3 Identification and classification of strata conditions and associated suitable temporary and face area support systems

3.1 Introduction

Enabling output 2 requires the identification and classification of strata conditions and determination of the most suitable face area support systems for particular strata conditions.

The term “strata condition” is a poorly defined term used loosely in the mining industry and which, consequently, has various connotations. It is typically taken as a descriptive measure of the severity of mining induced fracturing and geological discontinuities in strata surrounding an underground excavation and the potential for rockfalls.

The determination of strata conditions usually involves a consideration of geology, mechanical properties of the rock mass and empirical observations of the rock mass behaviour. Quantitative rock mass classification systems have also been used to characterise the rock mass. Güler *et al.* (1998) discussed the use of rock mass classification systems in defining geotechnical areas for South African gold and platinum mines. Their major finding was that no single rock mass classification system is suitable due to the many and varied parameters required. Furthermore, certain parameters are applicable only to particular geotechnical areas, therefore a logical assessment of the rock mass was considered essential.

3.1.1 Definition of the face area

The working area is the area extending from the face to the sweeping line (Figure 3.1.1). This working area is divided into the face area and the permanent area. The face area is defined as the area between the stope face and up to 4,0 m away from the face (Jager, 1999). The permanent area extends to the sweeping line. Permanent support types are sometimes installed in the face area, where they serve initially as temporary support. Only the conditions occurring within the face area are considered. Variations in strata conditions with distance from face or time are, therefore, not considered, as these will not impact on face area support requirements.

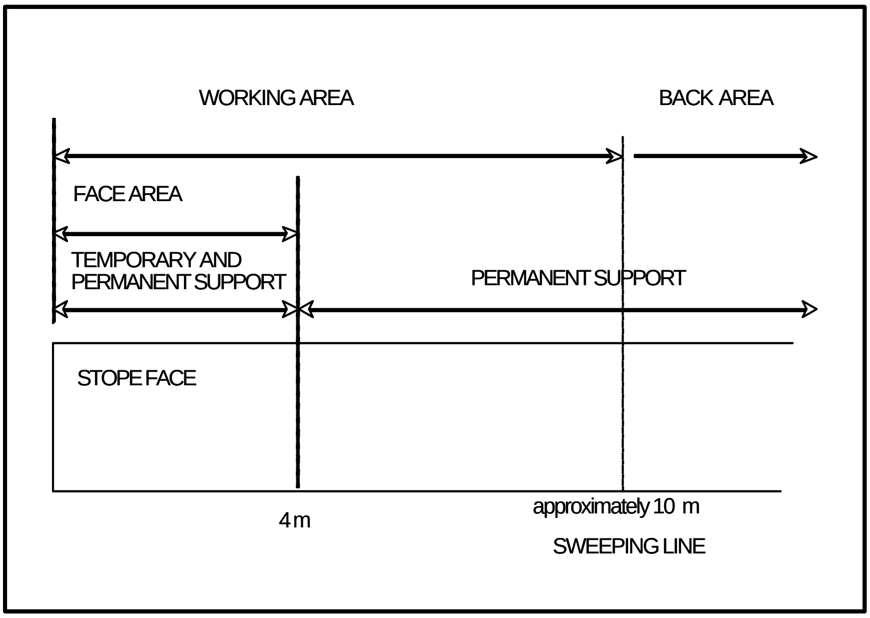


Figure 3.1.1 Diagram illustrating the face area.

3.1.2 Methodology

Due to the loose definition of the term “strata condition”, it is deemed necessary to define the term for the purpose of this project. Strata conditions refer to the hangingwall rock mass condition (manifested by the degree of discontinuity) and its potential stability/instability when considering the impact on choice of temporary support.

The hangingwall above the face area may be considered as a beam of variable dimensions dependent on its thickness. This beam has an inherent strength dependent on the rock type and composition. The beam will be intersected by various primary and secondary geological features, and mining induced fractures in deeper mines, resulting in a discontinuous rock mass.

Primary geological features are original features resulting from deposition or emplacement of rocks and include igneous layering, lithological variations and the various types of bedding. Secondary geological features reflect subsequent deformation or metamorphism and include faults, joints, intrusives and parting potential of the structures.

Features affecting the degree of discontinuity of the hangingwall beam are classed as reef parallel or reef perpendicular. Reef parallel structures are those oriented parallel or sub-parallel to the reef plane. Examples include igneous layering, various types of bedding, bedding parallel joints and fractures and bedding parallel faults. Reef perpendicular structures are those oriented perpendicular or sub-perpendicular to the reef plane. Examples include mining induced fractures, joints, micro-faults and strike-slip faults. Mining induced fracturing will be discussed and treated separately from other reef perpendicular structures.

Each of the above parameters will be classed as problematic or non-problematic. “Problematic” refers to particular parameters influencing strata conditions adversely to an extent where they have to be considered when choosing support types. “Non-problematic” refers to those parameters that have a minor influence on strata conditions and consequently different support types may be used.

A spreadsheet outlining the various classes of strata conditions is produced together with the most suitable support types for each class.

3.2 Mining environments/conditions

3.2.1 Introduction

South African gold and platinum deposits are either stratiform or layered and are represented by igneous stratification or sedimentary layering. The deposits are tabular in nature and extend from surface to over 5000 m below surface. Mining typically occurs over a wide depth range of over 3000 m.

COMRO (1988) identified three mining environments, viz. shallow, intermediate and deep based on various criteria. Mining conditions and hence strata conditions will vary according to the type of environment encountered. In this project intermediate and deep level environments will be considered together, as their characteristics are broadly similar. Boundaries are approximate and a degree of overlap may occur, as variations due to local conditions are possible. The typical conditions for the various mining environments are listed in Table 3.2.1.

Table 3.2.1 Mining environments and associated conditions.

	SHALLOW	INTERMEDIATE	DEEP
TYPICAL DEPTH (m)	< 1000	1000 - 2250	> 2250
STRESS FRACTURING	minimal or none	moderate	intense
STOPE CLOSURE RATE (mm/m face advance)	low < 10	moderate 10 to 30	high > 30
ERR (average regional) (MJ/m ^{2})	< 8	8-40	> 40*
REGIONAL SUPPORT	essential	optional	essential
ROCKBURST HAZARD	minimal	moderate to severe	severe
INFLUENCE OF GEOLOGY ON HANGINGWALL STABILITY	strong	moderate	moderate
VERTICAL VIRGIN STRESS LEVELS (MPa)	low < 25	moderate 25-60	high > 60

* if regional support not used

Figure 3.2.1 illustrates the process to follow for determining mining environments and conditions.

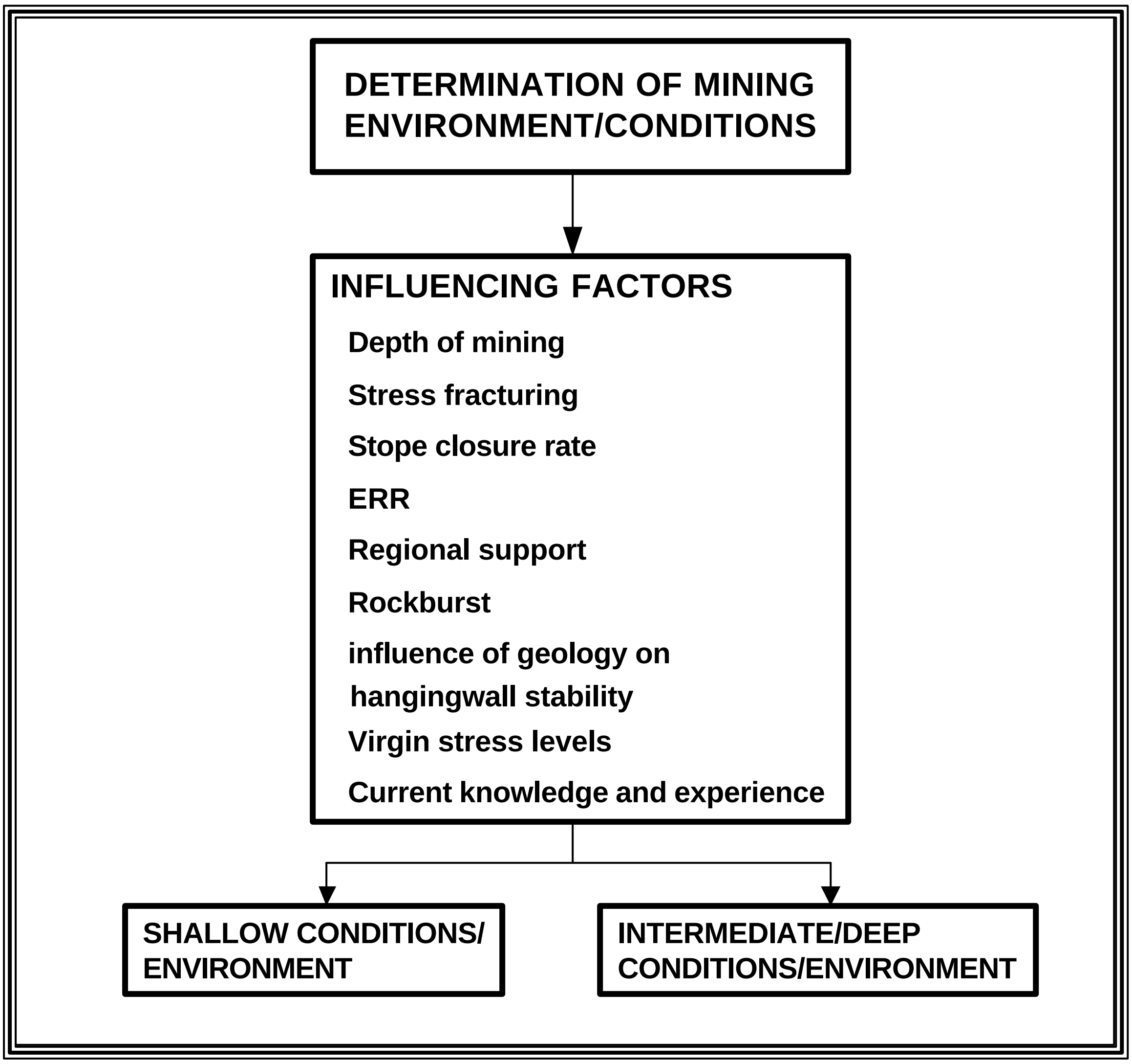


Figure 3.2.1 Flowchart determining mining environments and conditions.

3.2.2 Shallow mining environments

Shallow mining environments are characterized by low virgin stresses, low closure rates, low ERR, minimal or no stress fracturing, low clamping stresses and a minimal rockburst hazard. Choquet and Hadjigeorgiou (1993), Bieniawski (1984), and COMRO (1988), all mention the strong influence of geology on hangingwall stability.

Economic deposits that fall into this category may include the Merensky Reef and UG2 of the Bushveld Igneous Complex, the Kimberley Reef in the Evander goldfield, the VCR in the Klerksdorp goldfield, the South and Composite Reefs of the Central Rand goldfield, and the UE1A reef at Randfontein Estates gold mine.

3.2.3 Intermediate and deep mining environments

Intermediate and deep mining environments are characterized by moderate to high virgin stresses, moderate to intense fracturing, higher closure rates, moderate influences of geology on the hangingwall, higher ERR and a moderate to severe rockburst hazard.

Economic deposits in this category may include portions of the Merensky and UG2 Reefs, and most gold-bearing reefs of the Witwatersrand.

3.3 Parameters influencing strata conditions in shallow environments

3.3.1 Introduction

The classification of strata conditions needs to account for both gold and platinum deposits. General parameters influencing strata conditions will be discussed with a consideration of current strata control problems.

Important parameters affecting strata conditions include:

- Rock type and strength.
- Primary and secondary geological structures and their characteristics.

Other important parameters affecting strata conditions, but considered to have only a localised effect and therefore atypical, include:

- Major faults.
- Intrusives, including dykes and sills.
- Reef geometry.
- Groundwater.
- Abnormal stress conditions including high horizontal stresses.

These latter factors will not be discussed further.

3.3.2 Rock type and UCS

Hangingwall rock types encountered in current gold and platinum mines include igneous rocks and low grade metamorphosed sedimentary rocks. There are consequently a variety of rock types exhibiting a range of rock mass behaviours. It is proposed to differentiate between weak and strong rock types for this study. This differentiation would help to identify areas prone to hangingwall punching by support units, unravelling around support units and areas requiring some form of areal support or headboards on support units.

Variations in UCS may be significant for a single rock type due to compositional variation and the informal method of distinguishing rock types. Stavrapoulou (1982) found that the inhomogeneous nature of Witwatersrand quartzite resulted in different mechanical properties for what was considered to be the same rock type. The strength of quartzite was found to increase with increasing quartz content. Changes in sedimentary rock types and their metamorphosed equivalents due to proximal-distal relationships and other sedimentary controls need to be identified. Juxtaposition of different rock types may result in 'weaker' behaviour as in quartzites intercalated with shale partings.

Roberts *et al.* (1996) identified large variations in UCS for what was considered to be the same rock type (lava). This resulted in the differentiation of "soft" from "hard" lava.

Güler *et al.* (1998) proposed a classification of UCS of intact rock as summarised in Table 3.3.1.

Table 3.3.1 Classification of UCS of intact rock (after Güler et al., 1998).

Category	UCS (MPa)	Description
1	Less than 120	Soft
2	Between 120 and 200	Medium
3	Above 200	Hard

Daehnke *et al.* (1998) considered 200 MPa as the cut-off between "soft" and "hard" lithologies.

Uniaxial compressive strength (UCS) will be used as a guide to differentiate weaker and stronger rock types. The 200 MPa cut-off may be used as a guideline to differentiate weak from strong. There may be overlaps between the two categories and current knowledge and experiences of rock engineering practitioners are required.

Weak rock types are those prone to hangingwall punching by support units and unravelling around support units. Figure 3.3.1 illustrates a support unit with headboard attached, punching into a weak, argillaceous quartzite hangingwall. Strong rock types, which are highly discontinuous and therefore prone to punching and unravelling, should also be regarded as weak.

The UCS is only one measure and is generally relevant in shallow mining environment, where stress fracturing is minimal. When dealing with rock dissected with stress fractures the UCS is of lesser importance. In a deep mining environment mining induced stress fractures decrease the rock mass strength significantly. Further work needs to be conducted to identify improved parameters describing the strength of a highly discontinuous rock mass.

Figure 3.3.1 Punching of support unit into weak, argillaceous hangingwall (face to the right).

The following rock types are tentatively considered as weak:

- Westonaria Formation lava
- Argillaceous quartzite
- Shale
- Argillaceous conglomerate.

Figure 3.3.2 illustrates the fragmented nature of a weak hangingwall, soft lava in this case. Unravelling has occurred up to 2 m above the reef contact.

Figure 3.3.2 Weak hangingwall – Westonaria Formation lava (face to the left).

The following rock types are considered strong:

- Alberton Formation lava
- Siliceous quartzite
- Siliceous conglomerate

Figure 3.3.3 illustrates the relatively unfragmented nature of a strong hangingwall (siliceous quartzite). Although fractured, only minor unravelling up to bedding planes is seen.

Figure 3.3.3 Strong hangingwall – siliceous quartzite (face to the right).

Complications may arise where variations in stratigraphy result in a weaker rock type overlying a stronger rock type or vice versa. Fallout thicknesses and ejection thicknesses may be used as a guide to determine the vertical stratigraphic extent that needs to be considered.

Daehnke *et al.* (1998) conducted a regional geotechnical area classification for various Witwatersrand and Bushveld reefs based mainly on footwall and hangingwall rock types. This information may be used as a guide when considering rock types and UCS.

3.3.3 Reef parallel structures

3.3.3.1 Introduction

The following structures will be discussed together with their parting potential:

- bedding, lithological contacts and igneous layering
- bedding parallel joints, bedding parallel faults

3.3.3.2 Bedding and lithological contacts

Bedding is produced by changes in sedimentation patterns and is identified by changes in:

- sediment grain size
- mineralogy and composition

Thickness of beds may be described as in Table 3.3.2 (modified after Tucker, 1990).

Table 3.3.2   Terminology for thickness of beds (modified after Tucker, 1990).

THICKNESS (cm)	TERMINOLOGY
> 100	VERY THICK
30 - 100	THICK
10 - 30	MEDIUM
1 - 10	THIN
0,3 - 1	THICKLY LAMINATED
< 0,3	THINLY LAMINATED

Lithological contacts and igneous layering define surfaces across which rock types change. These contacts may be sharp or gradational. Widely differing properties of adjacent rock types may lead to competency contrasts.

Categories of bedding with thicknesses of up to 30 cm result in a more discontinuous nature of the hangingwall. Bedding planes are often characterised by soft, micaceous infill material. Alteration often occurs along bedding planes introducing softer secondary minerals.

Figure 3.3.4 illustrates fallout defined by a ripple-marked bedding plane with a shale parting. Figure 3.3.5 also illustrates fallout along a shaly bedding plane. Earlier fallout along a thinner underlying bed has resulted in instability of the overlying thicker bed.

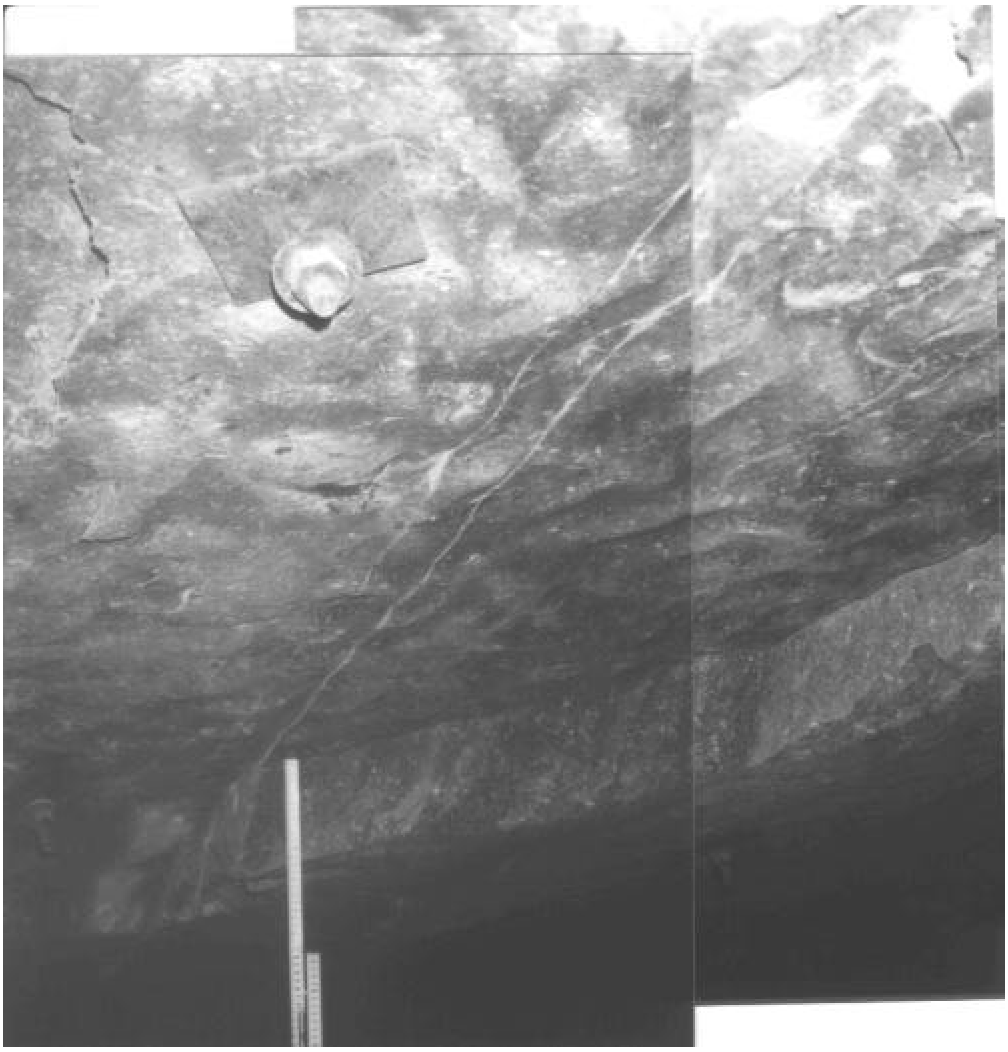


Figure 3.3.4 Ripple-marked bedding plane defining fallout (face to the right).

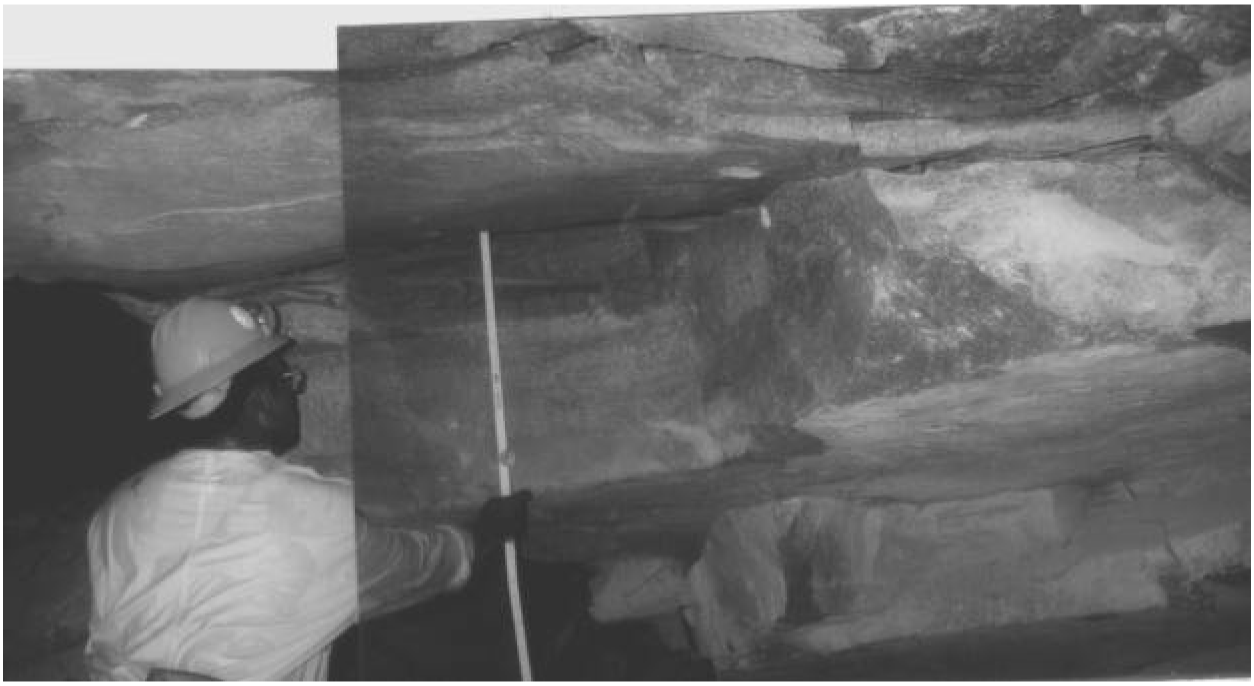


Figure 3.3.5 Fallout along a shaly bedding plane (face to the right).

The structures described above may form where:

- preferential alteration occurs,
- bedding parallel faulting and shearing occurs,
- competency contrasts result in differing conditions of rock types across the structure.

3.3.3.3 Bedding parallel faults and joints

Bedding parallel faults and joints usually occur along bedding planes or lithological contacts along zones of pre-existing weakness. Fracturing of a bedding parallel nature may occur in certain circumstances.

3.3.3.4 Parting potential

A parting plane is a generic term for all surfaces across which cohesive strengths are low. They typically coincide with bedding planes and lithological contacts. The ease with which parting may occur is termed parting potential (Turner, 1985). Weaker surfaces will have a moderate to good parting potential whereas welded stronger surfaces will have a poor parting potential.

The following surfaces on gold mines have moderate to good parting potentials:

- micaceous or argillaceous bedding planes
- lithological contacts between different rock types, e.g. quartzite and shale, quartzite and lava
- bedding parallel joints and faults that are not welded

Figure 3.3.5 illustrates a bedding plane with good parting potential. The shale parting varies in thickness from 1 to 10 cm and is very soft and friable due to bedding parallel movement.

Surfaces on platinum mines with good parting potentials include:

- lithological contacts, e.g. chromitite stringers and the surrounding pyroxenite in the UG2 hangingwall and occasionally Merensky pyroxenite and overlying noritic rocks in the Merensky hangingwall
- bedding parallel joints and faults that are not welded

The parting potential of partings may vary considerably laterally corresponding to variations in the underlying geological characteristics. The occurrence of these features closer to the hangingwall of the stope will be more problematic than features further away from the hangingwall.

3.3.4 Reef perpendicular structures

3.3.4.1 Introduction

The major structures of a reef perpendicular nature include joints and strike slip faults.

3.3.4.2 Joints

Joints are defined as fractures of geological origin along which no appreciable displacement has occurred. They typically occur as parallel or sub-parallel joint sets. The following characteristics of joints have the potential to influence the degree of discontinuity of the hangingwall:

- number of joint sets and their relative orientations
- spacing (density) of individual joint sets
- persistence of individual joint sets

The potential for instability may be influenced by:

- relative condition, shape and surface characteristics of joints
- aperture and infilling characteristics
- orientation of joint sets with respect to each other

The degree of discontinuity of the hangingwall relative to the influencing characteristics is illustrated in Figure 3.3.6.

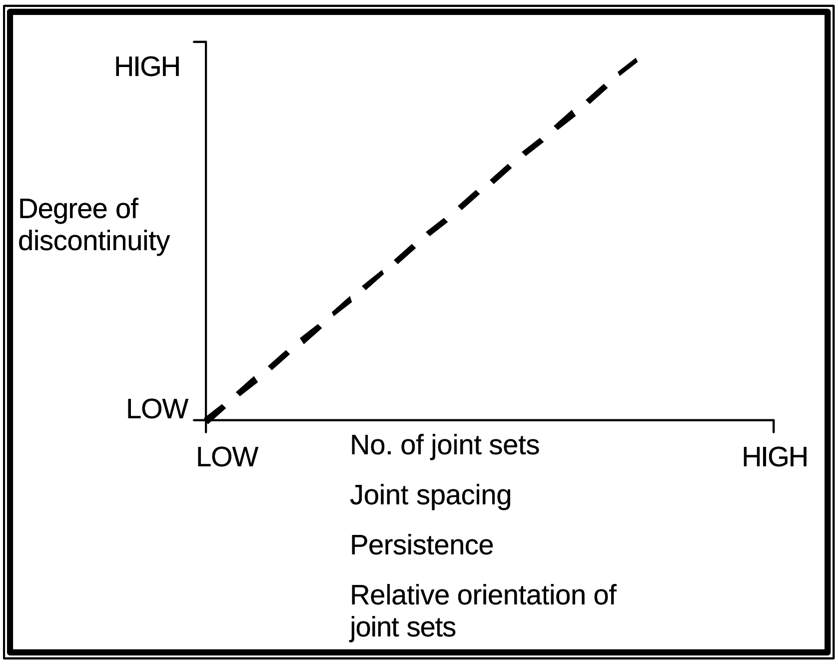


Figure 3.3.6 Influence of joint characteristics on the degree of discontinuity.

The number of joint sets and their relative orientations must be examined together. Sub-parallel joint sets result in a lower degree of discontinuity than intersecting joint sets. Güler *et al.* (1998) classified discontinuities into five categories based on the number of discontinuities and their relative orientations. A similar approach was adopted here and is illustrated in Table 3.3.3.

Table 3.3.3 Classification of discontinuities based on discontinuity sets (after Güler et al., 1998).

	Category 1	Category II	Category III	Category IV	Category V
	No intersecting discontinuities	One set of discontinuities	Two intersecting sets of discontinuities	Three intersecting sets of discontinuities	Four or more intersecting sets of discontinuities
Description	Very good	Good	Fair	Poor	Bad

The categorisation was based on the relative orientation of the discontinuity sets to form blocks with potential for instability.

Joint spacing is another important factor influencing the degree of discontinuity of the hangingwall. Generally, the closer joints are spaced, the higher the degree of discontinuity. The following classification may be used to describe spacing:

Table 3.3.4 Classification of discontinuity spacing (after Brady and Brown, 1985).

DESCRIPTION	SPACING (mm)
Extremely close	< 20
Very close	20 - 60
Close	60 - 200
Moderate	200 - 600
Wide	600 - 2000
Very wide	2000 - 6000
Extremely wide	> 6000

Spacing often follows a negative exponential distribution as reported by Brady and Brown (1985), Daehnke *et al*. (1998), and Priest (1993). A larger number of closer spaced discontinuities and increasingly smaller number of wider spaced discontinuities will be found. Joint spacing varies considerably, especially in the vicinity of faults, intrusives and folds. It is therefore important to obtain a representative sample of joint spacing near to and away from geological features.

Joint persistence is only important when examined relative to the size of the face area. Persistent joints will extend past the boundaries of the face area and impersistent joints will terminate within the face area. Persistence affects the size and number of keyblocks formed. Daehnke *et al*. (1998) used JBlock software to determine the effect of joint lengths on block sizes formed in the hangingwall. Longer joints were found to form larger blocks. Maximum joint lengths greater than three to five times the average length were found to have little effect on block size distributions.

Joint stability is influenced by aperture, shape, roughness and infilling which serve to increase or decrease shear strengths of joint surfaces. Clean, closed joints have higher shear strengths than open, filled joints. Roughness increases shear strength at low apertures. Larger apertures with soft infilling of clay, fault gouge, chlorite or serpentinized material will decrease shear strengths. Infilling of strong materials like quartz or pyrite may increase shear strengths.

The parameter J_a/J_r may be useful in describing the quality of the joint surface, where J_a is the joint alteration number and J_r is the joint roughness number.

3.3.4.3 Strike slip faults

These faults have their major displacement along the plane of the reef, i.e. in a lateral sense. Vertical displacement is absent or minimal. The characteristics influencing joint stability are also applicable to strike faults.

3.3.4.4 Summary of reef parallel and perpendicular structures

Strata conditions may vary depending on the influence of one or more reef parallel or perpendicular structures. The degree of influence of the various structures may vary with one or several having a major influence. It is therefore necessary to determine the major influence of those critical structures relative to those with minor influence. Current knowledge and experience is therefore an important input in determining problematic and non-problematic strata conditions.

The following reef parallel situations may be considered to be problematic:

- bedding thicknesses less than 30 cm in the immediate hangingwall
- bedding planes occupied by argillaceous or micaceous laminations and alteration products
- structures with moderate to good parting potential
- the location of critical structures in the immediate hangingwall

Reef perpendicular situations which may be problematic, include:

- three or more intersecting discontinuity sets
- closely spaced discontinuities
- discontinuities with soft, weak infilling, open apertures and smooth surfaces.

A combination of a reef parallel structure (chromitite seam with good parting potential) and reef perpendicular structures (various joints) is illustrated in Figure 3.3.7.

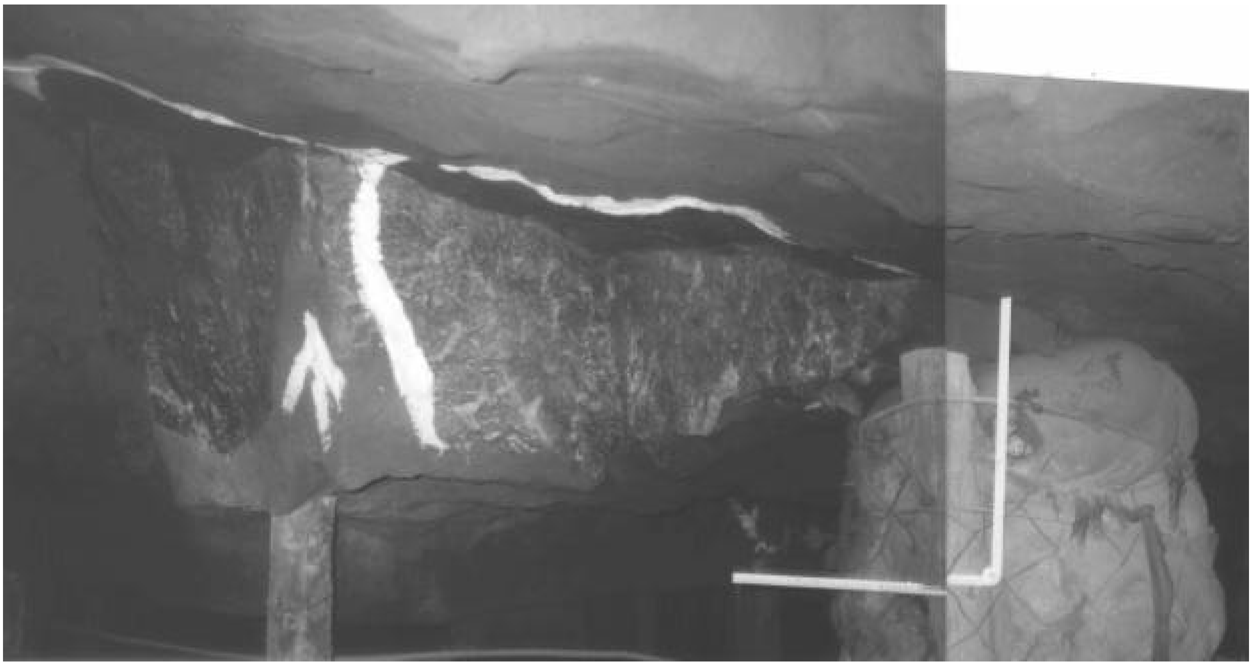


Figure 3.3.7 The effect of reef parallel and reef perpendicular structures on the hangingwall conditions.

3.4 Strata conditions and support requirements for shallow environments

3.4.1 Support types

The following face area support types can be used:

- Stiff (minepoles, etc.)
- Pre-stressed stiff (mechanical props, etc.)
- Pre-stressed with variable yield (hydraulic props, etc.)
- Headboards – directional, non-directional and wide (gully wing headboard, etc.)
- Areal support (headboards, base plates, etc.)
- Tendon support – various types (mechanical anchored rebar, cable bolt, etc.)

These are based on current support practices (Daehnke *et al.*, 1998), available support types and support types currently being tested (Acheampong, 1999).

Stiff support includes passive types of support that provide a high initial support resistance, e.g. mine poles and mechanical props. Pre-stressed types are active types of support with varying degrees of yield. Headboards are available as integral parts of certain support types or as add-on units. Non-directional units are round or square. Directional units are rectangular and should be installed in a particular direction. Wide headboards are proposed for greater areal coverage and would also be directional. Areal support refers to support types giving large areal coverage, e.g. safety nets. Tendon support types include cables and bolts, either grouted or non-grouted, and continuous friction tendons. The various support types are discussed in more detail in Chapter 2.

3.4.2 Recommended support types for various classes of strata conditions

Table 3.4.1 lists the various possible classes of strata conditions and the recommended support types for each class. It is assumed that support resistance and ejection thickness requirements will be met by appropriate spacing of the units. All support types must be used where more than one type is indicated. Alternatives are indicated in brackets. For stoping widths greater than 1,5 m, downgrading is necessary to account for excessive lengths of support units (see Chapter 2).

Table 3.4.1 Classes of strata conditions and recommended support types for each class.

CLASS	1	2	3	4
ROCK MASS	WEAK	WEAK	WEAK	WEAK
REEF PARALLEL STRUCTURE	PROBLEMATIC	PROBLEMATIC	NON-PROBLEMATIC	NON-PROBLEMATIC
REEF PERPENDICULAR STRUCTURE	PROBLEMATIC	NON-PROBLEMATIC	PROBLEMATIC	NON-PROBLEMATIC
S.W.: < 1,8 m	2 + 8 (7)	2 + 6 + 10	2 + 7 (6)	1 (2)
> 1,8 m	2 + 8 (7)	2 + 10	2 + 11 (6)	1 (13) (11)

CLASS	5	6	7	8
ROCK MASS	STRONG	STRONG	STRONG	STRONG
REEF PARALLEL STRUCTURE	PROBLEMATIC	PROBLEMATIC	NON-PROBLEMATIC	NON-PROBLEMATIC
REEF PERPENDICULAR STRUCTURE	PROBLEMATIC	NON-PROBLEMATIC	PROBLEMATIC	NON-PROBLEMATIC
S.W.: > 1,8 m	1 + 7 (10)	2 + 9	2 + 6	1
< 1,8 m	2 + 7 (11)	1 + 12	1 + 11	1 (12)

SUPPORT TYPES			
STIFF (eg Mechanical props, Timber sprags)	1	AERIAL SUPPORT (eg Net, Evermine)	8
STIFF - P/S (eg Yieldable camlock prop)	2	CABLE - NON-GROUTED (eg Cablebolt, Conebolt)	9
PS Elongate (eg Ebenhaeser)	3	CABLE - GROUTED (eg Cement grouted cablebolt, Conebolt)	10
Rapid Yield Hydraulic Prop	4	BAR - GROUTED (eg Cement grouted expansion shell, Cablebolt)	11
HEADBOARDS - NON DIRECTIONAL (eg Pots)	5	BAR - NON- GROUTED (eg Expansion shell)	12
HEADBOARDS - DIRECTIONAL	6	CONTINUOUSLY FRICTION COUPLED TENDONS (eg Split sets)	13
HEADBOARDS - (WIDE)	7	RESIN GROUTED CYLINDRICAL TUBES	14

3.5 Parameters influencing strata conditions in intermediate and deep environments

3.5.1 Introduction

Gold and platinum deposits are included in this category. Important parameters to be discussed include:

- rock type and UCS
- mining induced fracturing
- reef parallel structures
- reef perpendicular structures

3.5.2 Rock type and UCS

A similar distinction is made, as for shallow environments, between weak and strong rocks. Current knowledge and rock practitioners' experience is an essential input.

Competent rock types with UCS of > 200 MPa (siliceous quartzite) often fracture more intensively than argillaceous quartzites (Daehnke *et al.,* 1998, and Turner, 1985) resulting in a highly discontinuous hangingwall. The latter may be more prone to punching and unravelling and would have to be classed as a weak rock.

3.5.3 Mining induced fracturing

In intermediate and deep gold mines, induced stresses acting on excavations are large enough to cause failure in the surrounding rock by fracturing. Fracturing is generally systematic and symmetric about the plane of the stope (Gay and Jager, 1986).

Gay and Jager (1986) further identified five distinct sets of fractures: four types of extension fractures and one type of shear fracture. Extension fractures do not normally exhibit any displacement parallel to the fracture surface and are clean. Shear fractures exhibit signs of displacement, and dusty and gouge-filled surfaces. The characteristics of the various types are outlined in Table 3.5.1:

Table 3.5.1 Characteristics of mining induced fractures in deep gold mine stopes (amended after Daehnke et al., 1998). Types 1 to 4 represent extension fractures while type 5 represents shear fractures.

TYPE	SPACING		PERSISTENCE (STRIKE)		PERSISTENCE (INTO HW)	DIP
	AVERAGE	RANGE	AVERAGE	RANGE	RANGE	RANGE
1	0,07	0,05 - 0,5	1,5	0,8 - 6,0	1,0 - 5,0	60^{O}-80^{O}
2	0,07	0,05 - 0.5	1,5	0,8 - 4,0	1,0 - 5,0	60^{O}-80^{O}
3	0,03	0,01 - 1,0	1,5	0,8 - 4,0	1,0	20^{O}-40^{O}
4	0,3	0,1 - 0,5	0,3	0,1 - 0,5	1,0	0^{O}-20^{O}
5	3,0	1,0 - 6,0	8	5,0 - 30,0	7,0	100^{O}-120^{O}

The fractures are generally face parallel but may vary up to 20º from the face. Hagan (1980) identified “longwall parallel fractures” that were aligned parallel to the general longwall direction rather than to individual stope faces. Fracture strikes also vary with locality, specifically near lead-lag configurations, near strike pillars and at the top and bottom portions of panels. Only face parallel fractures are considered in this study.

Gay and Jager (1986) summarised the relative time of formation and position relative to the stope face at which fractures develop. Shear fractures are the first to develop ahead of the face. Type 1 extension fractures also develop ahead of the face, either with shear fractures or soon after. Type 2 occurs next and closer to the face. Type 3 occurs at the face followed by type 4 behind the face. The relative proportion of each set varies between stopes and is dependent on local geological conditions, mining configurations and stresses.

Face parallel fracturing in this study includes type 1 and 2, 3 and 5. Type 4 forms behind the face and thus has little effect on the immediate face area. Types 1 and 2 generally dip at greater than 50º (see Table 3.5.1). Face parallel fractures exhibit a bifurcating nature along their strike direction due to their overlapping nature. They are considered to be persistent for the length of the panel.

Daehnke *et al.* (1998) indicated that for specified conditions (clamping stress = 1 MPa and friction coefficient = 40º), fractures dipping at greater than 50º were more stable than those dipping at less than 50º. A reduction in hangingwall clamping stresses results in an increase in hangingwall instability and only the steepest fractures are stable. Squelch (1994) measured maximum horizontal stresses between 0,7 m and 2,5 m into the hangingwall of between 1 and 10 MPa for conventionally supported stopes. Face parallel fracturing will therefore be divided into those with dips greater than 50º and those less than 50º with horizontal stresses generally assumed to be adequate to clamp the hangingwall.

3.5.4 Reef parallel structures

Reef parallel structures are similar to those discussed for shallow environments. Higher horizontal stresses are, however, present and influence reef parallel structures to a greater extent. Shearing along bedding planes, lithological contacts, bedding parallel faults and joints will destroy any cohesion and increase their parting potential resulting in an increase in hangingwall instability. Daehnke *et al.* (1998) found that decreasing the beam thickness resulted in a reduction in maximum stable spans and consequently an increase in hangingwall instability.

The following hangingwall characteristics would be problematic:

- occurrence of bedding parallel faults and joints
- medium bedded to thinly laminated rocks
- argillaceous laminations in quartzites
- lithological contacts within 2 m of the hangingwall
- moderate to good parting potential of the structures listed above
- shallow dipping fractures

3.5.5 Reef perpendicular structures

These structures (joints and strike slip faults) have been introduced in Chapter 2. The distinction between problematic and non-problematic structures is likewise similar.

3.6 Strata conditions and support requirements for intermediate/deep environments

Table 3.6.1 lists the various possible classes of strata conditions and the recommended support types for each class.

Table 3.6.1 Classes of strata conditions and the recommended support types for each class for intermediate/deep mines.

CLASS	1	2	3	4	5	6	7	8
ROCK MASS	WEAK	WEAK	WEAK	WEAK	WEAK	WEAK	WEAK	WEAK
M.I. FRACTURE (DIP)	$>50^{o}$	$>50^{o}$	$>50^{o}$	$>50^{o}$	$<50^{o}$	$<50^{o}$	$<50^{o}$	$<50^{o}$
REEF PERPENDICULAR STRUCT.	PROB.	PROB.	NON-PROB.	NON-PROB.	PROB.	PROB.	NON-PROB.	NON-PROB.
REEF PARALLEL STRUCT.	PROB.	NON-PROB.	NON-PROB.	PROB.	PROB.	NON-PROB.	NON-PROB.	PROB.
ROCKBURST: < 1,8 m	3(4) + 8	3 + 7(6)	3 + 7	3(4) + 7	4 + 6(7) + 8	4 + 6(7)(8)	3(4) + 7	4(3) + 7(6)
> 1,8 m	2(1) + 14(8)	2(1) + 7	1(2) + 7	1(3) + 7	2(1) + 14(8)	1(2) + 6(7)(8)	1(2) + 7	1(2) + 7(6)
ROCKFALLS: < 1,8 m	3(4) + 8	2(1) + 7	3(2) + 7(6)	3 + 7	3(4) + 6 + 8	3(2) + 6(7)(8)	3 + 7	2(3) + 6(7)
> 1,8 m	2(1) + 14(8)	1 + 7(6)	1(2) + 7	1 + 7	1(2) + 14(8)	1(2) + 6(7)(8)	1(2) + 7	1(2) + 7

CLASS	9	10	11	12	13	14	15	16
ROCK MASS	STRONG	STRONG	STRONG	STRONG	STRONG	STRONG	STRONG	STRONG
M.I. FRACTURE (DIP)	$>50^{o}$	$>50^{o}$	$>50^{o}$	$>50^{o}$	$<50^{o}$	$<50^{o}$	$<50^{o}$	$<50^{o}$
REEF PERPENDICULAR STRUCT.	PROB.	PROB.	NON-PROB.	NON-PROB.	PROB.	PROB.	NON-PROB.	NON-PROB.
REEF PARALLEL STRUCT.	PROB.	NON-PROB.	NON-PROB.	PROB.	PROB.	NON-PROB.	NON-PROB.	PROB.
ROCKBURST: <1,8 m	3(4)+8	4 + 6	3(4)	4(3) + 5(6)	4 + 8 + 6	4 + 6 + 8	3(4) + 6	3 + 6
>1,8 m	1(2)+8(10)	1(2) + 6	2(1)	2(1) + 9	3(2) + 6	2(3) +6+ 9(6)	2(3) + 10	3 + 9
ROCKFALLS: <1,8 m	2(3)+8	1 + 6	1	2 + 5	1 + 8 + 6	3(4) + 6	2 + 6	2 + 6
>1,8 m	1(2)+8(10)	1 + 6	1	2 + 12 + 9	1 + 8 + 10	2 + 6 + 12	2 + 12	3 + 12

SUPPORT TYPES			
STIFF (eg Mechanical props, Timber sprags)	1	AERIAL SUPPORT (eg Net, Evermine)	8
STIFF - P/S (eg Yieldable camlock prop)	2	CABLE - NON-GROUTED (eg Cablebolt, Conebolt)	9
PS Elongate (eg Ebenhaeser)	3	CABLE - GROUTED (eg Cement grouted cablebolt, Conebolt)	10
Rapid Yield Hydraulic Prop	4	BAR - GROUTED (eg Cement grouted expansion shell, Cablebolt)	11
HEADBOARDS - NON DIRECTIONAL (eg Pots)	5	BAR - NON- GROUTED (eg Expansion shell)	12
HEADBOARDS - DIRECTIONAL	6	CONTINUOUSLY FRICTION COUPLED TENDONS (eg Split sets)	13
HEADBOARDS - (WIDE)	7	RESIN GROUTED CYLINDRICAL TUBES	14

3.7 Possible alternative support technologies in concept that may assist in alleviating rockfall and rockburst problems

3.7.1 Introduction

The alternative face support systems that have been identified have one characteristic in common, areal coverage. The advantages and disadvantages of these systems are discussed in this section. Some of these systems are designed for shallow to intermediate mining conditions, but there are others that can be installed in deep mines where seismic activity is expected. The systems that are considered in this section are:

1) twin beam support system,
2) powered support for hard rock mining,
3) membrane support (Evermine),
4) mesh support for stopes,
5) strapping between tendon support,
6) large headboards and base plates,
7) safety nets,
8) Fortrac Geogrids,
9) modified stope support system using hydraulic props,
10) rock tendon support,
11) inflatable supports.

The advantages and disadvantages of each of these systems are given below.

3.7.3 Twin beam support system

3.7.3.1 Introduction

This system is similar in concept to that used by Megkon (Tarr *et al.*, 1988), to develop a Mobile Stope Support System (MSSS) (Figure 3.7.1). The MSSS project was subsequently stopped and the information was archived. This information was used to design a simplified twin beam support system as an internal CSIR project (Kuijpers personal communication, 1999).

3.7.3.2 Discussion

The twin beam support system consists of two modified hydraulic props connected by a steel beam, Figure 3.7.2. The beam has rails, which carry two hydraulic props and greatly reduces the effort in moving the props by allowing the hydraulic props to slide along the rails, and is used, by drawing the rear prop up to the front prop to walk the structure forward. The beam extends ahead of the hydraulic props to provide areal support close to the face. The long axis of the beam will be installed perpendicular to the face and the extension fractures (on strike in a breast mining panel), such that it will support a number of potentially unstable hangingwall blocks. Using this system, the hydraulic props and the beam can be moved with the mining cycle, while maintaining areal coverage. This system is currently being tested. The envisaged advantages and disadvantages of this system are given below.

Figure 3.7.1 Twin beam support system.

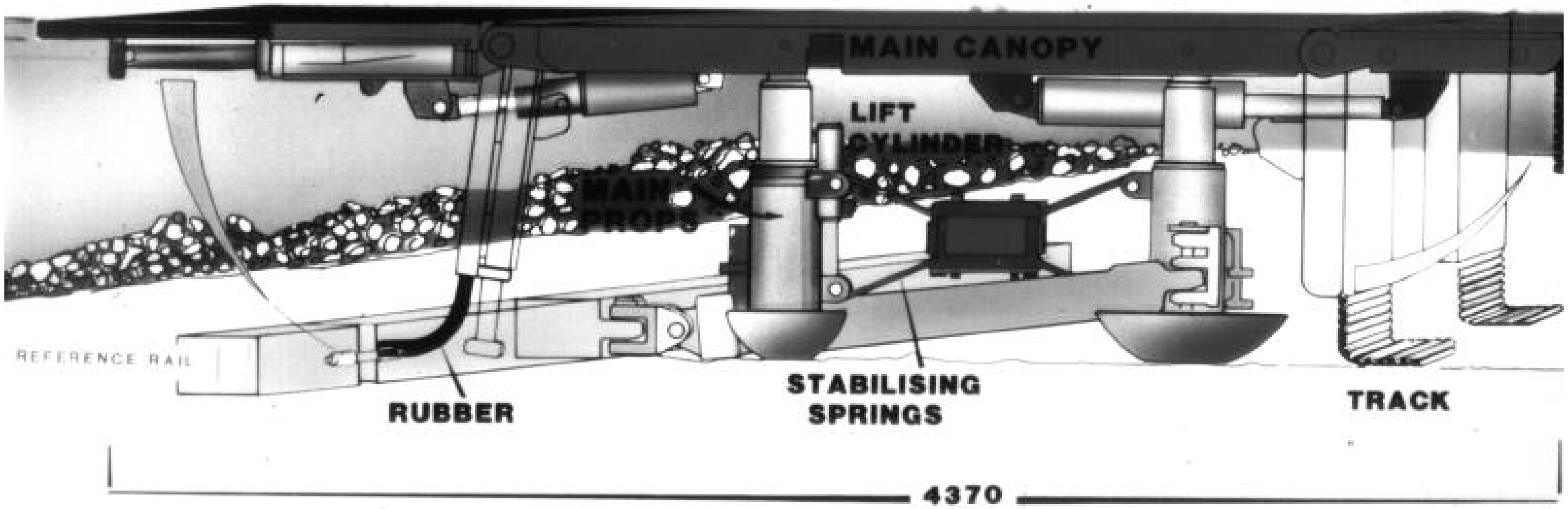


Figure 3.7.2 The prototype of the Mobile Stope Support System (MSSS).

Advantages

1) Provides continuous hangingwall support, thus decreasing the probability of a rockfall occurring.
2) Once the system is installed, no further lifting of heavy equipment is required since the system can be moved with the mining cycle by sliding the different sections along the rails.
3) The hydraulic props can yield and support a hangingwall moving at a velocity (caused by dynamic loading) of 3 m/s.
4) The structure can be used to support a blast barricade.
5) Provides good areal coverage of the stope face area.
6) Moving of the support will not be time consuming and would not require great physical effort.
7) The system can be applied in both rockfall and rockburst conditions.
8) The system can be upgraded to provide further areal coverage between the different systems by using a Safety Net type system.
9) Due to the ease of moving the support forward, the drilling operation can commence sooner. Thus, the whole mining operation will be more effective.
10) The workers will feel safer with this system compared to just having single support units.

Disadvantages

1) Point loading of the beam can occur due to the undulating nature of a stope hangingwall.
2) Transportation of the system will be time consuming due to the mass of the beam.
3) The first installation will require lifting of heavy equipment, therefore, at first glance, the workers may react negatively towards the system.
4) The first installation process will be time consuming due to the heavy equipment.
5) A modification of conventional hydraulic props is required.
6) Maintenance must be done on the hydraulic props.
7) Extremely disciplined teams must operate the system. If the hydraulic props give problems, they must be immediately removed and repaired. The cost of running a system such as this is also a function of the management and discipline of the teams involved.
8) This system cannot be used in very high stope widths.
9) In very low stoping widths the beam may interfere with scraper cleaning.

3.7.3 Powered support for hard rock mining

3.7.3.1 Introduction

Powered supports are mobile hydraulic support systems used at the face area in modern longwall coal mining. Their functions are fourfold: they hold and keep the roof up, they advance the chain conveyor (AFC), they advance themselves, and provide a safe environment for all mining activities. The most important requirement for successful longwall mining is the selection and application of powered supports.

3.7.3.2 Discussion

Powered support systems are classified according to the following: 1) the presence or absence of a curving shield; 2) the number of ways of arranging the hydraulic legs. Curving shields may or may not be included in the support; if they have been included then the support is a shield support, if not the support is a Frame or a Chock. The support capacity of these systems is based on the number of hydraulic legs and the way they are installed; i.e. the more hydraulic legs the higher the support capacity. Vertically installed hydraulics have a higher efficiency of application than inclined ones. From the above concept there are four types of powered support that have been developed over the years: the frame, chock, shield and the chock shield. These are discussed in detail below.

The Frame

Figure 3.7.3 shows the design of the frame. The frame consists of two sets of hydraulic legs, the primary and the secondary set. There is a double-acting ram installed between each set. The piston of the ram is connected to the secondary set and the cylinder to the primary set. During movement of the support, the primary support is set against the roof while the secondary set is lowered and pushed forward by the piston. Once the required position has been reached, the secondary set is set against the roof while the primary set is lowered and pulled forward by the cylinder. The distance of each advance ranges from approximately 50 to 90 cm. A disadvantage of the frame is that it is structurally less stable and there is a large uncovered space between the two pieces of canopy, which allows broken rocks to fall through. It also requires frequent maintenance.

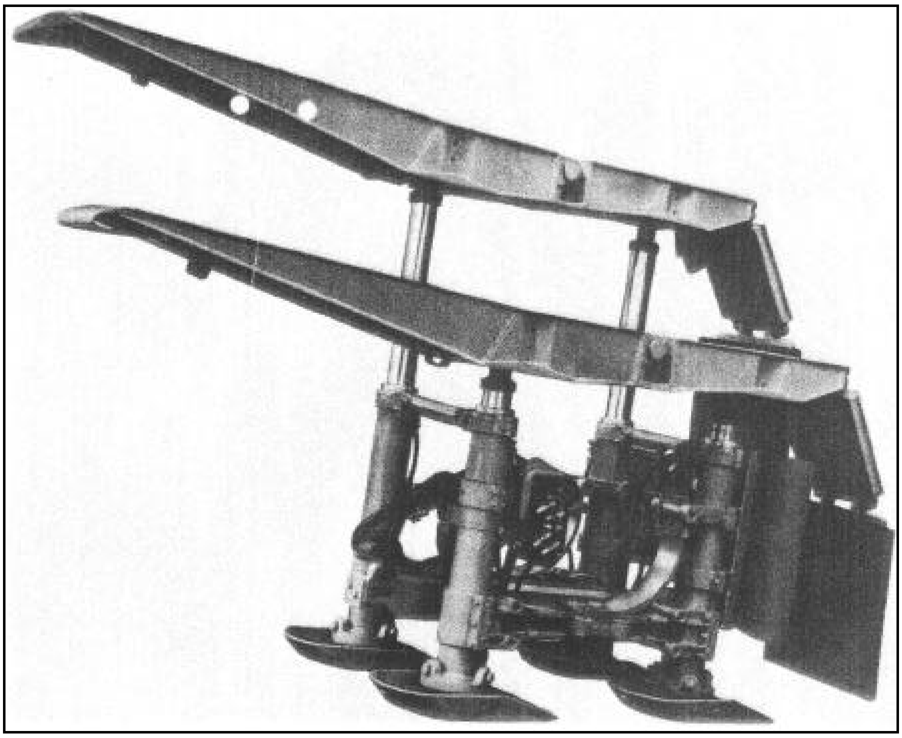


Figure 3.7.3 Layout of Frame support.

The Chock

The chock is characterised by a solid piece canopy and base that may either be a solid piece or two separate parts connected by steel bars at the rear and/or front ends, as shown in Figure 3.7.4. The number of hydraulic legs ranges from three to six, but the four-leg chocks are the most popular ones. All hydraulic legs are installed vertically between the base and the canopy. The double-acting hydraulic ram is used to push the chain conveyor and pull the chock in a whole unit. This setup is also used in the shields and chock shields. The chock has been reinforced with a box-shaped steel frame between the base and each leg. This box-shaped steel frame is used for longitudinal stability of the chock. The chocks are suitable for medium to hard roof conditions.

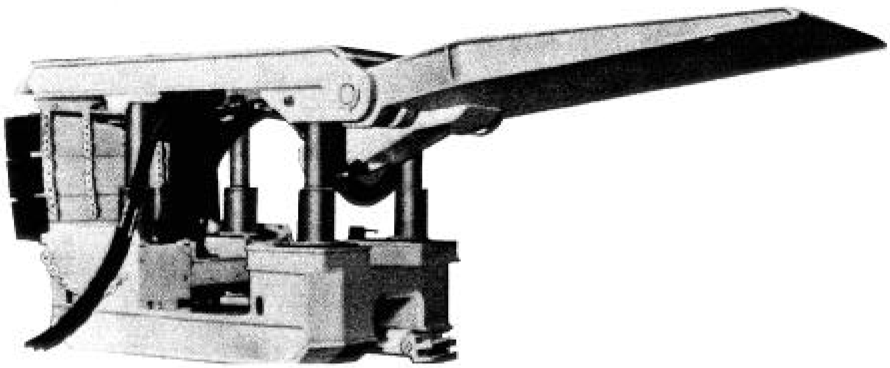
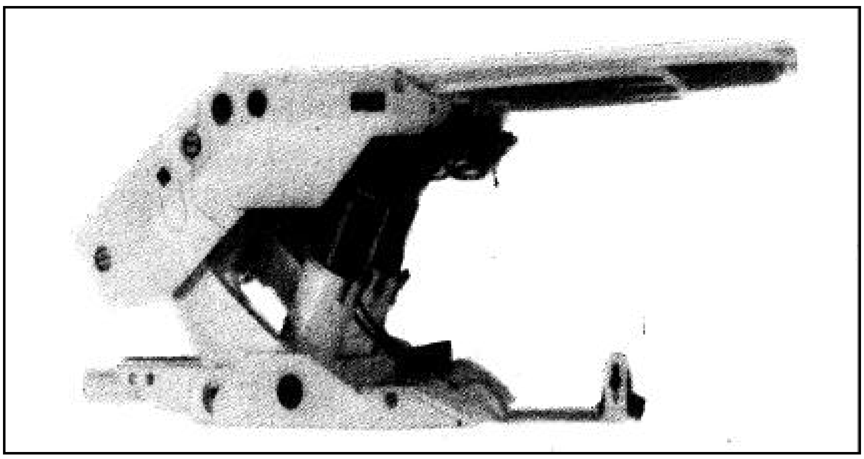


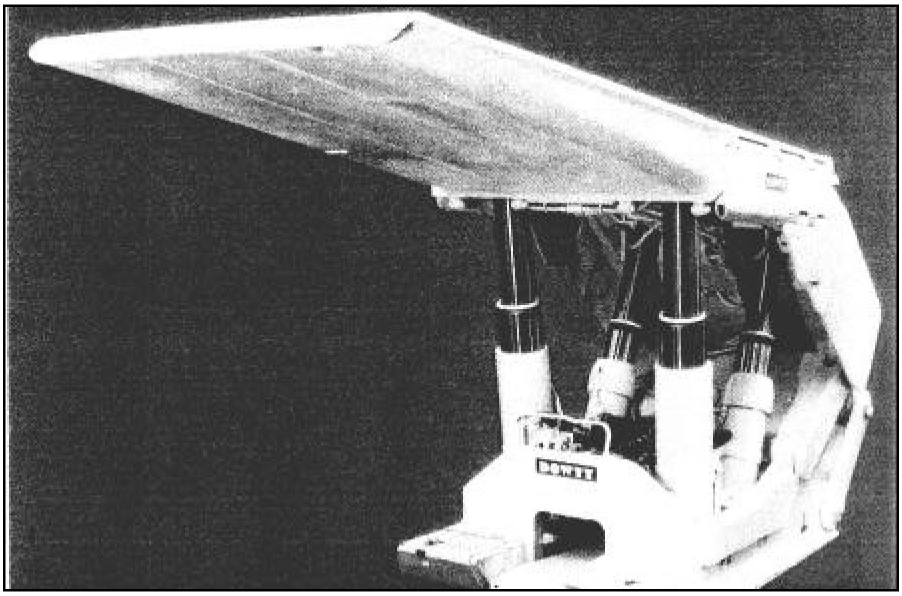
Figure 3.7.4 Chock support.

The Shields

Shields are characterized by the addition of an inclined caving shield. The caving shields are hinge-jointed to the canopy and the base, making them the most stable support. The hydraulic legs of the shield are generally inclined. The pin connections between the legs and the canopy, and between the legs and the base in a shield support, make it possible for the angle of inclination of the hydraulic legs to vary. The major advantage of the shield is that it completely seals off the gob and prevents rock debris from getting into the face side of the support.



(a)



(b)

Figure 3.7.5 (a) 2F$_h$/420$_{0.75}$ shield and (b) 4V/500$_{0.8}$ shield.

Chock shield

The chock shield (Figure 3.7.6) is characterized by four (or six) hydraulic legs, which are all installed at the same time. There are regular four or six leg chock shields in which all legs are vertical and parallel, while others form V or X shapes. Some canopies consist of a single piece, while others consist of two pieces with a hydraulic ram at the hinge joint. The chock shield has the advantage of both the chock and the shield because it combines the features of both. The chock shield has the highest supporting efficiency and is suitable for hard roof conditions.

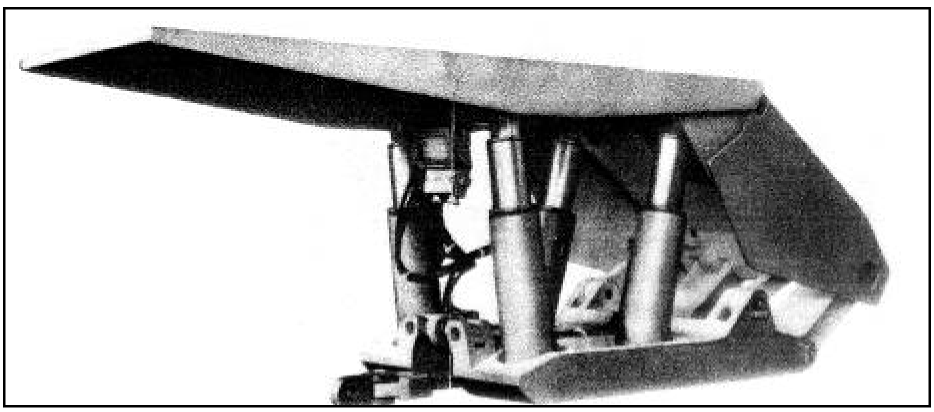


Figure 3.7.6 4V/600$_{0.85}$ chock shield.

3.7.3.3 Performance of powered supports

Setting load versus Yield Load

Once the support has been set, the pressurized fluid is introduced into the front chambers of the leg piston. This fluid pushes the legs until the canopy touches the roof. Once the canopy touches the roof, the pressurized fluid rapidly increases to the working pressure of the hydraulic pump. The control valve is turned off and the fluid is locked in. The fluid is now operating at the working pressure of the pump and this is the setting pressure of the support.

The capacity of the powered supports is conventionally designated by its yield load. For instance, a 500-ton shield support means that the shield has a yield load of 500 tons. The yield load cannot be represented by the contact area between the canopy and the roof because each model has a unique canopy style and the spacing between the support is not the same. The load per tributary area of the roof supported is commonly employed. Therefore, the setting load density is the support load at the time of setting divided by the area of the roof that is supported. The yield load density is the support load per unit area of the supported roof at the time of yielding, (Peng, 1986). The total roof to floor convergence at the face area is a measure of the intensity of the roof activity and the stability. This has been used to correlate with the adequacy of setting load. The relation between the roof to floor convergence and setting load density is a hyperbolic curve shown in Figure 3.7.7 below.

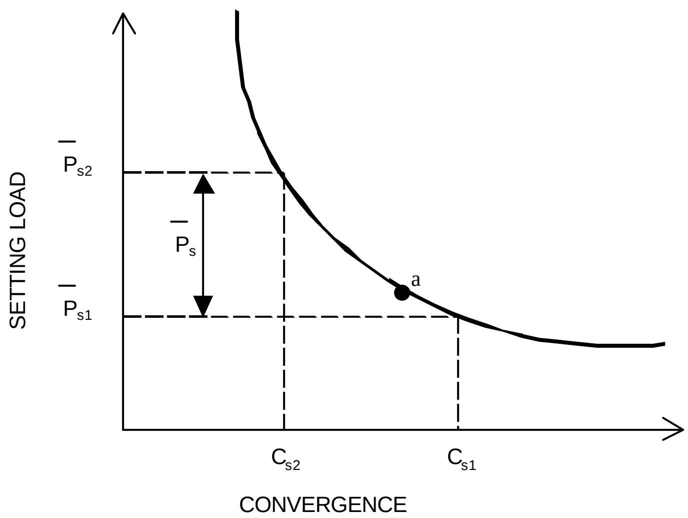


Figure 3.7.7 Setting load versus roof convergence.

Working Conditions of Power Supports

After the support is set against the roof and the floor, its load or resistance varies from period to period as a result of interaction between the roof, the support and the floor. There are distinctive patterns of variations in support resistance that can be used to evaluate the working conditions of the supports and the strata behaviour. Figure 3.7.8 shows a typical variation of support resistance in a mining cycle. Once the support has been advanced and reset, an initial setting load P_s is rapidly achieved within a short time period. As the support starts to interact with the roof, the support resistance increases rapidly in a period of t_a until it reaches a relative equilibrium. This is the rapid increase period (s-a), then comes the relatively stable period (a-b). When the cutting action of the shearer approaches the support, the resistance increases from (b-c) and is called the cutting influence period. Finally as the neighboring support is disengaged and advanced, the roof load is suddenly transferred to the support thereby rapidly increasing the support resistance in a short time period (c-d). As the support is released for advancing, its resistance drops almost instantaneously to zero (d-e). Figure 3.7.8 shows the sequence of load variation with respect to time.

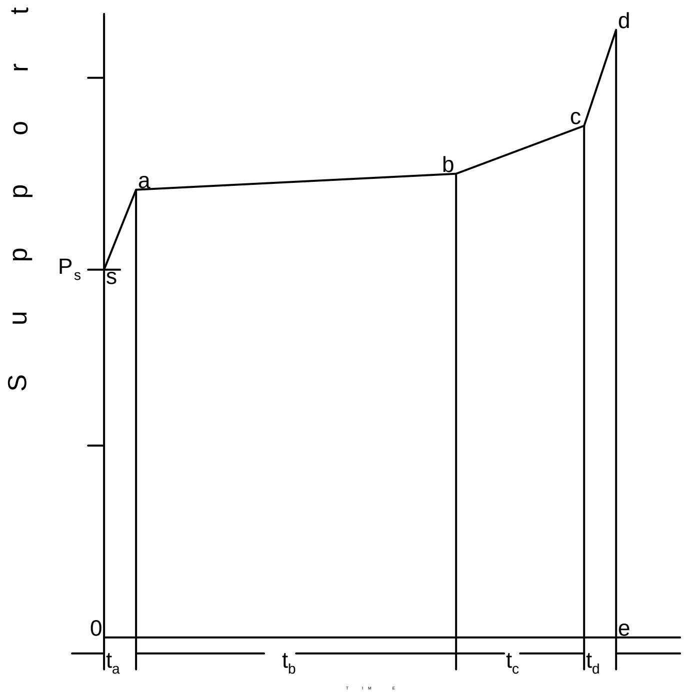


Figure 3.7.8 Typical form of variations in support resistance (Peng et al., 1982).

Advantages

1) Complete areal coverage along the face area.
2) The system provides active support to the hangingwall.
3) The canopy can be used to support the blast barricade.
4) The system can be used with an impact ripper at ultra depths.
5) All activities are mechanized, thus fewer workers are needed.
6) The probability of an accident occurring is lower, since fewer workers are needed.
7) No other artificial support is needed.

Disadvantages

1) The transportation and assembly of the system is time consuming.
2) The initial capital is extremely high.
3) For seismically active stopes the system needs to be modified to allow rapid and controlled yielding.
4) The high closure rates at depth can cause the system to become trapped.
5) The system cannot be used in stopes where the footwall is not smooth.
6) The system must be robust to withstand the effects of blasting. Thus, the cost will increase.
7) The impact ripper can be used with this system only where the ground is extremely fractured and no blasting is needed.
8) Intensive maintenance is required.

9) The system requires the back area to cave. Thus, no worker must be allowed in the back area because there is no support.
10) The system is difficult to operate in steeply dipping stopes.
11) It is difficult to negotiate discontinuities intersecting the stope (eg. faults).

3.7.4 Membrane support (Evermine)

3.7.4.1 Discussion

Evermine is a membrane support type (see Figure 3.7.9) that is currently being tested as pillar reinforcement on the coal, platinum and gold mines. Laboratory testing has been carried to investigate its energy absorption capabilities. This support must be applied to the rock mass before unravelling has occurred. If a running dyke has been intersected, Evermine cannot be applied to prevent it from further unravelling. Thus, the condition of the rock mass must be such that Evermine is applied before large deformation has occurred, otherwise it should not be used at all. The surface to which it will be applied to must be properly prepared, i.e. barring of loose rock and washing off the dust on the surface. Evermine will not adhere to the rock if there is dust on the rock surface (Wojno, 1999). The application of Evermine to the stope hangingwall will have advantages as well as disadvantages. These are summarized below.

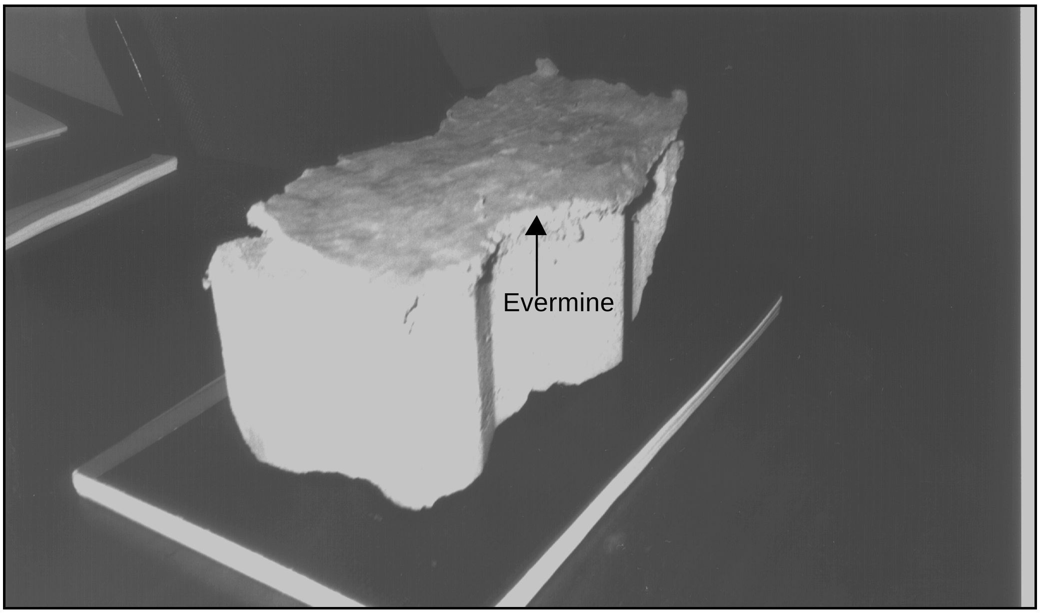


Figure 3.7.9 Photo showing cured Evermine on brick after dynamic load test.

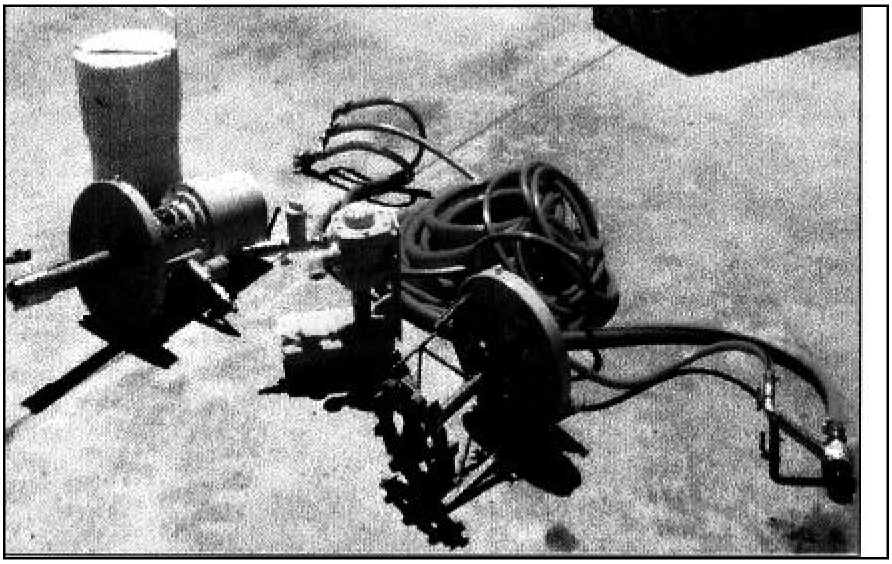


Figure 3.7.10 Evermine mixer equipment:a mixer, its gearbox and pneumatic motor, stem pump, hoses and nozzle.

Figure 3.7.11 Note ease of application and zero rebound.

Advantages

1) No dust involved in the application process (Figure 3.7.11).
2) No rebound occurs when Evermine is applied (Figure 3.7.11).

3) Mixing time (of the products) is maximum two minutes.
4) Makes use of the Spedel pump principle, which is widely used in the mining industry.
5) Application can be done at compressed air pressures as low 2 bars.
6) The membrane is only 2-3 mm in thickness.
7) Provides complete areal coverage.
8) Laboratory tests have shown that Evermine increases the energy absorption capabilities of layers of concrete blocks almost three-fold.
9) Only three workers are required to perform the application of Evermine.
10) Evermine can be applied at a rate of 60 m^2 per hour by a gang of three workers, compared to 30 m^2 of shotcrete in a five-hour shift by a gang of six workers.
11) Good adhesion is obtained within a short time period.
12) The complete system is compact and easy to transport underground (Figure 3.7.10).
13) This system can be used in any stope width.
14) Teams can be used on a contract basis for application of Evermine.

Disadvantages

1) Evermine cannot be applied everywhere, only where no unravelling, nor large deformations have taken place.
2) Material cost is approximately R 55 per square metre.
3) Evermine applied right up to the stope face can be partially damaged by the blast (fly-rock).
4) The discontinuities cannot be observed since the support is not transparent.
5) During barring, no areal support will be installed to protect the workers.

3.7.5 Wire mesh for stope support

As currently practiced in tunnels, wire mesh can be used with tendon support to prevent rockfalls from injuring workers. The mesh will give complete areal coverage of the stope hangingwall. There are a number of disadvantages and advantages to this system. These are discussed below.

Advantages

1) Complete areal coverage.
2) The mesh can be used with different types of tendons, i.e. cable anchors, cone bolts, shepherd's crook, etc.
3) Different tendon types can be chosen to suit certain conditions, e.g. cone bolts can be used in seismically active areas.
4) This support can be installed at the face (where workers spend most of their time).
5) This support would not be in the path of scrapers, therefore the chance of it being accidentally removed by scrapers is very small. Pack and elongate support in stopes can be dislodged by scrapers.
6) The mesh and lace support with yielding tendons have energy absorption capabilities.

Disadvantages

1) If a worker is trapped underground, the rescue operation is made more difficult because the rescuers must cut through the mesh to reach the injured persons.
2) Drilling is difficult in low stope widths.
3) Installation procedure is time consuming.
4) Blast damage may occur.

3.7.6 Strapping between tendon support

Straps consist of interconnected steel bands of about 30 cm in width and 2 m or more in length. They are used to prevent the fall of rocks between the tendons, Figure 3.7.12. Tendons interconnected by straps are used in some Australian and Canadian mines. The axis along which the strapping is installed depends on the conditions of the rock mass. If the dominant fractures that are problematic occur parallel to the stope face, then the strapping can be installed perpendicular to these fractures. The advantages and disadvantages are discussed below.

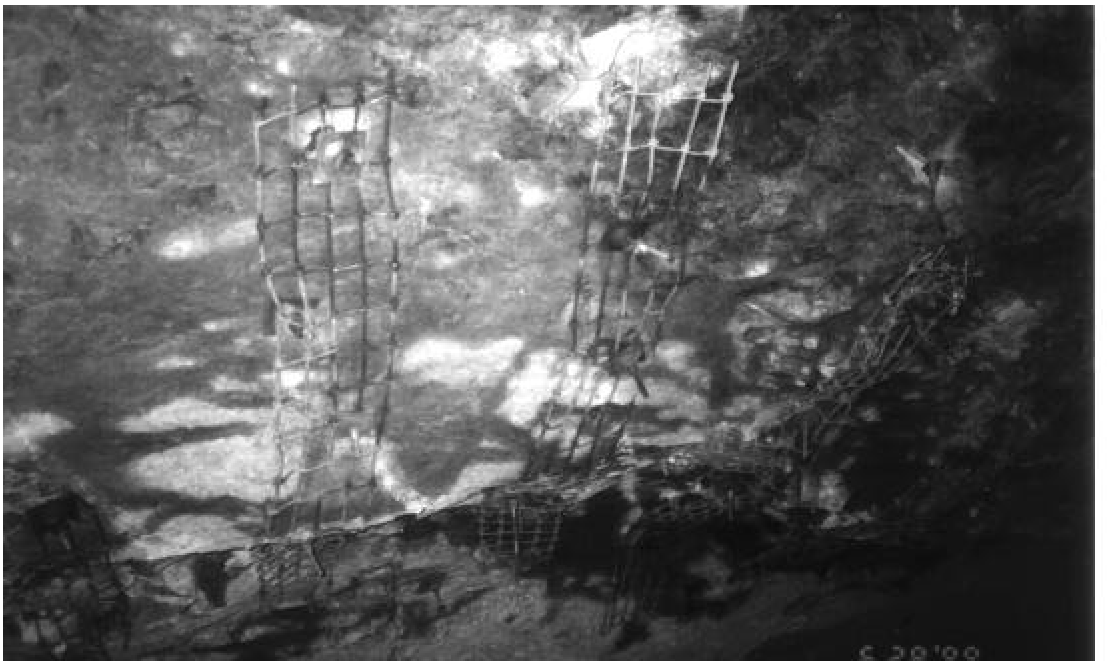


Figure 3.7.12 Strapping between tendons.

Advantages

1) Strapping can be used with different types of tendons i.e. cable anchors, cone bolts, shepherd's crook, etc.
2) Different tendon types can be chosen to suit certain conditions, e.g. cone bolts can be used in seismically active areas.
3) This support can be installed on the face (where workers spend most of their time).

4) This support would not be in the path of scrapers, therefore the chances of being accidentally removed by scrapers are small. Pack and elongate support in stopes can be dislodged by scrapers.

Disadvantages

1) If a worker is trapped underground, the rescue operation is made difficult because the rescuers must cut through the strapping to get to the injured persons.
2) Drilling difficulty in low stope widths.
3) Installation procedure is time consuming.
4) Blast damage may occur.

3.7.7 Large headboards and base plates

3.7.7.1 Introduction

When designing stope support systems it must be borne in mind that stopes tend to be potentially unstable due to rockfalls and rockbursts, and that there is a high concentration of workers in stoping areas. Therefore, adequate support of stopes is crucial in ensuring the safety of men, equipment and materials, high productivity and the minimum of reef dilution. The cost effectiveness of the support system must also be considered.

3.7.7.2 Discussion

Although different types and makes of headboards were designed, they were all designed to spread the load evenly and provide greater areal coverage. Only one type of headboard is mentioned in this section.

A RYHP headboard (gully wing type headboards), shown in Figure 3.7.13 below, are cumbersome to install but provide more than 1,5 m of areal coverage and theoretically should not interfere with the operations of the scraper. These headboards are however not in use.

The Base Plate

The base plate is designed to distribute the load at the bolt head uniformly into the surrounding rock. To maintain the elasticity of the rockbolt system, there are several types of base plates used in the mining industry. The choice of the base plate is crucial. There are three common types of base plates used in the mines, namely the Flat plate, the Dome plate, and the Triangular bell plate (Stillborg, 1986). The Flat plate can be used when the rock surface is smooth and the bolt is installed perpendicular to the surface of the rock. These plates also give the rockbolt system greater flexibility, and provide a high degree of areal coverage and stability of the hangingwall.

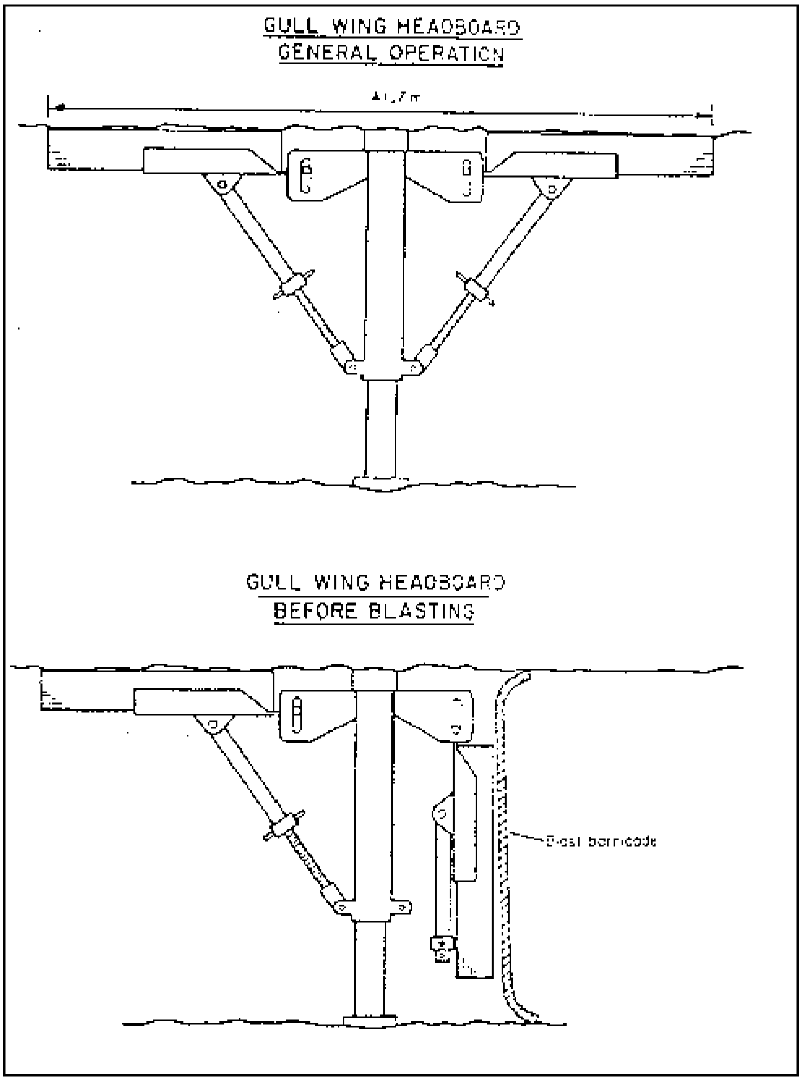


Figure 3.7.13 Gully wing headboard in general operation and just before blasting (*after Spearing, 1992*).

Advantages

1) Provides uniformly distributed hangingwall support, thus decreasing the probability of a rockfall occurring. The headboards can yield and support a hangingwall moving at a velocity (caused by dynamic loading) of 3 m/s.
2) The wing can be used to support the blast barricade.
3) The headboards provide good areal coverage of the stope face area.
4) Moving of the support will not be time consuming.
5) The headboards can be applied in both rockfall and rockburst conditions.
6) Light and easier to handle.
7) It is easy to install.
8) It is flexible to handle uneven hangingwalls.
9) It has a wide areal support span.

3.7.8 SAFETY NET

3.7.8.1 Introduction

The safety net has a rectangular shape and is currently being tested for use in the stope face area. The net is placed between the temporary support at the stope face and the first row of permanent support, as well as between the first and second row of permanent support. The net covers up to 85 % of exposed hangingwall between four elongates. The outside perimeter of the net consists of webbing with a strength of 4,5 tons. The inside webbing has a strength of 2,5 tons. Straps made of 4,5 tons webbing are attached to the four corners of the net and each strap has a hook attached to it. When installing the net, the straps are rapped around an elongate and hooked onto the net's perimeter. The net is tensioned by hand and kept as close as possible to the hangingwall.

3.7.8.2 Discussion

Two nets were taken to Tautona Mine, and installed in 120 East 3 and East 2 stope faces. The stope width of these panels was approximately 1 m. One net was lost during the night shift and could not be retrieved. The other net with a strap was moved between East 2 and East 3 faces and installed between the first row and second row of permanent support at an average distance of 2.2 m from the stope face. After 13 blasts the net was caught by a scraper and damaged. The figures below show the installation and performance of the net.

Figure 3.7.14 Net installed with straps.

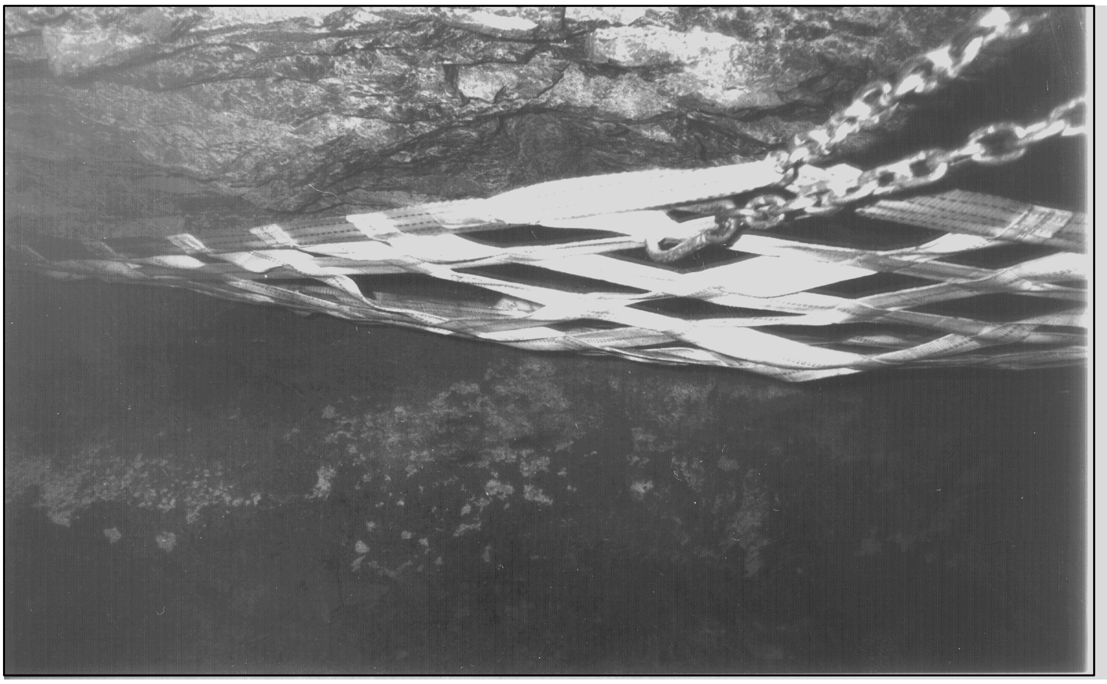


Figure 3.7.15 Net installed with chains.

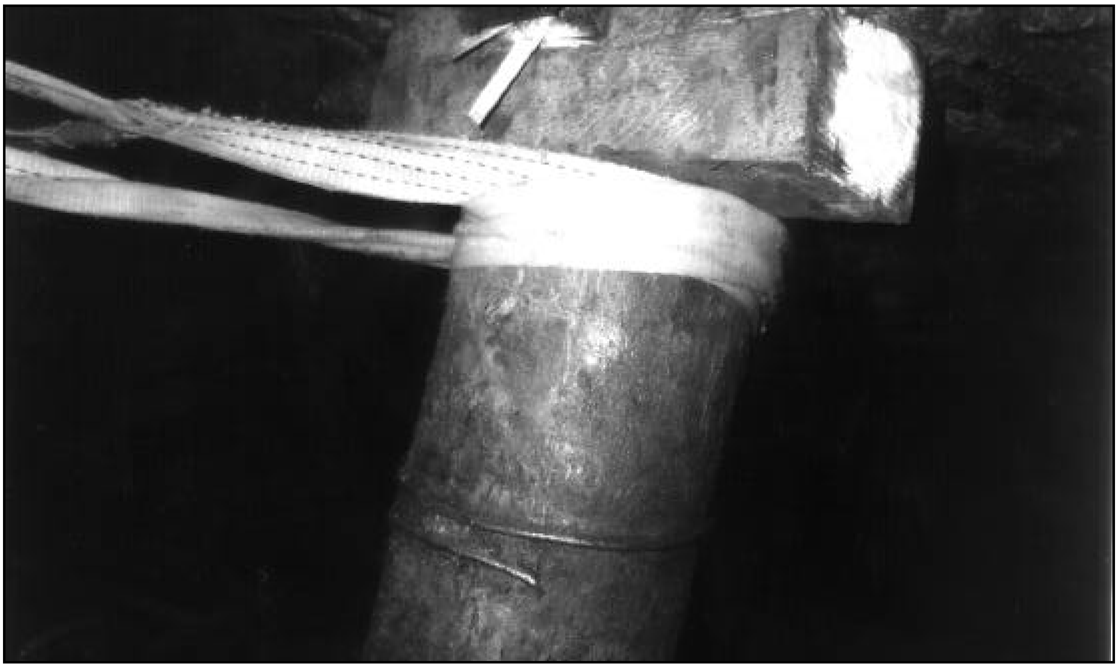


Figure 3.7.16 Straps around elongate did not slip down elongate.

Figure 3.7.17 Net pulled tight by hand only.

Figure 3.7.18 Net showing signs of wear after seven blasts.

Figure 3.7.19 Damage to net from blast.

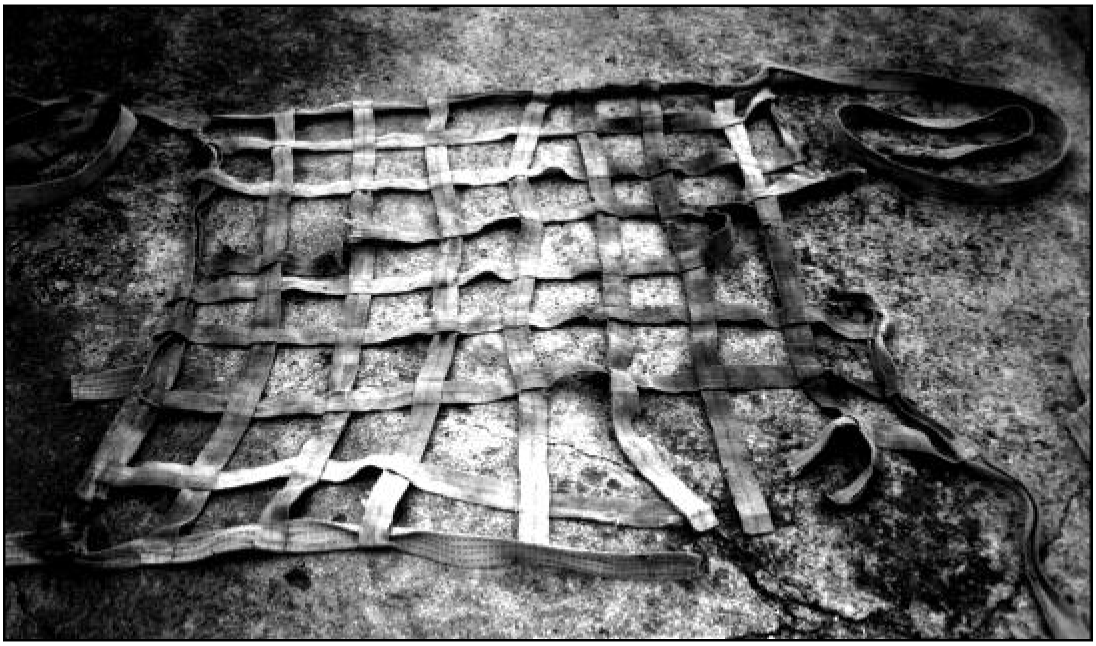


Figure 3.7.20 Note perimeter webbing broken as a result of having been caught by the scraper.

Advantages

1) Easy to transport.
2) No bolting or pinning of the roof is required.
3) High areal coverage at the stope face.
4) Easy to install.

Disadvantages

1) The net can be damaged by scrapers.
2) It offers no additional support resistance.

3.7.9 Fortrac Geogrids (Syddell, 1998)

3.7.9.1 Introduction

The Fortrac Geogrid comprise high strength polyester yarns (see Figure 3.7.21) woven into an interlocking pattern and then coated in plastic mesh. These nets have been used in civil engineering for road and runway construction and they are now also used for longwall and roadway support moves in coal mining. The use of the Fortrac Geogrids in Australian coal Mines is discussed below.

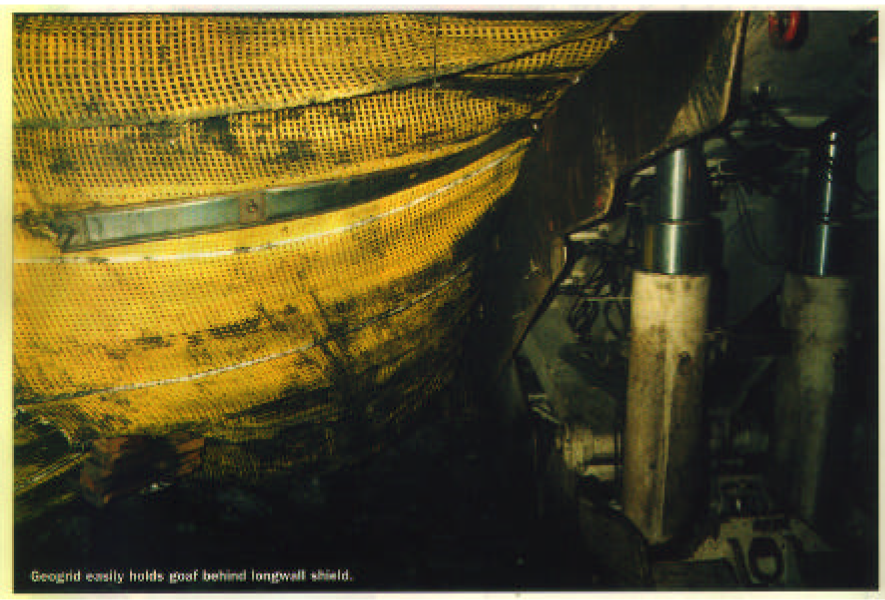


Figure 3.7.21 The Geogrid easily holds the goaf behind the longwall.

Figure 3.7.22 Rolling up the mesh on surface at Springvale colliery.

3.7.9.2 Discussion

The lengths of the Geogrids are approximately 100 m (see Figure 3.7.22). They are assembled into a "one-piece carpet roll" which is tailor-made for each longwall, depending on local conditions and goaf material. High strength nylon ropes are used instead of steel ropes to reinforce the mesh. This makes it easier to install. A small diameter steel rope is used in the installation process to keep the mesh straight and to ensure that it is positioned correctly. The carpet roll is folded into a transportable length, approximately 6 m by 1,5 m by 1 m, and delivered to the mine site. The mesh can be pre-loaded onto underground transport vehicles so that there is no need to reload at the mine site. The assembly is then pulled across the faceline using the shearer. The process takes approximately 20 minutes. The mesh is pinned to the roof by the canopies, so there is no need to bolt the mesh to the roof. When each cut is taken, the mesh is held tightly against the canopies, the winches are released (see Figure 3.7.23), the roll is dropped down, the zip ties cut and the support advanced.

Figure 3.7.23 View along longwall showing the boat winches; every second support positioning the mesh roll tightly against the canopies.

The time taken from pulling the roll onto the face to the next cut is approximately two to six hours, depending on the face conditions. It is suggested that several days can be saved when compared to the use of conventional methods and materials.
The Geogrid provides a strong yet flexible restraint for the goaf material, especially during the pull-off phase. This allows efficient recovery of the longwall support. The Geogrids have also improved transfer times significantly, and have also proved to be popular with the longwall operators who find the system easy and safe to operate. The system has been successfully used at seam heights of 1,8 m to 4 m.

Advantages of Fortrac Geogrid for longwall coal mining

1) Ease of transport.
2) Speed of erection.
3) Safety is improved.
4) The assembly requires no bolting or pinning the roof.
5) The mesh has a tensile strength of 20 ton per linear metre, which is 10 times that of comparable mesh systems.
6) The mesh improves recovery times.

7) The mesh system is designed to suit the needs of each longwall.
8) Durability of the system (the system is proven in thick seams).

Disadvantages

1) The system is used with powered support, which is not cost effective.
2) Labour intensive.

3.7.10 Modified stope support system using hydraulic props

3.7.10.1 Introduction

There are several support systems used in the stope face area which can reduce the rockfall hazard or rockburst damage. These include props of various types, which are used in conjunction with headboards as discussed above. The common problem in the use of prop face support systems is cleaning of the stope face after blasting.

Removable props are installed in the stope face area after the cleaning shift and removed prior to the blast. Removable props are the most common form of face area support systems. The blast-on props option, in which the props are installed 1,0 m from the stope face and remain in the stope face during the blast and subsequent cleaning by waterjet assisted face scraping, will be discussed in this section.

3.7.10.2 Discussion

20/40 t blast-on hydraulic props with 800 mm long headboards

These support systems are designed primarily for the control of rockfalls in the stope face area. They consist of 20/40 t blast-on hydraulic props with 800 mm long headboards, which provide a high degree of areal coverage and reduce prop blast out. The headboards are installed slightly up dip of strike in order to cover the face parallel fractures and this allows the headboards to point into the blast. This type of face support system is always in place and has the potential to reduce dilution and to control stoping width.

This support system comprises three rows of props spaced 1,0 m apart on strike and 1,5 m apart on dip. The face support system is installed diagonally as shown in Figure 3.7.24 to allow for cleaning and drilling. The permanent support system consisting of profile props is installed behind the first three rows of the 20/40 t hydraulic props. Mat packs 1,1 m^2 in size are installed on the shoulder of the gully. The gully is supported by shepherd’s crooks installed 1,5 m from the gully face on a strike and dip spacing of 1,0 m at 0,5 m from the gully shoulder.

Diagonal barricades are installed between the second and third rows of hydraulic props. The barricades prevent the rock, blasted from the face, being thrown into the back area. A water jet is used to move the rock into the path of the scraper operating along the stope face.

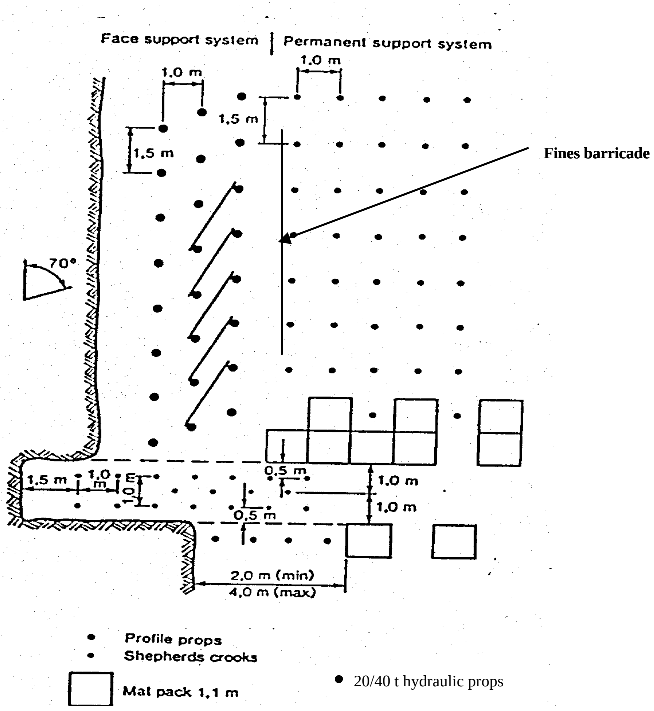


Figure 3.7.24 ***20/40 t blast-on face support system (after Glisson and Roberts, 1991).***

Advantages

1) They provide high degree of areal coverage.
2) Reduced prop blast out.
3) Props remain in the stope face area at all times actively assisting in strata control.
4) Reduce dilution.
5) Do not disturb the hangingwall until the props are removed at the back row.
6) Capability of rapid yielding during rockburst.
7) Require less time for installation.
8) Easy to handle.
9) Good control of fly-rock.
10) Efficient cleaning and sweeping.

3.7.11 Rock tendon support system

3.7.11.1 Introduction

The support system that incorporates tendons such as rock bolts, cable anchors, etc. is widely used in tunnels. The system is usually used in conjunction with mesh and lace or shotcrete support in blocky rock mass conditions. A few mines are using tendons as stope face support. The principle that most of the support systems involving tendons are based on is that the tendons are anchored in stable rock above the unstable strata. This system is however not suitable for highly fractured zones such as the Carbon Leader stopes. Some tendons have been tested in the laboratory and have shown that they have energy absorption capabilities (Cone bolt). There are, however, installation problems in low stope widths due to lack of space to drill the tendon holes.

Advantages

1) Provide good areal coverage.
2) It have energy absorption capability.
3) Can be used with big base plates.
4) Work well in low to moderately jointed rocks.
5) Do not interfere with stope cleaning.

Disadvantages

1) Installation problems in low stope widths.
2) Dangerous in low stope widths (workers can injure themselves against protruding bolts).

3.7.12 Inflatable support (Wojno *et al.*, 1998)

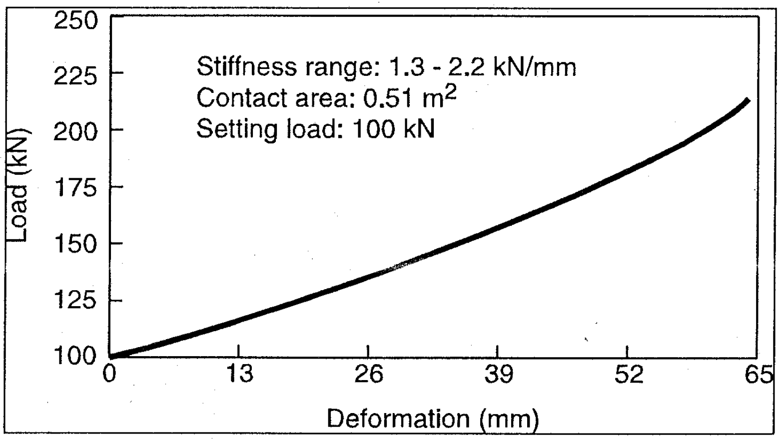
3.7.12.1 Introduction

It has become apparent that one of the essential requirements to improve the safety and stability of highly fractured stope gullies is the better control of support characteristics of timber packs.

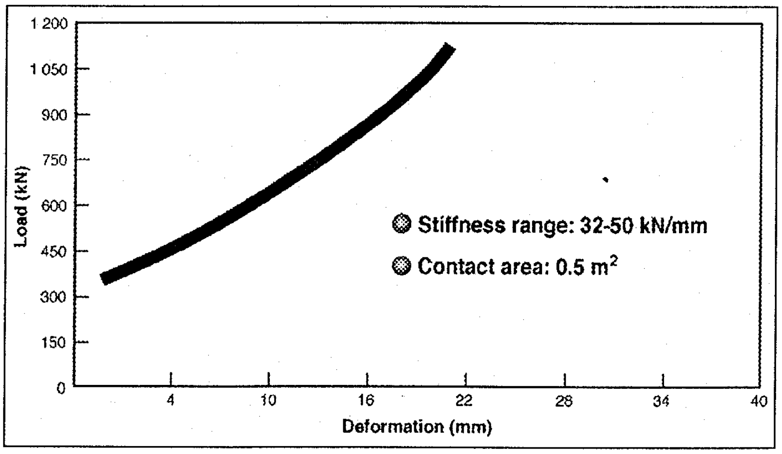
3.7.12.2 Discussion

Squelch *et al.* (1995) recommended a force-deformation curve for a deep mine gully support. The pack force should rise slowly with increasing stope closure up to a maximum of 1000 kN and should remain stable until about 700 mm of stope closure has been reached. It is from this force-deformation curve that a new type of support is being developed, namely inflatable packs.

The inflatable system has not yet been tested underground, but the results from the laboratory tests are presented and the envisaged advantages and disadvantages are listed. The load-deformation characteristics of two types of prototype inflatables are shown in Figure 3.7.25 (a) and (b). The air-filled support offered a rather soft performance with its stiffness ranging from about 1.3 kN per mm to 2.2 kN per mm deformation of the inflatable unit. The water filled unit offered much higher stiffness ranging from about 32 kN per mm to 50kN per mm deformation. This support characteristic compares very favourably with the load deformation curves of a number of typical gully packs with stiffness ranging from approximately 4 kN/mm to 16 kN/mm.



(a)



(b)

Figure 3.7.25 **(a) Performance characteristic of the air-filled support unit.**
(b) Performance characteristic of the water filled support unit.

Envisaged Advantages

1) Improve stability of gully shoulders.
2) Improve face conditions.
3) Better stope width control.
4) Increased face advance.
5) Reduce dilution.
6) Reduce number of lost blasts.
7) Reduce ventilation requirements.
8) Re-useable units.
9) Reduce support labour.

Envisaged Disadvantages

1) Susceptibility to sudden pressure loss due to bursting.
2) Could be punctured by fly-rock.

3.8 Conclusions and Recommendations

To summarise, the objective of enabling output 2 is the identification and classification of strata conditions and thereafter the determination of the most suitable temporary support systems for particular strata conditions. The term “strata” condition is defined as the hangingwall rock mass condition (manifested by the degree of discontinuity) and its integrity when considering the impact on choice of temporary support.

Mining environments and conditions are classed as shallow and intermediate/deep using various criteria. Strata conditions for each of the environments are divided into various classes using rock type, UCS, reef parallel structures, reef perpendicular structures and mining induced fracturing. The use of classes is necessary to preclude the possible inference that some strata conditions are safer than others. Support types are assigned to the various classes of strata conditions after consideration for stoping width and rockfall or rockburst conditions.

The use of tendons is generally recommended for strata conditions entailing a strong hangingwall, minimum hangingwall fracturing, but with problematic roof parallel discontinuities. In other strata conditions appropriate columnar support types with adequate areal coverage is recommendable. In particular, the application of complete areal coverage in conditions such as weak and fractured hangingwall is strongly suggested.

The hangingwall rock mass typically has a complex, variable nature resulting in varying degrees of discontinuity and hence changes in strata conditions. The experience and knowledge of rock mechanic practitioners is an important and essential input in determining the nature of the parameters influencing strata conditions.

The investigation into alternative support technologies has shown that systems such as the twin beam support system, the safety net, and large headboards are suitable and practical to reduce the rock related hazard at the stope face. Other technologies (e.g. powered support, shields) have the potential to significantly increase worker safety, however at this stage their application in a narrow, tabular hard rock stoping environment requires further research endeavours.