2 Review of temporary support practice and systems

2.1 Literature review

2.1.1 Immediate face area support requirements

The problem of stope support has been shown (Cook, 1966) to depend upon the effectiveness with which the fractured rock, especially in the hangingwall, can be kept in place to reduce the amount of damage and the production delays resulting from rockfalls and rockbursts. The primary function, therefore, of an immediate stope face support system, is to maintain the integrity of the fractured hangingwall beam and thus prevent falls of loose rock for a specific period. This is achieved by applying sufficient force to the immediate hangingwall to generate frictional forces between individual segments of the hangingwall beam, to a lesser extent by restricting bed separation, and by directly supporting any loosened blocks. The nature and extent of the generated forces will greatly depend on the local ground conditions. The low closure rate in shallow stoping conditions requires support that can exert a resisting force on the hangingwall and footwall at small deformations. In deep stoping conditions, the support is required to exert clamping forces on the immediate fractured hangingwall before loosening of any inherently unstable blocks can take place. It is therefore generally required that, irrespective of mining depth, initial forces are rapidly generated by support systems. This opinion is shared by a number of researchers (Gay and Jager, 1987; Roberts *et al.*, 1993).

The support system in the face area must, in addition, be able to provide areal coverage to prevent falls of loose rock. This is particularly important where discontinuities delineating unstable keyblocks are unfavourably oriented. Ideally, therefore, the support spacing should be less than the smallest significant keyblock, even if the reactionary forces afforded by the support units are sufficient for a wider spacing of support units (Jager and Roberts, 1986).

Further requirements of immediate face area support systems are resistance to blast damage, availability, and non-interference with face drilling and cleaning operations.

The nature of loading anticipated in the stope face area will greatly influence the magnitude of the force exerted on the support system. Two types of loading conditions, namely rockfall and rockburst criteria, are generally identified. The rate and amount of closure acting on the support unit and the associated force deformation characteristics of the unit determine the reactionary force exerted by the support element. Several support types are available that can be pre-stressed during the installation process, thereby providing immediate active support.

2.1.2 Support resistance required to control rockfalls in the vicinity of the stope face

The prevention of falls of ground in the stope face vicinity relies on the generation of a support resistance to stabilise the hangingwall. The determination of the required support resistance criterion was undertaken by back analysis of falls of ground (Roberts, 1995). The analysis indicates that 50 % of falls are less than 0,45 m thick, 85 % are less than 1,0 m and 95 % are less than 2,0 m. The support resistance (density of rock * ejection thickness * acceleration due to gravity) required to support a tabular block of rock 2,0 m thick is 54 kN/m^2. COMRO (1988) recommended that a support resistance of 50 kN/m^2 be taken as a support resistance criterion, or requirement to prevent the majority of rockfalls in the stope face area. In further work (Roberts, 1995), the rockfall criterion was modified to reflect reef specific support resistance criteria (see Section 2.1.4).

The support system is evaluated once the support resistance criterion has been set. This is done based on the tributary area theory, which is applied to determine if the load bearing requirements of the support system are met.

2.1.3 Determination of the rockburst energy absorption criteria for the stope face area

In stopes subject to seismicity and rockbursts, a general energy absorption criterion has been applied, which requires that the support system should be capable of absorbing 60 kJ of energy per square metre of hangingwall. As a consequence, the support system must have a yielding capability. The basis of this criterion was a support resistance of 200 kN/m^2, required to arrest the hangingwall displaced through 0,3 m at an initial velocity of 3,0 m/s during a rockburst and in the process absorb 60 kJ/m^2 (Roberts *et al.*, 1993). Therefore, the energy absorption capacity of stope support system needs to be evaluated against the energy absorption requirement of 60 kJ/m^2 (COMRO, 1988). As in the case of the rockfall criterion, the energy absorption requirements for specific ground control districts have recently been modified (see Section 2.1.4).

2.1.4 Current estimate of support resistance and energy absorption requirements

Roberts (1995) made use of a comprehensive accident database recording all rock related fatalities on the gold mines since 1990. Cumulative percentage fallout heights were determined and a criterion set such that the support system caters for 95 % of all rockfalls. The criterion has been updated (Daehnke *et al.*, 1998) to include more recent fatality data, and Table 2.1.1 shows the fallout thickness for the 95 % frequency level, i.e. 95 % of all falls were the indicated thickness or less. The table also shows the associated support resistance criteria.

Table 2.1.1 Fallout thickness for the various reefs at 95 % frequency level.

Reef type	Fall-out thickness	Support resistance
Carbon Leader Reef	1,0 m	27,5 kN/m^2
VCR	1,2 m	31,8 kN/m^2
Vaal Reef	1,2 m	31,8 kN/m^2
Basal Reef	1,8 m	47,7 kN/m^2

Using block ejection thicknesses for the different reefs, the minimum energy absorption requirement per square metre of stope hangingwall, that a support system should provide to stabilise the stope hangingwall in 95 % of cases, has been suggested by Daehnke *et al.* (1998). The ejection velocity is assumed to be 3 m/s and it is further assumed that, during the dynamic event, the hangingwall displaces 0,2 m downwards.

Therefore:

$$\text{Energy absorption criterion} = 1/2mv^2 + mgh \qquad 2.1.1$$

where:

m = 2700 kg * ejection thickness
v = 3,0 m/s
h = 0,2 m
g = 9,8 m/s^2

The energy absorption criterion is detailed for specific reefs in Table 2.1.2.

Table 2.1.2 Ejection thickness for the various reefs at 95 % frequency level and the associated energy absorption criteria.

Reef type	Ejection thickness	Energy absorption
Carbon Leader Reef	2,2 m	38,4 kJ/m^2
VCR	1,8 m	31,4 kJ/m^2
Vaal Reef	1,2 m	20,9 kJ/m^2
Basal Reef	2,6 m	45,4 kJ/m^2

Further criteria that need to be met by rockburst resistant support systems are:

1. To ensure post-rockburst stability, the load carried by the support unit after the rockburst should exceed the dead weight of the tributary area supported by the unit.

2. The post-rockburst stoping width should exceed 0,6 m to allow movement of, and prevent injury to, mine personnel.

2.2 Classification of immediate face area support systems

The classification of face area support systems is based on the method of stabilising the rock adjacent to the excavation. The first category, termed external support, is taken to include all methods, which essentially provide surface restraint to the rock mass by installation of structural elements on the excavation boundary, i.e. the application of a reactive force at the surface of the excavation. The second category, termed rock reinforcement, includes methods which modify the internal behaviour of the rock mass by installation of structural elements within the rock mass.

2.2.1 Classification of external support systems used in the immediate face area

The load deformation behaviour of support units is used in the classification of external support systems. On this basis, three broad types are identified as timber props, mechanical props and hydraulic props.

2.2.1.1 Timber props

These props are generally machined timber poles of diameter 150 mm to 200 mm with one or both ends modified, either by tapering or by grooving, to allow them to yield at a controlled rate. Figure 2.2.1 shows the range and difference between laboratory and underground force-deformation curves for various types of yielding timber props. The curves show that in the laboratory the props display good initial stiffness (16 to 64 kN/mm) followed by high levels of deformation at a fairly constant yield force (300 to 600 kN). The actual *in situ* (underground) support resistance of this type of support is lower than that calculated from laboratory tests.

Despite the lower *in situ* performance, timber props provide adequate support in the immediate face area under normal conditions. Other advantages include ease of transport and installation, and non-retention of gold-bearing fines. As a result, timber props have been used extensively in the past in narrow reefs, i.e. reefs less than 2,0 m thick (Gay and Jager, 1986).

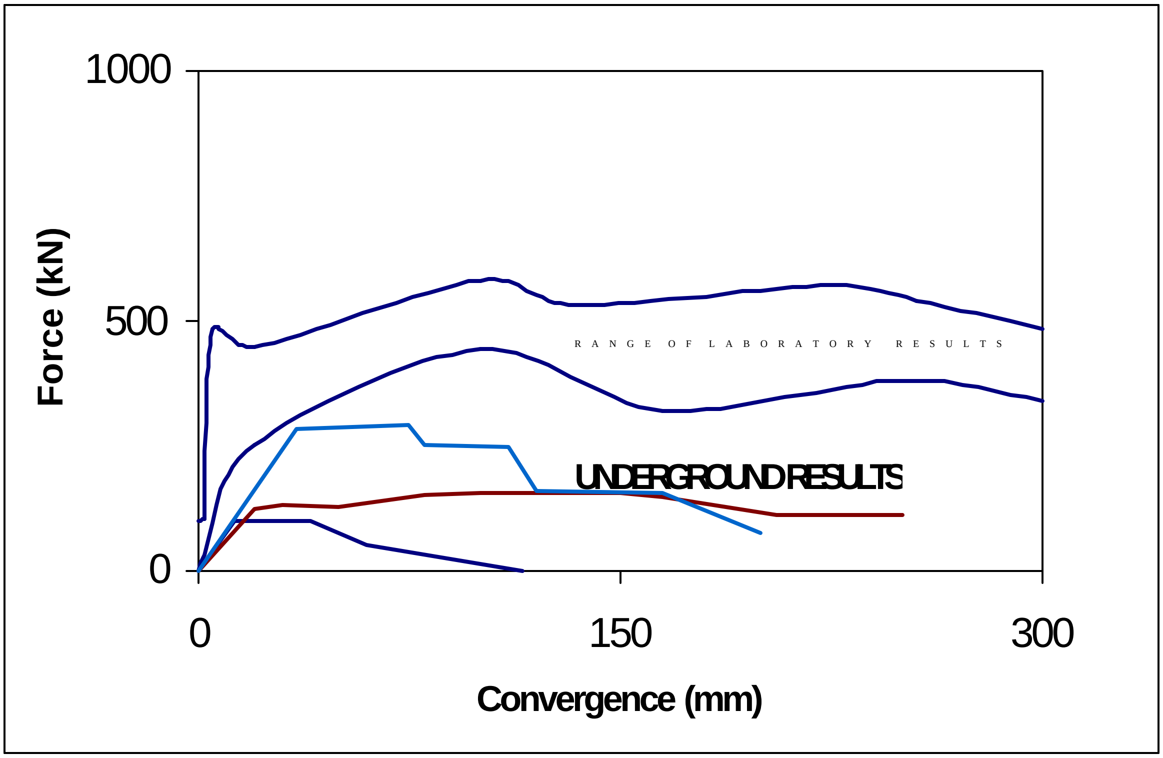


Figure 2.2.1 Force-compression curves for yielding timber props as determined in laboratory tests and underground (after Jager and Roberts, 1986).

2.2.1.2 Mechanical props

The use of mechanical props (instead of mine poles) for temporary face support has increased in the past, especially in the West Rand mining district, where convergence rates are low (Gay and Jager, 1986).

Some of the advantages of these props over mine poles are that they are reusable, they are not left behind as the face advances but are moved forward regularly, and that they can apply initial setting forces of 50 to 100 kN to the hangingwall. Mechanical props incorporate a load release device for worker protection, which allows the prop to be retracted remotely when subjected to loads of up to 150 kN. The major drawback to these props is their limited yieldability (see Figure 2.2.2). Disadvantages of the mechanical prop are that they cannot yield and can become trapped in high closure areas, and they cannot absorb significant amounts of energy therefore they cannot contribute to the energy absorption capacity in rockburst support.

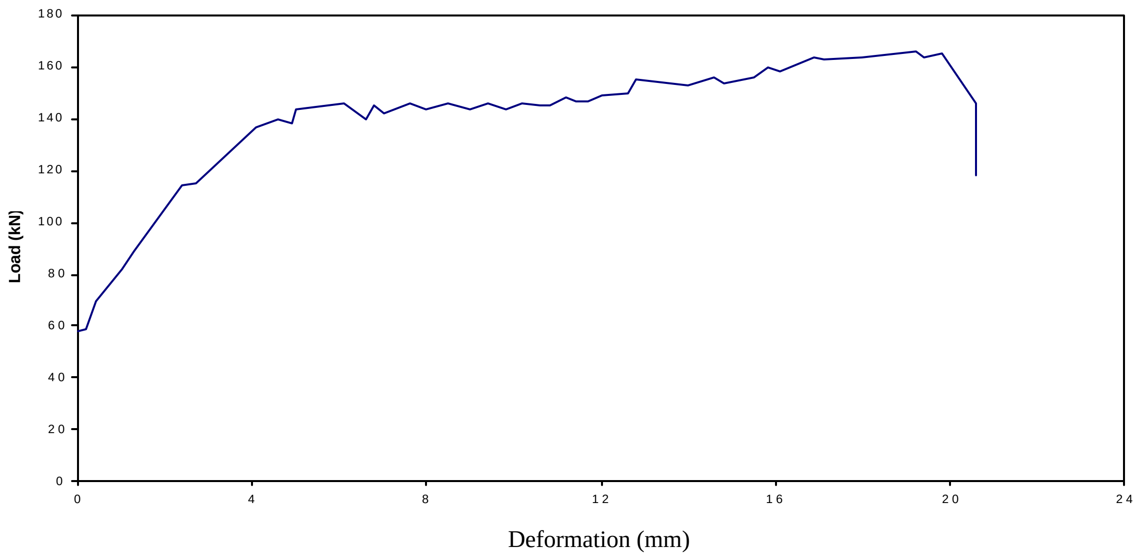


Figure 2.2.2 Typical force-compression curve for an SS7 Mechanical prop.

2.2.1.3 Hydraulic props

Hydraulic props as a face area support system were primarily developed for use in rockburst-prone mines due to their energy absorption capability and rapid yielding characteristics. The early variety of these props were designed for a setting force of 400 kN and a yield force of 400 kN at a closure rate of up to 1,7 m/s. Current designs of hydraulic props allow a setting force below 200 kN and a yield force of 400 kN at closure rates of 3 m/s. In intensely fractured hangingwall conditions, the use of hydraulic props without an appropriate headboard could cause hangingwall punching (Jager and Roberts, 1986).

2.2.2 Classification of rock tendons

2.2.2.1 Introduction

Tendons refer to any form of reinforcement in the form of a bar, cable or tube inserted into boreholes in the rock mass and anchored by grouting, friction or mechanical means.

A tendon installed in a borehole is subjected to larger displacements closer to the skin of the hangingwall than at a point some distance into the rock mass. The difference between these displacements results in a tensile force in the tendon. If the tendon is coupled to the rock, either at both ends or along its entire length, then it will actively reduce these dilations by holding the rock together.

Another mode of operation of the tendon is suspension. The entire tendon can be tied back to relatively stable ground deeper into the rock mass. In this function, loads are applied through transfer from the bulging rock in the following manner:

1. Rock movement, which requires load transfer from the unstable rock to the reinforcing element.

2. Transfer of load via the reinforcing element from the unstable surface region to a stable interior region.

3. Transfer of the reinforcing element load to a stable rock mass.

On this basis, three categories of tendons are identified. Theses are:

- Continuous mechanically coupled tendons (CMC) – fully grouted bars and cables.
- Continuous frictionally coupled (CFC) – Split Sets and Swellex bolts.
- Discrete mechanically and frictionally coupled (DMFC) – Mechanical or resin end-anchored bars and cables.

2.2.2.2 Continuous mechanically coupled (CMC) tendons

These types of reinforcing elements rely on a securing agent, typically cement grout or resin, which fills the annulus between the element and the borehole wall. Loads are hence transferred from the rock mass to the tendon via the grout along the entire length of the tendon. Reinforcing elements used in conjunction with grouts often have variable cross sectional shapes along the length so that a geometric interference is created, thus increasing the strength of the grout-tendon bond.

They can be described as passive but stiff support units requiring deformation of the rock to generate support forces. The most widely used type in South African mines is the Shepherd's Crook (i.e. a smooth or rebar bent into a shape) and is mostly used in tunnels. Figure 2.2.3 shows a load-deformation curve of a resin-grouted rebar loaded in tension across a joint.

Certain types of CMC can be pre-tensioned. Typically this would be a resin bolt where a fast setting resin capsule would be inserted first to end anchor the bolt followed by slow setting resin capsule until the hole is full. The bolt can then be pre-tensioned before the slow setting resin hardens. It is believed that pre-tensioning enhances the integrity of the rock mass. Further, the reinforcement system will be active and stiffer when subjected to load (Stillborg, 1994). Shepherd's Crooks and cable bolts are generally not tensioned. They have little yield capability, as strain in the tendon is concentrated in the area providing the deformation (e.g. a dilating fracture).

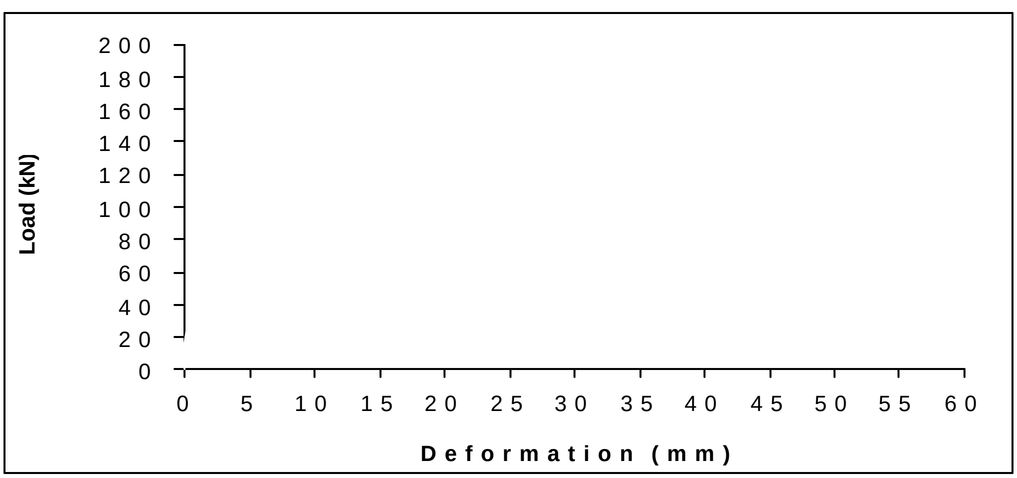


Figure 2.2.3 Resin grouted rebar – tensile loading across a joint (after Stillborg, 1994).

2.2.2.3 Continuous friction coupled (CFC) tendons

With these types of tendons, the reinforcing element is placed in direct contact with the rock. This means that they are coupled by a frictional interface between the rock and the tendon itself. They have a radial pre-stress, which provides a force normal to the friction interface. The most commonly used types are the Split Set and Swellex systems. The Split Set is an oversized split tube forced into an undersize hole, while the Swellex rockbolt is an undersized element expanded into an oversize hole by water pressure.

Figure 2.2.4 illustrates a typical result from a tensile loading test across a joint for a Split Set. It can be seen from the graph that the bolt starts to slide at about 50 kN after the frictional resistance is exceeded. The sliding is preceded by no measurable rockbolt deformation. The strain is distributed over a significant length of the unit and the whole unit can slide, thus providing some degree of yieldability. However, the amount of yield depends on rock conditions and borehole geometry. It should be noted that the yield range of the split set is limited as applied load cannot be maintained after rockbolt deformation.

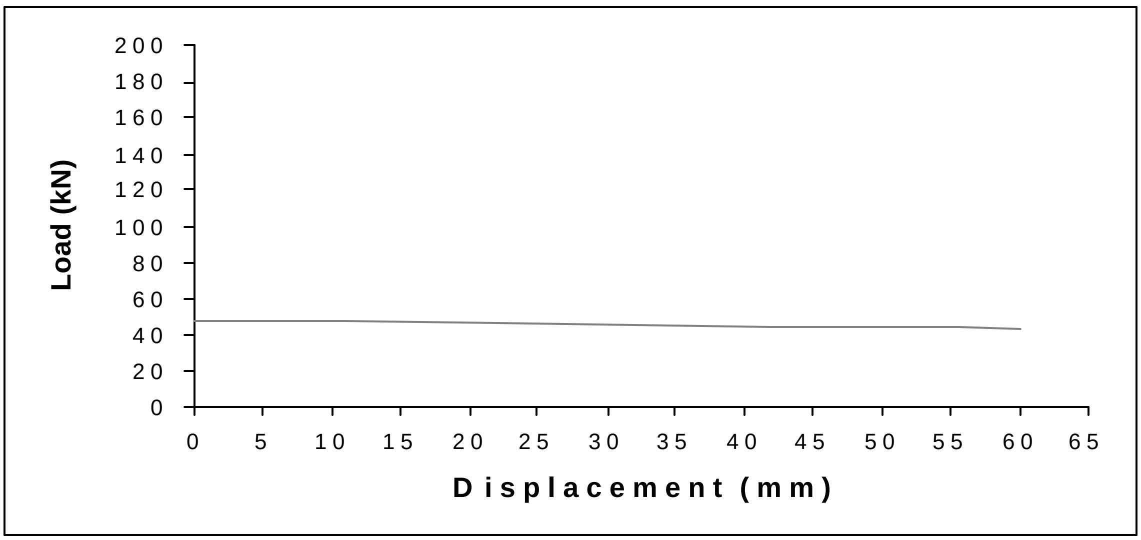


Figure 2.2.4 Split Set, type SS 39 – tensile loading across a joint (after Stillborg, 1994).

2.2.2.4 Discrete mechanically and friction coupled (CMFC) tendons

Discrete mechanical and friction coupled tendons are coupled to the rock over a discrete length of the tendon, usually concentrated on or located at the end of the tendon within the rock mass. This coupling, which must be sufficiently strong to mobilise the full strength of the reinforcing element, is either in the form of some sort of mechanical device, e.g. an expansion shell (discrete friction coupled), or by means of grout, usually resinous, at the anchoring end (discrete mechanical coupled). An integral part of these devices is a base plate or bearing plate, which allows load to be transferred from the rock to the tendon. Such tendons are active in that they can be tensioned upon installation. Examples of CMFC tendons are the expansion shell bolt, end anchored cable bolt, resinous plain bar, deformed bar, etc. Strain generated in the tendon by dilation of the rock mass can be distributed over the full length of the tendon, thus providing some degree of yieldability.

Often, however, due to deficiencies in material, manufacture or handling, weaknesses occur at positions along the bar where strain can be concentrated and necking occurs, which leads to premature failure.

A load-displacement curve of a typical expansion shell rockbolt is shown in Figure 2.2.5.

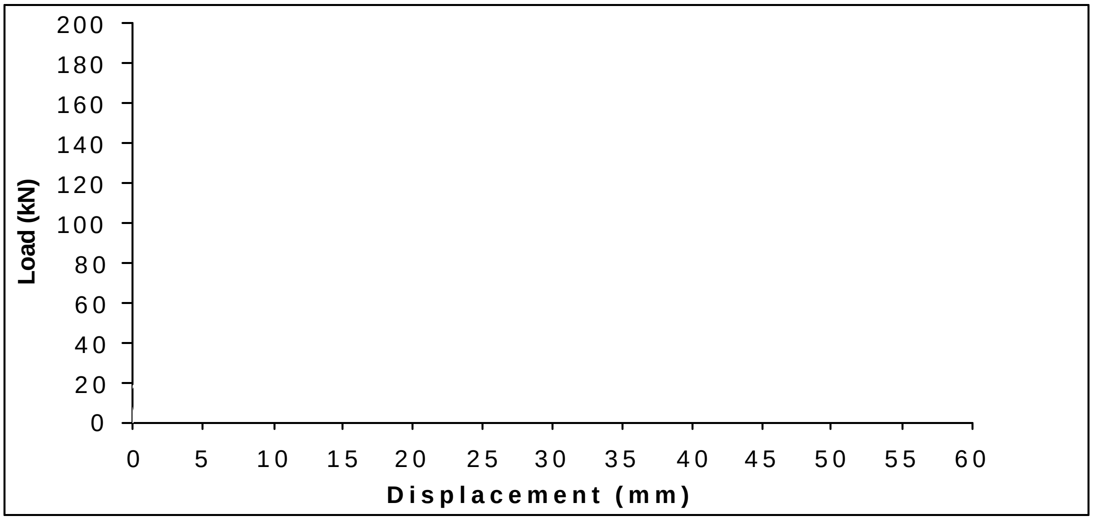


Figure 2.2.5 Expansion shell anchored rockbolt - tensile loading across a joint (after Stillborg, 1994).

A special type of CMFC is the conebolt. The principle on which this grouted unit is designed is that a protuberance at the end of the bar is drawn (by dilation of the rock mass anywhere along the supported thickness) through the smaller diameter hole left in the grout by the bar. This causes crushing of the grout and frictional resistance, resulting in a high yield force. Provided the borehole is not sheared by more than the annular gap between the borehole and the bar, conebolts are capable of yielding by large amounts.

2.3 Overview of currently available temporary and face area support systems

2.3.1 Introduction

Several visits and consultations were made with relevant personnel, particularly rock engineering staff, in the Carletonville, East Rand, Bushveld and Klerksdorp mining districts. The objectives of the consultations were firstly to acquire a general overview of current temporary and face area support practices, with the view to establishing their design considerations and performance requirements. Secondly, the objective was to identify mines requiring more detailed subsequent visits as far as rockbolting in the stope face area is concerned. A checklist was employed in the review. Details of these consultations are summarised in Appendix 1.

The following are some salient findings from the visits:

2.3.1.1 Design criteria

It appears that energy absorption and support resistance are the criteria most often used in the design of face area support systems. Different mines use different energy absorption and support resistance criteria (Appendix 1). The depth of instability used in defining the thickness of the unstable beam to be supported in the gold mines is mostly historical, i.e. from accident statistics. A further approach is to use the depth of parting planes above the reef contact as the depth of instability. As local ground conditions vary from one area to the other, even for the same rock type and for the same mine, it would be expected that the criteria, and for that matter the support requirements, should be varied accordingly. This is generally not the case, as the process of properly delineating ground control districts has not been fully accomplished.

What appears to warrant a change in choice of support for the same reef type and mine is a change in stoping width (see Appendix 1). The current rating of stopes at Vaal Reefs according to their risk potentials (with the view to appropriate spacing of support units) comes closer to solving the variation in local ground conditions. A Risk Assessment Programme, developed for Amandelbult and used in identifying high-risk areas requiring special support specifications, is recommended.

2.3.1.2 Factors influencing choice

In addition to the above, cost of temporary and face area support systems, ease of integration into the mining cycle, ease of transportation and ease of installation are some of the factors influencing the choice of support systems. For instance, at Kloof Gold mine there is now only a limited use of hydraulic props due to punching of the hangingwall, failure of valves and difficulty in maintaining the props. The present shift to contractors to installing and maintaining hydraulic props has reduced this problem. There is also a shift from conventional Camlock props towards the use of yieldable units (e.g. yieldable Camlock props, Ebenhauser props, etc.), due to buckling experienced with the former. On some mines, for example Libanon, Kloof and Elandsrand, the use of rockbolting as a stope face support system may be feasible, but has not yet been tried and implemented due to the negative worker perception of tendons in the stope face (refer to Appendix 1). Thoroughly planned implementation strategies, involving full participation of the work force and field trials, should be considered for such situations.

2.3.1.3 Types of temporary and face area support

From the survey, the most commonly used temporary and face area support units are the following:

- Mine poles
- Mechanical props
- Hydraulic props
- Elongates
- Tendons (very limited)

Mine poles are mostly not removed before the blast. Due to lack of pre-stressing and the comparatively limited hangingwall deformation before the blast, minepoles are often blasted out. Night shift may also enter and work under large unsupported roofs. Minepoles should be used in conjunction with headboards, especially where the hangingwall is friable or highly discontinuous (this is stipulated in most codes of practice). Pre-stressable mine poles have found wide application in the platinum mines as both temporary and permanent support.

Since mechanical props are not left behind as the face advances, but are moved forward regularly and are thus cost effective, they are used extensively in mines were convergence rates are low. Their limited yieldability has, however, restricted their use in some deeper mines (for example, Kloof mine). Yieldable Camlock props are now available and are currently being tried on some mines (for example Elandsrand and Kloof mines).

Rapid yielding hydraulic props are mostly used in the intermediate- to deep-level gold mines due to their yieldability and preloading abilities. It was observed that some mines, who had used hydraulic props extensively before, are now moving away to other forms of support. The reasons given for this are their comparatively high mass, installation difficulty, potential losses, and difficulty in maintenance. In some mines, hydraulic props are used as the only support in the immediate face area and in others it is behind other units (e.g. mine poles).

Elongates used as face area support are mostly the pre-stressable types (pre-stress range is between 120 to 250 kN). Elongates are used in some mines as the only temporary or permanent support (mostly in the platinum mines), whereas they are used in some gold mines to support the stope when the primary temporary support exceeds a certain distance from the face. For instance, at Mponeng Mine pre-stressed elongates are used as a face support system once the distance from the face to permanent support exceeds 2,6 m, otherwise Camlock props are used.

Rockbolts are hardly ever used as the only immediate stope face area support system. Exceptions, however, do exist (Kloof Gold Mine).

2.3.1.4 Support density and areal coverage

The designed dip and strike spacing between individual temporary support units and their load carrying capabilities appear to meet the set design criteria in most mines. If the spacing between these support units, and between them and the face, increases, the load on the units will increase, exceeding their designed yield force capacity and in the event of a rockfall or rockburst, the supports may fail. The likelihood of unfavourably orientated keyblocks to fall will increase if barring is not done properly. Because most temporary supports are replaced by permanent types, little attention is given to the proper installation and spacing of permanent support (see Figure 2.3.1). What seems to define the spacing is the amount of units available, number of workers present, the time of shift left for drilling, etc.

Analysis of accident data in the gold mining industry during the period 1990 to 1996 indicates that, for the support units associated with rockburst fatalities, a total of 207 failed and 841 were ineffective (i.e. could not hold up the hangingwall). Similarly, a total of 54 support units failed and 247 units were ineffective during rockfalls. This means that the number of support units that failed as a result of excess loading is relatively small. Individual support units can therefore be assumed to be sufficiently strong and/or yieldable to accommodate the majority of rockfalls and rockbursts. In severe rockburst conditions, however, it has been found that a high percentage of support units have failed.

It can therefore be concluded that the main cause of support failure may not be attributed to the inadequacy of support units. The most obvious cause for failure of the support system appears to be lack of areal coverage and too widely spaced support units.

The understanding of the behaviour of the immediate stope hangingwall is essential in determining the likelihood of rockfalls and extent of damage should rockbursts occur. The most crucial aspect of this behaviour is the integrity (coherence) of the hangingwall. Obviously, a totally solid hangingwall does not require any support, while an intensely fragmented hangingwall may only be stabilised if 100 % coverage can be provided.

The current support design methodology does not account for support areal coverage. The ideal temporary support system is one that will provide or incorporate areal coverage. The temporary support systems currently in use mostly employ headboards and similar forms of load spreaders as a form of areal coverage. They are often applied in such a way that the direction of these devices is perpendicular to the typical mining induced, face-parallel fractures and, therefore, efficiency of such spreaders seems to be maximised. These headboards do not span the entire distance between support units and therefore their effectiveness decreases with increasing hangingwall friability. In many cases their load carrying capacity can also be questioned. Belt straps (75 mm wide) tied between elongates were once used with some success at Elandsrand gold mine to minimise the unraveling of blocky hangingwalls in the stope face area (Banning, 1999).

2.3.1.5 Installation and removal

Good contact of the support with the hangingwall and footwall, and at the correct angle with respect to unstable blocks (i.e. perpendicular to the dip of strata), is important, particularly in the immediate stope face area. Good initial stiffness addresses the former. However, because the immediate face area is where movement of people is greatest, the unintentional interference with these units is inevitable. It has been observed that in some cases support units are installed on loose rocks on the footwall or against loose hangingwall rock. The deleterious effect of this is that, in the case of a seismic event, they may be dislodged by vibration and therefore there will be nothing in place to actually restrain the rock movement following the event.

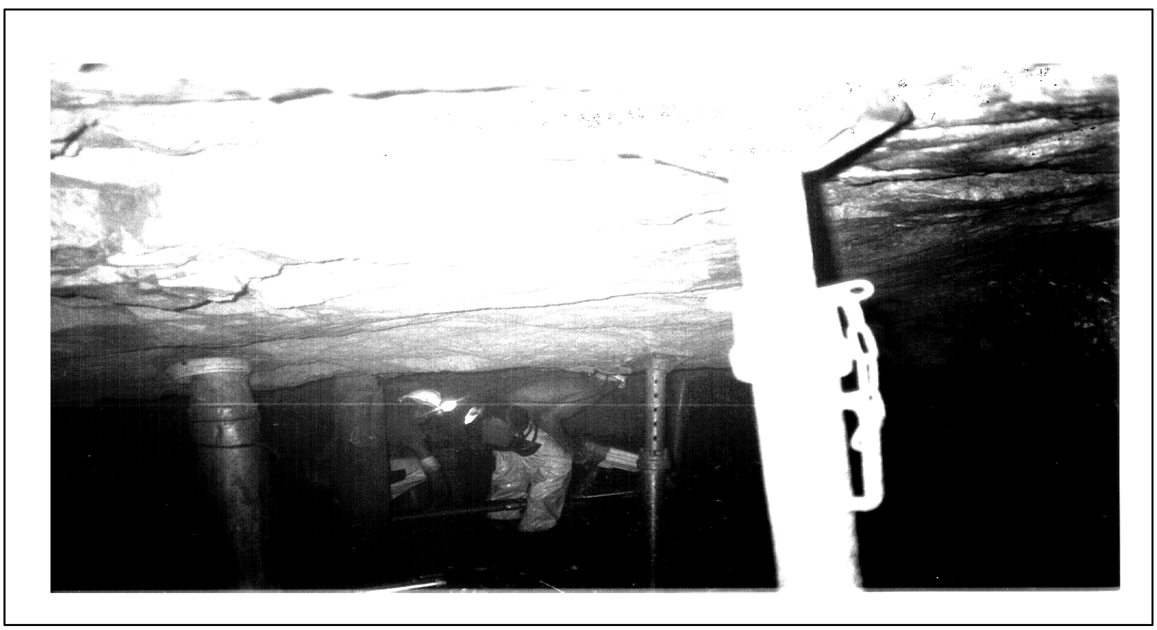


Figure 2.3.1 Temporary support spaced as per mine standards but without areal coverage under blocky ground conditions.

It was found during the survey that the ease of installation and removal of temporary support units is very important for the successful implementation of these units. For example, workers may prefer to use mechanical props or mine poles as opposed to rapid yielding hydraulic props, even when these types of units are inappropriate for the strata conditions. From personal knowledge, two separate fatalities occurred within a period of one month on a mine because correct procedures were not followed in removing hydraulic props. Since then the perception about the use of hydraulic props in those sections of the mine has been negative.

2.3.2 Current application of rockbolting in the immediate face area

The use of rockbolting as face area support is very limited in the gold and platinum mines as compared to the usage of external support systems (such as mine poles, hydraulic props, Camlock props, etc.). Evander, Western Platinum, Vaal Reefs, Randfontein Estates and Beatrix mines use rockbolts in the stope face area at varying distances from the stope face and at different spacings. The rockbolting practices of these mines are reviewed below.

2.3.2.1 Evander Mine (Kinross)

There exists a 20-30 cm thick beam that may be up to 1 m above the reef-hangingwall quartzite contact. The beam is separated from the quartzite (UCS of 170 MPa) by a well developed, persistent argillaceous parting. The approach therefore is the application of threaded expansion shell rockbolts, to bolt the 20-30 cm beam to the competent thick mass of quartzite above it. Currently, about 60-65 % of all panels at Kinross mine are on rockbolting. The practice is that the rockbolts are drilled 70° to the hangingwall over the stope face and not more than 1,3 m from the face after the blast. The choice of 70° is to accommodate both the bedding planes and the 75º faceward dipping extension fractures (see Figure 2.3.2). The length of the rockbolts used is mostly 0,9 m and occasionally 1,2 m, depending on the height of the beam above the reef-hangingwall contact.

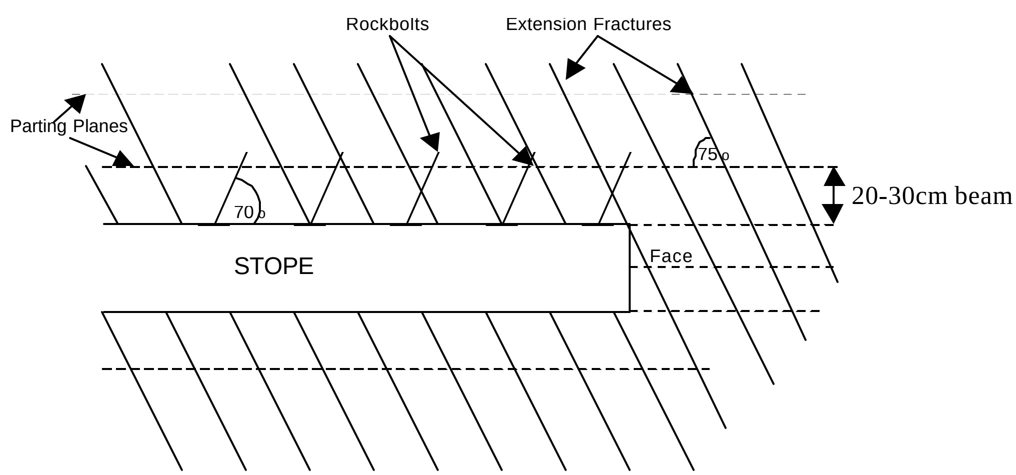


Figure 2.3.2 Inclination of rockbolts with respect to orientation of parting planes and extension fractures, Kinross Mine.

An underground visit was made to a rockbolting panel at a depth of 1800 m at No. 7 shaft (Kinross mine). The panel had not been mined for months but had recently been re-opened and therefore fracturing was intense. This, coupled with the extensive cross-bedding at the stope face (see Figure 2.3.3), might explain why the rockbolting within 5 m of the face appeared to be unsuccessful in that stope (many fall-outs in between rockbolts). It was, however, very successful in undercutting operations (Figure 2.3.4). Bolt loosening after the blast was common (see Figure 2.3.5). It appeared that, had it not been for the intense cross-bedding in this area, which was associated with variable beam thickness resulting in inadequate bolt length in some places, the rockbolt system could have been used effectively in stabilising the strata (Figure 2.3.6). It could not be established whether the loose bearing plates were due to poor setting of the anchor, blast induced vibration, inadequate pretensioning or frittering of rock from beneath the bearing plates. All of these issues need to be addressed by adequate training and supervision, and the availability of installation equipment in good working condition should be addressed.

Figure 2.3.3 Variability in hangingwall beam thickness (Kinross Mine). A worker pushing a rockbolt into a hole, prior to tensioning, at a stope face.

Figure 2.3.4 Rockbolts used successfully in undercutting.

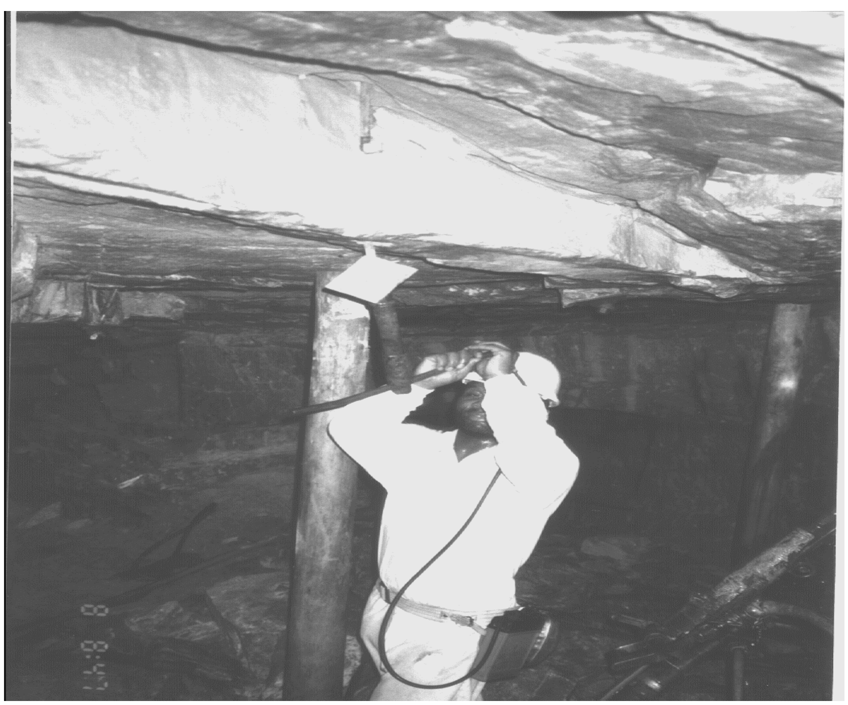


Figure 2.3.5 Re-tensioning a loose bearing plate.

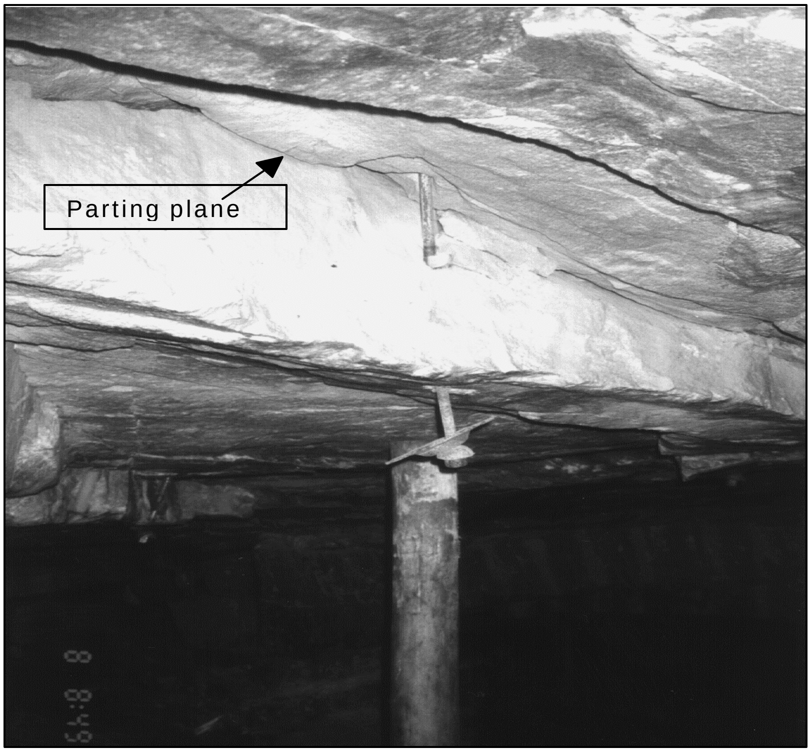


Figure 2.3.6 Hangingwall layering ideal for rockbolting (Kinross Mine). Note loose bearing plate after blast.

2.3.2.2 Western Platinum mine

Rockbolting is used effectively in the mining of the UG2 reef, where thin persistent chromite layers form weak partings in the hangingwall. The distance of these to the excavation ranges from 0,5 to 4,0 m.

A visit was made to 24W7 stope at a depth of 758 m below surface where rockbolts were successfully used in the stope face area. The approach in stabilising the strata includes the use of rockbolts to clamp a parting 30-50 cm above the UG2 and the hangingwall pyroxenite contact. Figures 2.3.7 and 2.3.8 show the hangingwall beams being bolted and the drill rig used. The drill rig permits the installation of rockbolts in stoping widths of less than 1,0 m. The 0,9 m long rockbolts are installed as close as possible to the face and are anchored by expansion shells (Figure 2.3.9 a). Apart from the problem with tensioning (Figure 2.3.10), rockbolting as applied in the stope visited, appears to have been able to stabilise the strata within the stope.

The use of long end-anchored cablebolts pre-stressed to 120 -200 kN is currently being tried at the B3 decline, where the distance to the chromite stringers above the reef-pyroxenite contact does not permit the use of rockbolts.

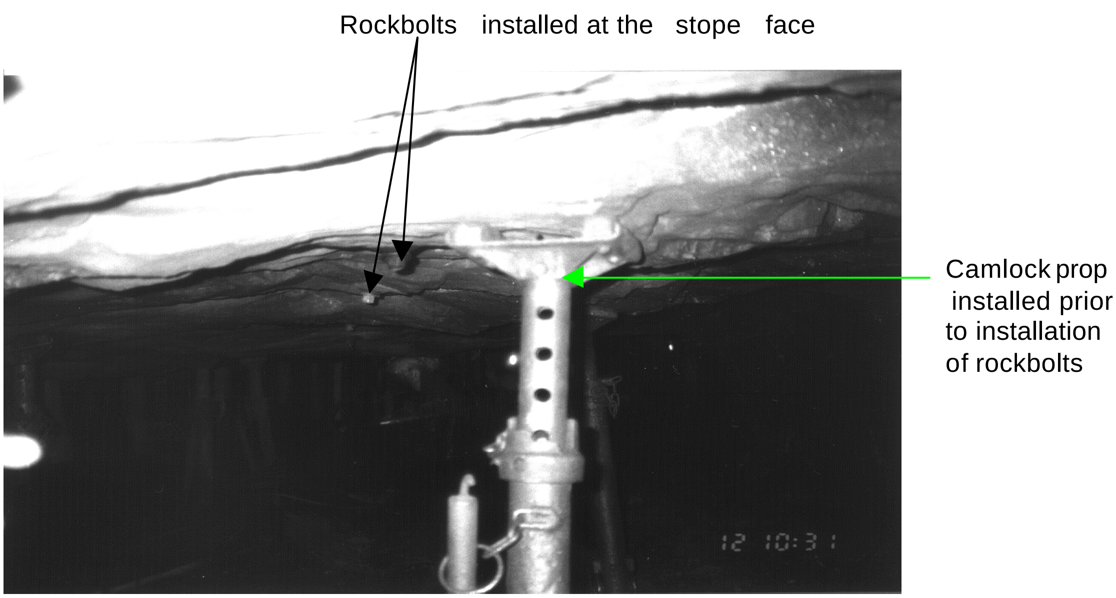


Figure 2.3.7 Layers of roof being bolted together (24 West Stope, North Shaft of Western Platinum Mine).

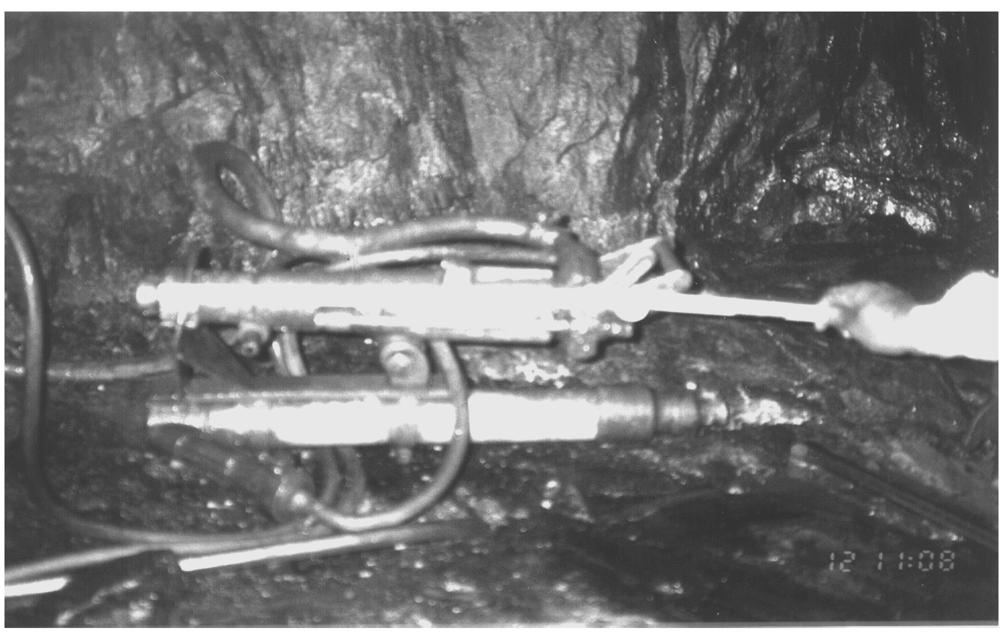
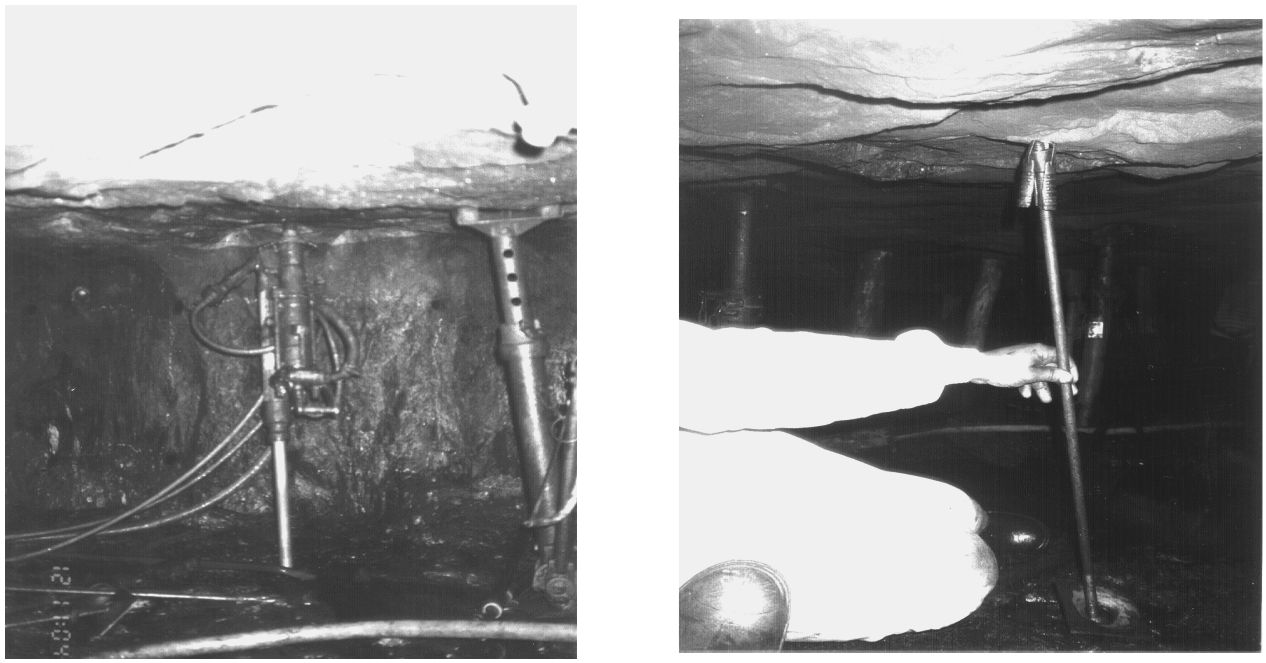


Figure 2.3.8 Drill rig ideal for rockbolting in narrow stoping width stopes (24 West-stope, North Shaft of Western Platinum Mine).



(a) ***(b)***

Figure 2.3.9 (a) Application of rockbolt as an immediate face area support.
(b) 0,9 m expansion shell rockbolt used in the stope.
(24 West-stope, North Shaft of Western Platinum Mine)

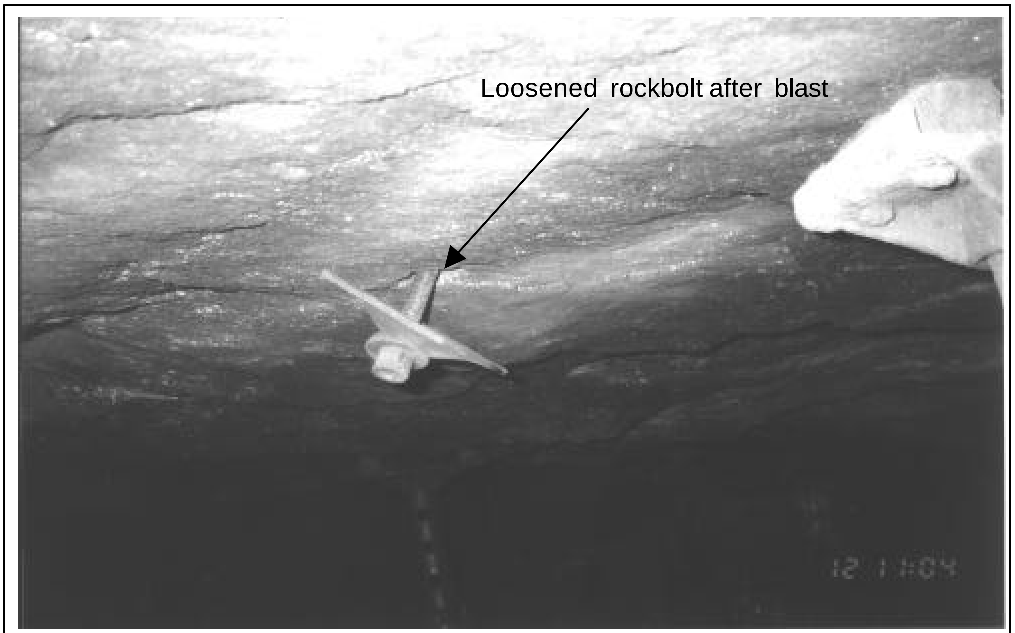


Figure 2.3.10 Improper tensioning of rockbolt at stope face.

2.3.2.3 Vaal Reefs

At the No. 10 Shaft, where the Ventersdorp Contact Reef (VCR) is mined at a depth range between 1 050 m and 1 500 m, rockbolting is applied to the hangingwall strata. The function of rockbolting in this application is to tie back and clamp the strata below pronounced, undulating bedding plane faults, which are often situated close to the VCR/hangingwall contact (Figure 2.3.11) to the more competent rock above. Secondly, the current design of in-stope pillars defines a tensile zone extending approximately 1,25 m above the reef. Expansion shell rockbolts (1,5 m long) are therefore employed to clamp the undulating bedding plane fault as well as the tensile zone.

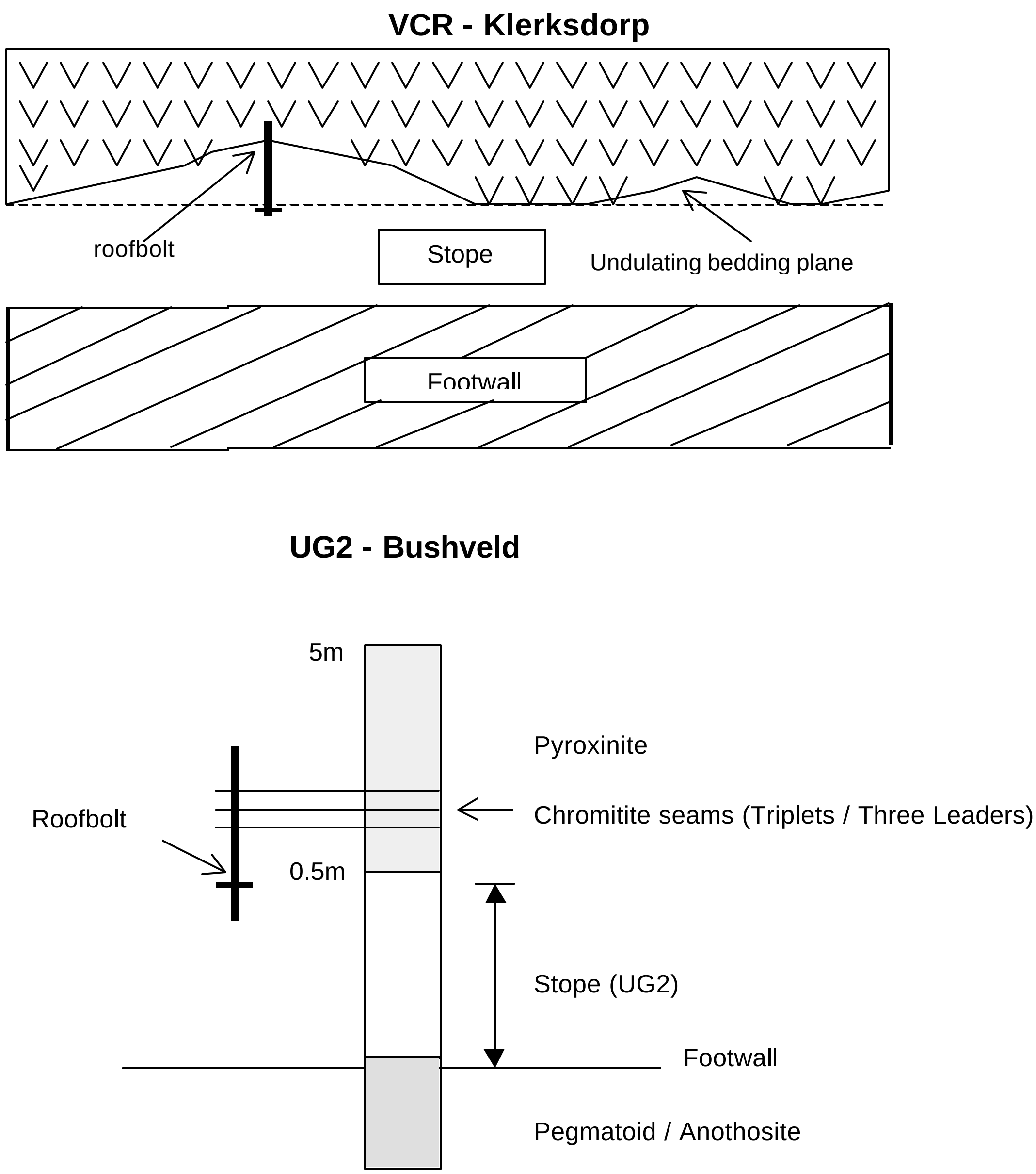


Figure 2.3.11 ***Roofbolting of the undulating bedding fault, VCR Klerksdorp and a geological section through UG2, Western Platinum mine, indicating application of rockbolting.***

About 23 of the 74 panels at No. 10 Shaft make use of rockbolts as face area support. This number could be increased, however the current design of the drill rigs used permits the practical use of rockbolts only at stoping widths greater than 1,8 m. Pre-stressable profile props are used when the stoping width is less than 1,8 m.

The expansion shell bolts are installed at 90º to the dip of the strata (influence of fracture orientation is almost negligible) and are column grouted with medium set cement capsules. They are thereafter pre-tensioned to approximately 10 –20 kN by means of a spanner. The bolts are installed about 3 m from the face to prevent blast damage to the grout, which is not fully set by the time the face is blasted.

2.3.2.4 Randfontein Estates

The presence of a hangingwall parting at a depth of 50-60 cm above the upper elsburg reef-hangingwall lava contact at some sections of No. 3 Shaft of Randfontein Estates Gold mine has necessitated the use of expansion shell rockbolts. At No. 4 Shaft, long cable anchors are used in wide stoping width areas (5 to 10 m) to support the unstable hangingwall lava.

2.3.2.5 Amandebult mine

At the Amandebult section of Rustenburg Platinum Mines, 0,9 m full column grouted anchor rockbolts (expansion shell rockbolts) are used to cater for the laminated hangingwall strata, comprising Chromite leaders within the first 30 cm of the immediate hangingwall. The rockbolts are used to prevent strata separation between the face and the first row of pre-stressed elongate support.

2.3.2.6 Other in-stope applications of rockbolts

Rockbolts have been used under special conditions at West Driefontein and Kloof gold mines with some success.

Four metre long full column grouted pre-stressed long anchors, together with 2,0 m shepherd crooks, are employed to support wide reef panels in the VCR and Main Reef stopes at No. 7 Shaft of West Driefontein Gold Mine. They are installed as close as possible to the face and later serve as the permanent support. The grout used is quick setting and sets before blasting commences. The hangingwall is layered and competent.

At No. 3 Shaft of Kloof gold mine, hollow cylindrical steel tendons were used successfully at a depth of 2 700 m to support a 20 m panel of 1,3 m stoping width. The tendon has a coupling mechanism and can be coupled to any length. The tendons were used as the drill steels and left in the hole after drilling was completed. Quick-setting resin was then injected through the steel tube into the surrounding rock mass and in the process the beam of hangingwall around the borehole was strengthened. The hangingwall comprised soft tuffaceous lava. It was the only method of strata stabilisation and was very successful (Geyser, 1999).

2.3.2.7 Conclusions

From the forgone discussions the following rockbolt practices and issues are prevalent on the mines currently using them.

Loading conditions
Rockbolts are currently used under quasi-static conditions. There is no application of rockbolts in a stope, where loading conditions are predominantly dynamic (i.e. high levels of seismicity).

Depth of mining
Apart from Kloof Gold Mine, where a special tendon was successfully applied at a depth of about 2700 m, tendons are currently used at a depth range up to about 1 600 m. At these depths, the influence of stress fracturing is comparatively low.

Strength of hangingwall
Rockbolts are currently used in situations where the UCS of the hangingwall rock is in excess of 170 MPa. This is probably due to the type of rockbolt mostly used (expansion shell rockbolt). At UCS < 170 Mpa, the rock can be crushed by the anchoring device and rockbolt slippage is likely.

Stoping width
Apart from Western Platinum Mines, where rockbolts are successfully used in a stoping width of 0,9 m, most mines are only able to use rockbolts when the stoping width is generally greater than 1 m. A modification to the drill machine used and a change to cable-bolts could permit the use of rockbolts in narrow stoping widths. For instance, at the No. 10 shaft of Vaal Reefs, rockbolts cannot be used at stoping widths less than 1,8 m although strata conditions are suitable for rockbolting. This is due to the design of the drill rigs employed for the installation of rockbolts.

Hangingwall parting
The current use of rockbolts is based primarily on the presence of at least one pronounced hangingwall parting at a reasonable distance (0,2 to 3 m) from the reef-hangingwall contact. Tendons are applied to clamp these partings and suspend the lower strata from the competent rock above the parting.

Type of tendon
Expansion shell rockbolts (refer to Section 2.2.2.4) are the most widely used rockbolt type in the stope face area. This is due to fact that they can be tensioned on installation to generate initial forces (10 – 20 kN).

Installation distance from the face
The distance at which tendons are installed from the face varied from one mine to the other and ranged from 0 m to 3.5 m before the blast. The reasons for the variability in distances include the type(s) of face area support(s) used and installation cycle, sensitivity of installation to blast and whether or not grouting is used. In those mines where the practice is to install rockbolts as close as possible to the face, other temporary support systems (e.g. mechanical props) are employed to complement the rockbolts. The additional temporary support units offer protection to the machine operators while they drill and install the rockbolts.

Tensioning
Loosening of rockbolt anchors from the walls of the borehole after the blast appears to be a major problem in most of the mines. This is due to the current tensioning method, which is done manually with a simple tool such as a spanner. Comparatively low forces of 10 -20 kN are generated with this method and, therefore, after the blast, they have to be retensioned. It is worth noting that, if multiple bolts are to lose their grip after the blast, the potential for a beam failure is high, especially if there

are no other in-stope support units. It also poses a safety hazard in the stope, as workers may unintentionally strike themselves against them. It is highly recommended to use a torque wrench when tensioning rockbolts and ensure optimum pre-tension forces, thereby minimising rockbolt dislodging during blasting.

Another reason for the loss of rockbolt grip against the walls of the borehole after a blast is incorrect borehole diameter or incorrect installation angle. Also, the proper installation of tendons is compromised under steep-dipping stope conditions due to the practical difficulty of drilling holes upwards in stopes.

Other problems associated with the use of tendons in stopes
The correct angle of possible installation, quality control and sensitivity of installation practice and susceptibility of tendons to blasting has been mentioned above to affect the performance of tendons. The following are other perceived problems associated with tendons in the stope face area:

- Length of possible installation (undulating parting planes make an accurate estimate of required bolt length difficult).
- Worker perception (cannot see action of tendons externally as compared to units like poles, props, etc.).
- Localisation of failure (bolting action creates fracturing around base plate).
- Poor areal coverage.
- Questionable reliability under dynamic loading conditions.

In spite of the above problems, rockbolts, as they are being used currently, can be more successful under rockfall conditions if attention can be given to their installation, tensioning and spacing. The benefits of rockbolting are numerous, which include ease of integration with other support types, light in weight, minimal congestion at the immediate stope face area, ease of cleaning of the face and reduced risk of rockfalls in the face area during cleaning.

2.4 Temporary and face area support in high stoping widths

2.4.1 Introduction

Stoping widths within the platinum and gold mines vary considerably, from ±1 m upto greater than 3 m. High stoping widths are dictated either by the thickness of the reef, or by instability in the hangingwall, or due to poor mining practices.

Under high stoping width conditions, and in highly stressed conditions, the presence of a high stope face is in itself a safety hazard. Barring of loose rock is difficult and any loose rock left in place is easily dislodged by small seismic events or even by the vibration caused by drilling operations.

Most importantly, the force-deformation behaviour of support units can be substantially modified by height. Generally, with increasing height, the stability and the integrity of the support system can become compromised. Current support practices in high stoping width conditions and the effect of height (stoping width) on the following commonly used temporary support systems is discussed (Roberts, 1999): hydraulic props, mechanical props, and elongates.

2.4.2 Hydraulic props

The practical use of hydraulic props in stopes with stoping widths in excess of 2,0 m is limited by their weight as handling is difficult and individual props become a safety hazard by toppling before installation is completed. The stability of hydraulic props and extensions has been greatly improved in recent years by the adoption of conical extension pieces. However, the blast-out rate of these props increases with increasing prop length or stope width. It has been shown that for lightweight 80 kN blast-on props, the blast-out rate for the front line of props increases significantly for stoping widths above 1,5 m.

Figure 2.4.1 shows the prop blast-out rate for the front row of 80 kN props for increasing stoping width. At a stoping width of up to 1,5 m, the blast-out rate is at or below 15 %, which is a viable percentage within production constraints. However, at stoping widths exceeding 1,5 m, the blast-out rate increases markedly and the prop face support system becomes impractical because of this. On the basis of this data, this support system is viable only up to a stoping width of 1,5 m, although undercutting, when required, has been successfully undertaken at stoping widths in excess of 1,5 m. Below a stoping width of 1,5 m, a line of these props installed at increasing distance from the stope face shows a decrease in the blast-out rate. Figure 2.4.2 shows the front row of props in a stoping width of less than 1,5 m for various distances from the stope face. When the front-line props were installed 1,2 m or more from the stope face, a blast-out rate of less than 15 % occurred, while installation of props closer than 1,2 m from the face resulted in increasingly high blast-out rates. This data therefore defines the closest practical distance that the front row of props can be installed from the stope face to be 1,2 m.

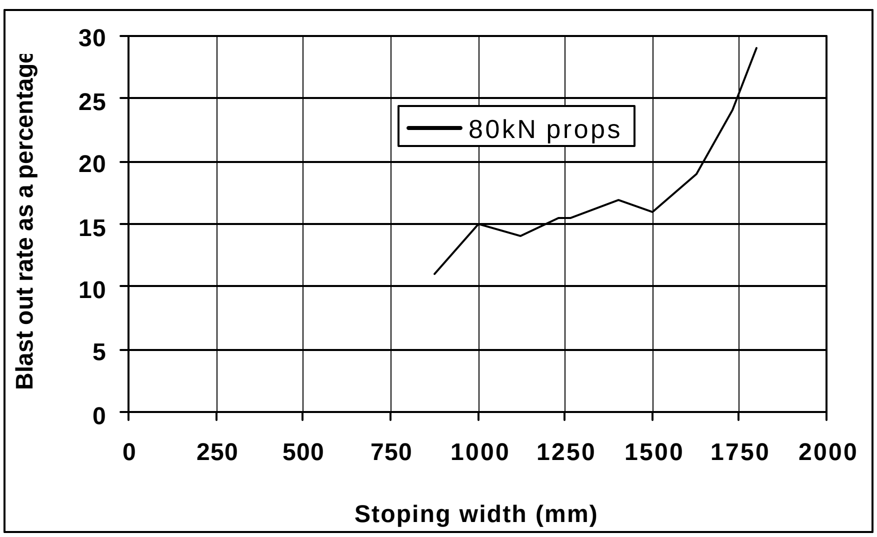


Figure 2.4.1 Front line prop blast-out rate versus stoping width, 80 kN props at an installation distance of 1 m from the face.

Both the above parameters define the working limits of this face support system, namely the 80 kN hydraulic props should not operate in stoping areas where the stoping width exceeds 1,5 m and that the front row should not be closer than 1,2 m from the stope face.

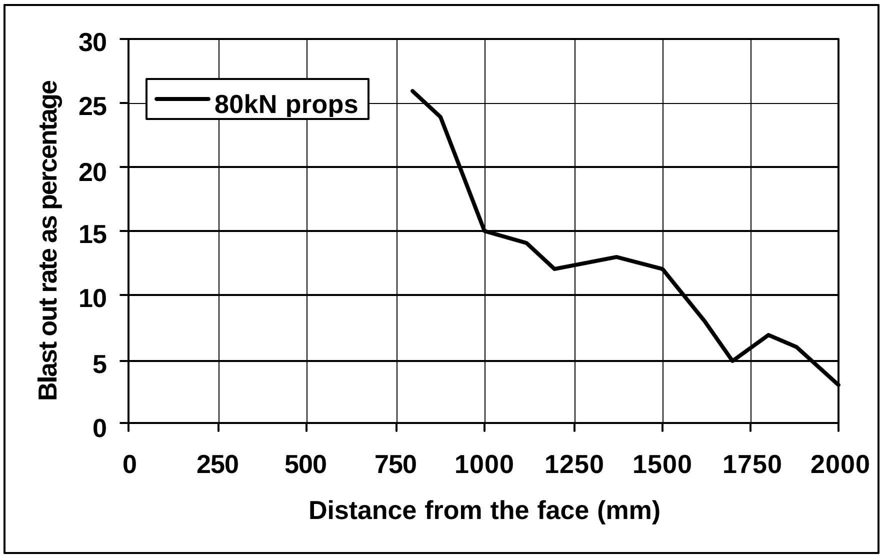


Figure 2.4.2 Front line prop blast-out rate versus distance to stope face (80 kN props).

It should be mentioned that the 80 kN props are currently not in use in the industry. The 200/400 kN props which are mostly used today could probably follow the same trend in respect of the blast out rate versus prop distance from face and increasing stoping width (Roberts 2000). Further work to confirm this is required.

2.4.3 Mechanical props

Testing of mechanical props with respect to their stability with increasing length was undertaken as part of SIMRAC project GAP032 (Roberts, 1995). A variety of mechanical props were tested, primarily to determine the buckling potential with increasing length and, secondly, to determine their ability to absorb energy.

Figure 2.4.3 shows two varieties of medium duty mechanical props, which were tested with increasing prop length. A round bar (12 mm diameter) was inserted between the prop headboard and the platen, 50 mm from the axis of symmetry of the mechanical prop. This was done in order to introduce a small amount of asymmetry to the testing procedure, which would be common underground. Between the lengths of 2,3 m and 2,7 m, the props failed by buckling at loads below 120 kN and at progressively lower loads with increasing lengths. It was recommended that the range of lengths between 2,3 m and 2,7 m should be regarded as the upper limit in terms of length for the use of these props.

Despite the buckling failure of mechanical props beyond 2,7 m, the survey revealed that they are widely used in stoping widths in excess of 2,0 m and, on a particular mine, they are used in stopes as high 5 m. (see Figure 2.4.4 and Appendix 1). The reason for their wide use in high stoping width conditions at shallow to intermediate depth may be due to the fact that they are available in varying lengths and are lighter as compared to hydraulic props.

Figure 2.4.3 The force-length relationship for two types of medium duty mechanical props.

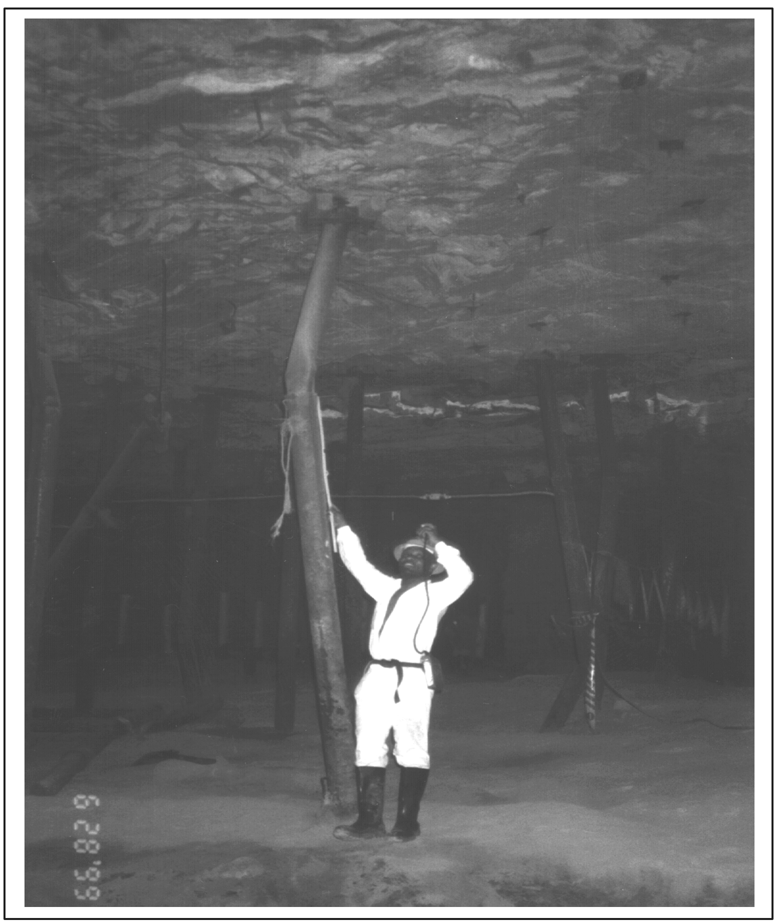


Figure 2.4.4 Roc Props used as a face area support in a 4,5 m height stope at Beatrix Mine (note buckling of prop).

2.4.4 Elongates

On certain mines, mine poles are used as temporary support when the stoping width is high (>2 m). The practical problems with these types of supports include getting them installed at the desired angle and equipping them with a headboard, especially when the stoping width exceeds 3,0 m.

The stability of mine poles and yielding timber elongate decreases with increasing height and with increasing amount of stope closure. This was determined by means of underground measurements in which the percentage of buckled yielding timber elongates was determined for various lengths and for various amount of stope closure.

Figure 2.4.5 shows this in graphical form, where each curve represents a different amount of stope closure acting on a Profile prop.

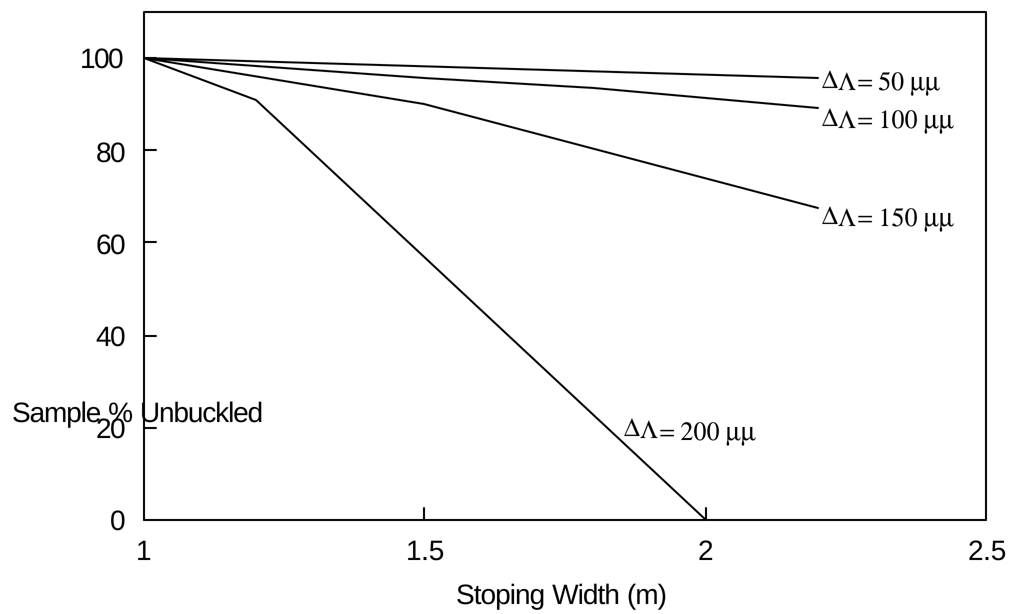


Figure 2.4.5 **The buckling potential of yielding timber elongates (200 mm diameter Profile prop) as a function of increasing length (stoping width) and stope closure (D*L*).**

It appears, therefore, that the height to diameter ratio of this particular elongate (Profile prop) should not exceed 10. The investigation from which the data making up Figure 2.4.5 is derived involved measuring many hundreds of yielding timber elongates which had been exposed to various amounts of stope closure. A variety of yielding timber elongate types was included in the survey. Each behaved differently with respect to the buckling potential for different amounts of stope closure. Some had a much higher buckling potential for a given stope closure than the profile prop example shown above. With respect to the current generation of yielding timber elongates, no information in this form is available. Similar investigations with respect to the behaviour of elongate timber support under dynamic loading have not been undertaken to date and are urgently required as the evaluation of buckling potential is an integral part of stope design.

From the survey, it appears that tendons are the preferred support system under very high stoping width conditions (>3 m). For example at the No. 7 shaft of West Driefontein gold mine where high stoping widths in excess of 5 m exist, a combination of full column grouted pre-stressed long anchor and Shepherd Crooks are employed in stabilising the immediate face area hangingwall. Tendons are also employed in wide reef stopes at the No. 4 shaft of Randfontein Estates gold mine. At Libanon gold mine, 1,5 m column grouted Shepherd Crooks are used in situations where the stoping width exceeds 2,0 m. Grouted Shepherd Crooks were used with success in reducing high stoping widths at Western Deep Levels West mine (now Savuka mine).

On some other mines where it is impractical to use any temporary support system under high stoping width conditions, the practice has been to install tall packs as close as possible to the face and then pre-stress them before drilling and blasting commences. It is, however, known that the stability of these support units decreases with increasing height. For example, it is commonly assumed that timber packs whose height to width ratio exceeds 2 are considered unstable, particularly during dynamic closure. This appears to be a qualitative assessment from underground experience and is probably correct. However, the pack type has an influence on the buckling potential with the stiff end-grained packs having a higher buckling potential than the conventional mat packs. It will only be possible to evaluate these issues under rockburst conditions once a dynamic pack tester is available.

In conclusion, the practical and most effective means of stabilising the immediate stope face area in wide reef stopes is worth investigating because, from the forgone discussion it appears that the current support practices in wide reef stopes are not sufficiently effective. In situations where rockbolting has been part of the support strategy (example Randfontein Estates, West Driefontein, Libanon, etc.), the hangingwall conditions are relatively stable even in stopes as high as 10 m.

2.4.5 Corrections for high stoping widths

In order to design support systems for high stoping widths, the force versus deformation curves obtained by means of laboratory compression tests need to be modified to account for the increased buckling potential. The work by Roberts (1995) based on underground data has resulted in a buckling adjustment formula. In the Support Design Analysis program (SDA) developed by CSIR Mining Technology, the following buckling adjustment is applied to timber elongates:

$$F = F_o \frac{\left[100 - 10^{10d}(h-1)\right]}{100} \qquad (2.4.1)$$

where: F = adjusted force
F_o = original force determined by means of laboratory tests on 1 m elongates
h = underground stoping width
d = displacement/compression

Furthermore, if $h \leq 1$ then $F = F_o$, while if $h \geq 2{,}5$ then $F = 0$.

Figure 2.4.6 graphically depicts the buckling adjustment of timber elongates for various values of displacement/compression (d).

As is apparent from Figure 2.4.5, timber elongates are assumed to fail at lengths in excess of 2,5 m. Whilst this might be an over-conservative assumption for rockfall conditions, it is applicable in rockburst conditions where timber elongates are particularly prone to buckle during rapid stope closure (Roberts, 1999).

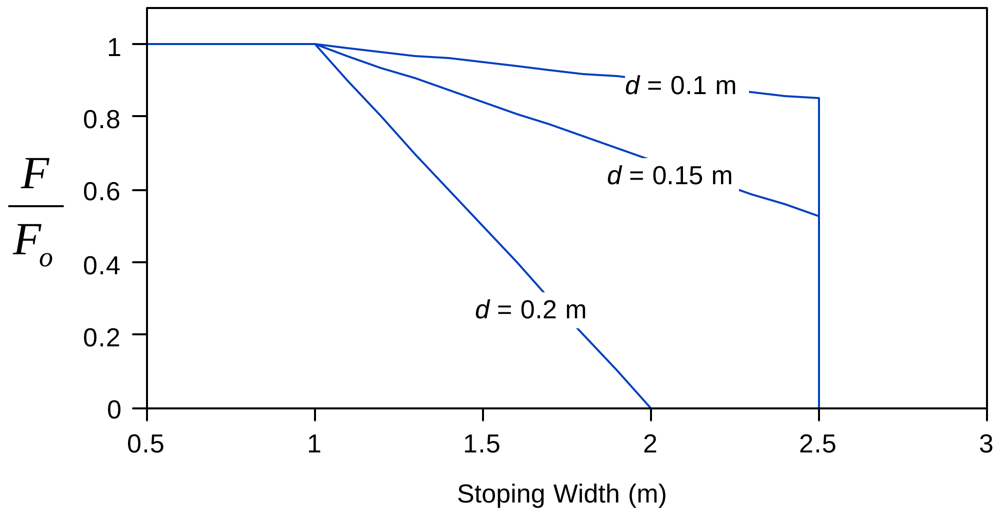


Figure 2.4.6 Buckling adjustment versus stoping width.

To gain further insights into the effect of high stoping widths on the buckling potential of elongates, an approach based on column analysis is adopted. Elongates can be considered as columns, where the stress may be considered partly due to compression and partly due to bending. Hence, two column theories are used, namely (i) the Euler column theory for *long columns*, and (ii) the Johnson column theory for *short columns* (Shigley, 1986). The expressions for the critical load at the onset of buckling for the two column theories are given below:

Euler column:

$$P_{cr} = \frac{A\pi^2 E}{(l/k)^2} \tag{2.4.2}$$

Johnson column:

$$P_{cr} = A\left[S_y - \frac{1}{E}\left(\frac{S_y}{2\pi}\frac{l}{k}\right)^2\right] \tag{2.4.3}$$

where:
- P_{cr} = critical load
- A = area of column
- k = radius of gyration
- l = column length (stoping width)
- E = Young's Modulus
- S_y = yield strength of the column material

Extensive laboratory compression tests on mine poles (Daehnke *et al.*, 2000) indicate an average Young's Modulus of E = 4.0 GPa (taken as the average slope of the approximately linear force-deformation curve during the initial loading phase). The average yield strength of the mine poles was found to be S_y = 22.7 MPa. The Young's Modulus and yield strength are downgraded from a laboratory compression rate of 1 mm/min to typical underground closure rates of 10 mm/day, giving adjusted values of E = 3.3 GPa and S_y = 18.6 MPa (see Roberts, 1995, for compression rate downgrading functions).

Figure 2.4.7 gives values of s_{crit} (normalised with respect to s_{crit} at a slenderness ration of 5). The slenderness ratio is defined as $l = L/d$, where L and d are the length and diameter of the prop, respectively.

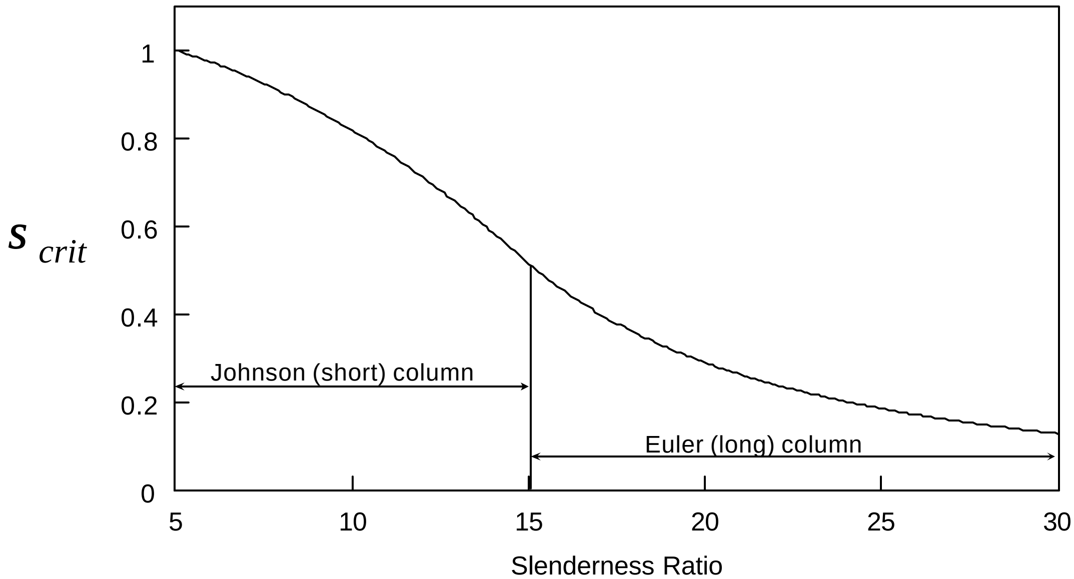


Figure 2.4.7 Normalised s_{crit} versus slenderness ratio based on the Johnson and Euler column buckling theories.

The use of Figure 2.4.7 is illustrated by means of an example: Compression tests on a minepole (L = 1,0 m, d = 0,15 m and $\therefore l$ = 6,7) indicated a peak load capacity of 260 kN (i.e. s_{crit} = 14,7 MPa) before the onset of buckling failure. To estimate the peak load capacity of a mine pole with L = 2,0 m, d = 0,15 m ($\therefore l$ = 13,3), the ratio of normalised values of $s_{crit}^{l=13.3} = 0.63$ and $s_{crit}^{l=6.7} = 0.95$ (determined from Figure 7.2.6) is used to downgrade the load capacity. Thus $s_{crit}^{l=13.3} \div s_{crit}^{l=6.7} = 0.66$ times the peak load capacity of the 1 m long mine pole (260 kN, 14,7 MPa) gives a load capacity of 172 kN (9,7 MPa) for the 2 m long mine pole before the onset of buckling.

The column theories applied here are applicable for estimating the increased buckling potential as the length of mine poles is increased. In the case of the new generation yielding elongates, the adjustment of load capacity for increased elongate length is complicated by the yielding mechanism of the prop. At this stage no general formulae to adjust for increased buckling potential, which are applicable to all yielding elongates, have been developed. It is recommended that *in situ* and laboratory force-deformation data of elongates tested at the appropriate length be used to quantify actual elongate performance. If the elongate performance data for the required stoping width is not available, further tests need to be conducted, making use of elongates of the corresponding length.

In addition, it is important to recognise that the support resistance generated by a given amount of closure acting on a support unit decreases with increasing length of support unit. Assuming a linear stress-strain relationship during the initial loading phase, i.e.

$$s = Ee = E\frac{\Delta L}{L}\ . \qquad (2.4.4)$$

where:
- s = stress transmitted by support unit
- E = support stiffness
- e = support strain
- L = installed support length (stoping width)
- DL = closure acting on support unit

From Equation 2.4.4 it is apparent that, when increasing L (for constant DL), the stress transmitted by the support unit is decreased. Furthermore, the support stiffness is generally decreased with increasing support length, and hence the load generated by the support unit is further decreased. It is therefore recommended that the higher the stoping width, the greater the degree of downgrading of support behaviour to ensure a desired stress-strain relationship.

2.5 Load-deformation behaviour of commonly used temporary support types

2.5.1 Introduction

The load-deformation behaviour of commonly used support types under static and dynamic loading conditions is well documented in recent SIMRAC funded projects such as the ‘Stope Face Support Systems’, GAP 330 (Daehnke *et al.*, 1998), ‘Testing of Tunnel Support: Dynamic Load Testing of Rockbolts’ (GAP 423) and ‘Tunnel Support’, GAP 335 (Haile *et al.*, 1998), projects.

In the discussion that follows, expertise developed in the above projects is employed in evaluating the commonly used temporary support systems based on their load-deformation characteristics.

2.5.2 Elongate behaviour

The load-compression curves for most commonly used elongates at the stope face area are given in Appendix 2. For the rapid displacement tests at a rate of 3,0 m/s, the units are initially compressed slowly for 50 mm then subjected to 3,0 m/s for at least 200 mm and the test completed at the slow rate until failure. The slow displacement tests are done at a rate of 15 mm/minute. Results detailing the test procedure and full results of all the tests for different inclined platen angles are published in GAP 330 (Daehnke *et al.*, 1998), together with some underground evaluations.

The pencil prop is a timber-based unit with a separate pre-stressing device and some machining in an attempt to control the yielding behaviour (Appendix 2). Being a basic prop with timber as its main constituent, it exhibits a greater degree of variability in its performance, as can be seen in Appendix 2.

Consistency of performance is displayed by the Rocprop, which is an engineered steel prop (see Appendix 2).

The laboratory test results showed that, in general, for the same support type the performance of the longer units (1,6 m) was similar to the shorter units (1,0 – 1,2 m).

In general, the performance of elongates tested in laboratory presses exceeded the elongate performance measured underground (see Figures 2.5.1 and 2.5.2). The primary reason for this discrepancy is the reduced underground loading rate (5 – 30 mm/day), compared to the faster loading rate used for the laboratory compression tests (10 – 30 mm/min). Further details of the influence of compression rate are given by Roberts (1995) and Daehnke *et al.*, (1998).

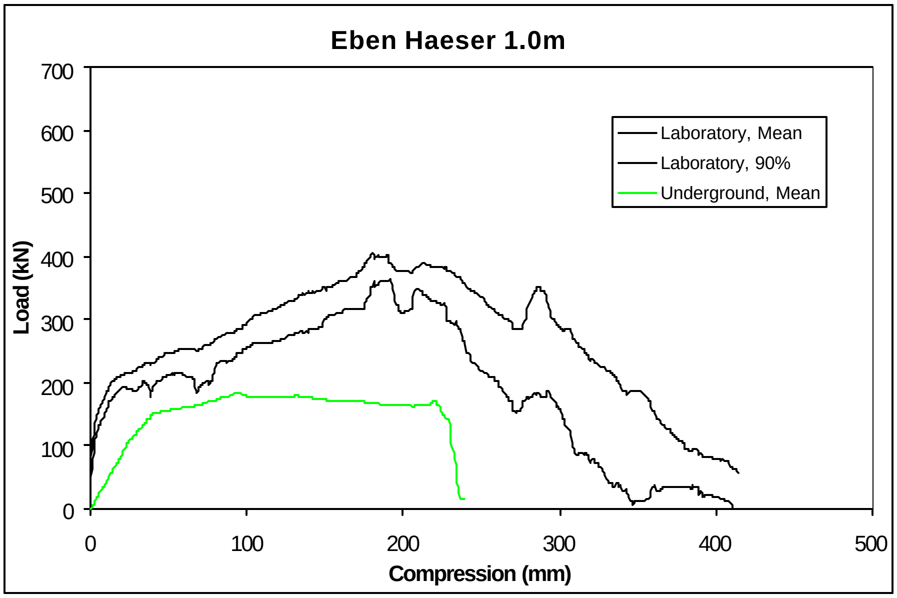


Figure 2.5.1 Comparison of laboratory slow test results with underground performance of Ebenhaeser props.

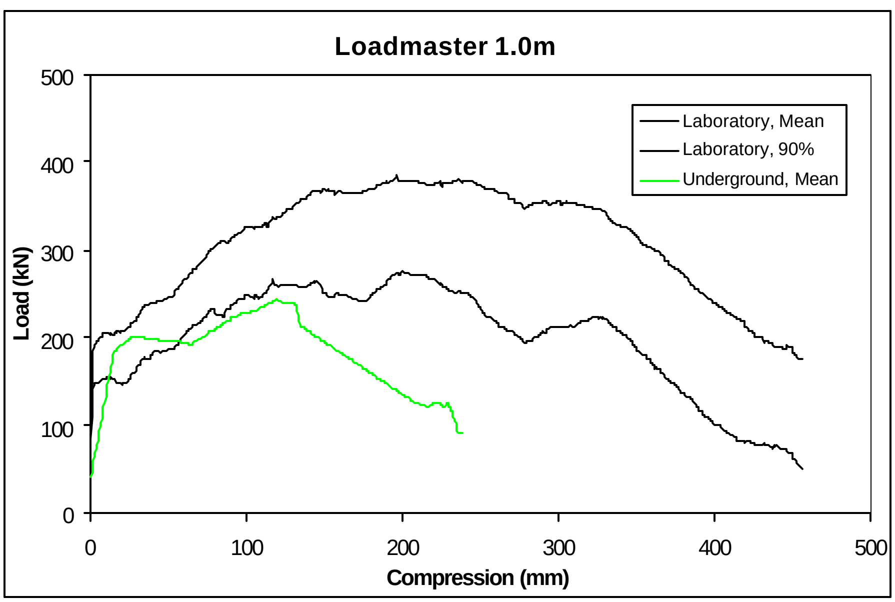


Figure 2.5.2 Comparison of laboratory slow test results with underground performance of Loadmaster props.

2.5.3 Mechanical prop behaviour

Figure 2.5.3 shows a force-deformation curve for a typical medium duty mechanical prop subjected to a 3 m/s deformation rate. The area under the curve, i.e. the ability to absorb energy (3.2 kJ), is extremely limited. These props, therefore, have little energy absorption capabilities, and thus a reduced function as support in rockburst conditions.

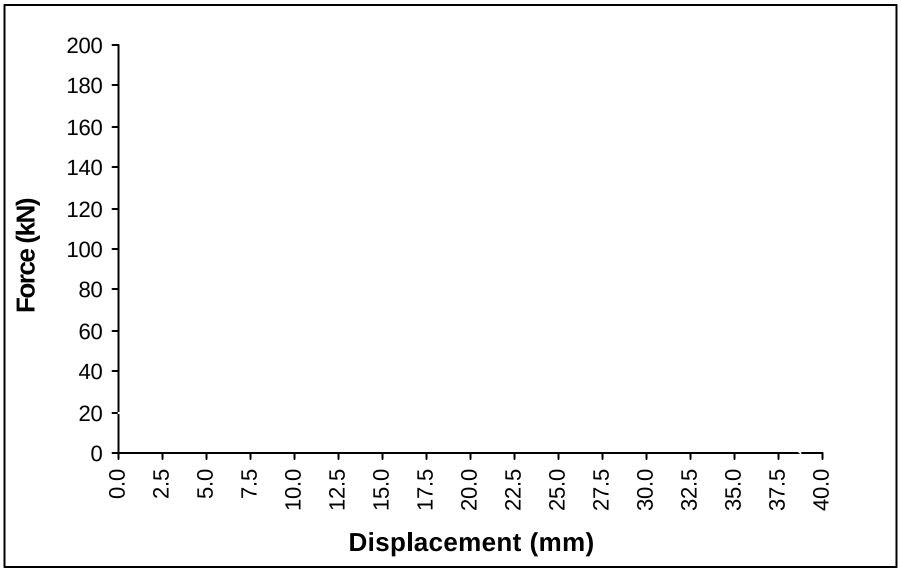


Figure 2.5.3 The force-deformation curve of a 1.5 m long medium duty mechanical prop.

The yieldable Camlock prop has been introduced recently. The discussion here is based on a recent test (refer to Figure 2.5.4) done at the CSIR Miningtek's TerraTek 2000 kN servo controlled testing machine on a Camlock M3A Yielding Prop.

The results show a constant yield load during the slow displacement. This load was recorded as being 100 kN on average during slow displacement. Load integrity was maintained during dynamic displacement. The average load during the first dynamic displacement was 88 kN at a maximum rate of 3,4 m/s. The average load during the second dynamic displacement was 92 kN at a maximum rate of 3,2 m/s. After each dynamic loading, the prop returned to its constant yield during subsequent slow displacement.

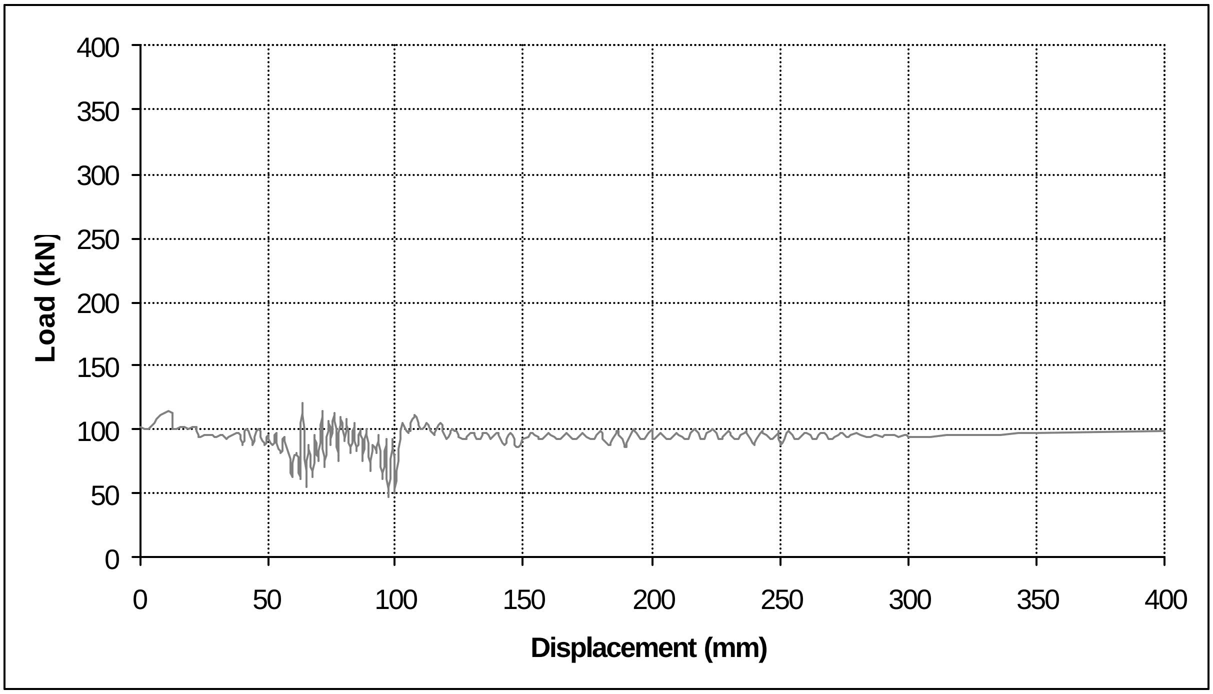


Figure 2.5.4 The force-deformation curve of a yieldable Camlock prop.

2.5.4 Tendon behaviour

The static and dynamic shear load-deformation behaviour of tendons described here is based on extracts from SIMRAC project GAP 335 (Haile *et al.*, 1998). Because tendons are subjected to both shear and tensile failure during their operation in stopes, it has been considered necessary to determine their performance under both shear and tensile loading conditions.

Tests were conducted in a single shear test rig. The rig incorporated ‘rock rings’ at the shear interface to represent the rock mass surrounding the support drill hole. The inner diameter of the ring was representative of a typical borehole (40 mm). Both static loading and dynamic loading were applied to the apparatus and the load and deformation response of the system monitored. Figures 2.5.5 and 2.5.6 show the results of the test programme for the shear and tensile behaviour of some commonly used tendons. From Figure 2.5.5 it is apparent that the cone bolt and the grouted rope systems have generally higher capacity (energy absorption). This may be explained by the ability of these systems to yield and thus allow greater deformation along the shear plane. In the case of the cone bolt this is due to the debonded nature of the rockbolt system; with the grouted cable it was observed that the inner layers of the cable were able to yield relative to the outer layers where failure was initiated. The limit of shear deformation of the other systems was defined by the diameter of the borehole (40 mm). At a shear deformation approaching that of the diameter of the borehole, the surfaces of the rock annulus were found to result in pinching of the rock bolt bar and rapid localisation of failure at this point. It must be mentioned that the expansion shell rockbolt was not tested. Its shearing deformation would have also been defined by the diameter of the borehole and might have even failed at a lower shear displacement.

Table 2.5.1 shows a summary of the load displacement performance of the tendons tested. As can be seen from the table, tests conducted under dynamic loading conditions (3 m/s) were generally found to have similar deformation characteristics, although the peak load and deformation limit were slightly reduced due to the stiffer nature of the system under rapid loading conditions.

Figure 2.5.5 Shear characteristics of typical simulated rock bolt installations under static laboratory shear loading.

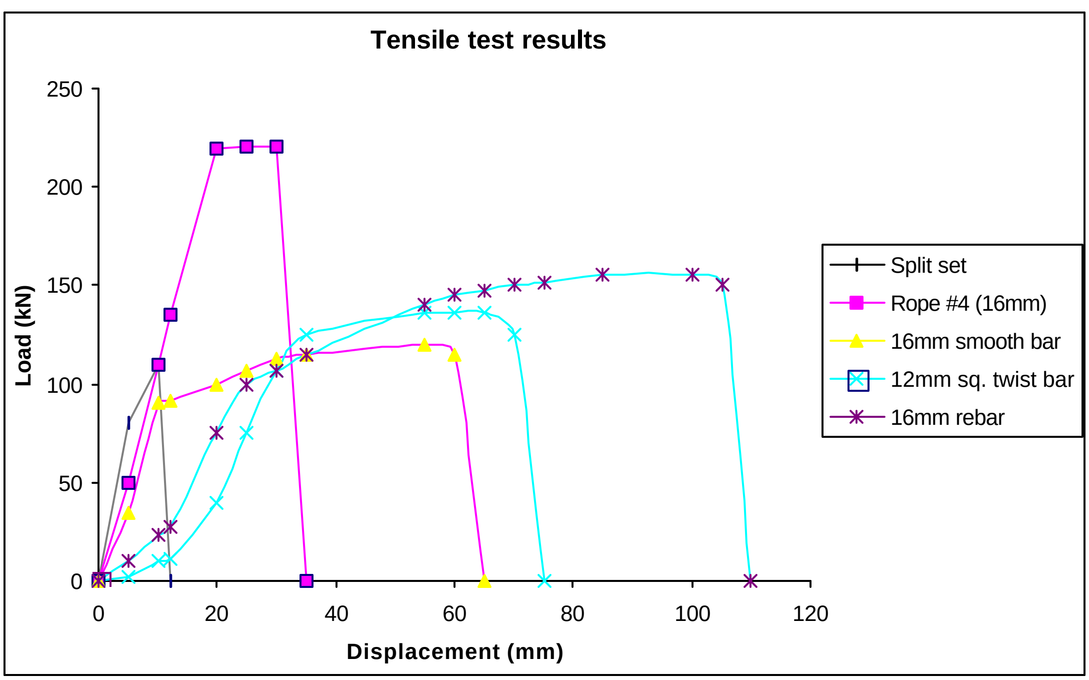


Figure 2.5.6 Direct tensile tests of support units.

Table 2.5.1   Summary of shear test results.

Tendon	Tensile Strength (kN)	Static Shear Strength (kN)	Static Shear Strength (%of UTS)	Dynamic Shear Strength (kN)	Dynamic Shear Strength (%of UTS)	Dynamic Shear Strength (% static)	Loss in Strength (%)	Static Disp. At Failure (mm)	Dynamic Disp. At Failure (mm)
SB	116	120	103.4	122	105.2	101.7	0.0	34	33
RB	156	140	89.7	124	79.5	88.6	11.4	33	32
TB	139	137	98.6	63	45.3	46.0	54.0	33	32
VB	120	85	70.8	38	31.7	44.7	55.3	11-20	10-15
CB	116	190	163.8	120	103.4	63.2	33.6	68	33-65
R1	117	116	99.1	76	65.0	65.5	34.5	40	-
R2	135	142	105.2	73	54.1	51.4	48.6	42	23-30
R3	243	242	99.6	179	73.7	74.0	26.0	37	33
R4	221	227	102.7	118	53.4	52.0	48.0	39	-
SS	110	97.7	88.8	60	54.5	61.4	38.6	31	26-33

Legend:

SB - Smooth bar (16mm); RB – Rebar (16 mm); VB - Twist bar (12 mm sq.);
CB – Cone bolt (16 mm); R1 – Rope # 1 (12 mm); R2 – Rope # 2 (12 mm);
R3 – Rope # 3 (14 mm); R4 – Rope # 4 (16 mm); SS – Split Set (SS 39);