)	POISSON'S RATIO ν	UNIAXIAL COMPRESSIVE STRENGTH q_u (MPa)	UNIAXIAL TENSILE STRENGTH $-T_s$ (MPa)	FRICTION ANGLE ϕ	E/q_u
BASALT*	2.8	5.0	61.0	0.16	127.0	19.0	35	480
CONCRETE	2.4	-	25.0	0.20	35.0	0.50	45	1000
GABBRO*	2.6	-	76.0	0.19	-	-	38	-
GRANITE*	2.63	1.6	52.0	0.16	161.0	9.0	47	323
MELBOURNE MUDSTONE	2.3	20.0	0.40	0.20	2.0	0.20	23 - 38	200
PLASTER	1.3	40.0	7.0	0.2	25.0	1.7	35	280
QUARTZITE*	2.6	1.9	42.0	0.16	292.0	11.0	48	144
SANDSTONE*	2.2	14.0	20.0	0.40	100.0	3.0	44	200
SCHIST*	2.7	2.0	8.0	0.04	44.0	-	24 - 42	182
SILTSTONE*	2.2	12.0	12.0	0.2	24.0	0.9	32	500

* from Kulhawy (1975)

The use of a constant dilation angle has meant that the simpler Patton model of

$$\tau = \sigma_n \tan(\phi_b + i) \qquad (3.1.1)$$

could be used to determine the frictional component of the shear strength-deformation curve.

The important parameters governing the behaviour of the bar during shearing have been assumed to be the diameter of the bar (D_b), the yield strength of the bar (σ_y), Young's modulus of the bar (E_b), uniaxial compression strength (q_{ug}) and uniaxial tensile strength (T_{sg}) of the grout, and modulus of the grout (E_g), the angle the bar makes with the direction of shearing (α_b), the diameter of the hole (D_h) (and hence thickness of grout), the length of embedment of the bar (ℓ_e) and the initial tension in the bar (T).

An initial series of tests was performed using plaster to simulate the rock. The grout used was an epoxy preparation and the reinforcing bar was aluminium. The tests investigated the influence of dilation angle and normal stress on the resultant contribution of the bar to the shear strength.

A second series of tests was perfotmed on natural rock joints (basalt) with epoxy grout and a reinforcing bar of aluminium. In this series the dilation angle and normal stress were varied. The differences between this series and the final series are the randomness of the joint profile and the stiffness and strength of the rock mass. From the two series of tests the effect of dilation, normal stress and rock strength could be readily identified. In both these series the bar was installed at 90° to the direction of shearing.

A third series of experiments using a steel block to simulate the rock investigated the influence of the angle of the dowel to the direction of shearing (α_b) and also investigated the influence of the hole diameter (D_h).

Strain gauging of bars allowed a quantitative assessment of the axial and flexural behaviour of the bars during shearing. The results support the theoretical development in Chapter 4. The three series of tests described above, all used a large direct shear machine described by Williams (1980) and modified to handle constant normal load tests.

The effects of bar strength (σ_y) and length of embedment (ℓ_e) were investigated in a fourth series of tests using a simple shear machine (Guillotine). This series of tests involved deformed bar and prestressing strand bonded in a cement grout.

Strength tests were performed on plaster, basalt, epoxy grout, cement grout, aluminium bar, deformed bar, and prestressing strand. Pull out tests were performed to investigate the depth of embedment required in the plaster samples.

3.2 DIRECT SHEAR MACHINE

3.2.1 Introduction

The direct shear machine (DSM) used for the tests was originally designed by Williams (1980) as a constant normal stiffness machine (CNS) and subsequently modified for this work to handle a constant normal load.

The principles of the machine are shown in Figure 3.2.1.

(a) the top shear box is held in a fixed position such that it cannot displace or rotate.
(b) the bottom box is subjected to a shearing force and it is free to undergo shear displacement and normal displacement.
(c) the normal force on the sample is obtained through a series of levers as shown in Figure 3.2.1.

The test program kept the range of normal stresses between 100 kPa and 1 MPa on a shear sample area of approximately 150 mm by 200 mm.

3.2.2 Specification

In general the DSM performed according to the specifications determined by Williams (1980) for the CNS machine. These were:

(a) a maximum sample size of 150 mm wide x 200 mm long and 75 mm deep.
(b) a maximum allowable dilation of 10 mm.
(c) a maximum shear displacement of 50 mm.
(d) the maximum capacity of the machine was designed to be 180 kN normal force and 150 kN shear force. For the tests described in this thesis, however, the maximum normal force required was 28 kN and the maximum shear force 60 kN.
(e) by coupling the loading of the direct shear machine to a triaxial compression loading frame using hydraulic jacks, shear displacement rates could be varied from 0.001 mm/min to 1 mm/min.

3.10 SUMMARY AND CONCLUSIONS

The test results presented indicate that both axial and flexural action contribute to the behaviour of the bar. It appears that for bars installed at acute angles to the direction of shearing or where dilation of the joint occurs, the tensile behaviour may be more important, while for bars installed at angles greater than 90^{o} to the direction of shearing, the flexural or dowel behaviour is more important. Yield is generated in the bar at very small displacements, typically less than 1.5mm.

The tests have investigated the effect of rock strength on the behaviour of the bar. In the softer material, all the deformation has taken place in the "rock", while for the harder rocks "failure" of the bar is initiated by the guillotining effect of the borehole on the bar. The effects of bar diameter, hole diameter bar type and strength have been demonstrated.

The effect of normal stress has been shown to be insignificant in the range tested. This is supported by the work of Azuar (1977). Profiling of the rough joint surface has demonstrated that the approach developed by Dight and Chiu (1981) is relatively accurate in predicting the joint dilation. Statistical procedures, however, suffer from the averaging process which can mean missing important changes in the profile.

The effect of embedment length on the behaviour of dowels was investigated. A method to predict embedment length and anchor behaviour is developed in Appendix B.

Farmer's (1975) theory on the distribution of shear stress along a bar has been shown to be in excellent agreement where the assumed boundary conditions are met.

The direct shear machine proved to be satisfactory, although modifications have been suggested to improve its performance.

The Guillotine rig proved to be excellent. It was extremely easy to use, and allowed a great number of variables to be investigated. Modifications to the rig have been suggested to increase its applications. The use of strand as untensioned reinforcing was shown to be very satisfactory.

CHAPTER 4

THEORETICAL BEHAVIOUR OF FULLY GROUTED ROCK BOLTS

4.1 INTRODUCTION

The behaviour of fully grouted rock bolts has been the subject of some intense investigation, particularly in the underground coal mining industry. (e.g. Raffoux (1971), Fairhurst and Singh (1974), Haas (1976, 1981), Gerdeen *et al.* (1977), Kwitowski *et al.* (1980), Radcliffe *et al.* (1980). The work has mainly concentrated on the behaviour of the reinforcing in tension, although lately some attention has been given to the behaviour of reinforcing in shear.

Further theoretical and experimental evidence has been provided by Heuzé and Goodman (1973), Bjurström (1974), Azuar (1977), and Fuller and Cox (1978). Heuzé and Goodman (1973) equate the increase in joint strength to the yield strength of the bar in shear plus a component related to the compressive stress of the rock as a result of confinement. Their approach however, as discussed in Chapter 2, does not satisfy equilibrium.

Bjurström (1974) recognised that two mechanisms were operating - dowel behaviour and tension behaviour. He appears to have used a plastic design approach (ASCE (1961)) to develop a theory to account for the dowel action, assuming that the tension effects were very small. For a bar installed at an acute angle to the direction of shearing, he assumed that the tensile behaviour dominated. Haas (1976) considered the geometric effects of shear displacement on mechanically anchored bars, while Fuller and Cox (1978) extended this approach to fully grouted bars. However in order to evaluate the theory developed by Fuller and Cox information on the location of the plastic hinge is necessary. They have suggested that this can be determined from pullout tests, and thus equate the location of the plastic hinge to the length of debonding. However this is considered doubtful in view of the fact that a plastic hinge occurs as a result of flexural action of the bar. Azuar (1977) recognised that for bars installed perpendicular to the directions of shearing frictional effects were negligible. This is not consistent with the tensile theories which attribute part of the strength increase to a frictional component.

An analogous problem to that being considered here has been studied in concrete where investigations into dowel behaviour have been going on since the thirties (Friberg (1940)) with more recent work by Hofbeck *et al.* (1969), Johnston and Zia (1971) and Dulacska (1972). The theoretical approach in the work on concrete has largely been confined to the behaviour of a beam on an elastic foundation (Hetenyi (1946)) although Dulacska (*loc. cit.*) made use of the plastic design approach (ASCE, Commentary on Plastic Design in Steel (1961)).

Some authors are critical of the use of passive reinforcement (Lang (1972), Sage (1977), and Hoek and Brown (1981)) because of difficulties in design and problems with installation. Evidence presented in this chapter should overcome these criticisms.

4.2 EXPECTED LOAD AT FAILURE

A bar can fail in shear, tension or a combination of shear and tension. In order to demonstrate this phenomenon, Chesson *et al.* (1965) and Fisher and Struik (1974) have conducted experiments on bolted connections orientated at different angles to the direction of shearing, as shown in Figure 4.1. They developed an equation to predict the load at failure which takes account of the contributions of shear and tension with respect to the angle of the bolt. This equation takes the form of an ellipse

$$\left(\frac{S}{T_y}\right)^2 + \left(\frac{T}{T_y}\right)^2 = 1.0 \qquad (4.1)$$

where S = shear force in the bolt

T = tensile force in the bolt

T_y = yield strength of the bolt

It is common in structural engineering to assume that steel behaves as a Tresca material. Hence if T = 0, then S = 0.5 T_y. An interaction diagram based on this equation is shown in Figure 4.2. This diagram has been reinterpreted by plotting $(\frac{P}{T_y})$ versus the angle of the bar (θ_δ) in Figure 4.3, where P is the load measured during the test. On the same figure are plotted curves of the shear and tensile components of equation (4.1). Also on the same figure is a curve showing the load recorded for a bar in simple shear. The purpose of presenting this figure is to show that the angle of the bar at the point of failure has a significant effect on the load at failure. In particular if the bar is behaving in pure shear,

4.7 SUMMARY AND CONCLUSIONS

A theory has been developed to account for the dowel behaviour and tensile behaviour of a fully grouted rock bolt subjected to shear. Account has been taken of the geometric properties of the rock, and bar, and mechanical properties of the rock, grout and bar.

The theory has been evaluated by comparing predictions of load-deformation behaviour of fully grouted reinforcing bars in shear with experimental results. In general, the theory is a good predictor of the measured behaviour, while in specific cases explanations have been offered to account for apparent discrepencies. Given that the theory involves some crude approximations to the problem of a bar crushing into the rock or grout the theoretical description of the behaviour of fully grout bars in shear is sufficiently close to be within a correct order of magnitude.

A sensitivity analysis has been performed to demonstrate the relative effect of the input parameters. In particular, the effect of pretensioning does not significantly change the effect of the bar in shear, which is its primary mode of behaviour.

The development of the theory and its evaluation represents a significant step forward in the design and use of fully grouted rock bolts in soft and hard rocks.

CHAPTER 5

A NUMERICAL INVESTIGATION INTO THE BEHAVIOUR OF REINFORCED SLOPES IN JOINTED ROCK

5.1 INTRODUCTION

This chapter presents the results of a series of numerical experiments on an idealized blocky mass, using a numerical technique developed by Burman (1974). The method has been derived from the finite element method (FEM) by concentrating on the joint (or discontinuous) behaviour and suppressing the intact rock mass behaviour resulting in computational efficiencies and achieving an improved method to simulate complex jointed systems.

Cundall (1971) has developed a similar approach to the analysis of discontinua, by concentrating on the behaviour of the rock joints. However Cundall considers the kinematic behaviour of the rock blocks, and then calculates the force-displacement relationships using a dynamic relaxation procedure. Cundall's approach is useful for modelling post failure behaviour.

Both methods highlight the importance of block rotation. Block rotation has also been recognised by Chappell (1972), where the load redistribution caused by block rotation may lead to the possible development of a collapse mechanism. The rotation causes non uniformity of normal stress along joint planes and plays an important role in the development of "tension" cracks.

The purpose of using this program has been to investigate, numerically, the influence of surface reinforcement on the behaviour of an excavated slope. The slope is a simple imbricated model as used by Krsmanovic and Milic (1964), Brown and Trollope (1970), John (1970), Walker (1972) and Burman *et al.* (1975), as shown in Figure 5.1.1.

Surface reinforcing has been discussed by Hoek and Londe (1974) with particular reference to the use of shot-crete or gunite reinforced with steel mesh. This method has been successfully applied in open pit mines (Barron *et al.* (1970)) however no satisfactory design method has been developed. Reed (1974) considered the use of short lengths of steel embedded between long lengths, as shown in Figure 5.1.2, to improve stability in underground openings. In surface exposures this could also be considered as a precursor to the use of shotcrete.

Burman, Trollope and Philp (1975) used the program to investigate the behaviour of an excavated slope with both plastic and brittle joint properties. From their study, it was readily apparent that if the rock slope was not predisposed to failing along a single defined plane of weakness as shown in Figure 5.1.3a, then a sequential failure mode could be discerned as shown in Figure 5.1.3b thus casting doubt on the limit equilibrium approach. However their conclusion is based on the assumption that peak strength values are used in design calculations rather than residual strength values. In the author's experience, residual values tend to be used however.

The problems considered by Burman *et al.* (1975) could be broadly classified into 2 categories. The first category involved an "intact" block sliding on a well defined weakness plane. The second category used the same block geometry, but with uniform joint properties. These two categories of problems have been investigated in this study as they form a datum for considering variations induced in the slope behaviour when passive reinforcement is introduced.

The next section briefly summarizes the numerical approach. Section 5.3 briefly outlines the previous approaches to the modelling of rock bolting. Section 5.4 provides background to the problems to be analysed.

5.2 METHOD OF ANALYSIS

Burman (1974) described a numerical approach designed to concentrate on the behaviour of discontinuities between blocks. To achieve this he formulated the normal finite element equations ignoring the block deformations and considering the joint elements as lineal rectangular units. A typical element is shown in Figure 5.2.1. This element models the interaction of the adjacent rigid blocks with centroidal nodal positions 1 and 2. The length, thickness and orientation of the element are defined relative to node 1 in a local coordinate system. Forces and displacements are associated in the global coordinate system with the nodal points, while stresses and strains are related to the centre of the joint in local coordinates. The block has three degrees of freedom as shown in Figure 5.2.2.

5.7 EXCAVATION OF 65° SLOPE

Most slopes are not excavated with a vertical face. In this section a 65° slope is considered using the same mesh as in the previous sections, but "excavating" the face back to 65°. Figure 5.7.1 shows a full slope excavation in blocky material where the joint cohesion is 0 and joint friction angle is 30°. The figure shows a significant amount of instability at the crest with well developed tension cracks and inter block sliding. This observation is quite consistent with field experience.

Face reinforcement as shown in Figure 5.7.2 has increased the stability of the crest but has had very little effect on the balance of the slope. Just the effect of cutting the slope back to 65° has had a significant effect on the stability and reduced the likelihood of toe instability. This can be observed by comparing Figure 5.6.2 and Figure 5.7.1.

5.8 CONCLUSIONS

The numerical method has highlighted some of the effects of surface reinforcement and the relative influence on the slope behaviour of a change in the joint properties.

Face stabilization appears to be satisfactory where uniform joint properties have been assumed. The use of stabilizing key blocks may change the mode of behaviour.

"Tension" cracks appear to exist due to block rotations, and while in themselves may not suggest failure, they are susceptible to filling with surface water and significantly increasing the down dip forces. As many of the joints have been significantly mobilized, this would induce failure.

The analysis could be extended to consider heterogeneous joint properties and different geometries.

Attention to improving the program efficiency would also be welcome - and in particular the handling of non linear strength criteria such as Ladanyi and Archamault's (1970) variable dilation shear strength relationship.

CHAPTER 6

DESIGN OF A PASSIVELY REINFORCED SLOPE

6.1 INTRODUCTION

The proposal to use passive reinforcement was originally made to the mine by Prof. L. Endersbee in a letter to the mine in September, 1974. However the necessary theory to analyse the behaviour of passively reinforced rock joints has only recently been completed. Passive reinforcement at Savage River Mines was commenced in August, 1981, under the supervision of the Slope Stability Engineer, Ross Hastings.

This chapter presents an analysis of the stability of a bench in the mine using probability methods. The stability curve is compared to the experience curve data collected by the Senior Mine Geologist. The analysis then determines the required reinforcement to alter the expected behaviour of the pit slopes.

A preliminary study of the available data is made to compare the predicted behaviour of the passively reinforced slopes with the field performance.

6.10 CONCLUSIONS

This chapter has shown that geometry (i.e. orientation, spacing and length of joint sets, and orientations and dip of the excavated face) dominates the subsequent behaviour of the excavated slope. Hence a detailed geological survey is necessary for a complete design to be undertaken. The results of the survey are also essential to identify possible failure mechanisms and assess alternative designs.

Using a design philosophy based on limiting movements, fully grouted rock bolts act as shear keys and hence alter the effective cohesion of the unfavourable joint sets. The effect of altering the cohesion has quite a dramatic effect on the calculated probability of failure.

The preplacement of the reinforcing prior to excavating will ensure that the natural interlock within the rock mass will most effectively be retained. The design must be put into an economic context to assess the consequences of a decision on the overall mine plan.

Field evaluation of the design is an essential feature of assessing the success or failure or the approach and to identify the reasons for the performance.

CHAPTER 7

CONCLUSIONS

7.1 GENERAL

The main aims of this project have been to provide an understanding of the behaviour of fully grouted untensioned reinforcing and to determine whether the use of this type of reinforcing would improve the stability of rock slopes in open pit mines.

In order to satisfy these aims several investigations were undertaken. These included

- laborating testing of "passively" reinforced rock joints,
- theoretical analyses of the axial and flexural behaviour of fully grouted reinforcing,
- numerical experiments on unstabilized and stabilized rock slopes,
- development of a design procedure for "passively" reinforced rock slopes,
- evaluation of "passively" reinforced rock slopes in a mining environment.

The following sections summarize the major conclusions reached in this project.

7.2 BEHAVIOUR OF FULLY GROUTED UNTENSIONED ROCK BOLTS

Laboratory investigations identified that the behaviour of "passive" reinforcing was related primarily to the initial installed angle of the bar to the direction of shearing, and to the roughness of the rock joint. Other factors which also controlled the response of the bar and affected its contribution to the shear strength of the joint were

- diameter and stiffness of bar,
- diameter of hole,
- strength, stiffness and friction angle of grout,
- strength, stiffness and friction angle of rock.

Two mechanisms were identified as governing the behaviour of the bar. For bars installed at angles $< 90^{\circ}$ to the direction of shearing on smooth joints, the axial or tensile behaviour of the bar dominated its contribution to the shear strength of the joint. For bars installed at angles $\geq 90^{\circ}$

to the direction of shearing on smooth joints, the flexural or dowel action dominated the contribution of the bar to the shear strength of the joint. For rough rock joints, the angle of dilation, i which occurred as a result of shearing meant that the behaviour of bars installed at angles $< 90^\circ + i$ would be dominated by the axial or tensile action, while the behaviour of bars installed at angles $\geq 90^\circ + i$ to the direction of shearing would be governed by the flexural or dowel action of the bar. However it was apparent from the strain gauge results that both mechanisms interacted. The flexural behaviour of the bar, which was a response to the external loads, governed the incremental angle of the bar to the direction of shearing. This in turn governed the component of the axial load in the bar, in the direction of shearing. The axial load in the bar at the plastic hinge governed the flexural behaviour of the bar.

A theory was developed which was able to predict the experimental results obtained in this project, as well as interpret results for data published in the literature. The degree of correlation was very good. Sensitivity analyses identified that the effect of pretensioning the reinforcing had very little effect on the contribution of the bar to the shear strength of rock joint.

The experiments and theory confirmed that the behaviour of fully grouted rock bolts could be predicted in soft and hard rocks.

The experimental work also demonstrated that prestressing strand was satisfactory as "passive" reinforcing.

7.3 SURFACE STABILIZATION

The effect of surface stabilization was investigated in a series of numerical experiments. The results showed that for a uniform rock mass with the front face stabilized by altering the joint cohesion, significant improvements to the stability could be achieved.

7.4 DESIGN PROCEDURE

The design of a "passively" reinforced rock slope was developed based on limit equilibrium techniques and Monte Carlo simulations. Initial calculations on unreinforced slopes were verified using an "experience" curve obtained from Savage River Mines. The technique showed that a detailed knowledge of the geometry (spacing, length and orientation of joint sets, orientation and dips of slopes) was critical to a successful appraisal, and that geometry had a first order effect on the calculation of the probability of failure. Introduction of a method of design based on limiting joint movements provided the basis for determining the density of reinforcing required to improve the stability of the rock slope. The effect of the "passive" reinforcing was introduced into the calculations on the basis that the bars acted as shear keys, and hence altered the joint cohesion. Based on the altered probability of failure of the slope economic analyses were undertaken to identify the major cost components and savings of the reinforcing system. The major cost was the drill hole for the reinforcing while the major potential saving was the recovery of the ore at depth.

7.5 FIELD EVALUATION

Installation of the fully grouted bars was undertaken to assess the success of the method in the field. While the effectiveness of the method has still not been quantified several points can be made.

1) Benches have been retained in areas of the mine where this had not been possible on the previous mining pass, reducing the maintenance problems associated with clearing falling debris and improving the prospect of recovery of ore at depth.
2) The major cost of installation was the drill hole for the reinforcing.
3) Preplacement of the bars is essential to maximize the effectiveness of the system.
4) Strand is satisfactory as untensioned reinforcing, although consideration should be given to the use of anchor plates to reduce the incidence of unravelling adjacent to the collars of the bars.

7.6 AREAS OF FURTHER RESEARCH

In order to satisfy the requirements of this project a wide range of topics has been covered. Some of these have only been briefly investigated and require further work, while others were ignored. Further research in the following areas would be most useful

a) The pullout behaviour of rock anchors.
b) Improved efficiency of the block model program and extension to handle more complex problems.
c) Adaptation of the guillotine to handle bars installed at varying

angles to the direction of shearing.

d) A study of the behaviour of unreinforced and reinforced clay filled rock joints.