Technical note: Safety and cost implications in the spacing of support

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Synopsis

This note considers the most fundamental concepts of support design for deep underground openings and discusses them briefly in the context of tabular stoping and deep hard-rock tunnelling.

It concludes that the conventional re-bar, mesh, and lacing methods that have been widely used in deep gold mines are quite inappropriate for the support of highly stressed tunnels. The following is suggested as the criterion for adequate dynamic support: the energy absorbed by a support unit (yielding through some acceptable limited displacement) should exceed the kinetic energy generated by the mass of rock that is intended to be retained by that unit.

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Introduction

The design of support for deep underground openings is not a simple matter. The engineering considerations can be seriously complicated by the difficulty of specifying or estimating the load or the demand that might be imposed on the excavation. It is often difficult even to determine exactly what comprises the 'structure' that is at risk and that therefore requires support. This is particularly true when it is the geological environment that poses the threat dynamically by way of mine-induced seismicity, which often causes severe rockburst damage.

Although safety should be the primary concern in the design process, the over-riding consideration is often one of cost. This is particularly so because most components of the support system are not recoverable or re-usable and have to be regarded as consumable items.

For all the above reasons, it is necessary to consider the basic essentials of the design procedure. This note attempts to set out the most fundamental of these concepts and then to discuss them briefly in the context of tabular stoping and deep hard-rock tunnelling. It also shows how the important factors in costing are often overlooked both to the detriment of overall cost and, more importantly, to the total effectiveness of the system.

The treatment of the topic is not comprehensive, but is intended merely to promote a more critical awareness of the essential requirements of design, and perhaps to encourage rejection of the short-sighted cost considerations and 'blinkered' thinking that at present constrain the proper development and design of tunnel support for deep-level conditions.

Essential requirements of support design

The main objectives of mine operators when requesting the design of support for an excavation are

- to ensure the safety of the personnel
- to ensure the security of the equipment
- to keep the excavation functionally open
- to optimize the cost-effectiveness of the system that will best achieve the above goals.

In engineering terms, the essential requirements for the attainment of these objectives are as follows.

- The load-bearing elements of the support system must maintain their resistance through large deformations.
- The unsupported span must be limited so that the rock cannot collapse between the load-bearing elements or 'abutments'.
- Rock blocks or fragments must be contained between their 'point load' supports to prevent unravelling of the fractured rock, which would lead to failure of the system.

To ensure cost effectiveness, individual elements of the system must not be over-designed, and the various components must be functionally compatible or 'balanced'.

In most engineering disciplines, an essential underlying premise of any design is that, to ensure the stability or safe performance of a structure as a whole, no part of the structure can be allowed to fail.

This is not necessarily the case in underground mining. The structure has to be regarded as comprising the rock immediately surrounding the excavation, as well as the support elements in the opening, in the lining, or in the cladding, and those reinforcing the rock itself. More often than not, the rock surround is already in a failed state either as a result of the excavation process or because of induced stress. It can easily be shown that the process of initial rock fracture by indirect tensile extension of elemental flaws is not affected by economically practicable densities of support. Thus, rock-reinforcing systems utilizing stiff grouted bolts or even other forms of pre-stressed tendon do not prevent fracturing of 'reinforced' material in the same way as the corresponding components in structural concrete are designed to do. Instead, the tendons act as 'retention' elements that hold back the 'containing' elements of mesh, lacing, or other forms of cladding, as well as providing some constraint to maintain the integrity of the fractured surround.

It is self-evident and generally well understood that the spacing between the 'retention' elements in tunnel support, or the 'point load' supports in stoping, is crucial in determining the support effectiveness of the system. However, it seems that the dominant influence of the spacing between tendons on the cost of tunnel support is sometimes not properly evaluated in determining the overall cost-effectiveness. This note treats the matter of support spacing in tabular stopes in a broad, general way, but the problem of required working loads and distribution of tendons in tunnel support is examined more specifically and critically.

Stope support

In designing stope-support layouts, the value of reducing the distance between the workface and the closest line of hydraulic props or timber packs is generally well appreciated. In a working stope, it is necessary to provide space to accommodate both the blasted rock, and the rock-handling and cleaning arrangements. When the face is prepared for drilling, manoeuvrability is essential for the drilling equipment. Because of these unavoidable needs, the unsupported span is usually greater than pure safety requirements would dictate. Temporary support is generally used to reduce this unsupported span. Too frequently, however, compromises are made—usually in the interests of maintaining production rate and reducing costs, sometimes at the risk of not ensuring adequate safety. To a large extent the above factors explain the well-documented reality that the working area up to 5 m back from the face is the most hazardous location in a gold mine.

The threat of falls of ground behind the front row of support is also substantial, particularly where rockbursts occur. Stacey[1] has provided a useful insight into the effects of spacing distance and spacing geometry on the probability of falls of ground in tabular stopes.

For comparative purposes, the cost of permanent stope support is normally expressed in rands per ton or rands per unit area stoped. The reduction in unit cost that results from an increase in the distance between packs varies as the square of the inter-pack spacing. Thus, relatively small increases in the distance between supports are accompanied by considerable reductions in installed cost. If it were possible to increase this distance significantly, substantial savings could be effected. However, this would inevitably tend to increase the probability of rock falls between support packs or props. The consequent increase in risk can be evaluated from the probabilities given by Stacey[1].

To some extent it is possible to counter the effects of the resulting increase in unsupported span by some means, such as extended headboards on props or 'umbrella' decking of the top layer of packs. Obviously, the potential savings resulting from the increased spacing between the main support units is reduced by the amount spent on these additional measures. Such cost juggling must not ignore the engineering certainty that the resistance afforded by 'cantilevered' bars or beams diminishes as the fourth power of the length of the cantilever.

Tunnel support

The evaluation of the cost and safety implications of varying the spacing between supports in tunnels is almost completely uncomplicated by the difficulties outlined above for stope support.

Although the principles are very simple and follow the same law as in stoping, viz that the cost varies inversely as the square of the spacing, it is only when the arithmetical exercise is undertaken methodically that it becomes apparent how great the benefits actually are. It becomes easy to demonstrate that sometimes a complete change of support method may be justified, with considerable improvements in safety and cost.

One such instance is given by highly stressed tunnels, where the safety, functional capability, and cost-effectiveness of support provided by **yielding cone bolts** is improved to such a degree that these bolts should be used wherever conventional fully grouted re-bar support is now employed.

In some cases, such a change could result in a **tenfold** increase in the dynamic capability of the support. At the same time, the installed cost could be **reduced** significantly in absolute terms and not merely in terms of relative cost-effectiveness. The quantitative justification for this apparently extravagant claim is shown later by means of an example.

The importance of yieldability in tunnel support tendons was realized as much as 25 years ago (Ortlepp[2]), but the magnitude of the improved capability that they would provide was not appreciated until recently, when their performance capability at high rates of displacement was quantified (Ortlepp[3]). Even now, there appears to be a reluctance on the part of the deep mines of the world to contemplate the widespread change that is clearly indicated.

There are probably two real reasons for this resistance to change. Firstly, there is the inherent conservatism of the underground miner, who is understandably wary of change simply for the sake of change, particularly when old and tried methods have proved to be reliable. However, the inadequacy of stiff grouted support has been frequently demonstrated by the occurrence of broken tendons after even moderate rockbursts in tunnels, as shown in Figure 1. When the old methods have been shown to be completely deficient, an unwillingness to adopt improved systems is difficult to understand.

The second, less obvious reason is the previous lack of a proper engineering procedure for the design of tunnel support. Wagner[4] and Jager *et al.*[5] have specified performance requirements in terms of the minimum displacement rates that have to be met. However, no simple design procedure existed until recently, when it was suggested that the kinetic energy of the broken rock expelled from the tunnel walls essentially constituted the **maximum** loading of the retention elements of the 'structure'(Ortlepp [3,6]). The criterion for adequate dynamic support can thus be stated very simply: **the energy absorbed by a support unit (yielding through some acceptable limited displacement) should exceed the kinetic energy generated by the mass of rock that is intended to be retained by that unit.** The kinetic energy is one-half the product of the tributary mass and the square of the velocity with which its ejection is initiated.

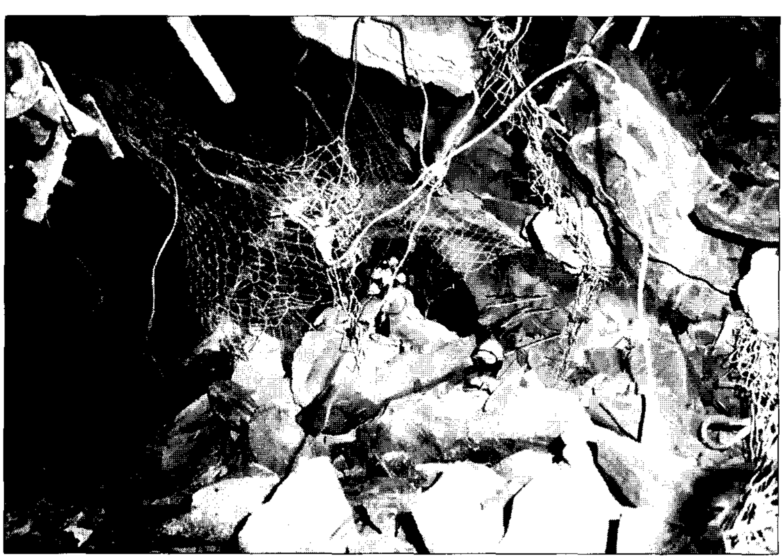

Figure 1—Damage to a tunnel supported by stiff grouted re-bar, mesh, and lacing after a moderate rockburst

It is important to emphasize that most types of yielding bolts, such as cone bolts, are only slightly less stiff in initial response than stiff grouted re-bar bolts. Their effectiveness in 'reinforcing' the rock is therefore only slightly less than that of stiff systems, while their 'retention' effect is maintained for a displacement as much as 50 times greater. Thus, a design that will cope with the large dynamic displacements of severe rockbursts will naturally be adequate also to contain the strongly driven, but much smaller, displacements caused by quasi-static dilatation processes.

A third reason advanced for the non-implementation of the yielding cone bolt is that it is substantially more expensive than the conventional stiff grouted reinforcing element that it seeks to replace. The fallacy of this argument is soon exposed if one considers that the inventory cost of a yielding bolt and a conventional bolt is, respectively, about 1/4 and 1/6 of the total installed cost. On the other hand, the energy-absorbing capability differs by almost two orders of magnitude.

Even in hard rock, a tunnel at depth is likely to be surrounded by extension fracturing sub-parallel to the tunnel walls. The 'structure' can be considered to be bounded by a fracture surface or potential separation surface enclosing a shell of fractured rock around the opening. The prime function of the rock reinforcement is to provide stability to this fractured shell.

A grouted re-bar intersecting such a bounding surface for 1 m or so deep into the rock mass would be stretched by any initial separation across the surface resulting from the dilation accompanying the fracture or from the ejection of the fractured shell during a rockburst. Because of the deformed, ribbed surface of the bar, the bond between grout and steel would be broken down, for possibly two or three bar diameters only, on either side of the fracture surface (Figure 2). No more than 80 mm or so of the length of the support would be subjected to the maximum elongation of about 15 per cent that the steel is capable of sustaining before the final 'necking' and failure that inevitably follow. The fracture of the bar thus occurs at a separation of about 12 to 15 mm. The energy absorbed in this process is given by the product of the total displacement and the average resistance of the steel:

13 mm x 100 kN = 1,3 kJ
(for a bar of 16 mm diameter).

Similarly, a re-bar of 25 mm diameter in the same circumstances can be shown to absorb about 3,3 kJ of energy before fracture.

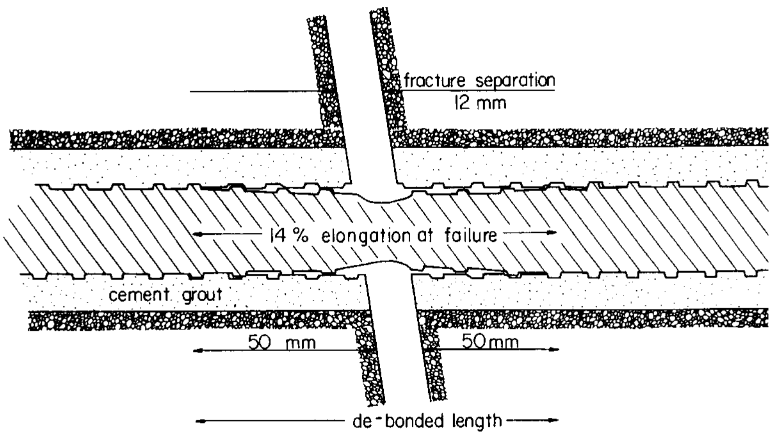

Figure 2—Section through a fully grouted re-bar, showing necking and start of fracture after limited opening of the separation surface

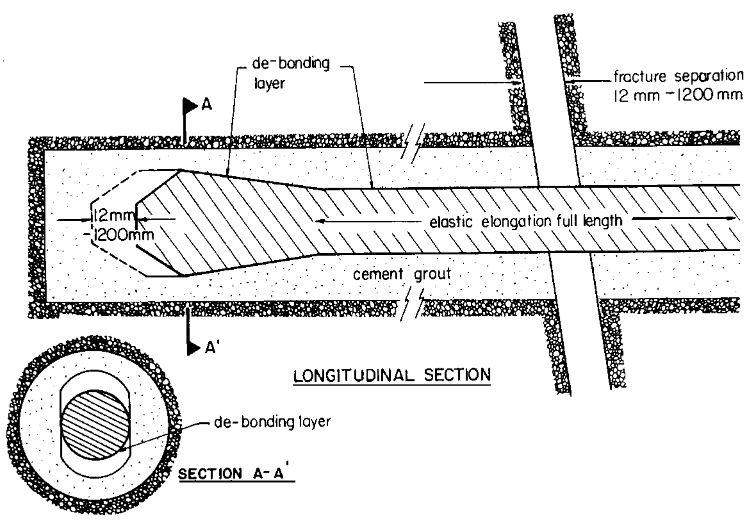

Figure 3—Section through a cone bolt, showing its ability to yield limited only by the length of the hole beyond the separation surface

The de-bonded cone bolt, on the other hand, is able to yield by sliding through the cement grout, dissipating energy by crushing the grout and by frictional heating (Figure 3). Theoretically, this dissipation process is limited only by the length of bolt that extends beyond the bounding surface. For every 100 mm of such sliding, about 8 kJ and 19 kJ of energy are dissipated by cone bolts of 16 mm and 22 mm diameter respectively. This is already six times more energy than is necessary to cause complete fracture of the equivalent diameters in fully bonded stiff re-bar. More often than not, 300 or 400 mm of displacement of each sidewall can be tolerated before a tunnel becomes unusable.

It is this large difference between yielding and stiff fully grouted tendons in respect of their energy-absorbing capacities that, if properly recognized in support design, enables large improvements in cost-effectiveness to be achieved. This potentially great improvement can best be illustrated by means of simple calculated examples based on realistic relative costs of components and consumables.

The assumed costs for the examples are as follows:

2,3 m x 16 mm re-bar shepherd's crook (ultimate tensile strength 110 kN, elongation 15%)	R 6,60
2,3 m x 16 mm yield-cone shepherd's crook (yield resistance 80 kN)	R 10,70
22 mm cone bolt (yield resistance 190 kN)	R 18,60
25 mm re-bar bolt (ultimate tensile strength 270 kN, elongation 15%)	R 12,20
150 x 150 x 6 flat washer	R 1,80
Cost of grout per hole	R 5,50
Cost of drilling hole (including labour, maintenance, consumables)	R 25,75

The excavation to be supported is a typical haulage in hard rock at depth where the fractured surround is 'reinforced' by different types of tendons at appropriate spacings. In practice, the fractured rock between the tendons would be contained by lacing and diamond mesh. The cost of this cladding would be similar, whether rigid or yielding tendons were used. Therefore, the cost of the mesh and lacing is not included in the comparisons that follow.

Example 1 : The relative effectiveness of 16 mm 'rigid' and yielding tendons at the same cost per square metre

If 16 mm conventional re-bar shepherd's crooks are spaced 1 m apart, the roof support would provide a static resistance of, say, 110 kN/m^2. This would be sufficient to carry the dead-weight load of about 2,1 m thickness of roof strata with a safety factor of 2,0. The function of the sidewall bolts under static conditions would be to restrain the excessive movement caused by dilational forces. Since it is very difficult to calculate or estimate these dilational forces, it is not possible to estimate the capability and safety factor in the case of the sidewall support. The cost of providing the total static capability would be R37,85 per square metre (Table I).

If, as a result of a seismic event in the proximity of the tunnel, a 1 m thickness of slab were ejected from the roof or sidewall at an initial velocity of 1 m/s, the kinetic energy per square metre of affected area would be

$$E = \frac{mv^2}{2} = \frac{2,65 \times 1 \times 1}{2} = 1,32 \text{ kN.}$$

Since the energy-absorption capability of the rigid tendon is 1,3 kJ, the density of reinforcement at a spacing of 1 m is just adequate (safety factor, $SF = 1,0$). It should be noted that the slightest additional strain, whether quasi-static or dynamic, would cause all the tendons to fracture, leading to complete failure of the support system.

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Table I

Performance capability of 16 mm tendons for equivalent support cost (R38 per square metre)

Tendon	Cost of materials R	Installed cost R	Support density at R38/m^2 m^2/unit	Hole spacing m	Energy absorbed kJ/m^2	Ejection velocity of 1 m slab m/s
16 mm re-bar	12,10	37,85	1,0	1,0	1,32 (0,012)*	1,0
16 mm cone bolt	16,20	42,00	1,11	1,05	18,0 (0,25)*	3,7

* Distance yielded (m) to absorb the stated energy

At R10,70 each, the unit cost of the yielding cone bolt in shepherd's crook form is 62 per cent greater than the re-bar equivalent, but the cost of the grout (R5,50) and the cost of the hole (R25,75) are the same in each case. The installed cost is thus R42,00, or 11 per cent greater, per hole than for the conventional shepherd's crook.

If the support density were decreased slightly, the total support cost per square metre would be reduced by the square of the increased distance between bolts. Table I shows that the spacing would have to be increased by 0,05 m to make the cost identical. The variation in inter-bolt spacing typically obtained in practice underground is shown in Figure 4, which demonstrates that an increase in spacing of 0,05 m would be completely undiscernible and insignificant.

The greatly improved capability of the yielding support is indicated in the last column of Table I, which shows that the 1 m thickness of fractured shell would have to be ejected at nearly 4 m/s to cause a displacement of the sidewall of as much as 0,25 m. Such an ejection velocity would occur only close to the origin of a large seismic event.

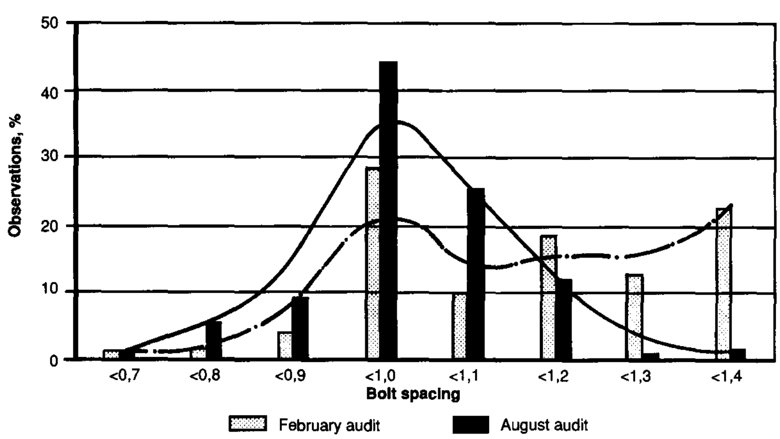

Figure 4—Distribution of distances (actual measurements) between bolts with a pattern of 1,0 m designed spacing (after Wilkes[7])

Example 2: Support cost necessary to contain substantial rockburst damage by means of yielding and fully bonded bolts

The concept of the cone bolt as a suitable form of yielding bolt was originated by the former Chamber of Mines Research Organization (COMRO). Studies by that organization showed[5] that the minimum density of yielding support to contain a severe rockburst was that which would absorb 25 kJ/m^2. The energy flux generated, as a function of initial ejection velocity, by various rock thicknesses that might be dislodged during moderate to severe rockbursts in tunnels is shown in Figure 5.

Example 1 showed that 16 mm cone bolts yielding at 80 kN would stop a 1,0 m thickness of fractured rock shell ejected at nearly 4 m/s after a displacement of only 0,25 m. This represents an energy-absorbing capability of 20 kJ/m^2. As such a duty can be achieved by the commonly used standard pattern of 1 m spacing, that system can be used as a suitable basis in the comparison of densities and, therefore, of the costs of alternative types and patterns of bolts. The installed cost of a system comprising 16 mm yielding bolts at a spacing of 1 m would be R46 per square metre (Table II).

Because of the considerably higher yielding resistance of 190 kN for 22 mm cone bolts, each bolt would absorb 47,5 kJ of energy after 0,25 m of sliding. Thus, it could contain the kinetic energy (*KE*) developed by 2,26 m^3 of rock moving at 4 m/s:

$$\begin{aligned} KE &= 1/2 \\ mv^2 &= 1/2 \times 2{,}26 \times 2{,}65 \times 4 \times 4 \\ &= 47{,}9 \text{ kJ.} \end{aligned}$$

To perform the base duty, viz to arrest 1,0 m thickness of surrounding rock imploded inwards at 4 m/s, each 22 mm cone bolt would absorb the energy generated by 2,26 m^2 of tunnel surface (i.e. a mass of 6 t). Table II shows that the cost of installing 22 mm bolts at that density would be only R20 per square metre, which is less than one-half the cost of the 16 mm cone-bolt system.

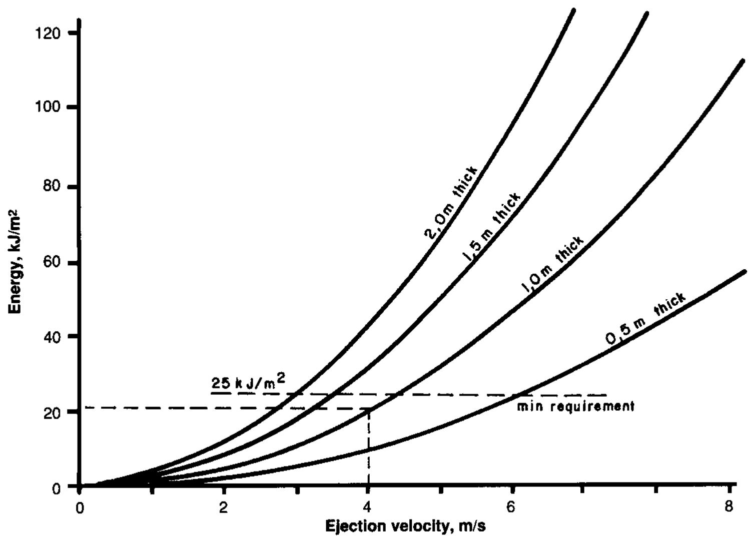

Figure 5—Kinetic energy as a function of ejection velocity for rock slabs of various thicknesses

If an overly simplistic approach were adopted (viz that, the higher the stress, the **stronger** would be the support required by the tunnel, and that the most economical way to achieve strong support would be to put as much steel as possible into each hole), then the alternative of conventional grouted re-bars of larger diameter might be considered.

Re-bars of 25 mm diameter are not used for support in South African gold-mine tunnels, but are used elsewhere where heavy support is deemed necessary. Although each bar would generate a very high initial resistance (as much as 270 kN), because of limited de-bonding, the total elongation that could be tolerated across any surface of separation would be limited to about 15 mm. The energy absorbed per bolt would be about 3,2 kJ, and it would thus require more than 6 bolts to absorb 20 kJ/m². The cost of drilling holes at 0,39 m spacing to install the required number of bolts to prevent a rockburst displacement of more than 15 mm would be R291 per square metre!

The total absurdity of the argument that suggests that cone bolts cannot be used to replace conventional shepherd's crooks because 'they cost nearly twice as much' is well illustrated if the 16 mm re-bar option (which would absorb the same amount of rockburst energy) is examined. Table II shows that the holes would have to be spaced no more than 250 mm apart. The cost of drilling 16 holes per square metre and grouting in the conventional shepherd's crooks would be R610 per square metre!

These two examples are not suggested as practical alternatives, but simply to illustrate the comparative costs for the effectiveness of corresponding support systems. The high density that would be required for stiff support is clearly not practical.

Conclusion

The rationale of the support design outlined in this note, quantified by means of simple numerical examples, indicates strongly that the conventional re-bar, mesh, and lacing methods that have been widely used in deep gold mines are quite **inappropriate** for the support of highly stressed tunnels.

Although much more work is necessary to optimize the recommended system and the yielding components, the reality and practicability of its greatly improved performance have already been demonstrated by prototype-scale dynamic tests[3].

Somewhat surprisingly, it appears that the cost of providing this greatly improved capability can actually be significantly less than the cost of the inappropriate system, even in absolute terms. ◆

Table II

The support cost of various tendons for equivalent performance (21 kJ/m²)

Tendon	Resistance or UTS kN	Energy absorption kJ	Support density m²/unit	Support spacing m	No. of holes per m²	Cost per m² R
22 mm cone bolt	190	47,5 (0,25)*	2,26	1,5	0,44	20
16 mm cone bolt	80	20,0 (0,25)*	0,95	0,98	1,05	46
25 mm re-bar	270	3,2 (0,015)*	0,152	0,39	6,58	291
16 mm re-bar	110	1,3 (0,012)*	0,062	0,25	16,13	610

* Distance yielded (m) to absorb the stated energy

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