FINAL REPORT

SIMRAC

COL504 : SIMPLE USER'S GUIDE ON ROOF SUPPORT INSTALLATION AND EVALUATION

Prepared by : Dr J.N. van der Merwe, Itasca Africa (Pty) Ltd

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Abstract

The report traces the chain of events from system design through to the installation of roofbolts in underground coal mines. The mathematical design aspects are covered in the main body of the report while the practical monitoring methods, of a mainly visual nature, are contained separately in an appendix, as is a suggested roof bolting audit method.

The design methods are aimed primarily at the rock engineer who is responsible for developing and maintaining a support strategy for a mine. It covers both the relatively simple suspension type problem as well as the more complex beam creation method. A new design method, based on fundamentals, was developed for the beam creation problem. Worked examples of calculations are supplied in an appendix.

Application of the suggested methods results in first order designs, to be tempered by local knowledge and adapted at the hand of monitoring. Rock is too variable a material to hope to have an optimal design at the first attempt, yet it is important to have a scientific basis for that first design. Using a scientific basis coupled to monitoring will eventually result in a support method that is as inexpensive as can be without sacrificing stability.

With regard to monitoring, the main emphasis was on simple, visual methods to evaluate the success of the support system as well as the quality of individual bolt installations. For system evaluation, the main clues are supplied by the rock, in the form of different types of roof falls and the positions and appearances of cracks. The visible portions of bolts, on the other hand, supply very good information about the quality of individual installations and in several cases, a possible diagnosis of the problem with installations.

The trouble shooting appendix is supplied in a stand alone format that can be reduced to a pocket size booklet to be carried underground. It has simple diagrams to indicate deviations with cryptic notes about the possible causes, potential negative consequences and advice on remedial actions. The intention is for this guide to be used on a daily basis by all supervisors.

The second appendix contains a suggested rating system to be used for support audits. It is more comprehensive than the trouble shooting guide, requiring a large number of observations of road width, condition of bolts, etc. This is intended for use at longer intervals than the trouble shooting guide, say for annual or bi-annual inspections by rock engineering or safety department personnel. The possible deviations are rated in terms of seriousness and the system culminates in the calculation of a single index indicating the quality of support installation.

Finally, the report emphasises the importance of communication, indicating real examples where a lack of communication resulted in the escalation of very simple problems into potentially serious matters.

Figures

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1 Introduction

A successfully supported roof is one that does not fall before there are no negative consequences to its failure. The negative consequences include injury to people, disruption of production and weakening of pillars.

In most cases artificial supports in the form of roofbolts are used to stabilise coal mine roofs, while control over mining dimensions remains part of the overall system. In many cases control over road width is sufficient to prevent roof failure.

A roof fall occurs when rock is dislodged from the roof strata and falls under the influence of gravity. The process of dislodgement is sometimes complex, as in cases where the roof strata is sufficiently weak to be fractured or seriously deformed by horizontal stresses. In other cases it is simpler.

The ideal process of roof support is to begin by determining the mode of roof failure in the specific circumstances, deciding how to prevent losses (i.e. accept failure but prevent the failed material from falling or prevent failure in the first place), designing a system, using the materials that will best perform the desired function, installing the system and finally monitoring.

There are a multitude of materials available with which to support the roof, including mechanically anchored bolts, split sets, resin bonded bolts, etc. All have characteristics making them suitable for different applications; none are of no use at all, not one is the universal answer to roof support.

Well installed bolts can sometimes compensate for a sub optimally designed system, but poorly installed bolts will almost certainly result in failure. The main thrust of this report is aimed at proper support installation, which is considered to be the single most important element of the entire process. The intention of the report is to enhance coal mine safety by providing guidance of the proper installation procedure, simple methods of monitoring installation efficiency of bolts that are already installed and equally simple methods of judging the appropriateness of the overall system.

However, the bolt is at the end of a sequence of events – the proper installation is a function of the intention of the designer of the system and that in turn depends on the failure mechanism. Therefore the loop can only be closed by also giving consideration to these other matters.

The international survey on bolting procedures yielded more controversy than agreement on a number of basic issues. For instance, a general principle in Australia is to stiffen bolt systems by reducing the annulus to the minimum (2 – 4 mm) while in the USA the annulus has become greater over the last few years (8 – 10 mm). There is also controversy regarding the amount of pretension to be applied; some Australian researchers advocate zero pretension, while there is a movement in the USA towards much higher pretension – up to 125 kN. One reason for the controversial views is that roof conditions are different and that different circumstances require different approaches.

There is, however, general agreement that bolt systems should be as stiff as possible and that bolts should be installed as quickly as possible. In the light of the controversies and the lengthy explanations required to place them in the proper perspective, it was decided to omit a section in the report on international trends. Instead, design procedures were developed that take cognisance of the controversial items in quantifiable ways. For instance, instead of assuming a high pretension on bolts, a method is presented to calculate the required amount of pretension for the existing circumstances. In the same manner, the bolting system is built around the existing annulus and resin characteristics instead of assuming a certain annulus.

This has the negative consequence of a more complex design method, but on the positive side it results in an engineered instead of assumed support system. With the use of computers and very basic programming skills the complexities should not represent an obstacle to the rock engineer, at whom the design section of the report is aimed.

A bibliography is included that lists useful articles and papers on USA and Australian applications. Note, however, that readers are cautioned against reading only or two of those and thereby getting a slanted view of practices in other countries. A second cautionary note is required: authors sometimes assume that readers are familiar with their underground conditions and do not explicitly describe them. The reader may then gain the impression that what is written is intended to be universally applicable. This is not necessarily the case, indeed some methods are intended for very special cases only. A broad and general guide is that papers written by mine or company employees tend to be valid for specific situations while those written by university or research institution employees are often more generic.

2 Typical South African Roof Conditions.

There are essentially three broad classes of roof conditions in South African Collieries. Each has its own characteristics and requires unique support philosophies. There are no clear boundaries between these; the classes to follow should be seen as representing the midpoints of fairly broad ranges of roof types, schematically shown in Figure 1. They are presented in ascending order of difficulty to support.

The suggested support design methods should be seen as first order methods, not final designs. The roof nature is too variable and the unknowns too many to even hope to deliver the correct and optimal design system with a desk calculation. The design procedure is to investigate, gather the important information, design, implement and then to monitor and adapt on a continuous basis. The loop is an ever repeating one: monitoring is part of the design process.

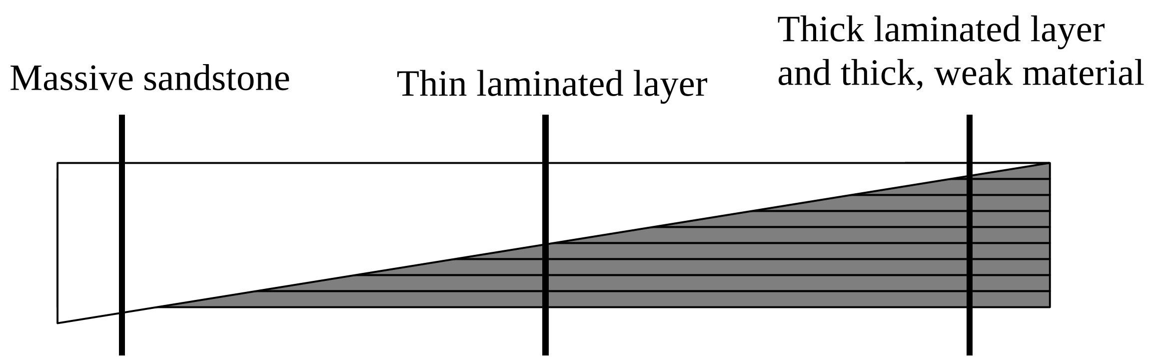


Figure 1. ***Graphic representation of roof types, indicating the gradual transition from one type to another.***

In this section, guidance will be given for the scientific design of support systems. The “science” has been simplified as far as possible. Everything is based on simple, fundamental concepts that are sometimes slightly expanded. It is recognised that the rock is imperfect and highly variable in nature and that underground mining differs from desk top drawings. There is thus little point in for instance designing support spacings to the millimetre. While this argument justifies the simplified approach that is followed, it does not distract from the fact that all materials follow the basic laws of nature. It is therefore important to do some form of calculation, however much it has been simplified.

2.1 Sandstone roof

A thick, continuous sandstone roof is relatively rare but does occur. In this context, the term “thick” implies thick enough to be self supporting over prevailing road widths and “continuous” means containing joints and other discontinuities at spacings wider than prevailing road widths.

There are potential hazards associated with this roof type: thin coal bands left underneath the sandstone and unexpected changes in geology leading to thinning of the sandstone layer or an increase in jointing or cross bedding.

2.2 Thin Coal Layer

The coal layer in the roof usually varies in thickness due to varying operator proficiency and seam thickness. It is left to prevent premature blunting of continuous miner picks by cutting into sandstone and to prevent possible methane ignitions by causing sparks and the thin, hot smear layer on the roof. The coal layer in the roof is best treated by barring it down, although most mines install roof bolts at wide spacings or employ spot bolting as support.

2.3 Increased jointing

Increased jointing, or a single joint in a roadway, is very often not detected until it is too late. The effect of a joint in a roadway, as shown in Figure 2, i.e. to increase the induced horizontal tension in the roof six-fold, is well known but is repeated here:

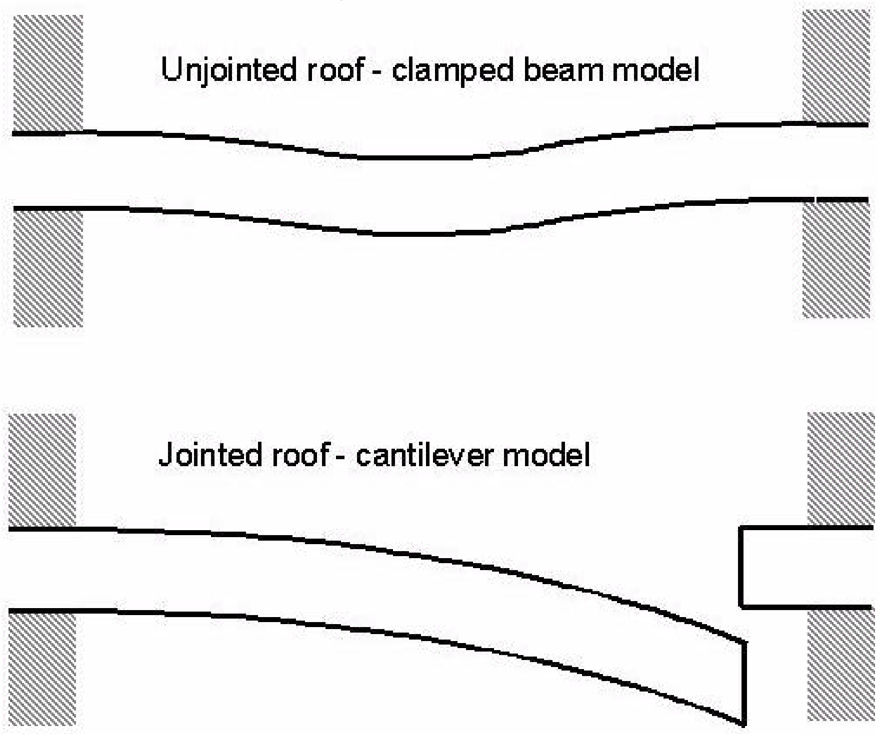


Figure 2. ***The effect of a joint in the roof is to change the basic behaviour from that of a clamped beam to a cantilver.***

$$s_{nj} = \frac{gL^2}{2t}$$ [clamped beam] (1)

and

$$s_{nj} = \frac{3gL^2}{t}$$ [cantilever] (2)

where:

s_j = horizontal tension in jointed roof

s_{nj} = horizontal tension in unjointed roof

L = span (road width)

t = sandstone layer

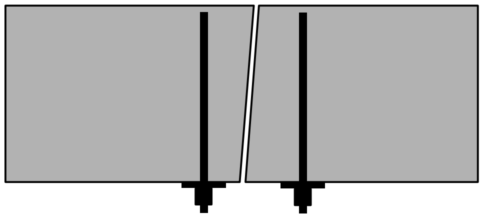
g = unit weight of roof material and beam

Therefore, it can be seen

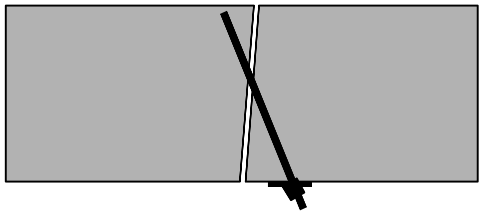
$$\frac{s_j}{s_{nj}} = 6$$ (3)

The spot bolting that is intended to support the thin coal layer in the roof will invariably not be able to support the sandstone beam if it is weakened by joints. Even increasing the bolt density will be ineffective as in the majority of cases the bolts are shorter than the beam thickness.

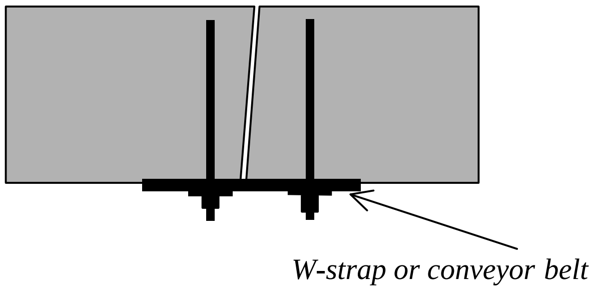
The only effective support measures are to employ one of the common joint support techniques, like installing inclined bolts to intersect the joint plane or installing short W-straps or conveyor belt strips across the joint. Note that one common joint support technique, installing short vertical bolts on either side of a steeply dipping joint, has no beneficial effect. Common joint support methods are shown in Figure 3.



Vertical bolts alongside steeply dipping bolts are ineffective



Inclined bolts intersecting the joint plane are more effective



Short lengths of W-strap or conveyor belt strips are also effective

Figure 3. Common methods of joint support. The use of vertical bolts next to steeply dipping joints are not effective.

In severe cases – a heavily jointed sandstone roof – long cable anchors, trusses or standing supports like mine poles, cluster sticks or sets are about the only successful roof supports.

The risk of roof falls can be minimised by adapting the mining layout to suit the geology. However, in most cases, this strategy has limited application because the zones of intensive jointing only become known after the mine has been established and it is then impractical to change the directions of major development. What can still be done is to minimise the widths of roads running parallel to the major joint direction or to stagger intersections in order to reduce the number of joints daylighting in roofs. Rectangular pillars with the long axis oriented perpendicularly to the joint direction can also be implemented, see Figure 4.

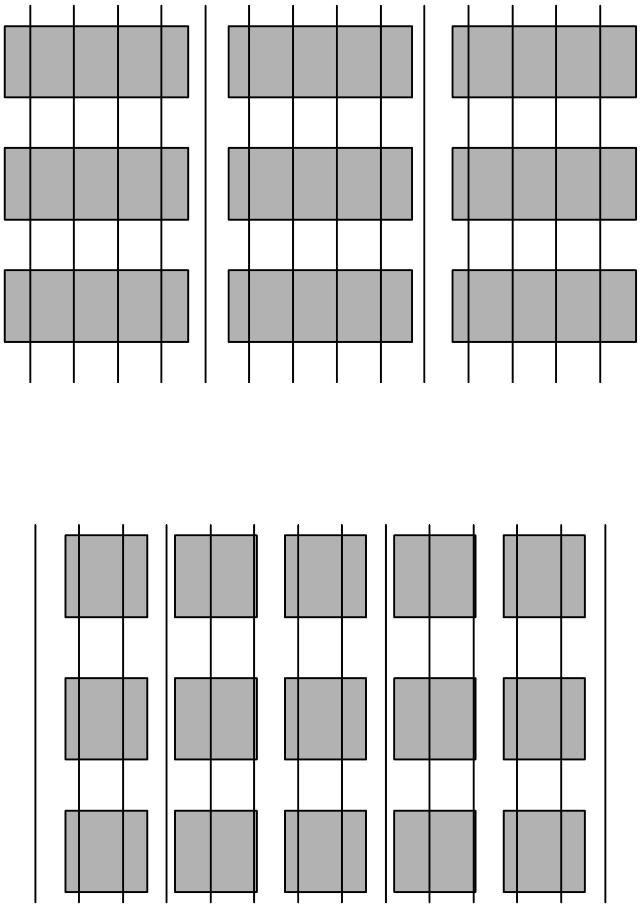


Figure 4. ***Road widths and pillar shapes can be adjusted to cater for areas of intensive jointing without changing the direction of sections.***

2.4 Thinning sandstone roof

The thickness of lithological roof units are not nearly as consistent as their sequence. The stability of a roof plate (here simplified to a clamped beam loaded by its own weight only) is dependent on its thickness and the road width, or because

$$s_t = \frac{gL^2F}{2t}, \qquad (4)$$

it follows that the maximum span over which a given beam will be stable, is

$$L = \sqrt{\frac{t2s_t}{gF}} \qquad (5)$$

which can be simplified to

$$L = 16{,}3\sqrt{t} \qquad (6)$$

for a sandstone with 5 MPa tensile strength, s_t, and safety factor, *F*, of 1,5.

Therefore, the minimum required thickness, *t*, of a sandstone beam must be:

$$t = \frac{L^2}{266} \qquad (7)$$

In simplified terms under these specific conditions this means that for a 6,6 m wide roadway the minimum thickness of a self supporting sandstone beam must be 16 cm.

The above is only valid for a sandstone unit that is overlain by another sandstone or other unit that is as thick or thicker and as stiff or stiffer than the sandstone. This will seldom be the case.

More often, there will be alternating layers of stiff and softer material, i.e. sandstone and shale. In this more common case where a sandstone is overlain by say a shale layer, the sandstone will be loaded by the shale. Equations (4) to (6) should then be adapted to cater for the additional loading. The amount of load that is transferred to the bottom layer is a function of the relative thicknesses of the beams and their material stiffnesses. A simplified approach erring on the conservative side is followed here.

For instance, if the sandstone is overlain by a shale of the same thickness as the sandstone, the safe assumption that the sandstone is loaded by the full weight of the shale can be made. Equations (4) to (7) then become:

$$s_t = \frac{2gL^2}{t} F \qquad (8)$$

$$L = \sqrt{\frac{t s_t}{2gF}} \qquad (9)$$

$$L = 11{,}5\sqrt{t} \qquad (10)$$

$$t = \frac{L^2}{133} \qquad (11)$$

Under these circumstances the required minimum thickness of a self supporting sandstone beam for a 6,6 m wide roadway is 33 cm.

In simplified generic terms, Equation (6) can be expressed as:

$$t = \frac{nL^2}{266} \qquad (12)$$

where $n = 1 + \frac{thickness\ of\ softer/thinner\ rock}{thickness\ of\ sandstone}$

2.5 Suggested precautions.

It has been shown that unexpected variations in geology can cause a dramatic change in the support requirements of a sandstone roof. Under favorable conditions, barring with or without light bolt support is sufficient. The sudden appearance of isolated joints will necessitate the installation of W-straps or similar type of supports at the joints, while dense jointing will require significantly longer bolts or cable anchors.

The key to the continued stability of a sandstone roof is in the geology and that is where monitoring should concentrate on. It is suggested that test holes should be drilled at intersections. Ideally the holes should be inspected with a petroscope, but observation of the drill chips during drilling by an experienced person could also be sufficient. Section 5 of the report (monitoring) contains more detailed recommendations in this regard.

2.6 Sandstone underlain by thin layer of laminated material

This is possibly the most common situation in South African coal mines, and also the least well supported one. The reason for this may be that the laminated layer, mostly consisting of alternating layers of stiff and soft material, may have a stiff and stable appearance. It's hazard is often under estimated because it is relatively thin.

What one tends to forget is that a 50 mm thick slab of rock measuring 2 x 2 m has a mass of 500 kg, more than enough to cause severe, or worse, injury when it falls.

The most common support philosophy for this type of situation is weight suspension. The most commonly overlooked parameter is the support spacing required to prevent falls between the bolts.

2.6.1 Design procedure

Note: the units used in this and following discussions are m, kN and kPa. Under this philosophy, the design procedure is relatively simple: it requires that the bolt system's support capacity must exceed the weight of the laminated material, that the spacing of bolts is dense enough to prevent falls between bolts and that the overlying sandstone beam must be thick enough to support itself plus any softer overlying layers and the laminated material suspended underneath. Figure 5 explains some of the symbols used in the following paragraphs.

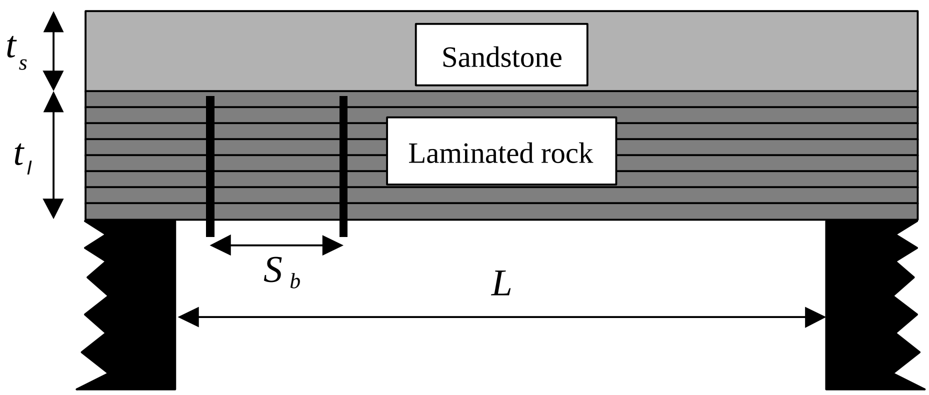


Figure 5. Sketch explaining some of the symbols used in the following text.

The required design procedure is as follows for resin anchors:

Step 1: Check integrity of the sandstone layer.

The required thickness in this case is

$$t = \frac{n_\lambda L^2}{266} \tag{13}$$

where $n_\lambda = 1 + \dfrac{t_{\lambda a}}{t_{sandstone}}$

Step 2: Calculate bolt spacing

Calculate maximum bolt spacing by

$$S_b = 16{,}3\sqrt{n_q t_{ave}}\ , \tag{14}$$

where $n_q = 1 + \dfrac{t_{stiff}}{t_{lam}}$ (15)

t_{lam} = combined thickness of layers in laminated zone

t_{stiff} = average thickness of stiff layers in laminated zone

t_{ave} = average thickness of all layers in laminated zone

Step 3: Bolt length

Calculate bolt length by

$$\lambda_b = t_{lam} + \lambda_a \tag{16}$$

where

λ_a = anchor length

Rudimentary experimental work is required to determine the anchor length. This consists of doing a number of pull tests on short resin capsules to determine the shear resistance, t , of the resin/rock interface.

Per definition,

$$t = \frac{P}{pd\lambda_r} \tag{17}$$

Where P is the load at which the bolt pulls out, d is the hole diameter and λ_r is the length of the resin/rock bond in the hole.

Then,

$$\lambda_a = \frac{25S_b^2 t_{lam}}{pdt} \tag{18}$$

Equation (16) can then be written as

$$\lambda_b = \left[1 + \frac{8S_b^2}{dt}\right] t_{lam} \tag{19}$$

In the case where mechanical anchors are used, the procedure is somewhat different because the anchor resistance is fixed.

Then, the first two steps remain the same, but the anchor length is merely the thickness of the laminated layer plus say 150 mm to ensure that the anchorage is in sandstone. Then follows a check to ensure that the spacing is such that the weight of the roof to be suspended does not exceed the anchor resistance.

Step 3 (mechanical anchors)
Determine the maximum spacing S_m, to ensure that the anchors do not pull out.

$$S_m = \sqrt{\frac{P}{16{,}7t_{lam}}}, \tag{20}$$

where P is the anchor resistance.

The spacing to be used is the <u>smaller</u> of S_b determined with Equation (14) or S_m determined with Equation (20).

In general, bolt patterns in coal mines are specified by stating a number of bolts per row and the spacing of rows in the direction of face advance. The “spacing” in the context of this paragraph is the maximum distance that two bolts are apart in any direction. Note also that this criterion cannot be satisfied by adjusting the pattern so that the area between bolts equals the square of the maximum spacing determined by equations (14) and (20). This will satisfy the weight criterion (thus anchors will not pull out) but could result in falls between the bolts if any dimension exceeds the calculated maximum spacing.

2.7 Thick laminated roof

Not all laminated roofs are prone to collapse. In some cases the cohesion and friction between layers are sufficient to allow the laminated zone to behave like a single beam. Where this is not known beyond doubt to be the case, it is better to assume that the laminations can move relative to one another and will act like a number of separate beams.

This type of roof can be supported by beam creation or suspension. Beam creation is the more sophisticated design procedure and often results in substantial savings because some of the rock properties (i.e. cohesion and friction between layers) are used to create a stable beam. However, there are a number of vitally important prerequisites that must be met before this design method can be used. If these prerequisite are not in place on a mine, then the weight suspension design method must be used.

2.7.1 Beam creation

The most important prerequisite is that the bolts have to be installed before the roof layers have started to sag or separate. Once sag has been initiated, the roof layers have already slid over one another and reduced the frictional resistance. The second prerequisite is that the support materials used and the roofbolting equipment on the mine must allow the required amounts of pretension to be applied to the roof.

The amount of roof sag depends on the road width, advance before bolts are installed and the time lapse between exposing the roof and installing the bolts. The influence of each will be summarised in the following sections:

2.7.2 Road width

The roof sag, h , is:

$$h = \frac{gL^4}{32\mathrm{E}t^2} \tag{21}$$

where $\mathrm{E} =$ Modulus of Elasticity of roof material.

Equation (21) shows that deflection is proportional to the fourth power of road width. Increasing the road width from 6,6m to 7,8m will thus double the amount of sag.

2.7.3 Cut-out distance

As in most other discussions about roof support, the theoretical aspects are simplified to those of a beam instead of a plate. The simplification is valid, provided that the length of the excavation is more than 1,4 times the width. Once that ratio has been exceeded, the roof behaviour is like that of a beam for all practical purposes.

The implication of this is that once the unsupported advance is more than 1,4 times the road width, further roof sag will be arrested. Therefore, for a 6,6m wide roadway there is little point to restrict the face advance to more than 9,3m because once the 9,3m distance has been exceeded, the full sag for that particular road width will already have occurred. This also implies that the narrower a roadway, the further it can advance because the sag will be arrested sooner.

2.7.4 Time lapse between roof exposure and support.

The stress redistribution following the creation of an excavation is immediate, but the failure process is time dependant.

The full roof sag does not manifest itself immediately as the roadway is driven. Exactly how long it takes is not yet known, but what is known is that the longer it takes before bolts are installed, the higher the probability of getting a roof fall. Rico et al (1997) measured continuous movement of a mudstone roof for more than six months. The failure process starts with the development of micro cracks that grow over time. Therefore, even if bolts are installed before the actual fall occurs, the roof has already been weakened and stress changes at a much later time may then result in an acceleration of the process, causing roof falls.

Restricting the cutting distance to 12 m does not have meaningful benefits for stabilisation from a dimensional viewpoint, as mentioned in the previous section, but it does mean that the time lapse between exposure and support is reduced. This could be a substantial benefit.

2.7.5 Design procedure

Beam creation in a laminated roof is based on the principle that the individual layers are bound together to form a single unit that acquires strength by virtue of its thickness. The bounding process hinges on preventing the individual laminae to slip relative to one another. This is achieved by installing bolts that do two things:

firstly, they act as pins and secondly, by tensioning them, they increase the normal stress on the layers to enhance the natural frictional resistance between the layers.

In the discussion to follow, it is assumed that the beam under discussion is loaded by material of the same thickness as the thickness of the beam and that the safety factor for roof support is 1,5. Also, it is assumed that the tensile strength of the roof material is 5 MPa, the shear strength is 8 MPa and Elastic Modulus is 13 GPa.

Step 1: Calculate the minimum thickness of the beam to be created

From

$$L = 16{,}3\sqrt{\frac{t_m}{n}} \quad , \qquad (22)$$

it follows that

$$t_m = 0{,}00753L^2 \qquad (23)$$

Step 2: Calculate maximum permissible sag in the centre of newly created beam:

The artificial beam will be allowed to undergo a certain amount of deflection before it fails. This maximum deflection is:

$$h = 1{,}2\ x\ 10^{-7}\ \frac{L^4}{t_m^2} \qquad (24)$$

Step 3: Determine position of bolt at edge

The relative displacement between laminations in a composite clamped beam is zero at the centre and reaches a maximum at the edges. Therefore, the closer to the edge, the more effective the bolt becomes. Due to practical limitations it is not always easy to install bolts right at the edge.

Most of the roofbolters used in South African collieries can only drill vertical holes, and often the closest that one can drill from the ribside is a distance equal to half of the roofbolter's width. This limitation can be overcome by turning the bolter so that it stands at 90° to the ribside, but then one has to be careful not to let the bolter assistant move in under unsupported roof. The required manoeuvering also slows down the support process.

Let the maximum practical distance be S_r .

Step 4: Calculate the maximum permissible inter layer displacement, $\Delta\lambda_d$*, see Figure 6*

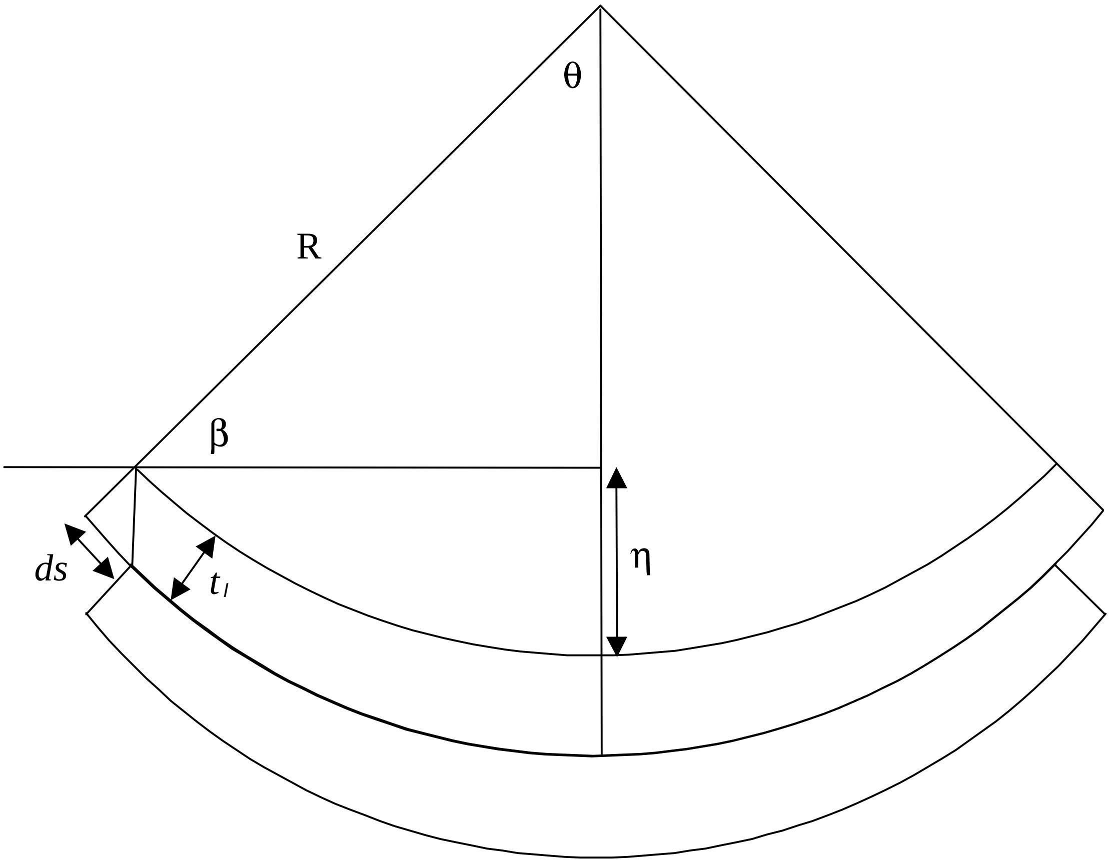


Figure 6. The simplified basis for the calculation of inter laminae slip in the roof.

$$\Delta\lambda_d = q t_{\lambda i} \qquad (25)$$

where $t_{\lambda i}$ is the average thickness of the individual laminae and

$$q = \frac{p}{2} - \arctan\left(\frac{R-h}{L/2 - S_r}\right) radians \qquad (26)$$

The radius of curvature of the roof, R , is:

$$R = \frac{L/2}{Cosb} \qquad (27)$$

and

$$b = \arctan\left(\frac{L/2}{h}\right) - \arctan\left(\frac{h}{L/2}\right) \qquad (28)$$

Step 5: Compare the maximum allowable displacement, $\Delta\lambda_d$*, with the possible displacement,* $\Delta\lambda_p$*.*

The possible displacement is the sum of two components, namely the resin shrinkage upon setting $\Delta\lambda_{rs}$, and the resin compression, $\Delta\lambda_{rc}$

$$\Delta\lambda_{rs} = F_v^{\,0,333}\left(d_h - d_b\right) \qquad (29)$$

where F_v = volumetric shrinkage factor of resin

d_h = hole diameter

d_b = bolt diameter

The resin compression is due to the shear stress generated in the beam, t_b - see Figure 7. This becomes a compressive stress on the resin column,

$$s_r = \frac{tS_b}{d_b} \qquad (31)$$

Where S_b is a chosen "seed" bolt spacing.

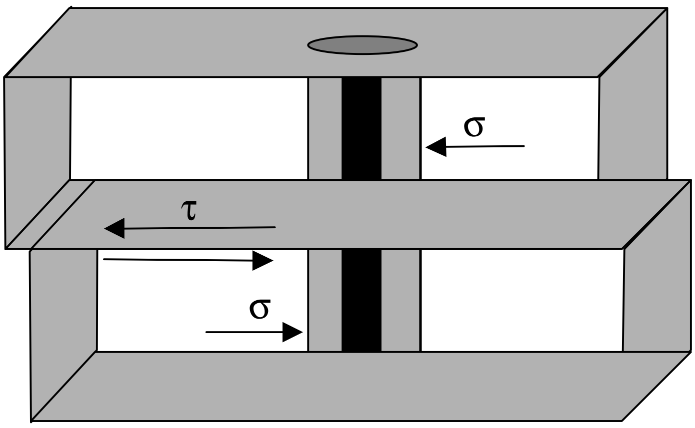


Figure 7. Shear displacement between roof layers causes a normal stress on the resin.

Then, the total resin compression due to the compressive stress is

$$\Delta\lambda_{rc} = \frac{s_r\left(d_h - d_b\right)}{E_r} \qquad (32)$$

Where E_r is the resin's Modulus of Elasticity.

Finally,

$$\Delta\lambda_p = \Delta\lambda_{rc} + \Delta\lambda_{rs} \qquad (33)$$

If $\Delta\lambda_p \langle \Delta\lambda_d$, the position of the edge bolt is confirmed. If not, one or more of the elements of the system have to change. It is noteworthy that one of the important elements of this system is the annulus. The greater the annulus, the more the resin can compress. Tadolini (1998) described the benefits of reducing the annulus in full column resin applications.

It is theoretically possible to create a stable beam with two side bolts per row only. However, due to variations in the efficiency of support installation and complexity of geological materials, it is required to add supplemental bolts. The suggested method to determine the spacing of the supplemental bolts is to use the procedure to prevent falls between bolts, as described in Section 2.6.1 of this report.

In beam creation, the edge bolt remains the pivot around the entire system is constructed.

Step 6: Determine bolt length

In beam creation, it is preferable to use a dual speed resin system. The length of the slow resin column must equal at least the thickness of the beam to be created. The length of the fast resin portion must be such that it can accommodate the pretension load on the bolt.

Internationally there is some controversy regarding the pretension to be applied for beam creation. At one extreme there is the argument that provided the bolts are installed very close to the face, no pretension should be applied; pretensioning of bolts installed right on the face can in fact result in bolt failure, according to Jafari (1994). At the other end of the scale is the argument forwarded by Stankus and Guo (1998) that a very high pretension – as high as 125 kN – combined with shorter bolts yields better results than longer bolts with the same pretension.

The core of the latter argument is that the roof state of vertical stress is tensile, and that pretensioned bolts create zones of compression at either end of the bolt. If the bolts are long, the two zones of compression are separated by a tensile zone in the centre, therefore two separate beams are created. If the bolts are shorter, the zones of compression overlap, resulting in the creation of a thicker, single beam that is stronger than the two separate thinner beams. The principle is shown in Figure 8.

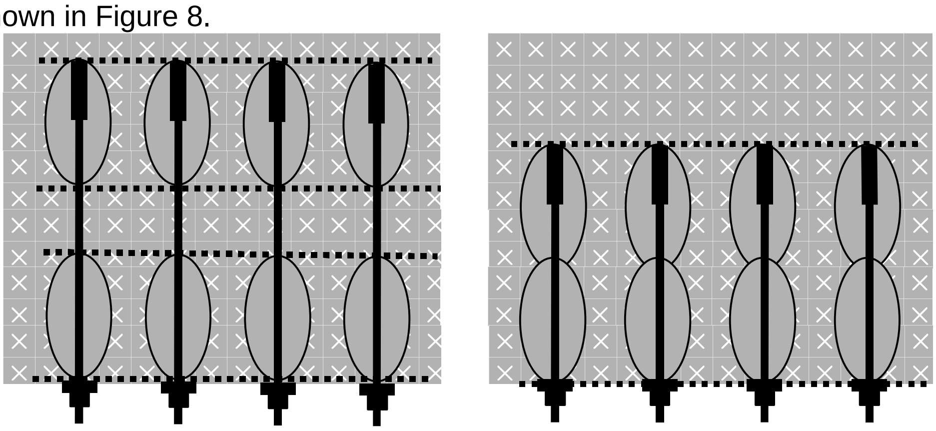


Figure 8. Shorter bolts with high pretension results in the creation of a thicker, stronger beam than longer bolts with the same pretension, due to the overlapping of compression zones in the roof, Stankus and Guo (1998)

This argument is fundamentally sound but requires proprietary software to produce practical bolting configurations. It is therefore proposed to base the design on a somewhat simplified approach.

If slip between the roof layers is to be prevented, the frictional resistance must at least equal the disturbing forces. The maximum disturbing shear, t_d, is:

$$t_d = \frac{3gL}{4} \tag{34}$$

The shear resistance, t_r, is

$$t_r = C + s_e \tan f \tag{35}$$

Where C = cohension between layers

s_e = effective stress applied by bolts

f = angle of friction of inter laminae contact plane.

The effective normal stress, s_e, is the balance between gravity induced tension and the compression supplied by the bolt's pretension. With the simplifying assumption that the pretension load is distributed evenly over the supported area per bolt,

$$s_e = \frac{F_b}{S_b^2} - gt_\lambda \tag{36}$$

Where F_b is the pretension of the bolt, and therefore the required pretension is

$$F_b = S_b^2 \left[\frac{\left(\frac{3gL}{4} - C \right)}{\tan_\Phi} + g\ t_{lam} \right] \tag{37}$$

The next step is to calculate the required length of anchor, λ_a, that will have the resistance calculated with eqn (37). Simply,

$$\lambda_a = \frac{F_b}{pdt} \tag{38}$$

where

λ_a = required anchor length

t = shear resistance of the resin/rock contact plane

The total bolt length is then the sum of the beam thickness and the anchor length, or

$$\lambda_b = \lambda_a + t_m \qquad (39)$$

Step 8: Optimisation of resin capsule lengths

The creation of the beam requires that the pretension must be applied to the bolt while the resin is still fluid over the thickness of the beam while the anchor portion must already have set. Therefore the anchor must be a fast resin, and the beam must have the slower resin.

$$\lambda_{frc} = \frac{d_h^2 - d_s^2}{d_c^2}.\lambda_a \qquad (39)$$

where λ_{frc} is the length of fast resin capsule and d_c is the diameter of the resin capsule.

Similarly, the length of the slow resin capsule, λ_{src}, is :

$$\lambda_{src} = \frac{d_h^2 - d_s^2}{d_c^2} t_m \qquad (40)$$

One is unlikely to be able to purchase the exact length of capsule that is required and some practical compromise will inevitably be required. The effects of any such compromise must, however, be considered by calculation before installation proceeds.

2.7.6 Suspension of thick weak roof

It has already been stated that the ideal support philosophy for a thick, weak roof is beam creation. Unfortunately this is not always possible for a number of reasons including excessive cut-out distances, non suitability of the available equipment or material, etc. The alternative, less sophisticated but equally effective philosophy is to accept that the roof will fail by whatever mechanism and to merely supply a basket in which the loose roof can be suspended without causing damage. The negative consequences are only realised in practice when the roof falls, not when it fails.

There are three sub classes in this division: standing supports, long cable anchors and short inclined bolts or trusses.

Standing supports include steel sets, arches and timber poles. These are no longer popular in South Africa (except in isolated bad spots like burnt coal, faulted zones, etc) but their use should not be discarded outright. This is especially true for the lower spectrum of mining heights.

Long cable anchors are more common than standing supports, but the design procedure is often not very scientific. It is not a complex procedure, as shown in the following paragraphs.

Step 1. Determine the load on the system

This is most practically done by observing the height of existing roof falls – they are more often than not restricted by the presence of an even slightly stronger roof layer or merely by the width of the falls, h_f, reaching a stable dimension. In most cases the need for cable anchors will not be foreseen, it usually being a reaction to increasing numbers of high roof falls, so that this information is usually available.

Then, the load, *W*, is

$$W = 37{,}5h_f \quad \text{kN per square metre} \qquad (41)$$

Note that equation (41) is based on the conservative simplification that the falls are vertically sided.

Step 2. Determine the anchor spacing

Long cable anchors are usually secondary supports, and it is therefore easier to obtain even spatial distributions of the anchors. The required spacing, s_l, is

$$s_l = \sqrt{\frac{R}{W}} \quad \text{m} \qquad (42)$$

where *R* is the strength per cable, in kN.

Step 3. Determine cable length

The cable lengths are determined in the same way as the bolt lengths for suspension of thin layers, described in Section 2.6.1 of this report. The additional consideration is that the length must be sufficient to ensure that anchoring is obtained in a strong layer higher up in the roof, or it must be at least three times the height of the roof falls.

Although cables are often installed with resin anchors, this is discouraged because the resin is seldom properly mixed by the cable. Mechanical anchors are easier to install, resulting in higher quality of support. Obtaining high anchorage loads with the mechanical anchors supplied with cable anchors is easier than with the smaller anchors supplied with normal bolts. Following tensioning of the cables they should be cement grouted very soon after tensioning. Tensioning should always precede grouting. The pretension load should equal half of the breaking strength of the cables.

Step 4: Supply areal cover

This design method does not cater for the prevention of roof falls between supports by adjusting the support spacing, as that would be prohibitively expensive. Some form of areal cover is thus also necessary. This can range from W-straps or other steel straps to wiremesh or wiremesh and lacing. The choice of material will be influenced by the nature of the immediate roof. A reasonable roof will not require more than strapping, while a friable roof should be meshed.

The application of wiremesh in coal mines is often negated by the method of application. The mesh has to be an integral part of the cable system. This can only be achieved by installing the cables through the mesh. Installing the cables first and then installing the mesh with separate short bolts is not sufficient as it will not transfer the load of the loose material to the cables. Once the mesh has been properly installed behind the cables, additional shorter bolts can be used to fix the mesh close to the rock – especially where shotcrete is also to be applied, this is good practice.

Short inclined bolts are essentially a way to provide the same effect as steel sets, without the legs. With long cable anchors, the anchorage is obtained above the weak zone. With inclined bolts, it is obtained beyond the edge, in the compressive zone just above the pillars. The concept is illustrated in Figure 9. Note that this is also the design method for support with roof trusses or cable trusses only.

The design procedure, as all suspension problems, is simply a matter of balancing the weight of the falls by the support resistance of the bolt system. The main advantage of the inclined bolt system is that the same anchorage is obtained with significantly shorter bolts as the full length of the bolt is used for anchoring; the “dead" length traversing the weak zone does not exist.

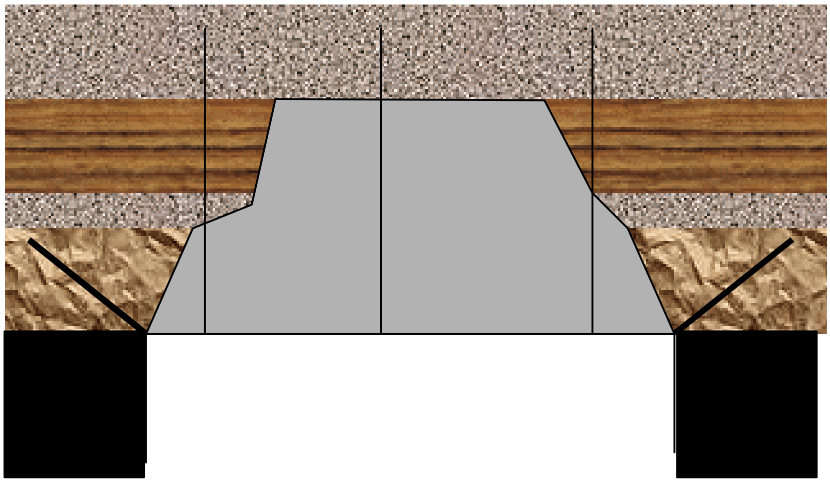


Figure 9. Bolts inclined over the pillars can result in the same anchorage as longer vertical bolts, because they do not traverse the potential fall height.

Step 1. Determine the load on the system

Use the same method as for long cable anchors. For this application, however, the load per running metre of roadway is the central parameter, and therefore the load equation becomes

$W = 25h_f L$ kN per running metre (42)

Step 2: Choose bolt length, calculate spacing

The support resistance is determined by a combination of hole diameter, d_h, bolt length, l_b, and bolt spacing. It is recommended to use minimum hole diameters of 28 mm with 20 mm bolts for this application. The hole length is a practical consideration and the maximum can be considered as fixed for any given situation while the spacing is the parameter that can be adjusted most easily. Therefore it is suggested to fix the diameters and length and calculate only the spacing. In simplified form, the spacing, s_b, for a safety factor of 1,5 is then

$$s_b = \frac{4{,}2 d_h l_b t_{res}}{W} \qquad (43)$$

where

τ_{res} = shear strength of resin/rock interface.

Equations (42) and (43) can be combined to yield the single equation for the determination of bolt spacings as a function of fall height:

$$s_b = \frac{0{,}17 d_h l_b t}{h_f L} \qquad (44)$$

If this spacing is too dense, adjust the system by increasing the hole and bolt diameters or increasing the hole lengths.

Step 3: Check for steel strength
It is possible for the load per bolt to exceed the strength of the bolt if the equations are used without checking. The bolt strength, F_b, should be greater than the bolt load, or

$$F_b > \frac{W}{2s_b}, \qquad (44a)$$

which can also be written as

$$F_b > \frac{18{,}75 h_f L}{s_b} \qquad (45)$$

Step 4: Supply a real cover
This support method is reliant on areal cover, as the basic idea is to install no intermediate bolts. Therefore, if trusses are to be replaced by bolts, W-straps or something similar is essential, not preferable. The remarks with regard to areal support in the previous section on long vertical cable anchors also apply to this section.

2.7.7 Effect of position of hole

The method described above is based on the assumption that the support holes are drilled in the corner of the roof. Additional benefit may be obtained if the inclined holes are drilled approximately 0,5 m from the corner, as shown in Figure 10. In doing that, the bolts penetrate the plane along which the fracture causing the roof falls will develop. They then also fullfill a preventative role as well as supporting the dead weight of potential falls.

However, an additional check is then necessary, to ensure that the shear strength of the steel, F_{sb}, is not exceeded. Therefore, it is important to check that

$$F_{sb} > \frac{18{,}75 h_f L}{s_b} \qquad (46)$$

In most cases it will be easier to drill holes right in the corner with hand held equipment or light rigs, but there are significant benefits to installing the bolts about 0,5 m from the corner.

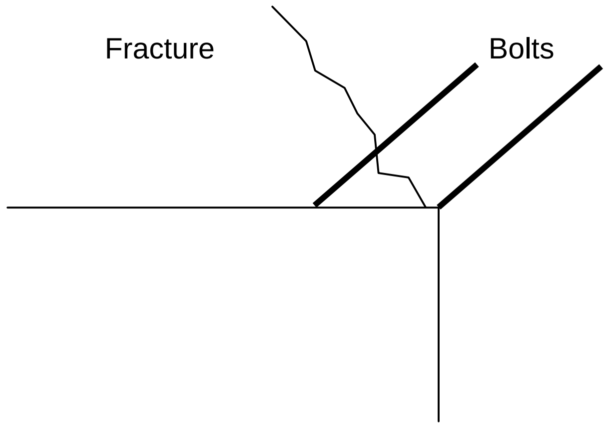


Figure 10 Cross section of the corner of a roadway, showing how an inclined bolt a short distance away from the ribside penetrates the fracture plane.

2.8 Concluding remarks on design methods

As stated in the introduction, the recommended design methods depend on a basic understanding of the roof type and consequently the expected failure mechanism. There is no such thing as the best design method – there are only effective methods for the prevailing conditions.

In the preceding descriptions three basic conditions were identified. They represent midpoints of "sections" in an infinite variety of gradual change. For instance, one question that was not answered was at what thickness of the laminated layer underneath a sandstone does one change over from a suspension philosophy to beam creation ? There is no technically correct answer to this question, but the user will be guided by practical matters like the maximum bolt length that can be installed, whether it is at all possible to install bolts before the roof has deflected, etc. In the grey zone, it should eventually become a matter of economics, i.e. in choosing between two equally safe roof support methods, simply pick the cheapest one.

The greatest danger to avoid is wishful thinking – the roof is what it is, not what we wish it is. Roof beams cannot be created if the bolts are installed after deflection has occurred, or if the bolters cannot supply the required pretension.

The work is not complete once the system has been designed. Monitoring of the performance of the system will in most cases highlight design shortcomings and changes in roof composition that have to be catered for by adaptation. The monitoring/correction action, discussed in Section 5 of this report, should be seen as a continual process, not a one-off step.

3 Selection of components

There are several types and combinations of roof support components available. While most are effective for certain types of applications, they have different degrees of efficiency for different applications. Not all are equally effective for all conditions. The classification to be used for this selection guide, is to view the different systems under the groupings of the two basic types of applications, i.e. suspension and beam creation.

3.1 Suspension application

As the only requirement for suspension systems is a certain capacity for load bearing, virtually any type of bolt can be used.

Mechanical anchors are sometimes acceptable, especially in hard sandstone where it is difficult to drill holes thinner than 32 mm diameter. The disadvantages are that in the absence of a grout filling, they are susceptible to corrosion. Also, anchors may creep and of course the anchor resistance is fixed at between 50 and 100 kN. What is not commonly appreciated is that once the bolt relaxes due to for instance frittering of the roof underneath the washer, the anchor itself may lose grip due to relaxation.

In cases where mechanical anchors are used, the bolt diameter only needs to be thick enough to be 1,5 times stronger than the required anchor resistance. In most cases a 16 mm bolt with a yield strength of 115 kN will be sufficient.

A very important element of any suspension system is the strength of the washer assembly, which includes the physical washer and the nut and thread. The washer must be able to withstand 80% of the system's required resistance before it deforms and 100% before it fails, usually by the nut pulling through the washer. The nut and thread must be stronger than the bolt.

The recommended test procedure is the following:

Design system, determine required resistance of bolt.

Install bolt underground in the chosen hole diameter and perform pull test on anchor, using double nuts and a 20mm thick steel washer at the protruding end of the bolt, as shown in Figure 11. Check whether the anchor offers the required resistance.

Fit double nuts to the end of the bolts and a single nut of the type to be used underground, to the other end. Perform pull test in a workshop. The steel body must fail before the thread fails. This also tests the breaking strength of the steel.

Fit the roof washer and nut to one end of the bolt, insert a 20 mm thick steel washer with a hole with diameter 1,5 times the diameter of holes to be drilled underground on the inside of the roof washer and fit double nuts to the anchor end, as shown in Figure 12. Perform pull tests in workshop, check loads at which washer deforms and fails.

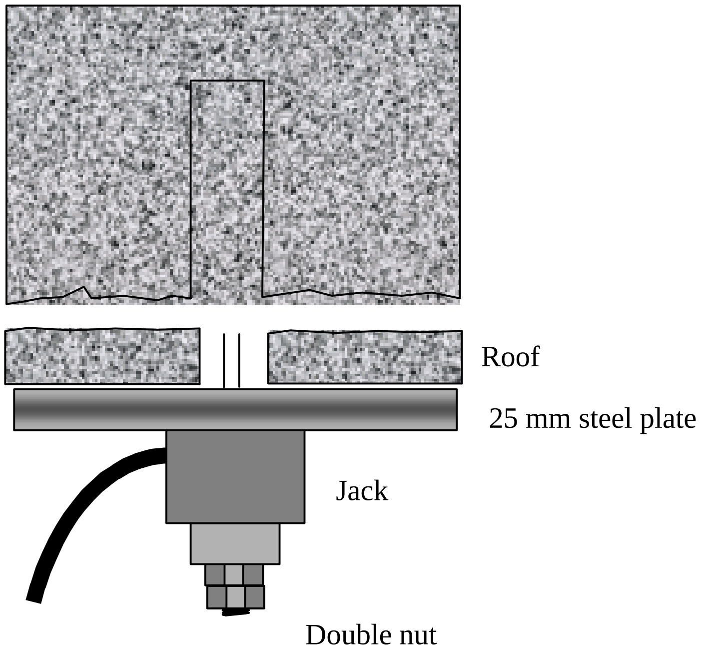


Figure 11.   Pull test on a mechanical anchor.

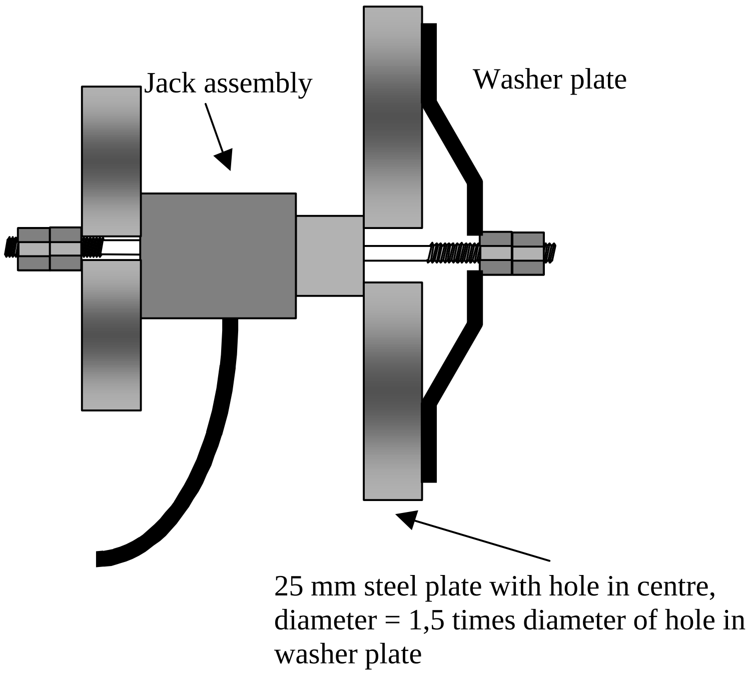


Figure 12.   Pull test on a washer plate

Point anchor resin bolts are equally effective for suspension systems, with the major advantage that the anchor resistance can be adjusted by varying the length of the resin anchor. Also, the anchor does not lose grip when the bolt relaxes.

The major disadvantages of resin point anchors as compared to mechanical anchors is that the installation procedure is more complex, requires more discipline and that seasonal fluctuations in temperature may require adjustments to the installation procedure. It is also often necessary to use a thicker bolt than would be required from a strength point of view, merely to ensure proper mixing of the resin.

As with mechanical anchors, the anchor resistance depends on the rock type into which the anchors are installed. It is therefore necessary to do a number of pull-out tests on short anchors in the actual rock where the support is to be installed to determine the resistance. For this test, it is important that the test anchors be short enough to fail, as the actual failure loads have to be recorded.

The notes about the importance of the washer and nut assembly in the section on mechanical anchors are applicable to resin point anchor systems as well, as are points 2 to 4 on the recommended test procedure.

In order to optimize resin performance, it is important to allow proper mixing of the components. This is achieved by balancing the hole and bolt diameters – the bolt should be between 4 and 8 mm smaller in diameter than the hole. This guide is based on practical experience. Commonly used systems are 16 mm bolts in 22 mm holes or 20 mm bolts in 28 mm holes. The combination of 25 mm holes with 16 mm rebar is also in use on some mines but is discouraged because of inconsistant resin mixing.

In beam creation applications it will often be necessary to apply relatively high pretension to the roof. It is then sometimes necessary to use thicker bolts, say 25 mm diameter.

In several situations, the system elements are determined by ease of drilling into the roof. There are situations where it is impractical to drill holes thinner than 28 mm, and in those situations 20 mm steel has to be used to allow proper resin mixing, whether or not the strength of the 20 mm bar is required. It is paradoxical that in several suspension type systems where a relatively light load is to be supported, the overlying roof beam is a strong sandstone into which 28 mm holes are drilled and 20 mm steel has to be used when 16 mm steel would have been adequate.

From the foregoing discussion, it can be deduced that steel with circular cross section can be replaced by other profiles – the important provision is satisfactory and consistant mixing of the resin. When alternative profiles with smaller cross sectional area (like quad-bar) are used, it is important to check that the actual strength of the steel member conforms to the requirements. It is also important to ensure that the combination of profile and direction of spinning during mixing is such that the resin is not displaced down the hole.

3.2 Beam creation

It is only theoretically possible to achieve beam creation with point anchor elements. Provided the required amount of pretension can be supplied, the beam will be stable – but only for as long as the pretension is maintained. Pretension is usually lost shortly after installation, by anchor slippage and/or frittering of the roof strata underneath the washer plates. When the pretension is lost, the normal stress on the lamination interfaces is also lost and the layers are free to slide. The inter laminae sliding will continue until the rock makes contact with the steel body of the bolts. This is after several millimeters of displacement (in the region of 6 to 16 mm relative displacement), and in several beam creation situations fractions of millimeters of displacement are sufficient to result in beam failure.

Full column dual speed resin systems are therefore suggested. With those, the pretension is locked in once the resin has set, and the void in the hole is filled by resin which restricts lateral inter laminae displacement. All the requirements mentioned in the section on point anchor resin systems are applicable, with the exception that there is now less emphasis on the washer assembly.

It will be prudent to place the same requirements on the washer assembly, but if significant savings can be achieved by using a washer assembly that is 20% less efficient, there is no real reason not to use the weaker washer. Theoretically, it is possible to create a stable beam without any washer at all, but because resin mixing is seldom perfect at the bottom end of the hole, practice dictates that there should be a washer of some description at least.

In beam creation, it is important to restrict the thickness of resin in the hole to the minimum. The thicker the resin, the more it can compress and consequently the greater the inter laminae slip will be.

The annulus must therefore be a minimum. So far, using commercially available products, it has been found difficult to achieve a smaller annulus than 5 mm – i.e. 20 mm rebar into a 25 mm hole, for any bolt longer than 1,2 m. Amongst other things, this is a function of the coarseness of the filler used in the resin. The coarser the filler, the more difficult it becomes to insert a bolt into a thin hole. Note that there is a trade-off in this situation, because in general the coarser resins are both stronger and stiffer. Both coarse and fine resins are available in South Africa.

The dual speed resins also suffer from the disadvantage that the order of resin capsule insertion into the holes must be correct, and that no matter what the length of the hole, at least two capsules are required. It is also not always possible to get the exact lengths of capsules that the system requires and consequently the anchor portions often have to be longer than required. This has negative cost consequences.

In Australia, single capsules with a fast resin at one end and the rest a slow resin, are commonly used. At the time of writing, they are not yet commercially available from the South African manufactures, although production trials are aleady under way.

3.3 Characteristics of resin

The resin used for roof support is supplied in a two component capsule, in which the resin (a poly unsaturated polyester) with a finely ground limestone filler is separated from the catalyst. The membrane is made of polyester material in which tears propagate easily once the material has been ruptured.

When the resin is mixed with the catalyst, it sets off a chemical reaction that causes the resin temperature to increase to approximately 80°C. At this elevated temperature, the short molecular chains join to form long chains and the resin's state changes from fluid to solid. During the joining up process, the links are weak and if broken, cannot recover.

It is therefore vitally important not to disturb the resin during the "holding" period.

The second important point is that the whole process is temperature controlled. If the resin is cold, the linking up process takes longer and consequently the holding period should be increased. This is especially important in the winter months, and even more so for sections working close to intake shafts. Resin should be stored in the section for at least 24 hours before being used, to reach ambient temperature.

Even then, it is often necessary to increase the holding time in winter. Several mines have increased their holding times for the year round to the maximum holding time in winter, to avoid confusion. Fig II shows the effect of ambient temperature on the reaction time of the resin. The information from which the diagram was constructed was supplied by Fasloc Resins (Pty) Ltd.

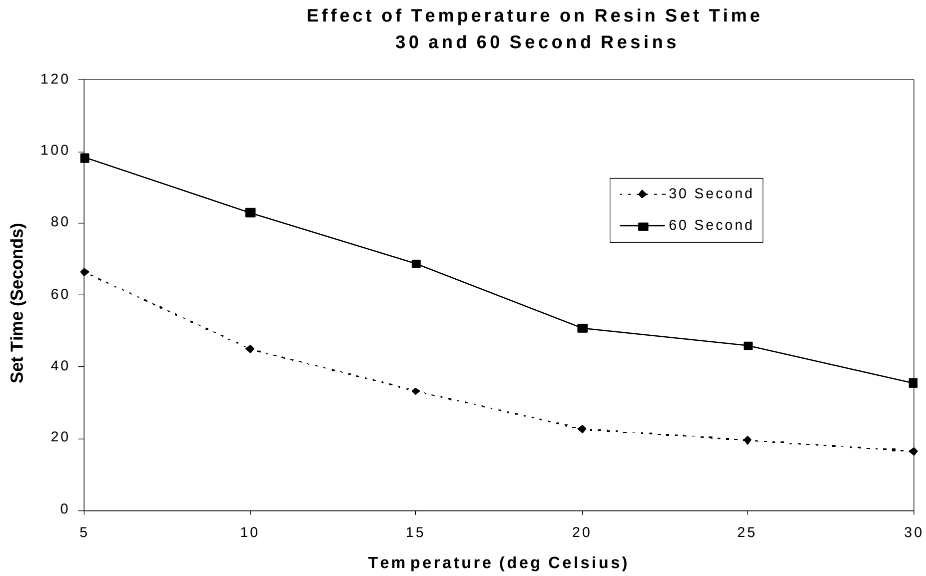


Figure 13. The effect of temperature on the gel time of commonly used resins.

Some attention needs to focus on the mechanism by which the bolt is spun to mix the resin. In most cases this is still done by means of a crimp nut. The nut is usually made of mild steel and the steel is high tensile 450 MPa (or higher) yield strength. The threads on the nut are deformed (crimped) to fail at a pre-determined torque, usually 50 to 70 N-m.

The crimp should be strong enough not to fail during the resin mixing phase. Once the resin has set, it should fail at a torque low enough not to damage the steel. The range of 50 to 70 N-m has been found satisfactory in practice.

There are serious disadvantages associated with crimp nuts. This has mainly to do with the degree of deformation of the threads. Small deviations can result in the failure torque being outside the acceptable range. If it fails prematurely the resin will not be properly mixed and if it is too strong, it will either result in bolt damage (if the roofbolter's to upper torque limit is set too high) or the roofbolter will stall before the bolt is tensioned if it is set too low.

There have been attempts at using alternatives to crimp nuts, unfortunately none less troublesome than crimp nuts themselves.

One alternative was to mold a resin cap into the nut, but it was found difficult to achieve a consistent thickness of the cap. Another was the shear pin, where the problem was that small remaining bits of pin after failure wedged themselves between the nut and the bolt, destroying the tread.

The notable exception is the nib bolt illustrated in Figure 14. The standard nut is threaded onto the bolt, then the thread on the bolt is deformed behind the nut. Spinning is done counter clockwise, and after the resin has set, pretensioning is done clockwise. This system is robust, being relatively insensitive to the degree of deformation on the threads, and as there is no tampering with the nut, there are no impedances to the pretensioning action.

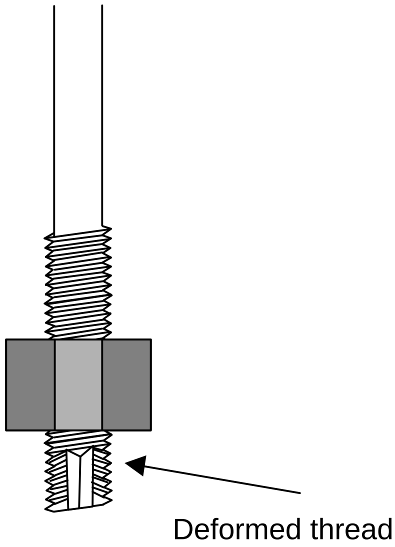


Figure 14. Sketch of the nib bolt.

This system requires that roofbolters should be able to rotate the steel clockwise and counter clockwise. On most roofbolters this merely requires that the existing counter clockwise rotation capability should be re-instated. There is also a tendency for the chuck of the roofbolter to screw out with the anti-clockwise rotation, but this is usually easily fixed in the workshops.

3.4 The size of washers and the use of head boards

Whether to use flat washers, shaped washers, spherical seat washers, large or small diameter washers, steel or timber head boards, etc depends on the function of the support, the nature of the immediate roof and the life expectancy of the excavation. Expensive spherical seat washers in a simple suspension application is wasted while using flat washers with inclined bolts in a beam creation system negates much of the effect of the system. The following table supplies general guidelines.

Table 1. General guidelines for washer types.

Support method	Nature of roof	Life of excavation	Washer type
Suspension	Friable	Short	Timber head board
Suspension	Friable	Long	Steel head board or large flat washer
Suspension	Strong	Short	Small flat steel washer
Suspension	Strong	Long	Small domed steel washer
Beam creation	Friable	Short	Timber head board
Beam creation	Friable	Long	Steel head board with domed washer
Suspension, inclined bolts	Friable	Short	Large domed washer
Suspension, inclined bolts	Strong	Long	Large spherical seat washers
Beam creation, inclined bolts	Strong or weak	Long or short	Large spherical seat washers

In Table 1, a short life means a few months only, ie typical bord and pillar production panels. Anything else is regarded as long life. A small washer means 100 mm square and a large washer means larger than 150 mm square.

In general, timber head boards play an important role in preventing falls of friable roof over the short term. They should not be used for any long term application because as the timber decays or dries out it causes the bolts to lose tension, with negative consequences for the system as a whole. What is often neglected when timber head boards are used with resin anchors, is to shorten the holes, resulting in anchor loss. Sections 4 and 5 contain more detailed discussions on hole lengths.

4 Installation techniques

Due to the importance of this section it will be sub divided into three parts for mechanical anchors, resin anchors with crimp nuts and resin installations with nib bolts. Each will form a complete section, duplicating the common items to avoid any possible confusion.

4.1 Mechanical anchors

Step 1: Position temporary supports according to the mine standards.

Step 2: Drill hole 5 cm shorter than total bolt length (or the length of the steel minus the thickness of the nut plus the washer and head board if it is used).

Step 3: Remove plastic cover from shell, twist shell lightly by hand so that it just begins to expand.

Step 4: Place bolt in hole

Step 5: Lift roofbolter chuck assembly, fit nut into chuck, lift all the way to the top, drop back 2cm.

Step 6: Rotate bolt slowly to expand anchor.

Step 7: When resistance increases, continue rotating until crimp breaks.

Step 8: Continue rotation until roofbolter stalls, or washer visibly bites into roof.

4.2 Resin anchor with crimp nut or shear pin

Step 1: Position temporary supports according to the mine standards.

Step 2: Drill hole 5 cm shorter than total bolt length (or the length of the steel minus the thickness of the nut plus the washer and head board if it is used).

Step 3: Insert resin cartridges into the hole, using the stopper to prevent them sliding out.

Step 4: Fit adapter to roofbolter chuck

Step 5: Insert bolt into hole - do not puncture resin capsules

Step 6: Lift boom of roofbolter, fit bolt to adapter.

Step 7: Lift boom right to the top without spinning, drop down 2 cm and commence spinning immediately. If the bolt bends during the lifting operation, rotate slowly to ease insertion. If dual speed resin is used, it is desirable to spin the bolt during the lifting operation. If single speed resin is used, the bolt should not be spun during lifting.

Step 8: Spin for stipulated mixing time – see information block on resin cartridge box.

Step 9: Stop, immediately lift bolt to top of hole and hold motionless for the stipulated waiting time (Note: the waiting period, incorporating any additions to cater for lower temperatures, should be prescribed per mine standards).

Step 10: Rotate again until crimp breaks and nut rotates up the bolt.

Step 11: Continue rotation until roofbolter stalls or washer visibility bites into roof.

4.3 Resin anchor with nib bolt.

Step 1: Position temporary supports according to the mine standards.

Step 2: Drill hole 7 cm shorter than total bolt length (or the length of the steel minus the thickness of the nut plus the washer and head board if it is used).

Step 3: Insert resin cartridges into the hole, using the stopper to prevent them sliding out.

Step 4: Fit adapter to roofbolter chuck

Step 5: Insert bolt into hole - do not puncture resin capsules

Step 6: Lift boom of roofbolter, fit bolt to adapter.

Step 7: Lift boom all the way to the top without spinning, drop 2cm and commence spinning counter clockwise immediately. If bolt bends during lifting operation, rotate slowly to ease insertion. If dual speed resin is used, it is desirable to spin the bolt during the lifting operation. If single speed resin is used, the bolt should not be spun during lifting.

Step 8: Spin counter clockwise for prescribed mixing period – see information block on resin cartridge box (Note: the waiting period, incorporating any additions to cater for lower temperature, should be prescribed per mine standards).

Step 9: Stop, immediately lift the bolt to the top of the hole and hold motionless for the prescribed waiting period.

Step 10: Switch rotating direction to clockwise, rotate nut clockwise until the roofbolter stalls or the washer visibly bites into the roof.

Note 1. For mechanical anchors and resin point anchor installations, the maximum torque setting on the roofbolters is usually 150 N-m, or such that the load transferred to the bolt is 50 kN. This should be tested in the workshop, tensioning a bolt through a hydraulic jack. The figures quoted with regard to torque settings correspond to common industry usage. This is in fact a complex matter, one that has perhaps not received the attention it deserves. For the purposes of breaking a crimp on a crimp nut and mixing the resin, the torque itself is the important parameter. For the other main purpose, tensioning the bolt, the important parameter is the tension transferred to the bolt as a result of the torque. There is no single relationship between the applied torque and the resultant tension, it being a function of the bolt diamemeter, the pitch of the thread on the bolt, the friction between the nut and the washer, etc.

Note 2. For beam creation purposes, the torque setting should be such to correspond to the required pre-tension load on the bolt.

Note 3. There are various methods to ensure that the stipulated mixing and waiting times are adhered to. The most successful appears to be red and green lights shining for predetermined time periods, fitted to camp lamp batteries and placed in the cabin of the roof bolter. The system works well for standard installations, but falls apart when small things go wrong, like a bolt slipping out of the adapter, etc. Some operators get irritated by the lights and then tend to cover them with a rag. Attempts to automate the spinning periods on the bolters have also been made, but none have been used successfully for a continuous time period.

Even with its shortcomings, the red and green lights on a cap lamp battery seem to be the best solution.

Note 4. Bolting should always start with one of the bolts on the edge, then moving progressively across to the other edge, as shown in Figure 15. The next row can then be installed starting at the edge where the previous row finished and moving progressively back to the opposite side.

(Theoretical considerations require both edge bolts to be installed before the center bolts, but underground this results in a tendency for the assistant to take a short cut under unsupported roof to the opposite side, and the repeated long tramming distances between installations delay the process.)

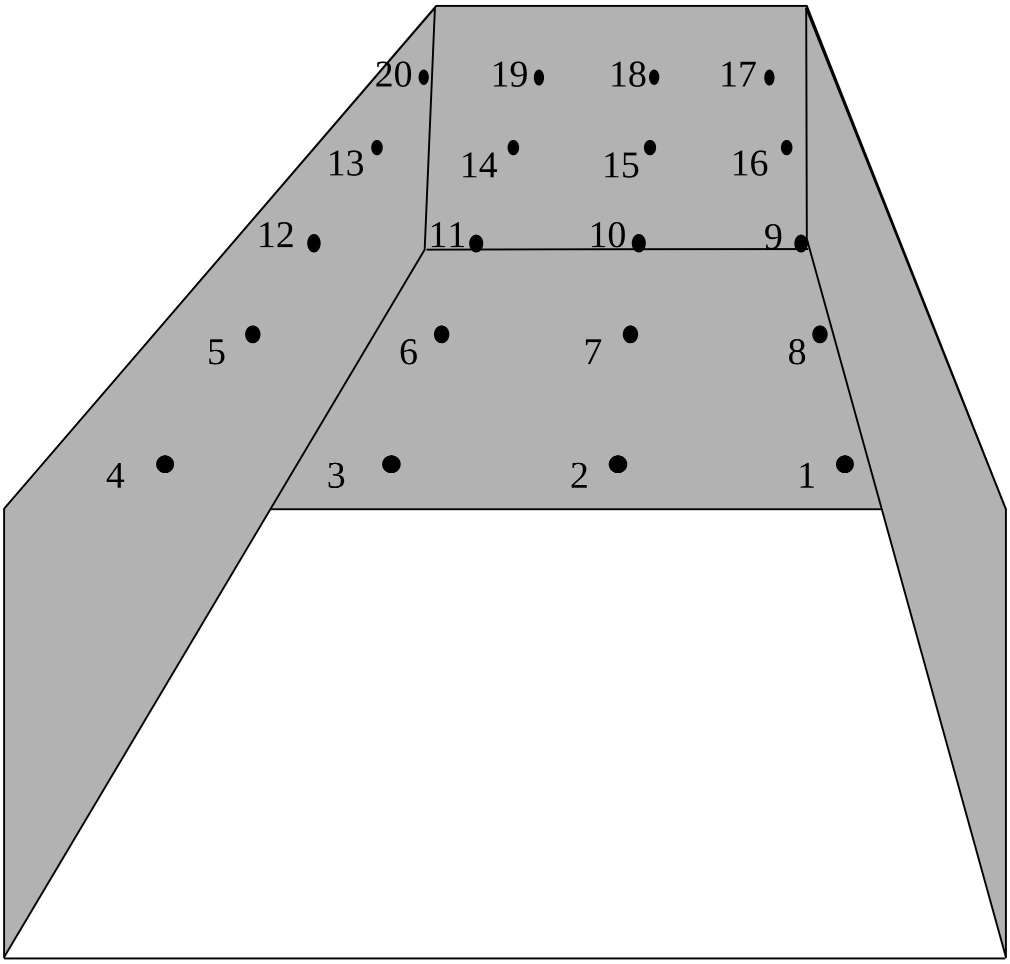


Figure 15. The recommended order of bolt installation.

Note 5. The installation procedure for continuous miners with on board bolters is the same as for stand alone bolters. On board bolting is just much more effective because it is done very close to the face and because it is possible to install the side bolts first without losing time or taking risks in the process.

5 Monitoring

Monitoring is the final step in the first cycle of the roofbolting process. It is a continuous process, not an act, as much part of the bolting exercise as drilling a hole into which to install a bolt. It should consist of three parts: monitoring of the support material prior to installation, monitoring of the suitability of the overall support system underground and monitoring of the quality of installations.

5.1 Monitoring of support material prior to installation.

The pre-installation monitoring should be done in three phases: prior to delivery in the factory, after delivery but prior to dispatch underground on the mine and finally after dispatch underground but prior to installation in the section.

5.1.1 Factory inspections

For the steel, unannounced factory visits should be made periodically and samples collected for destructive testing on the steel and checks on the dimensions of the bolts and the nuts.

Likewise, at the resin factory the gel times of the raw materials, quality certificates and delivery dates of the raw materials and storage of the finished products should be checked.

In both cases the quality control procedures of the manufacturing process and quality control reports should be inspected.

Operators should be asked direct questions regarding quality control procedures.

5.1.2 Post delivery monitoring on the mine

A pre-determined number of bolts - one per two thousand has been found sufficient according to Oldroyd (1998) - should be randomly collected for the testing of material quality. This can be done by means of non destructive testing to the required yield load. Dimensions of bolts should be checked with a vernier and the breaking torque of crimp nuts should be tested with a torque wrench. Bolts should be visually inspected for straightness and the angled crop at the end of bolt should be clean cut, without bent portions remaining.

Resin should be checked by weighing selected samples (one per thousand, i.e. one capsule out of every 20 boxes), measuring the dimensions, visually inspecting the colour markings, performing the stiffness test (i.e. a capsule held at 45° with 30cm protruding should not buckle), visually inspecting for leakage, and testing the gel times by hand mixing approximately 5 cm of a capsule's length and prodding to check when the mixture solidifies. The expiry dates on the boxes should be checked.

Resin boxes should not be stacked higher than 6 layers of boxes. They should not be stored outdoors – ideally, the temperatures of the store room should be maintained at about $25\,^{0}$C.

5.1.3 Underground checking prior to installation

The threads on bolts and nuts should be checked for damage by the roofbolter crew prior to loading the bolts on the bolter. Out of every batch of bolts delivered underground, the miner should measure and record the length of two and visually compare the rest for deviations.

One resin capsule out of every second box should be tested for stiffness and also for gel times. Every box should be visually examined for leakage of the capsules. Upon delivery, the expiry dates should be checked. Old resin must not be used.

5.2 Monitoring support system suitability underground

This section will be presented under two headings, i.e. observations of roof conditions indicating system deficiencies and measurements to give prior warning of collapse.

5.2.1 Observations indicating deficiencies of system

Open tensile crack approximately in centre of roadway, running along length of roadway (see Figure 16).

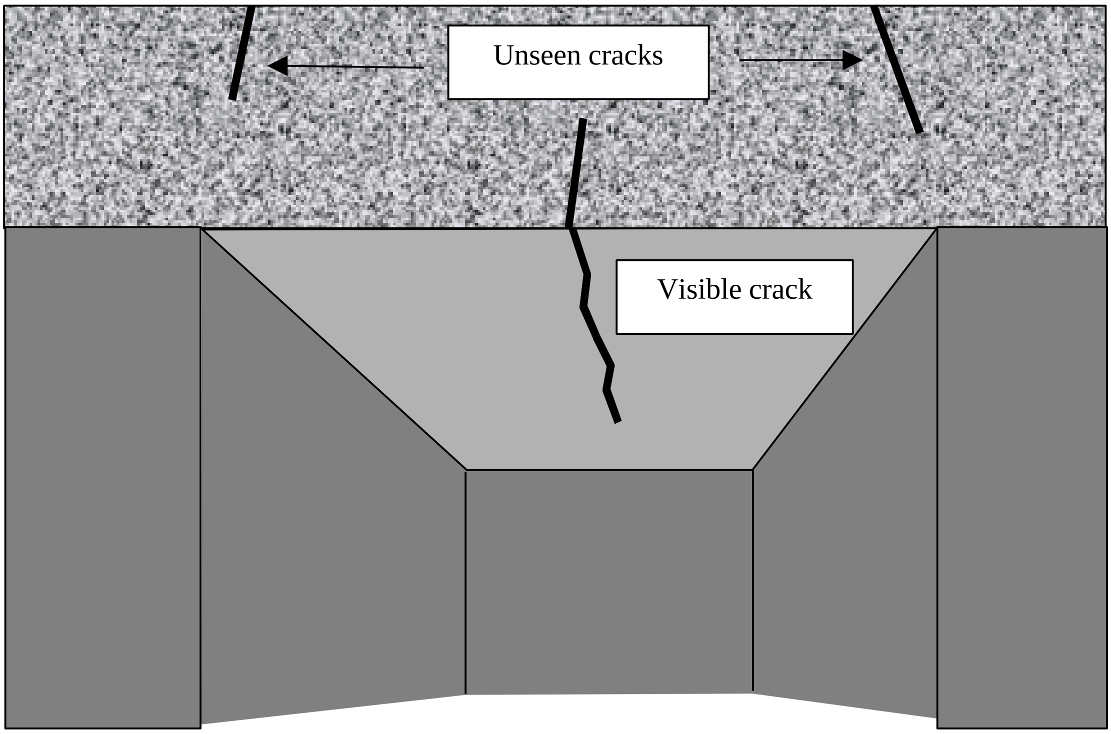


Figure 16. A tension crack, indicating one of the most dangerous situations underground.

This may be caused by bolts being too short, bolt spacing too wide or a resin problem. The way to isolate the exact cause is to calculate the weight of fallen roof per bolt (height of fall x bolt spacing in row x row spacing x 25). If the weight exceeds the designed anchor resistance of the portion of the bolt that pulled out (i.e. bolt length – fall height) then the bolts are too short or the spacings too wide. If the reverse is true, the resin shear resistance is less than designed: this indicates an installation problem (mostly under-or overspinning), a temperature problem, a resin material problem or a change in roof characteristics (i.e. the prior resin tests were done in sandstone and the real roof is mudstone).

Roof falls, bolts pulled out (see Figure 17)

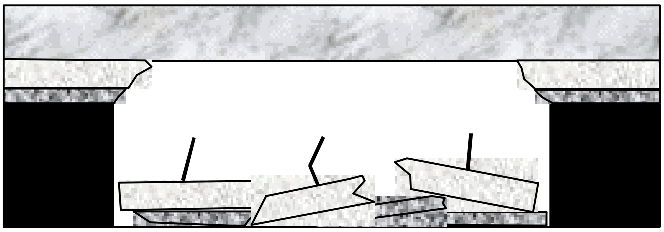


Figure 17. A fall where the bolts pulled out of the roof.

Obviously, if the fall height exceeds the bolt length in a suspension situation, the bolts are too short or the overlying sandstone beam failed because of excessive road width or the beam became thinner.

If the fall height exceeds the bolt length in beam creation, the beam failed because one or more of the system elements are sub optional (pre-tension, time of installation, spacing, etc).

Another potential cause is excessive loading caused by roof buckling, as described inter alia by Iannachioni etal (1998). This will for instance be the case where high horizontal stresses occur. While not common in south Africa, it has been observed in isolated cases in the vicinity of geological disturbances. Where this is the case, there will be other indicators of high horizontal stress, like guttering.

Falls occur between bolts, bolts hang free (shown in Figure 18)

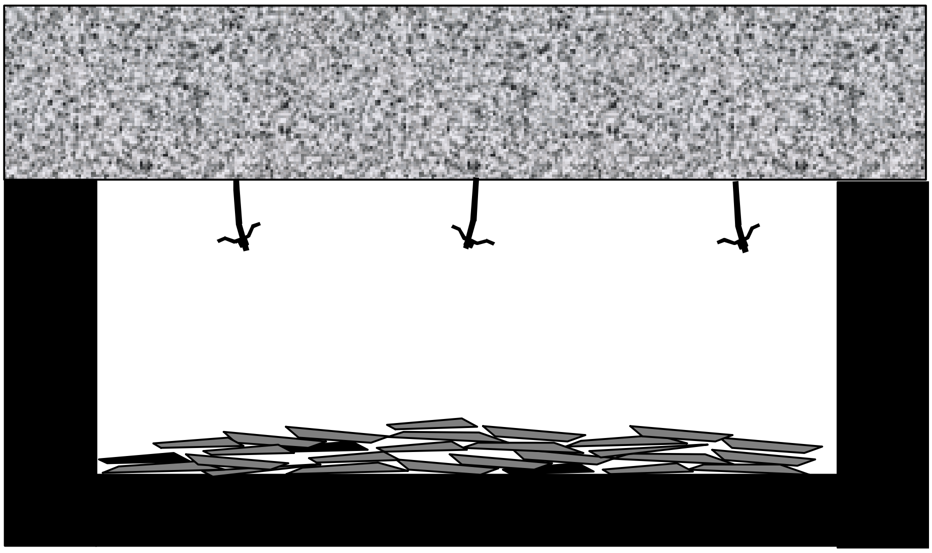


Figure 18. A fall of roof with free hanging bolts.

This usually indicates that the bolt spacing is too wide or more commonly that the bolts were not properly tensioned. Improper tensioning may be caused by a crimp nut problem, incorrect setting on the roof bolter, operator error or holes drilled too short. The latter will be indicated by the position of the nut - if it is at the end of the thread, too short holes is the problem.

Obviously, if the intended support should consist of full column resin, inspection of the free hanging ends should indicate whether or not the holes were filled.

If the bolts are broken, this indicates either that the bolt spacing is too wide or that the steel is too weak or too thin. Again, comparing the weight of fallen roof per bolt to the prescribed strength of the bolt will supply the answer.

Another common cause of this phenomonen is that the washers are too small or even too thin. It is also caused by wooden head boards either breaking or merely shrinking over time.

Falls between bolts, rock “columns” on top of washers (shown in Figure 19)

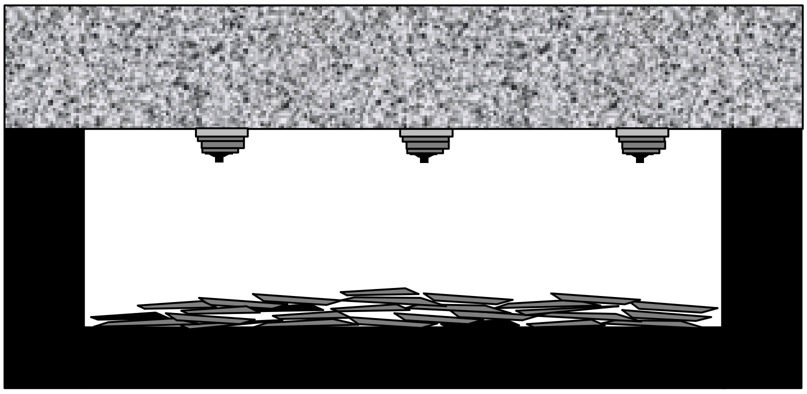


Figure 19. Fall of roof caused by excessive bolt spacing.

The predominant cause of this problem is that bolt spacings are too wide. This is again either an operator error or a design error, that can occur when the individual roof laminations become thinner and the support system is not adapted.

Another common cause is that washers are too small.

Guttering - thin horizontal slabbing, usually in the corner on one side (shown in Figure 20)

Figure 20. Guttering, caused by failure of the roof in horizontal compression.

Guttering is caused by the horizontal compressive stress in the roof exceeding the compressive strength of the roof rock. It may be caused by a high horizontal stress field or, especially if it is accompanied by a tension crack, by bending of the roof beam, in which case the road width should be decreased.

In all cases, however, the indicators of high horizontal stress only show that the horizontal stress has exceeded the compressive strength of the roof material. This may be the result of either a weakening of the material or an increase in the magnitude of horizontal stress. It could conceivably also be a combination of the two, but it does not necessarily indicate a sudden increase in the magnitude of horizontal stress.

For convenience, the probable causes of roof damage are summarised in Table 2 below. The "Seriousness" rating in the table reflects the seriousness of common causes of deviations, where a rating of 1 indicates the most serious potential problems arising from the deviation. The "Alarm" rating indicates the relative degree of danger the observation represents, a rating of 1 being the most dangerous.

Table 2. Summary of likely causes of commonly observed roof damage.

Cause / Observation	Roadway too wide	Spacings too wide	Bolts too short	Insufficient tension	High horizontal stress	Weak anchor	Weak steel	Support installed too late	Alarm rating
Tension crack	●		◑	○		○		◑	**1**
Guttering	◑				●			◑	**1**
Falls, bolts broken	○	◑			○		●		**2**
Falls, bolts pulled out	◑		○		○	●			**2**
Falls higher than bolt length	◑		●		○			○	**2**
Falls, bolts hanging free		◑		●		○		○	**3**
Falls, rock columns on top of bolts	○	●						◑	**3**
Seriousness rating	**1**	**3**	**4**	**6**	**4**	**5**	**7**	**2**	

Most likely cause ●

Potential cause ◑

Possible cause ○

5.2.2 Measurements to give prior warning of instability

There are a number of devices available with which roof deflection can be measured to supply prior warning of roof falls. They range from very accurate magnetic extensometers (very expensive) to more crude home made devices (less expensive). The optimum approach is to embark on an accurate measuring exercise to pin point the characteristic roof behaviour on a mine (or in each geotechanical district of a mine if there are more than one) and then to do less accurate customised monitoring in the sections, based on the results of the more accurate investigation.

The accurate investigation entails drilling long holes (typically more than 5 m) into the roof on the face and to install magnetic anchors into those at predetermined spacings. At the same time special roof bolts fitted with strain gauges are installed. As time goes by, and the roadway is developed further, the displacements between the anchors and the build up of load on the bolts are determined very accurately. This then yields information about where and when partings develop in the roof and which parts of roof bolts get loaded more.

This information can then be used as a scientific basis to design optimal bolting systems, and it also yields information on where to measure roof displacements with simpler instruments on an ongoing manner in the sections.

This approach is common in Australia and the USA (and is winning ground in the UK) but has not yet been regularly used in South Africa. One reason for this is possibly that in general roof conditions in South African collieries are better than in Australia . The same remarks can be applied to the USA, but where this type of investigation tends to be done by geotechnical consultants in Australia, it is commonly provided as a service by roof bolt suppliers in the USA.

The following slightly cruder approach is suggested for South African collieries until such time as the means to perform the more accurate analysis becomes commonly available and acceptable:

Observe the height of roof falls or drill holes into the roof and observe the positions where separation occurs with a petroscope (a relatively cheap instrument that enables one to visually observe the sides of a hole), illustrated in Figure 21

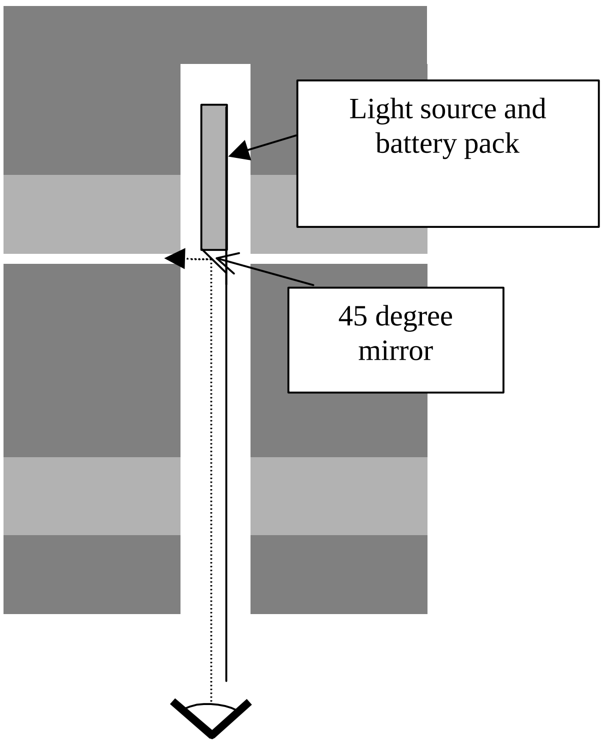


Figure 21. The petroscope.

This indicates a failing roof beam, and may be caused by the roadway being too wide (the most common cause) or, in a situation where the support is intended to create a beam, that one or more of the system elements is sub optimal (i.e. bolts are installed too late, bolts are too short, pretension is too low or bolt spacing is too wide).

When this crack is seen, bear in mind that the tensile stress on the upper side of the beam (that cannot be seen) is twice as high at the edges and that the beam there has also already failed. The immediate reaction should be to install mine poles where the failure is seen and to decrease the road width.

Install bolts with short anchors, anchored beyond the position where separation occurs (or the height of roof falls). The bolt should protrude by approximately 10 cm and a short length – about 5 to 10 cm - of pipe should be fitted to the mouth of the hole, as shown in Figure 22.

Measure the distance that the bolt protrudes and plot the results against time. Failure is imminent when the rate of shortening of the protruding end accelerates (see Figure 23) or when the total amount of shortening approaches the limit value obtained from Figure 24.

These installations should be done at all densely populated areas, like shaft bottom areas, underground workshops, main belt drive heads, etc and all other undefined places where there is reason to suspect that problems may occur. This includes for instance loading points in stooping. It should also be done at say 500 m intervals in main developments.

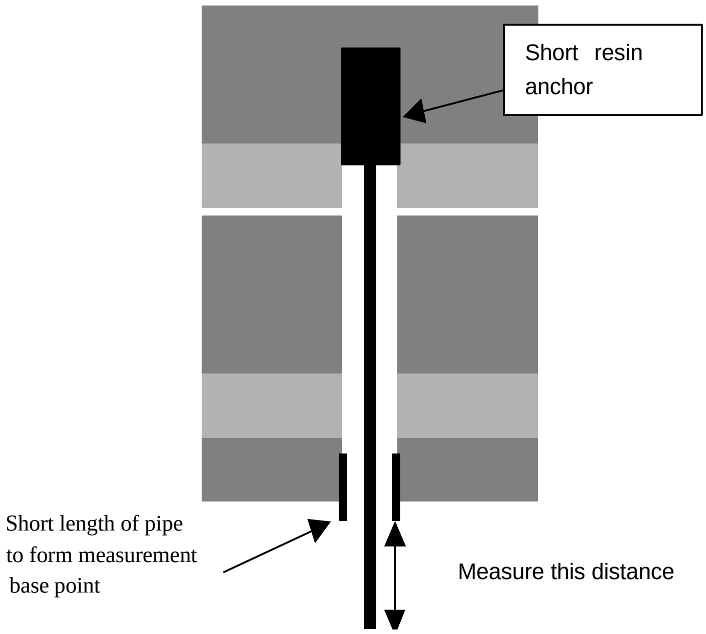


Figure 22. Diagram illustrating the principle of a simple yet effective extensometer.

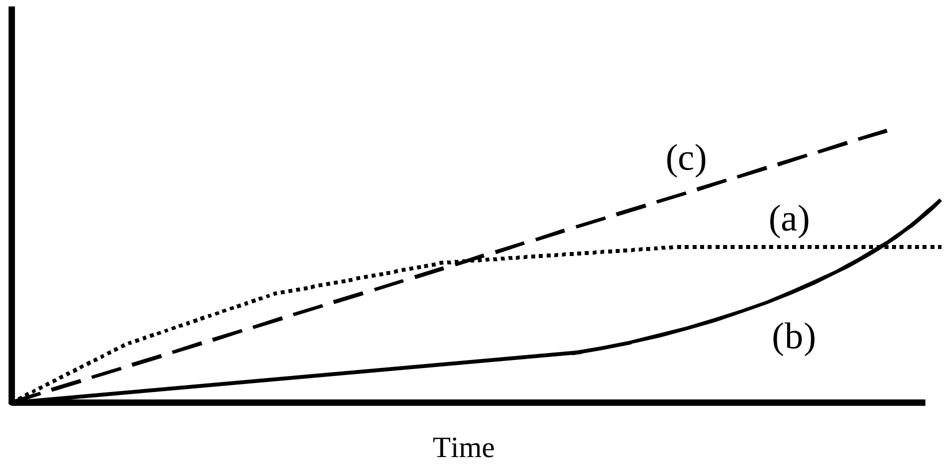
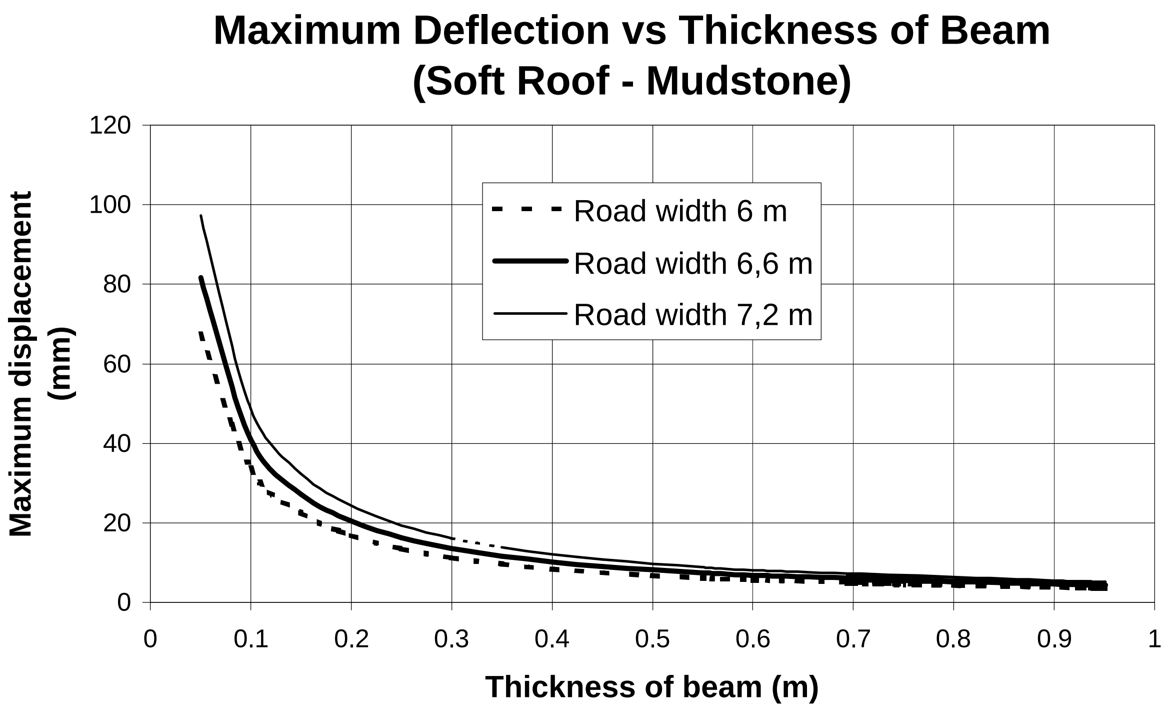


Figure 23. ***Three basic types of displacement vs time behaviour of roofs. Curve (a) represents a stable situation, requiring monitoring at long intervals. Curve (c) shows steady deformation, indicating failure when a maximum magnitude of displacement (see next figure) is exceeded. Curve (b) shows accelearation, typical behaviour indicating imminent failure.***

In the majority of working sections, there will already be satisfactory support systems. There, the major potential problem is that the roof composition may change, rendering the support system ineffective. Note that this may occur not only due to geological variations of the roof, but also for instance where the seam height changes and equipment restrictions forces one to cut higher into the roof.



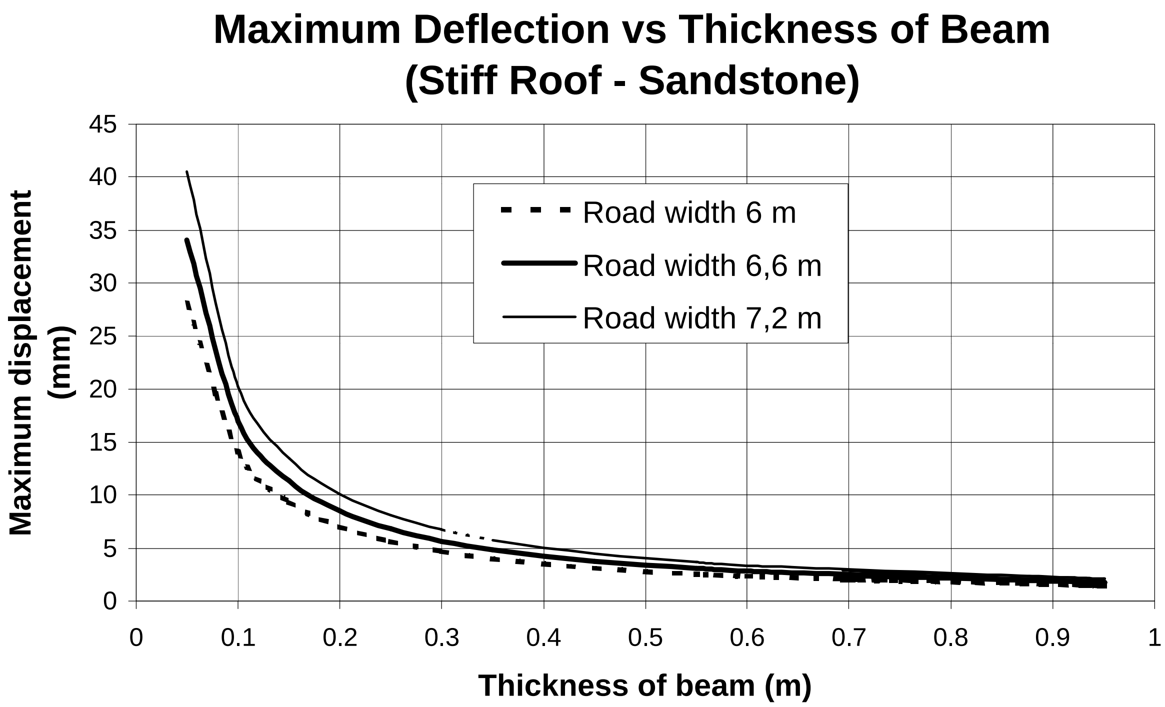


Figure 24. ***Maximum tolerable displacements in the centre of roadways for different situations. When the maximum is exceeded, failure is likely to occur. Note that stiffer roofs like sandstone can tolerate much less displacement, and thus give less warning than softer roofs like mudstone. Note further that these are TOTAL displacements, that include the displacement that has already taken place by the time the monitoring instruments are installed. It is very important that the monitoring instruments are installed right on the face.***

A very important element of monitoring is thus aimed at being continuously aware of the roof composition in all sections. Ideally, petroscope inspection holes should be drilled in every intersection.

However, some training is required to use a petroscope and it is laborious for some people, for instance those with multi-focal spectacles. It is therefore suggested to drill petroscope holes in one intersection once per month for observation by a specialist (rock engineer, geologist, mine surveyor – each with the necessary instruction) and to drill a bleeder hole of length 1,5 times the length of normal bolts in the section, and for the miner to observe the roof composition indirectly by the drill chips.

The colour and size of the chips, the rate of penetration and openings when the drill rod jumps suddenly should be observed.

Even the sound of drilling is useful to indicate a change from one rock type to another.

Yet another handy home made device is the convergence meter, which consists of a spring loaded telescopic two pipe arrangement, schematically shown in Figure 25. The disadvantage of these is that they cannot distinguish between floor heave and roof deflection, and tend to get in the way if left in any place for any period of time. However, they are excellent short term indicators of general instability with good application in stooping situations. Some, like the "ZAC" convergence meter, are fitted with compressed air canister hooters. A major advantage is that they are easily and quickly installed – it is literally a matter of standing them upright so that they make contact with the roof. They should also be used in all equipment retrieval operations after roof falls.

5.3 Monitoring of the quality of support installations

Perhaps the most important element of monitoring the quality of support installations is to observe the installation process. In this regard, overdrilling of holes and deviations in spinning and holding times of resin are the most common errors.

Incorrect hole lengths and diameters also occur frequently – the most practical way to ensure correct hole lengths is to place a short length of steel pipe over the drill rods (hose cut offs are also used, but don't last very long) so that the correct length of rod protrudes. The correct way to determine the hole length is to assemble a bolt with nut and washer (plus head board if it is to be used) and to measure the free length of the bolt.

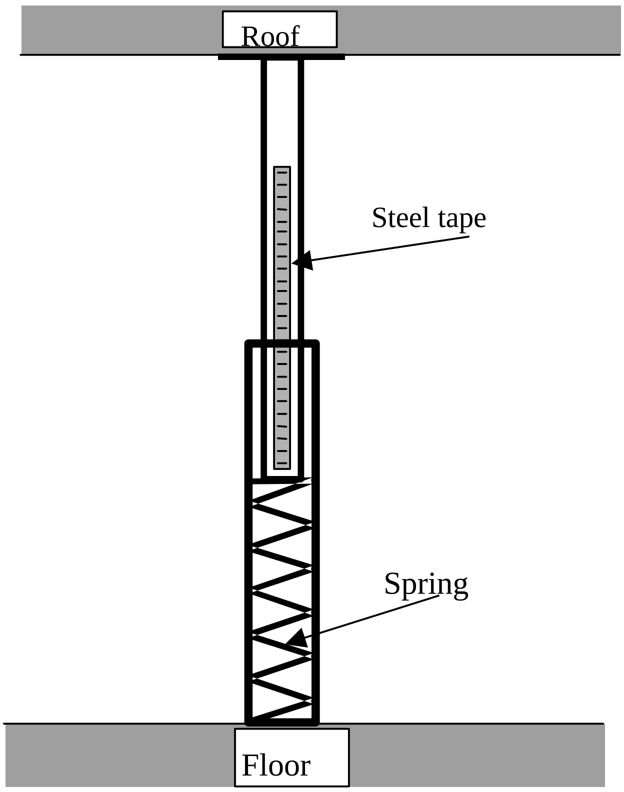


Figure 25. Another simple yet effective monitoring instrument is the convergence meter.

The economic impact of overdrilling holes should not be under estimated. Consider the following example: a large mine installs about 250 000 bolts per year. If all the holes are drilled to the length of the bolt without reducing the hole length to compensate for the nut and the washer, each hole is drilled about 50 mm too long. This results in a total overdrilled length of 12 500 m per year. In a hardish roof (a fairly common situation) this means a wastage of about 2 000 drill bits at a cost of more than R50 000 per year. The wasted resin that is paid for but does not do any work, equals about 38 000 capsules measuring 500 x 23 mm at a cost of another R50 000 plus. The direct wastage in terms of material is worth about R100 000. Much more serious is the hidden disadvantage that the entire mine is supported with anchors that are at least 100 mm shorter than intended.

The same argument is valid for incorrect drill bit diameters. It has been found that bit diameters can regularly be 2 mm wider than the stamp on the bit indicates. If bits that are intended to be 25 mm wide are in fact 27 mm wide, anchor lengths are reduced by 17%.

Resin anchor systems are particularly vulnerable to slightly inflated dimensions, as the following realistic example indicates:

A weak roof layer, 500 mm thick, in a 6,6 m wide roadway is to be supported with resin point anchored bolts at spacings of 2,5 m and three bolts per row. The required resistance in a suspension system is 68,75 kN per bolt. This can be achieved with an anchor length of 52 cm (28 mm hole, 1500 kPa resin/rock shear resistance). This requires a 31,9 cm long by 25 mm capsule.

Underground, the following happens: the capsules are 30 cm long (the closest practical length to 31,9 cm). The drill bits are 29,5 mm wide and the holes are overdrilled 50 mm. The total anchor length is now 30,6 cm and the weight it can suspend is 42,5 kN, only 62% of the design resistance. Note that even if the support designer had artificially increased the thickness of the weak layer and reduced the shear resistance of the resin to result in a system with a safety factor of 1,5, the real safety factor would still have been reduced to about 0,9.

This example would have caused one of those roof falls the cause of which cannot be easily determined. The “fat” in the safety factors are built in to cater for changes in roof material strength, thicknesses of layers, etc, in other words those variables over which man has no control. It is vitally important to control whatever we can.

It is thus required to do spot checks of the dimensions of the drill bits and bolts with a vernier and to have a template on the roofbolter for checks on every single drill bit that is to be used by the roofbolter operators. Checking the lengths of the holes and the bolts cannot be over emphasised.

Another common error is jerky operations, e.g. stopping to reposition the roofbolter halfway through the bolt insertion or spinning cycles. This invariably results in disturbing the resin during the holding period, or insufficient mixing. An interrupted installation cannot be a correct one.

The following visual observations of bolts after installation, illustrated in Figure 26, indicate certain common problems:

5.3.1 Excessive length of thread visible

In crimp nut applications, about 10-25 mm of thread should be visible, and about 20 – 40 mm in nib bolt applications. If more than that protrudes, it indicates that holes were drilled too short or that the bolt pulled out of the resin during tensioning. The latter indicates either a problem with the resin (perhaps temperature related) or that the crimp failed prematurely during mixing or that tensioning was done too soon.

The net result is the same : insufficient anchor strength.

5.3.2 No thread protruding

This usually indicates that crimps were not broken, which could be caused by either the crimps being too strong or the roof bolter torque setting too low. Crimps that are too strong to break with the correct torque setting on the roof bolter (± 150 N-m) represents a potentially very serious problem, as it is quite likely that the high torque resulted in micro fractures of the bolt, and the bolt will then fail at low loads. Typically, the bolts then fail at the position where the thread ends – this often happens during stooping.

If the crimps are broken, it indicates over drilled holes – in which case the anchor portion of the resin will be too short.

5.3.3 Loose washers

This indicates too short holes, untensioned bolts, too strong crimps or too low torque settings on the roofbolter. Even if the washer is loose due to the roof skin behind it becoming detached, that in itself was most likely caused by insufficient pretension.

The result of this is loosening of the roof layers.

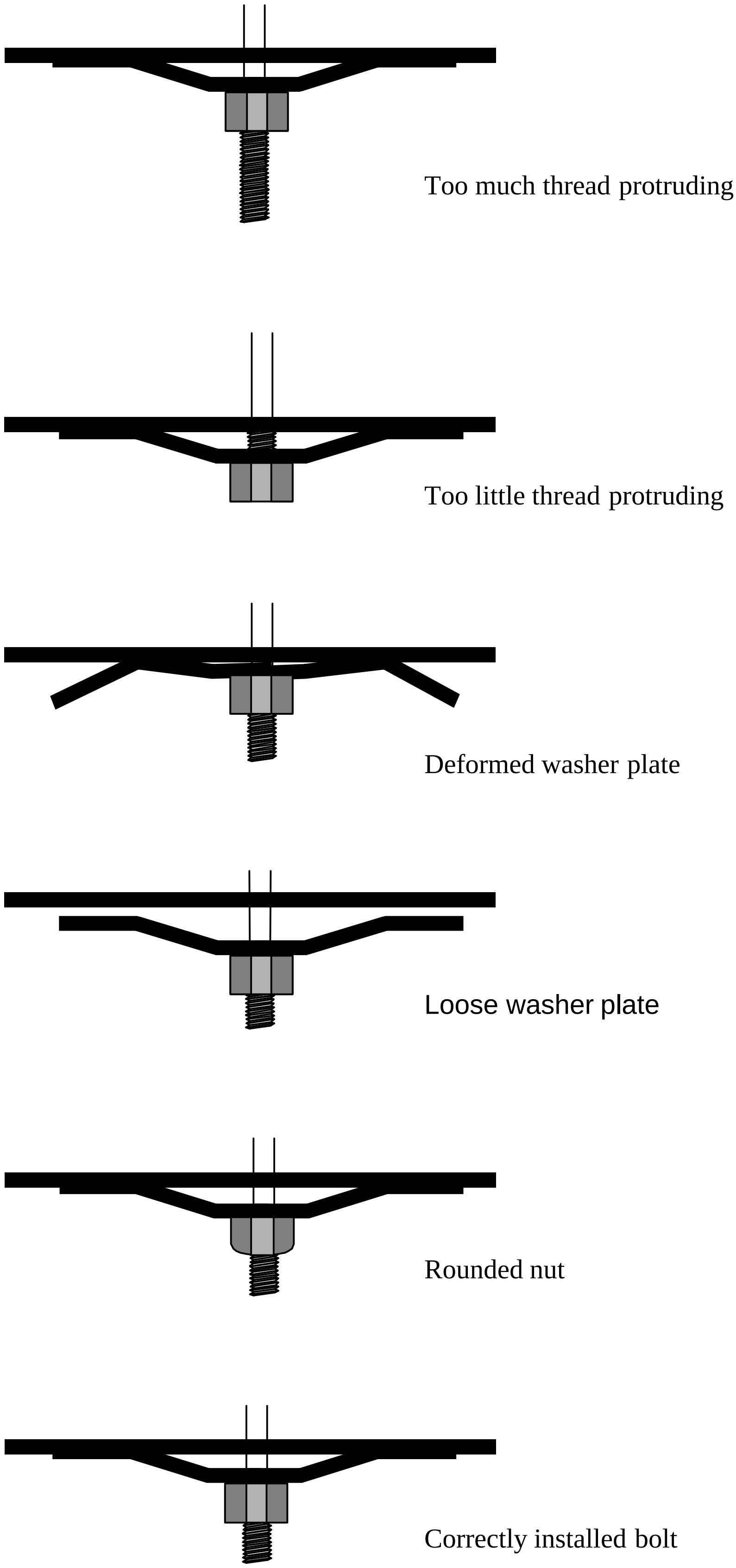


Figure 26. Commonly observed roofbolt indicators

5.3.4 Rounded nuts

This is most often caused by a worn adapter on the roofbolter, but may also be due to a too strong crimp or too high torque setting on the roofbolter. If the problem is with the adapter, the bolt could be under tensioned. If it is with the crimp, the bolt could be damaged by excessive torque and if the torque setting is too high the bolt could be over tensioned.

If the nut is rounded and the washer deformed or pulled into the roof, the torque setting is too high. If the nut is rounded and the washer not tight against the roof, it is either a too strong crimp or worn adapter.

5.3.5 Deformed washer

This indicates a washer that is too weak, too high torque setting or post installation roof movement.

For convenience, the probable causes of observed deviations are summarised in Table 3.

Table 3. Summary of probable causes of observed roofbolt installation defects.

Cause / **Observation**	Installation problems											System problems	
	Hole too short	Hole too long	Weak washer	Spin/wait times wrong	Temperature	Insufficient tension	Worn adaptor	Crimp too strong	Crimp too weak	Torque too high	torque too low	Spacing too wide	High stress
Too much thread protruding	●			◑ ⊕	◑ ⊕				○	○			
Too little thread protruding		●						◑ ⊕			○		
Loose washer	◑			○ ⊕	○	● ⊕		◑ ⊕			○		
Rounded nut							● ⊕	○		◑ ⊕			
Deformed washer			◑							○		○	● ⊕

● Most likely cause ◑ Possible cause ○ Potential cause ⊕ Most serious

5.4 Concluding remarks on visual observations

It is dangerous to base conclusions on single observations. Often, if it is an installation problem, the deviation will occur in patterns indicating that one crew is responsible. It is also important to look for combinations of deviations.

For instance, loose washers by themselves could have at least three causes, but the combination of loose washers and unbroken crimps and rounded nuts almost certainly indicate a too strong crimp. Also, if the crimp is broken and the nut is rounded and the washer is not tight, it is just about certain that the adapter is worn.

Table 4 contains a summary of the observations of the most common deviations of roof conditions and support installations, listing the possible negative consequences.

Table 4. Final potential results of non conformances

Non conformance	**System result**	**Final potential result**
Roadway too wide	Excessive tensile stress	Major collapse
Spacings too wide	Tensile stress between bolts	Falls between bolts
	Overloaded bolts	Bolts break, major falls
Bolts too short	Beam too thin	Major falls
Insufficient tension	Beam not created	Medium to major falls
High horizontal stress	Beam too thin	Several major falls
Weak anchor	Cannot suspend weight	Small to major falls
Weak steel	Cannot suspend weight	Major falls
Support installed too late	Beam not created	Major falls
	General deterioration prior to bolt installation	Falls between bolts
Hole too short	Beam too thin	Major falls
	Anchor too short	Major falls
Hole too long	Loss of resin, anchor too short	Major falls
Weak washer	Loss of suspension	Major falls
Spin/wait times wrong	Anchor too weak	Small to major falls
Temperature	Anchor too weak	Small to major falls
Worn adapter	Insufficient tension	Small to major falls
Crimp too strong	Insufficient tension	Small to major falls
	Damage to bolt	Major falls
Crimp too weak	Insufficient mixing, weak anchor	Major falls
Torque too high	Over tensioned bolt	Major falls
Torque too low	Insufficient tension	Small to major falls

6 Conclusions

The act of installing a roofbolt to stabilize the roof is the final stage of a long process, beginning with the identification of the most likely roof failure mechanism, followed by a primary design of a suitable support method and then the choice of materials and equipment. Failure of the system can occur due to an error at any stage of the process. However, well installed bolts will still perform some function even if the system design is sub optimal, while poorly installed bolts will serve no function and negate a perfectly designed system. It remains the most important step in the entire process.

The installation procedure has to be drawn up taking cognisance of the design philosophy – for instance, in beam creation the timing of installation and the application of pretension are much more important than in simple suspension applications. In deciding how to install a bolt it is thus necessary to know beforehand what the bolt is intended to do.

It has been attempted in this report to trace the whole chain of events, to treat support installation as a holistic entity. It has also been attempted to simplify monitoring by concentrating on visual observations as opposed to more sophisticated measurements. The intention is that by applying the suggested methods, any supervisor should know what the quality of his roof support is merely by looking at the roof. The evaluation of roof support should not be in the hands of a few specialists: it should become second nature to all who work underground.

Finally, the most important aspect of all is the training of roofbolt installation crews. All too often, installation procedures are laid down by a committee and forced onto operators without even explanation. Communication with the operators is vital. Two examples will be given to illustrate this point:

In one case, an audit indicated that holes were being over drilled (no threads protruded from the nuts). Instructions were given to crews to rectify the situation. The problem persisted. The crews were issued with short lengths of steel pipe to place over the drill rods, making it impossible to overdrill. They did not use the pipes. The final step, which should have been the first one, was to ask the crews why they insisted on over drilling. The answer was simple : it was not possible to insert the 1,8 m long 16 mm bolts without overdrilling the holes. The reason for that was that the mine had changed to a coarser resin to take advantage of the higher strength. What was obvious in retrospect, was simply not foreseen.

The second example was an attempt to regulate spinning and holding times by using glaring red and green lights, mounted to the roofbolter in a position where the operator could not help but see the lights. The irritation effect of working a whole day with bright lights flashing in the operator's face was never considered. The operators covered the lights with rags whenever the supervisors were not present. Again, the problem was only solved after very simple communication.

It will be wrong to allow roofbolter operators to design support systems, but it is essential to draw up the final installation procedures in consultation with them. Only when everyone understands which elements of the system are vital, is it possible to do this successfully. The key word, once more, is communication.

7 Acknowledgements

The SIMRAC organisation is gratefully acknowledged their for support to complete the work, especially the COLREAG sub-committee and the support material suppliers for their input at the workshop.

There were numerous discussions with colleagues in other countries on the broad subject of roof support. Although specific reference is not always made in the paper to general discussions, some colleagues deserve special mention for their input and explanations: Drs Syd Peng, Chris Mark, Steve Tadolini, Al Campoli, Chris Windsor, John Stankus and Jim Galvin.

The Project Report was reviewed by Dr Bernard Madden, who is likewise thanked for his input.

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APPENDIX I Coal Mine Roof Support Trouble Shooting Guide

ITASCA
AFRICA (PTY) LTD

SIMRAC

Coal Mine Roof Support Trouble Shooting Guide

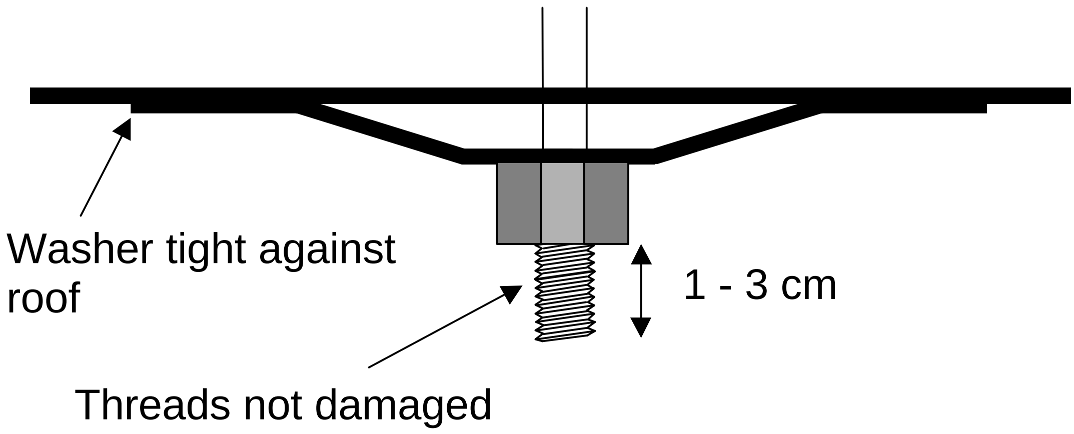
Prepared by: **Dr JN van der Merwe, ITASCA Africa (Pty) Ltd**

Date: **October 1998**

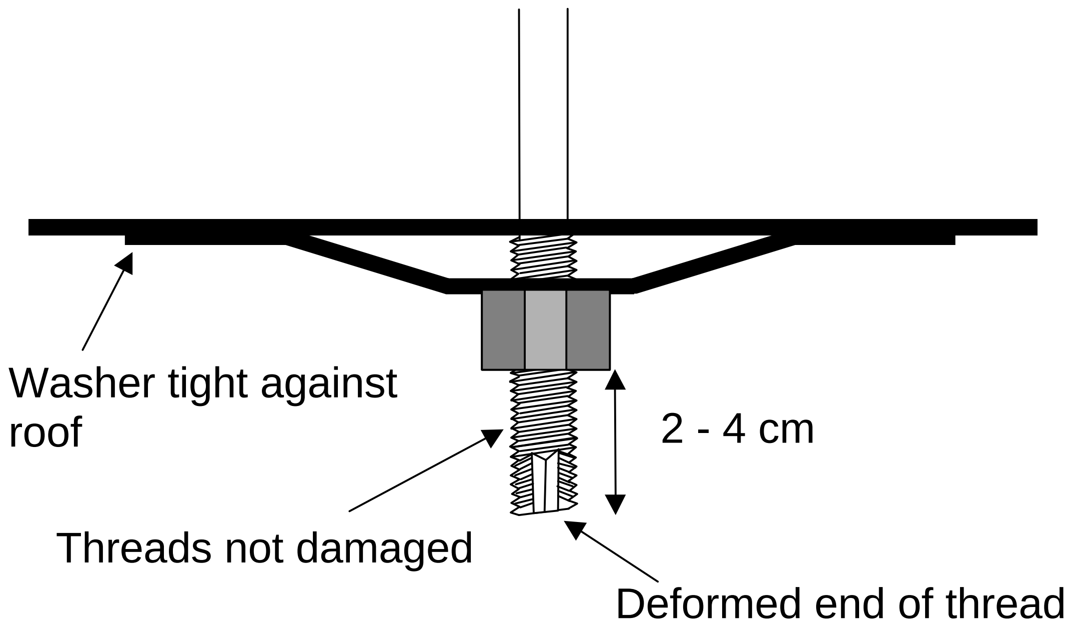
Name:______________________________

Section:	
Hole diameter	
Hole length	
Bolt diameter	
Bolt length	
Number of fast capsules / hole	
Number of slow capsules / hole	
Spinning time	
Holding time	
Bolts / row	
Row spacing	
Road width	
Mining height	
Special instructions:	

Correctly installed bolts

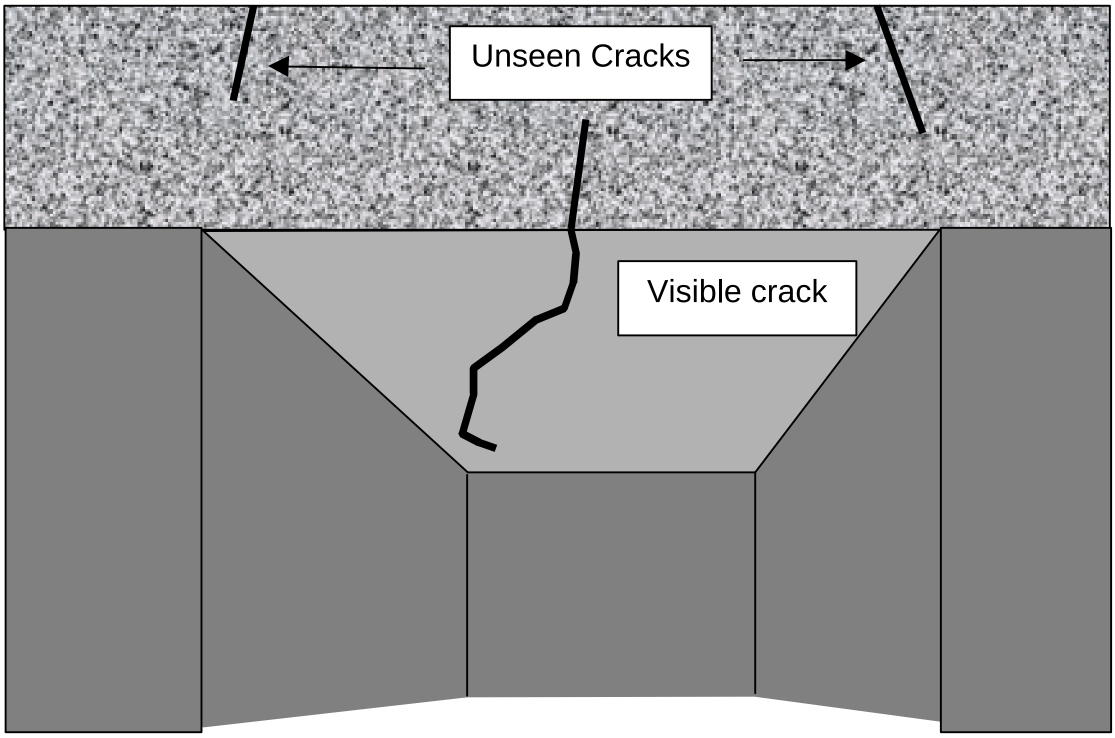


Crimp nut or shear pin



Nib bolt

Crack in centre of roadway



Possible Causes	Possible Consequences
Road too wide	**Major fall**
Geology changed	Major fall
Support went in too late	Major fall
Bolts too short	Major fall
Bolts not tensioned	Major fall
Spacing too wide	Major fall

Actions:

Decrease road width immediately
Install mine poles or other standing supports
Install long cable anchors
Call expert

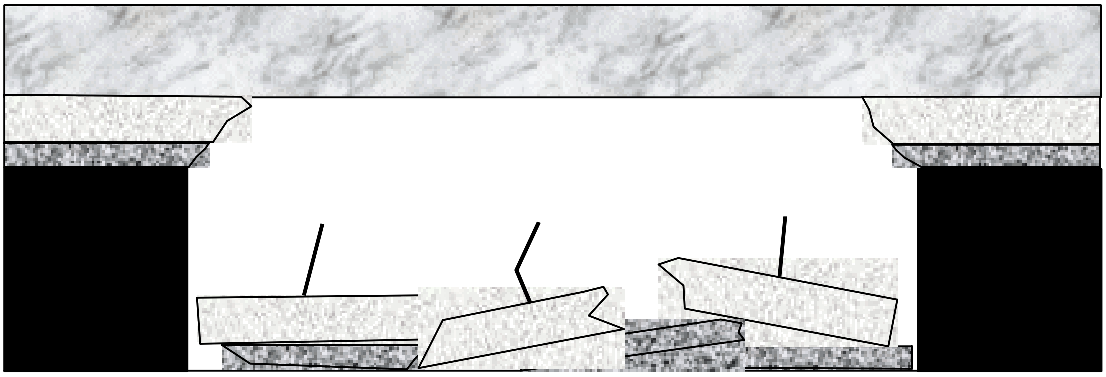
Guttering

Possible Causes	Possible Consequences
High horizontal stress	**Major fall**
Weaker strata	**Major fall**
Roof beam bending	Major fall
Bolts too far from edge	Major fall

Actions:

Decrease road width
Check bolt positions
Call expert

Major fall of roof, bolts pulled out

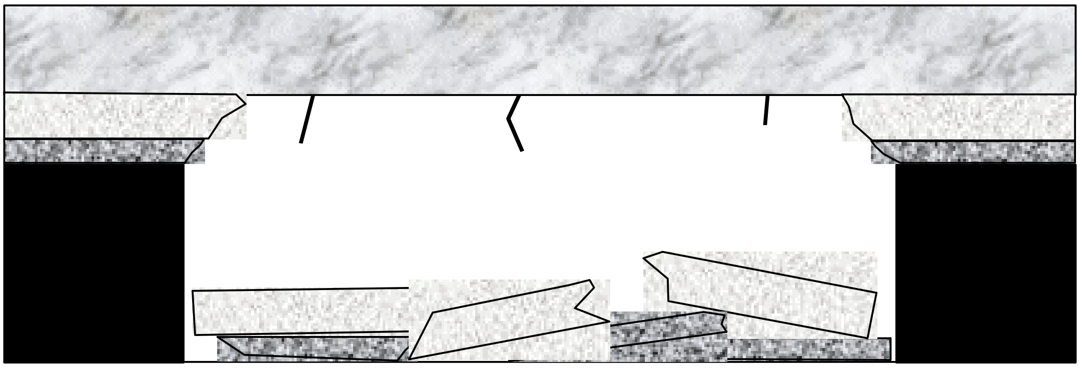


Possible Causes	**Possible Consequences**
Bolts too short	**More falls**
Roadway too wide	**More falls**
Insufficient anchor	More falls
Incorrect installation	More falls
Holes too long or wide	More falls

Actions:

Decrease road width
Check hole and bolt length
Check resin expiry date and temperature
Check installation procedure and material
Check drill bits

Major fall of roof, bolts broken

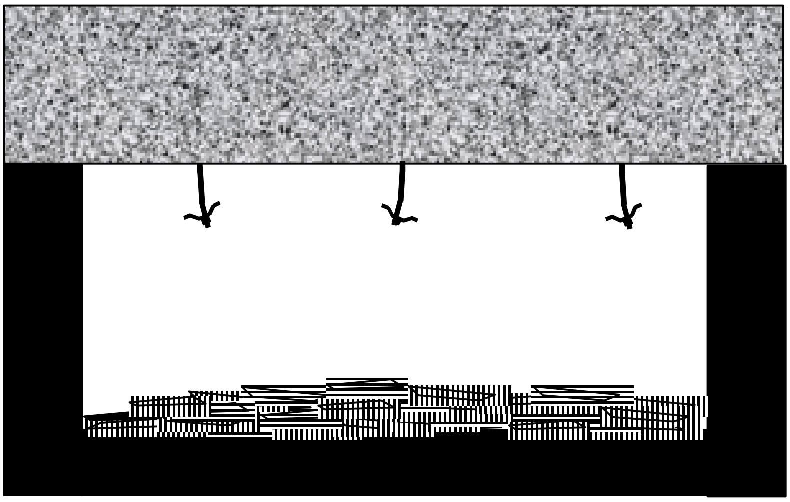


Possible Causes	Possible Consequences
Bolts weak	**More falls**
Support spacings too wide	More falls
Roadway too wide	More falls

Actions:

Check bolts, especially crimp nuts
Decrease bolt spacing
Decrease road width
Use thicker bolts

Falls of roof, bolts hanging loose

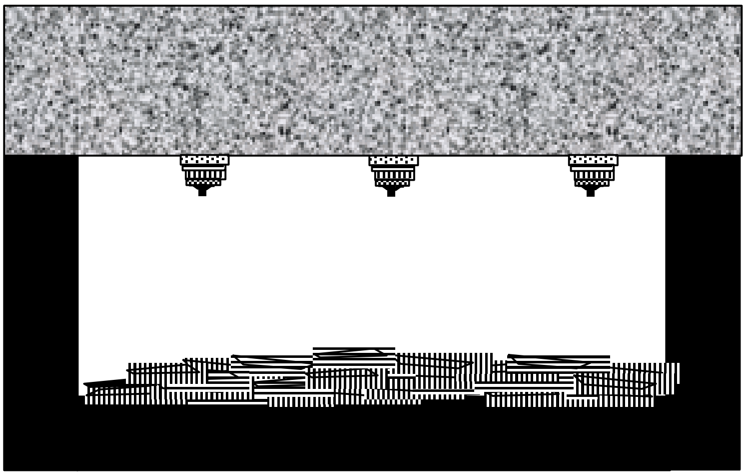


Possible Causes	Possible Consequences
Bolts not tensioned properly	**More falls**
Bolt spacings too wide	More falls
Geology changed	More falls
Washers too small	More falls

Actions:

Check installation procedure
Check crimp nuts
Check bolt spacing
Use bigger washers
Bar weak layer before installing support
Use head boards

Falls of roof between bolts, rock on top of washers

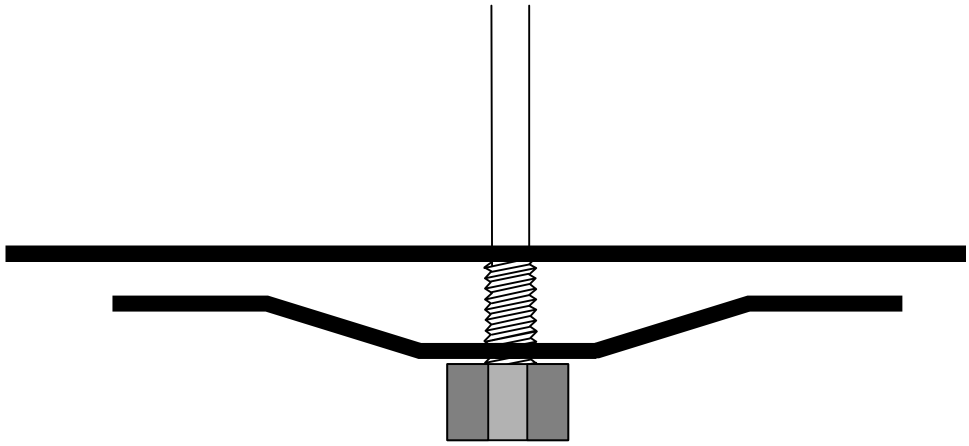


Possible Causes	Possible Consequences
Bolt spacings too wide	**Falls between bolts**
Weathering	Falls between bolts

Actions:

Decrease bolt spacing
Provide area cover
Bar weak layer before installing support

Loose washer plates, no threads protruding

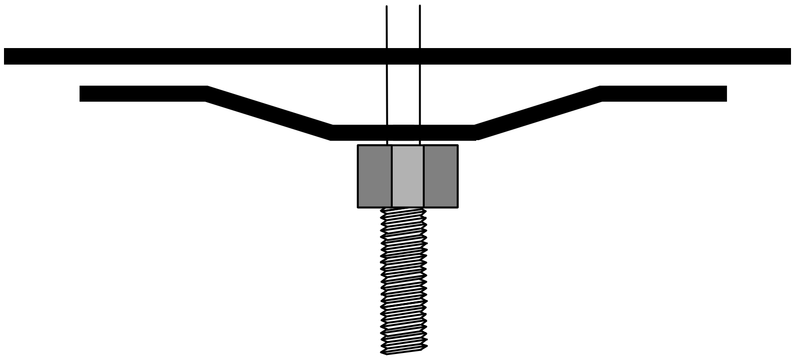


Possible Causes	Possible Consequences
Crimps too strong	Insufficient tension, falls of roof
Holes too long or too short	Insufficient tension, anchors too short
Roofbolter torque setting too low	Insufficient tension
Insufficient anchor	Loss of anchor
No or too little tensioning	Loose roof plates, falls of roof

Actions:

Check crimp nuts
Check roofbolter torque
Check hole lengths
Check installation procedure
Check resin expiry date and temperature

Loose washer, too much thread

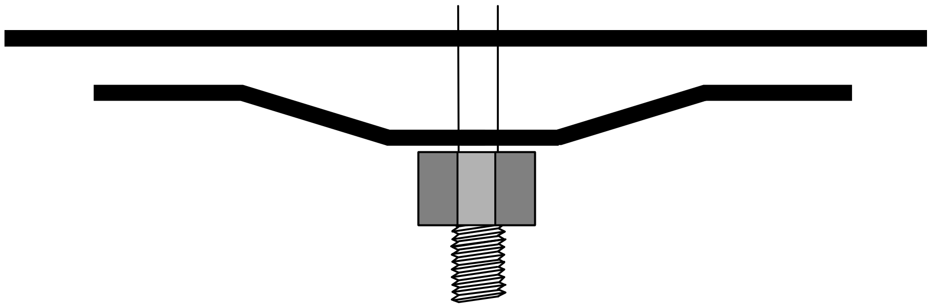


Possible Causes	Possible Consequences
Holding time too short	**Insufficient anchor**
Crimp too weak	Insufficient anchor
Anchor faulty	Insufficient anchor
Hole too short	

Actions:

Check installation procedure
Check crimp nuts
Check resin expiry date, temperature
Check hole length

Washer plates loose, correct thread

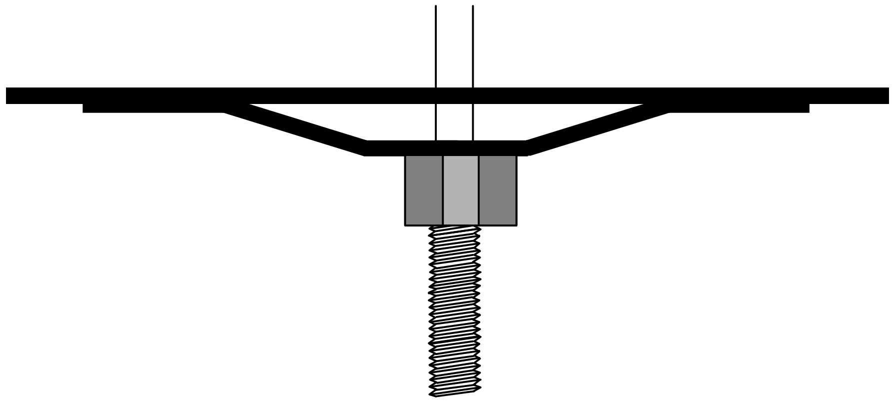


Possible Causes	Possible Consequences
Frittering of roof under washer	**Loss of tension**
Not properly tensioned	

Actions:

Retighten nuts
Re-install bolts
Check installation procedure

Washer tight, too much thread protruding (more than 3 cm for crimp nuts, or more than 5 cm for nib bolts)

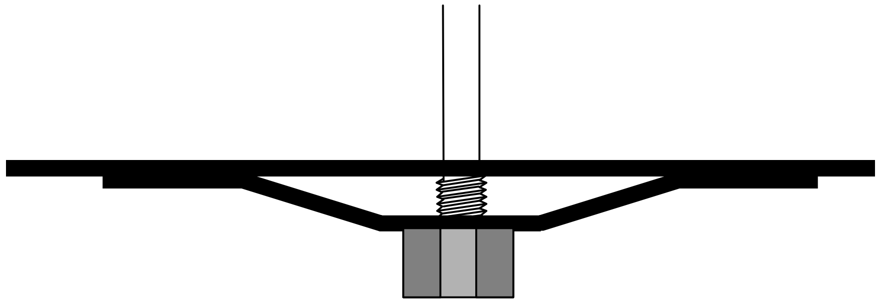


Possible Causes	**Possible Consequences**
Insufficient holding time	**Loss of anchor strength**
Holes too short	Bolts pull out
Anchor faulty	Bolts pull out

Actions:

Check installation procedure
Check resin expiry date and temperature
Check hole lengths

No or too little thread protruding (less than 1 cm for crimp nuts or 2 cm for nib bolts)

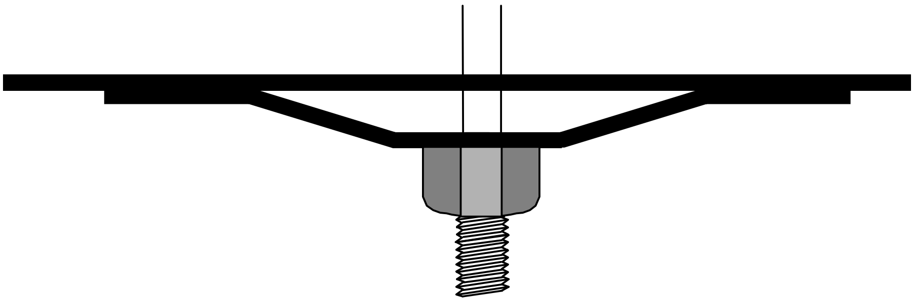


Possible Causes	Possible Consequences
Crimp too strong	**Insufficient tension, possibly damage to bolt**
Holes overdrilled	Anchor too short (resin pushed to back of hole)
Thread damaged	Bolt not tensioned
Torque setting too low	Bolt not tensioned

Actions:

Check crimp nuts
Check hole length
Check torque setting
Check threads

Rounded nuts, correct length of thread protruding

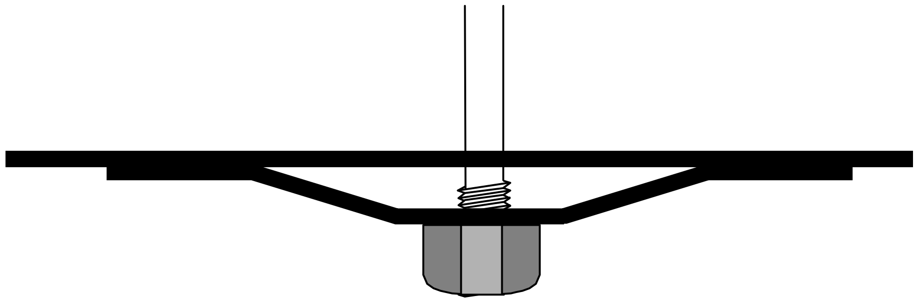


Possible Causes	Possible Consequences
Worn adaptor on roofbolter	**Insufficient tension**
Torque setting too high	Over tensioned bolt

Actions:

Check adaptor
Check torque setting

Rounded nuts, no or too little thread protruding

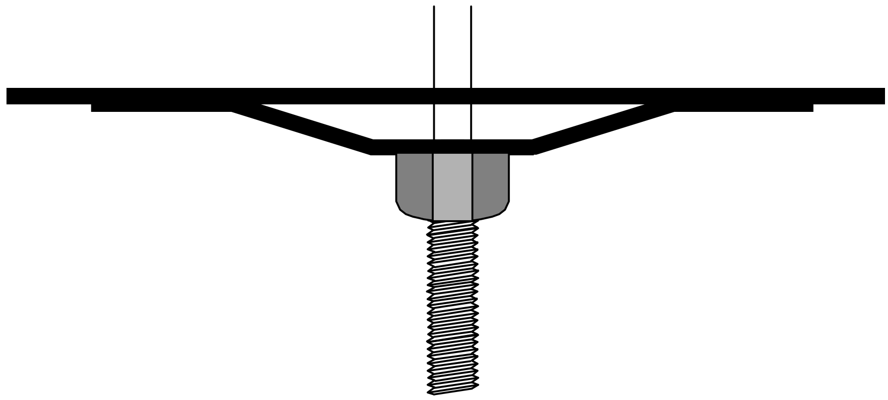


Possible Causes	**Possible Consequences**
Worn adaptor	**Insufficient tension on bolts**
Crimps too strong	Insufficient tension, possible bolt damage
Torque setting too high	Possible bolt damage

Actions:

Check adaptor
Check crimps
Check torque setting

Rounded nuts, too much thread protruding

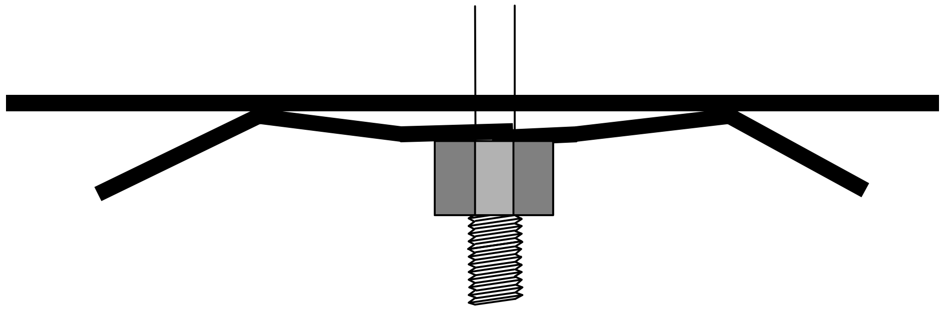


Possible Causes	Possible Consequences
Worn adaptor	**Insufficient tension**
Holes too short	Anchor too short

Actions:

Check adaptor
Check hole length

Deformed washer



Possible Causes	Possible Consequences
Roof movement after bolts were installed	**Bolt or washer may fail**
Torque setting too high	Bolt or washer may fail
New load on roof due to stooping	Bolt or washer may fail
Washers too thin	Washer may fail

Actions:

Check roof and ribsides for cracks
Check torque setting
Re-install support, barricade off
Check washer thickness

Probable causes of observed roof damage.

Use this guide to prioritise checklist

Observation \ Cause	Roadway too wide	Spacings too wide	Bolts too short	Insufficient tension	High horizontal stress	Weak anchor	Weak steel	Support installed too late	Alarm rating
Tension crack	●		◑	○		○		◑	**1**
Guttering	◑				●			◑	**1**
Falls, bolts broken	○	◑			○		●		**2**
Falls, bolts pulled out	◑		○		○	●			**2**
Falls higher than bolt length	◑		●		○			○	**2**
Falls, bolts hanging free		◑		●		○		○	**3**
Falls, rock columns on top of bolts	○	●						◑	**3**
Seriousness rating	**1**	**3**	**4**	**6**	**4**	**5**	**7**	**2**	

● Most likely cause ◑ Possible cause ○ Potential cause

Seriousness rating: 1 is the most serious non conformance

Alarm rating: 1 is the most dangerous situation

Probable causes of observed roofbolt defects.

Use this guide to prioritise checklist

Cause / Observation	Installation problems											System problems	
	Hole too short	Hole too long	Weak washer	Spin/wait times wrong	Temperature	Insufficient tension	Worn adaptor	Crimp too strong	Crimp too weak	Torque too high	torque too low	Spacing too wide	High stress
Too much thread protruding	●			◑ ⊕	◑ ⊕				○	○			
Too little thread protruding		●						◑ ⊕			○		
Loose washer	◑			○ ⊕	○	● ⊕		◑ ⊕			○		
Rounded nut							● ⊕	○		◑ ⊕			
Deformed washer			◑							○		○	● ⊕

● Most likely cause ◑ Possible cause ○ Potential cause ⊕ Most serious

Final potential results of non conformances

Non conformance	System result	Final potential result
Roadway too wide	Excessive tensile stress	Major collapse
Spacings too wide	Tensile stress between bolts	Falls between bolts
	Overloaded bolts	Bolts break, major falls
Bolts too short	Beam too thin	Major falls
Insufficient tension	Beam not created	Medium to major falls
High horizontal stress	Beam too thin	Several major falls
Weak anchor	Cannot suspend weight	Small to major falls
Weak steel	Cannot suspend weight	Major falls
Support installed too late	Beam not created	Major falls
	General deterioration prior to bolt installation	Falls between bolts
Hole too short	Beam too thin	Major falls
	Anchor too short	Major falls
Hole too long	Loss of resin, anchor too short	Major falls
Weak washer	Loss of suspension	Major falls
Spin/wait times wrong	Anchor too weak	Small to major falls
Temperature	Anchor too weak	Small to major falls
Worn adapter	Insufficient tension	Small to major falls
Crimp too strong	Insufficient tension	Small to major falls
	Damage to bolt	Major falls
Crimp too weak	Insufficient mixing, weak anchor	Major falls
Torque too high	Over tensioned bolt	Major falls
Torque too low	Insufficient tension	Small to major falls

Notes

APPENDIX II Roof Support Audit Guide

Roof Support Audit Guide

Mine:______________________ **Section:**________________________

Date:______________________

Present:____

Miner:_____________________ **Shiftboss:**_____________________

Mine Overseer:____________

No of roadways:____________ **Mining method:**_________________

Type of CM:________________ **Type(s) of bolter(s):**_____________

Drill (wet/dry):_____________ **Flush (wet/dry):**_________________

Single / double header:______ **No of shifts:**____________________

Section standards

Road width	
Mining height	
Bolts / row	
Row spacing	
Resin type	
Resin spin	
Resin hold	
Capsule length	
Capsule diameter	
Capsules / hole	
Hole length	
Hole diameter	
Drill bit diameter	
Bolt length	
Bolt diameter	
Crimp / nib	
Washer type	
Washer size	
Additional support	

Roof Support Audit Guide

A Observations of Installed Bolts

A.1   Road width

Measurements				Average
				Standard Deviation

A.2 Row Spacings

Linear Distance	No. of Rows	Ave Spacing	Average
			Standard Deviation

A.3 Bolts per row

Measurements						Average

A.4 Joints

Total joints found	
Total correctly supported	

Observations of installed bolts – Continued

A.5 Installed Bolts Observation Count Sheet

Too much thread	Crimp broken, too little thread	Crimp not broken	Loose washer	Rounded nut	Deformed washer	Correct
Total	Total	Total	Total	Total	Total	Total
Grand Total:						
%	%	%	%	%	%	%

B.Observations at Roofbolter

Ambient temperature:_______________
Resin temperature:_________________

B.1 Bolts											Ave	Std dev
Length												
Diameter												
Thread diameter												
Nut length												
Nut diameter												
Thread condition (Y/N)												

B.2 Resin											Ave	Std dev
Capsule diameter												
Capsule length												
Buckle test (Y/N)												
Leakage (Y/N)												
Set Time												
Condition of boxes (Y/N)												
Expiry dates												

Observations at Roofbolter - continued

B.3 At Roofbolter											Ave	Std dev
Bit diameter												
Hole length												
Spin time												
Hold time												
Capsules / hole												
Temporary Supports (Y/N)												
Barricades (Y/N)												
Length control in use (Y/N)												
Time control in use (Y/N)												
Adaptor condition (Y/N)												
Cut-out distance (estimate)												
Inspection hole drilled (Y/N)												
Extensometer (Y/N)												
Smooth installation (Y/N)												

Geological description of area (faults, dykes, joints, roof composition, petroscope observations)

Remarks: (Include on site rectifying action):

Roof Support Rating Calculation Sheet

Section:________________________   Date of inspection:______________

	Standard	Average	Std Dev	Formula	Result
Road width				$40\left[1-\frac{\text{Ave}+\text{Std.Dev.}-\text{Standard}}{\text{Standard}}\right]$	
Bolts / row				$20\left[1-\frac{\text{Standard - Ave.}}{\text{Standard}}\right]$	
Row spacings				$10\left[1-\frac{\text{Ave}+\text{Std.Dev.}-\text{Standard}}{\text{Standard}}\right]$	
% Correct installations	100		–	$30\left[\frac{\%\ \text{Correct}}{100}\right]$	
Sub Total					
Final Score = Sub Total X No. of Correctly Supported Joints / Total Number of Joints =					

Corrective Action:

(Refer back to section inspection sheets for diagnosis)

Step No	Action	Completion date
1		
2		
3		
4		
5		
6		
7		
8		
9		
10		

Date of follow-up inspection:_______________________

Rock Engineer   **Mine Overseer**   **Date**

APPENDIX III Calculation examples.

Appendix 3

Calculation examples.

The examples given in this appendix are intended to clarify the calculations done with the different formulae in the main body of the report. They are calculation examples only and should not be seen as hinting at suggested guidelines for support patterns.

Example 1: Solid sandstone beam.

Will a 20 cm thick sandstone, overlain by 35 cm of mudstone, be self supporting over a 6,6 m wide roadway?

$$n = 1 + \frac{0{,}35}{0{,}2}$$

$$= 2{,}75$$

$$t = \frac{2{,}75.6{,}6^2}{266} \qquad \text{(Equation 13)}$$

$$= 0{,}45$$

The required minimum thickness is 45 cm which is in excess of the existing 20 cm – the beam will thus fail.

Example 2: Suspension problem.

A zone of laminated material, total thickness 40 cm, is overlain by a massive sandstone layer of 2 m thickness. The laminated zone consists of alternating layers of sandstone and shale. The average thickness of the individual layers is 2 cm.

Resin or mechanical anchors are to be used. The shear resistance of the resin/rock interface has been found to be 2 700 kPa and the anchor resistance of mechanical anchors,

Use equations (13) through (20)

$$t_{lam} = 0{,}4$$

$$t_{stiff} = 0{,}02$$

$$t_{ave} = 0{,}02$$

Step 1: Check whether sandstone is thick enough

$$n_\lambda = 1 + \frac{0,4}{2} = 1,2$$

$$t = \frac{1,2.6,6^2}{266} \qquad \text{(Equation 13)}$$

$$= 0,16 \text{ m}$$

This is less than the existing 2 m, therefore the sandstone will be strong enough to suspend the laminated material.

Step 2: Maximum bolt spacings

$$n_q = 1 + \frac{0,02}{0,4} = 1,05 \qquad \text{(Equation 15)}$$

$$s_b = 16,3\sqrt{1,05.0,02} \qquad \text{(Equation 14)}$$

$$= 2,36 \text{ m}$$

To cater for irregularities in the support pattern, adopt a spacing of say 2 m.

Step 3: Calculate bolt length

For a 25 mm hole diameter,

$$\lambda_b = 0,4\left[1 + \frac{8.4}{0,025.2700}\right] \qquad \text{(Equation 19)}$$

$$= 0,59 \text{ m}$$

Therefore, 0,6m long bolts will be sufficient according to the calculation. In practice, however, the thickness of the laminated layer will seldom be consistant, and therefore a bolt length of say 0,9m should be used.

Step 4: Check spacing for mechanical anchors

$$S_m = \sqrt{\frac{65}{16,7.0,4}} \qquad \text{(Equation 20)}$$

$$= 3{,}11 \text{ m}$$

Because $S_m \rangle S_b$, the spacing $S_b = 2$ m should be used. Note that if an inferior anchor with pull out resistance of say 20 kN was used, S_m would be 1,7 m. As the least spacing is to be used, the bolt spacing will then have to be decreased to 1,7 m. The pull out resistance of a mechanical anchor often decreases over time and the pull out resistance of freshly installed mechanical anchors should not be used as is.

Example 3 Beam creation

In this example, a beam is to be created in weak roof material, which could be either say a thick mudstone or a thinly laminated roof with weak or no cohension between the roof layers. Say the required road width is 6,6 m.

Step 1: Thickness of beam to be created

$$t_m = 0{,}00753.6{,}6^2 \qquad \text{(Equation 23)}$$

$$= 0{,}33 \text{ m}$$

This is the minimum thickness, in practice increase it by a 1,5 safety factor to result in a 0,5m thick required beam.

Step 2: Maximum permissible sag

$$h = 1{,}2.10^{-7} \frac{6.6^4}{0{,}5^2} \qquad \text{(Equation 24)}$$

$$= 0{,}9 \text{ mm}$$

Step 3: Select position for edge bolt

Say $S_r = 0{,}5$ m

Step 4: Calculate maximum permissible inter layer slip at position of $S_r = 0{,}5$ *m.*

$$b = \arctan\left(\frac{3{,}3}{0{,}0009}\right) - \arctan\left(\frac{0{,}0009}{3{,}3}\right) \qquad \text{(Equation 28)}$$

$$= 89{,}969^{0}$$

$$R = \frac{3{,}3}{Cos\, 89{,}969} \qquad \text{(Equation 27)}$$

$$= 6099{,}228 \text{ m}$$

$$q = \frac{p}{2} - \arctan\left[\frac{6099{,}228 - 0{,}0009}{3.3 - 0{,}5}\right] \qquad \text{(Equation 26)}$$

$$= -0{,}00046\, Radians$$

Then,

$$\Delta\lambda_d = -0{,}00046.0{,}035 \qquad \text{(Equation 25)}$$

$$= 0{,}016 \text{ mm}$$

The minus sign may be omitted as it merely depends on whether one views the slip as that of the upper layer relative to the lower one or vice versa. Again allow a safety factor of 1,5 in which case the maximum allowable slip is 0,011mm.

Step 5 Calculate the possible slip

Say the volumetric shrinkage of the resin upon setting is 0.

Then, $\Delta\lambda_{rs} = 0$

Resin compression:

$$t = \frac{3.0{,}025\left(\frac{3{,}3 - 0{,}5}{3{,}3}\right).6{,}6}{4} \qquad \text{(Equation 29)}$$

$$= 0{,}105 \text{ MPa}$$

Assuming a 2,5 m bolt spacing,

$$s_r = \frac{0,105\,.\,2,5}{0,02} \qquad \text{(Equation 31)}$$

$$= 13,1 \text{ MPa}$$

The resin compression, $\Delta\lambda_{rc}$, is:

$$\Delta\lambda_{rc} = \frac{13,1(0,028 - 0,02)}{9000} \qquad \text{(Equation 32)}$$

$$= 0,012 \text{ mm}$$

This is just more than the allowable displacement of 0,011 mm and therefore one of the assumptions has to change. For instance, if the assumed bolt spacing is reduced to 2,0 m, then the resin compression reduces to 0,0093 mm, which is within the limit of allowable displacement.

Step 6: Calculate bolt length

Assuming inter laminae cohesion of 0,1 MPa and friction coefficient of 0,5, the required pretension is

$$F_b = 2^2 \left[\frac{\frac{3.0,025.6,6}{4} - 0,1}{0,5} + 0,025.0,5 \right] \text{MN} \qquad \text{(Equation 37)}$$

$$= 240 \text{ kN}$$

A pretension of 240 kN is clearly not achievable in practice. If the bolt spacing is then reduced to 1,5 m, the pretension reduces to 135 kN which is still very high but achievable with the right equipment. If a spacing of 1,0 m is adopted the required pretension reduces to 60 kN which is within the capability of most roofbolters used in South Africa.

The anchor length, assuming resin/rock shear resistance of 2700 kPa, is then:

$$\lambda_a = \frac{60}{p.0,028.2700} \qquad \text{(Equation 38)}$$

$$= 0,25 \text{ m}$$

and the total bolt length is

$$\lambda_b = 0,5 + 0,25$$

$$= 0,75 \text{ m}$$

This example demonstrates the principle that beam creation is best achieved with very small bolt spacings, while the bolts need not be very long. Again taking cognisance of natural variation, one would increase the bolt length to say 0,9 m or 1,0 m.

The example also demonstrates that some of the numbers are very small and that errors could arise from rounding off in the interim phases of the hand calculation. In practice it would be better to construct a small computer procedure to do the work.

Example 4: Suspension of thick, weak roof by inclined bolts.

In this example, a typical roof fall height of 2 m over a 6,6 m wide road has to be prevented by suspending the roof with inclined bolts.

Choose a bolt length of 1,5 m.

Then, the bolt spacing is simply:

$$S_b = 1,95 \text{ m} \qquad \text{(Equation 44)}$$

Check for tensile bolt failure – the load per bolt.

$$F = \frac{25.6,6.1,5}{2.1,95} \qquad \text{(Equation 44a)}$$

$$= 63 \text{ kN}$$

As this load is less than the tensile strength of a normally used M20 bolt, the bolts will not fail in tension.

APPENDIX IV **List of units, abbreviations and default parameters used in this report**

List of units, abbreviations and default parameters used in this report

1. **Units**

   The metric system of units is used. Except where stated differently, the following is used in particular:

   Length – m
   Force – kN
   Stress – kPa
   Bolt, hole and resin capsule diameters - mm
   Material strengths - MPa
   Material moduli - GPa

2. **In beam formulae**, the unit weight, $\gamma = \rho g$ = 25 kN/m^3 (0,025 MN/m^3), is used.

3. **Default values for material properties**

   Shear resistance of resin/rock contact plane = 1500 kPa
   Tensile strength of roof material = 5 MPa
   Shear strength of roof material = 8 MPa
   Modulus of Elasticity of roof material = 13 GPa
   Safety Factor of roof support systems = 1,5
   Yield strength of roofbolt = 450 MPa
   Modulus of Elasticity of resin = 15 GPa
   Tensile strength of stiff roof material (sandstone) = 5 MPa
   Shear strength of stiff roof material (sandstone) = 10 MPa
   Modulus of Elasticity of stiff roof material (sandstone) = 30 GPa
   Coefficient of friction of roof laminae = 0,5
   Cohesion of roof laminae = 0,1 MPa