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# EVALUATION OF SUPPORT SYSTEMS SUBJECT TO IMPULSE LOADING

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# EVALUATION OF SUPPORT SYSTEMS SUBJECT TO IMPULSE LOADING

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## ABSTRACT

When a rockburst occurs, a support system is subject to impulse loading. The peak particle velocity of the vibrations is the main criterion for accessing the damage potential. Peak particle velocity is dependent on rockburst magnitude and attenuates with distance from source.

The design of support systems, for rockburst conditions, is based on an energy balance approach. The energy imparted to a rock surface by the peak particle velocity has to be absorbed by stressing and stretching the support system. Supports with a yielding characteristic are better able to survive without major damage. The basic requirements for support systems are that they can yield at 2 m/s, accommodate rock displacements of 60 mm, and provide a support resistance of at least 60 $kN/m^2$.

There is ample evidence in Ontario mines on how well different support systems withstand rockburst conditions. Mechanical rockbolts are the most susceptible to failure, whereas lacing (a three-tier support system of grouted rebar, mesh and steel cables) is capable of surviving major bursts. In between these two extremes are the rebar, cable bolt and friction type support systems, generally in association with mesh.


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Key words: Rockbursts; Support systems; Design.

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# INTRODUCTION

Rockbursts first became a major problem in Ontario hardrock mines in the late 1930's. Two approaches were taken to tackle the problem. Mining methods were changed to include some aspect of backfilling, and the sequence of extraction was rigorously controlled to limit the chances of encountering rockburst-prone ground. Secondly, various types of support systems were tried to control the amount of damage and protect the labour force.

Concrete lining and conventional timber posts and beams, being used at that time, were ineffective in controlling rockburst damage. The first concept was to install stronger and more rigid support systems, especially with the introduction of mechanical rockbolts and rebar. These support systems were successful in controlling damage from the smaller magnitude rockbursts, but were ineffective against larger events. It was realized that the rigidity of the support was contributing to the problem. This led to the introduction of supports with yielding characteristics, especially with the introduction of the friction types of support. Also the importance of steel mesh, in conjunction with friction supports, was realized.

The rockburst problem is much more severe in South African gold mines than in Ontario mines. By necessity, support systems capable of withstanding rockbursts had to be found. This led to the development of rapid-yielding hydraulic props for stope support which could accommodate closure up to 2 m/s. These types of support are specific to South African gold mines (i.e., 1 m high stopes) and do not have general application to Ontario mines.

In haulage drifts a lacing support system was developed. This consists of grouted rebars in boreholes, wire mesh, and tensioned steel cables between rebars in a diamond configuration. This type of support has been very effective and has survived rockbursts of magnitude 4.0 only a few tens of metres away. These types of support systems have general application in Ontario mines, and the first lacing installations have recently been done at the Strathcona Mine (Davidge, et al. 1988). However, it is also an extremely expensive support system.

The development of these support techniques has been done in an intuitive manner with trial-and-error testing to fit specific applications. It was only in the early 1980's that Wagner in South Africa developed the theoretical basis for designing support systems to withstand rockbursts.

Between 1984 and 1987 in Ontario mines there have been 280 recorded seismic events of magnitude 2.0 or greater. The distribution by magnitude is shown in Figure 1. Approximately 96% of these events had a magnitude lower than 3.0. For a very large event, greater than 3.0, probably only lacing would survive if it occurred 10 m away. However, for the smaller events there is probably a range of cheaper support techniques which would control the damage. The purpose of the present research to to define the capabilities of a variety of support techniques. The preliminary criterion is what magnitude would the support withstand if the event occurred 10 m away.

## MECHANICS OF DYNAMIC LOADING

When a rockburst occurs underground, strain waves radiate from the source in a spherical pattern. There are two types of waves. P or compressional waves are radial vibrations in the same direction as the wave front, and travel at a velocity of about 6.2 km/sec in hardrock mines. S or shear waves are transverse vibrations perpendicular to the wave front and have a velocity of about 3.6 km/sec.

Predominantly, the peak particle velocity of these vibrations is in the S wave. Figure 2 shows the relationship between the maximum particle velocity in the P and S phase, from measurements taken on triaxial sensors at Quirke

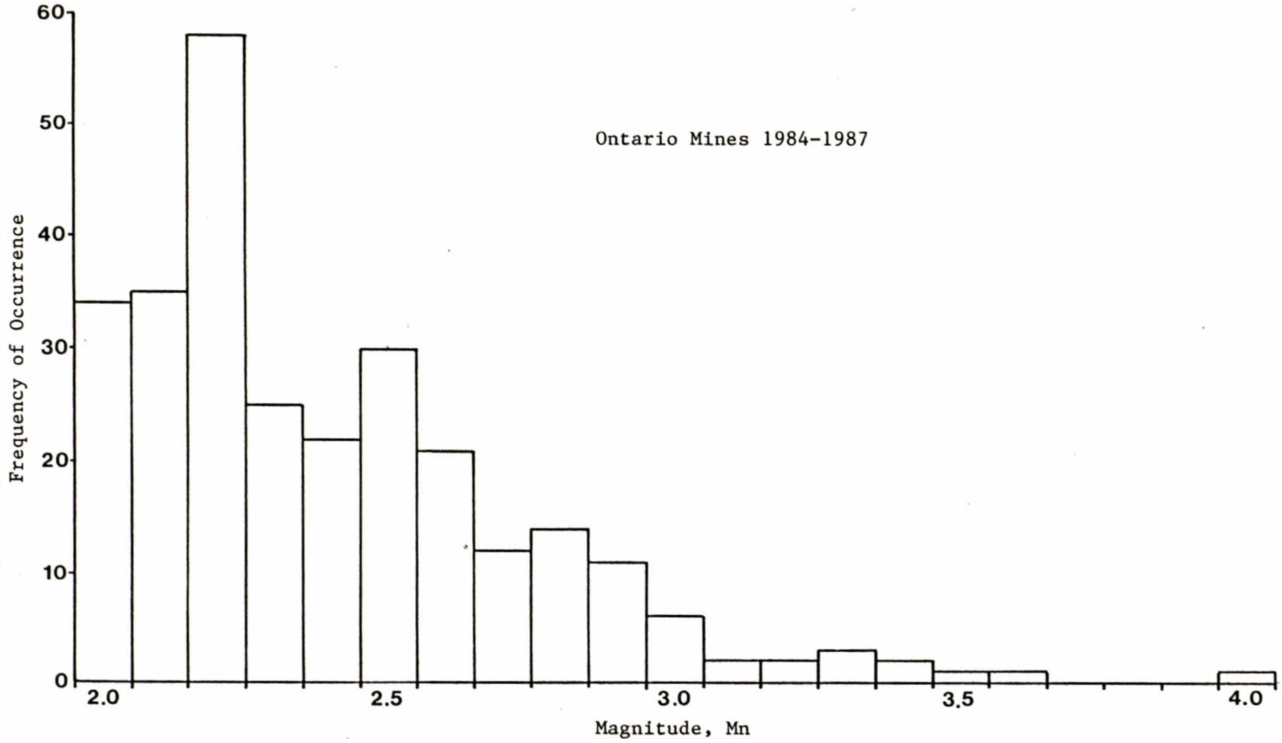


Fig. 1 - Distribution of seismic events in Ontario mines by magnitude.

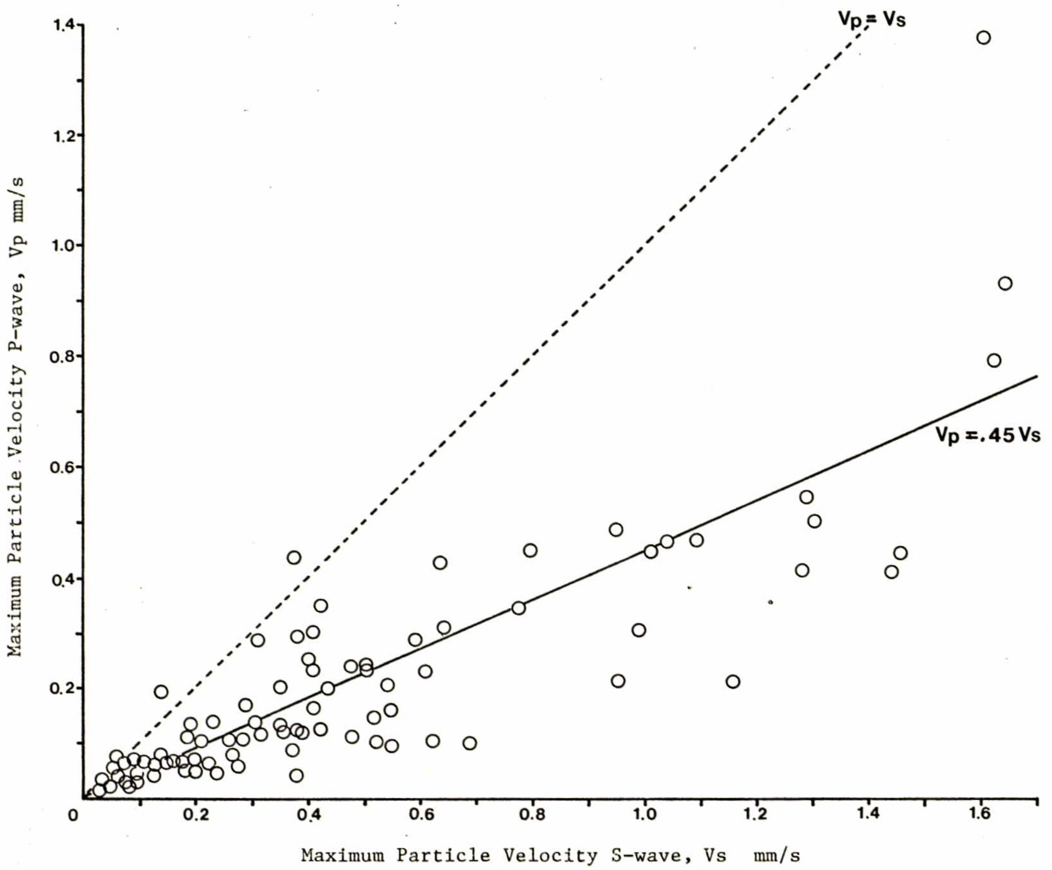


Fig. 2 - Maximum particle velocities in the P and S seismic waves.

Mine at Elliot Lake. In general, the maximum particle velocity in the P wave is 45% of the peak particle velocity in the S wave.

The peak particle velocity is the main criterion for assessing the damage to support systems and underground structures. A recent study in some Ontario mines (Hedley, 1988), used a relationship between peak particle velocity, distance from source and rockburst magnitude in the form:

$$\bar{V} = 4000 \left( \frac{R}{10^{M/3}} \right)^{-1.6} \qquad \text{Eq 1}$$

where, $\bar{V}$ = vector sum peak particle velocity, mm/s

R = distance from source, m

M = rockburst magnitude using the Nuttli scale.

This relationship is illustrated in Figure 3 for magnitudes 1.0 to 4.0.

Damage criteria, based on peak particle velocity, usually come from blasting investigations in tunnels. Lenhardt (1988) used blasting studies by Langefor and Kihlstrom (1963) to assess rockburst damage at the Western Deep Levels gold mine in South Africa. Blake and Cuvelier (1988) used similar criteria at Hecla's Lucky Friday Mine in the United States. Falls of loose ground occurred at velocities as low as 50 mm/s. Fracturing of intact rock started at about 300 mm/s, and severe damage at 600 mm/s. These damage criteria are expressed in Figure 4 in terms of the magnitude-velocity-distance relationship given in Equation 1. It should be noted that these damage criteria refer to unsupported rock, in general, supported rock can withstand much higher particle velocities.

An alternative method of analysis is to consider the dynamic stress pulses imparted to a rock structure or support. The equations for velocity can be converted into the radial ($\sigma$) and transverse ($\tau$) stress,

$$\sigma = \frac{\bar{V}p \; E(1-\nu)}{Cp(1+\nu)(1-2\nu)} \qquad \text{Eq 2}$$

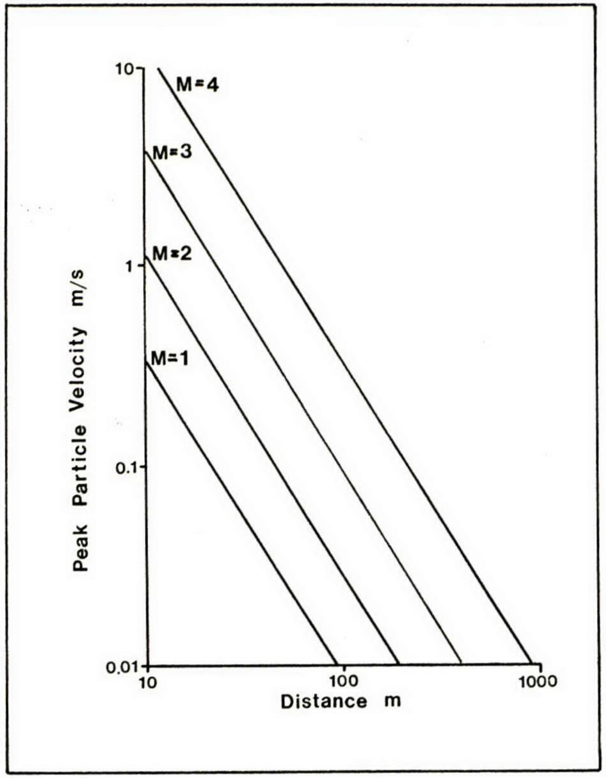


Fig. 3 - Peak particle velocity as a function of distance and magnitude.

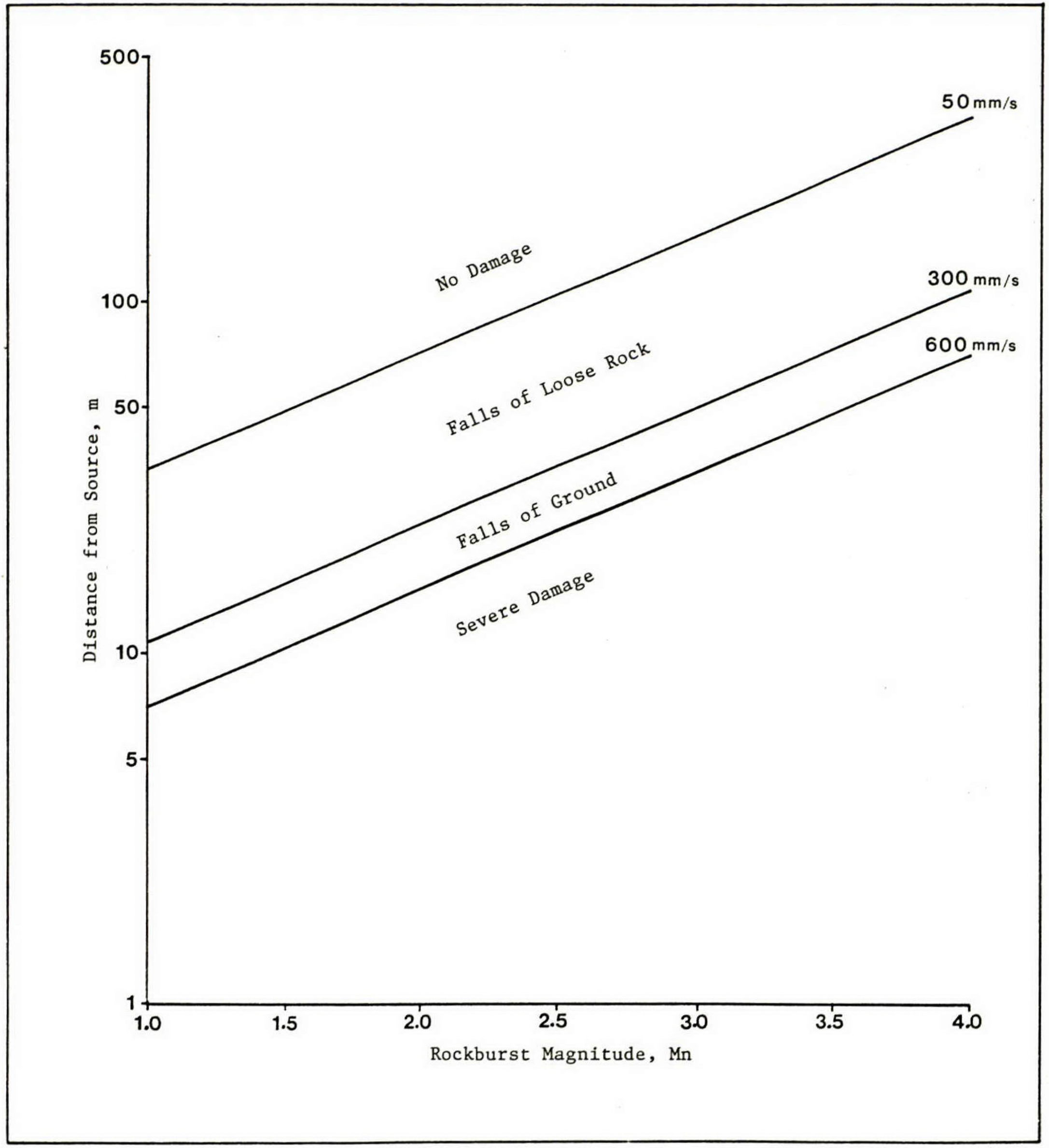


Fig. 4 - Damage criteria for unsupported tunnels from blasting studies.

and,

$$\tau = \frac{\dot{V}sG}{Cs} \qquad \text{Eq 3}$$

where, $\dot{V}p$ and $\dot{V}s$ = maximum particle velocities in P and S waves

E = elastic modulus ~70,000 MPa

G = shear modulus ~30,000 MPa

$\nu$ = Poisson's ration ~0.2

Cp = P wave velocity ~6,200 m/s

Cs = S wave velocity ~3,600 m/s.

If a rockburst of magnitude 3.0 occurred 10 m horizontally from a drift, a bolt in the side would experience a particle velocity and stress pulse in the P wave and a bolt in the roof in the S wave. Using typical parameters this would amount to 1.8 m/s and 4 m/s for particle velocities, and 23 MPa and 33 MPa stress pulses in the two bolts. These stress pulses would be on top of the static stresses already existing in the bolt.

The first theoretical study on the design of support systems in rockburst-prone ground was done by Wagner (1984) in South Africa. Subsequently, Roberts and Brummer (1988) elaborated on these design concepts. Figure 5 shows a drift supported by rock bolts in the walls and roof. At depth, the high stresses produce a fractured zone around the drift, typically 1 m deep. Consider a slab, in the roof and side walls, of mass Mr detached from the surrounding rock and being held by the bolts. When a rockburst occurs the work done by the slab at peak particle velocity, $\dot{V}$, equals the energy consumed in stressing and stretching the bolts. This can be expressed by:

$$\frac{Mr\ \dot{V}^2}{2} + Mr\ g\ c = 1/2\int_0^c Fs(x)dx \qquad \text{Eq 4}$$

where, Fs = force exerted by the bolt

c = amount of bolt stretch

g = gravity.

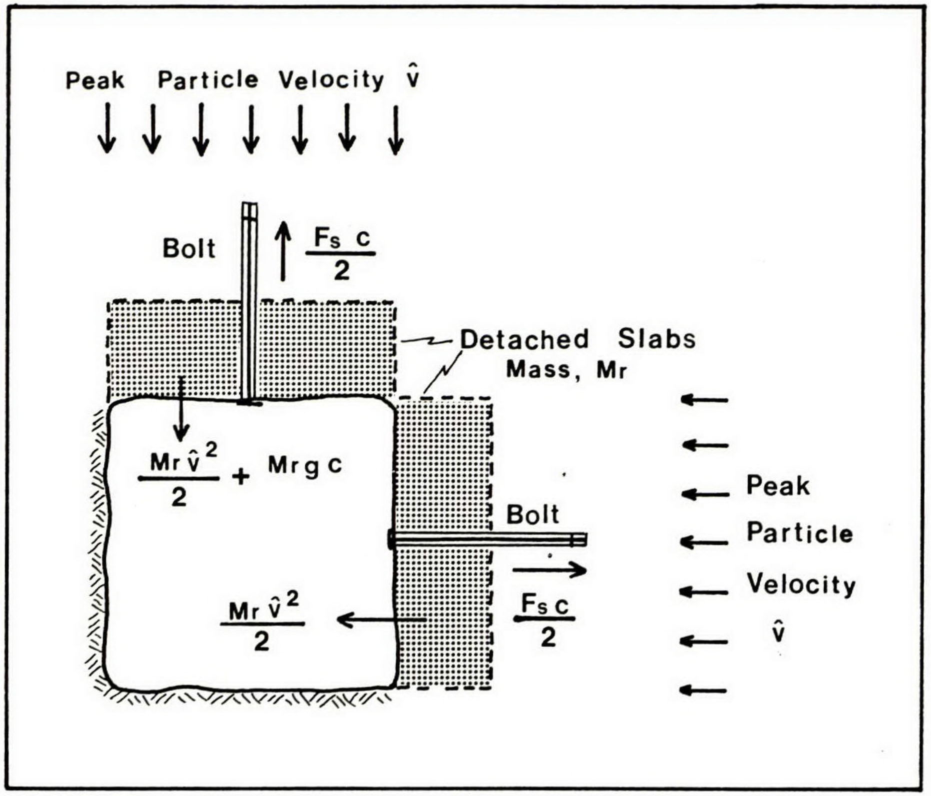


Fig. 5 - Forces imposed on rock slabs and the reaction of bolts during a rockburst.

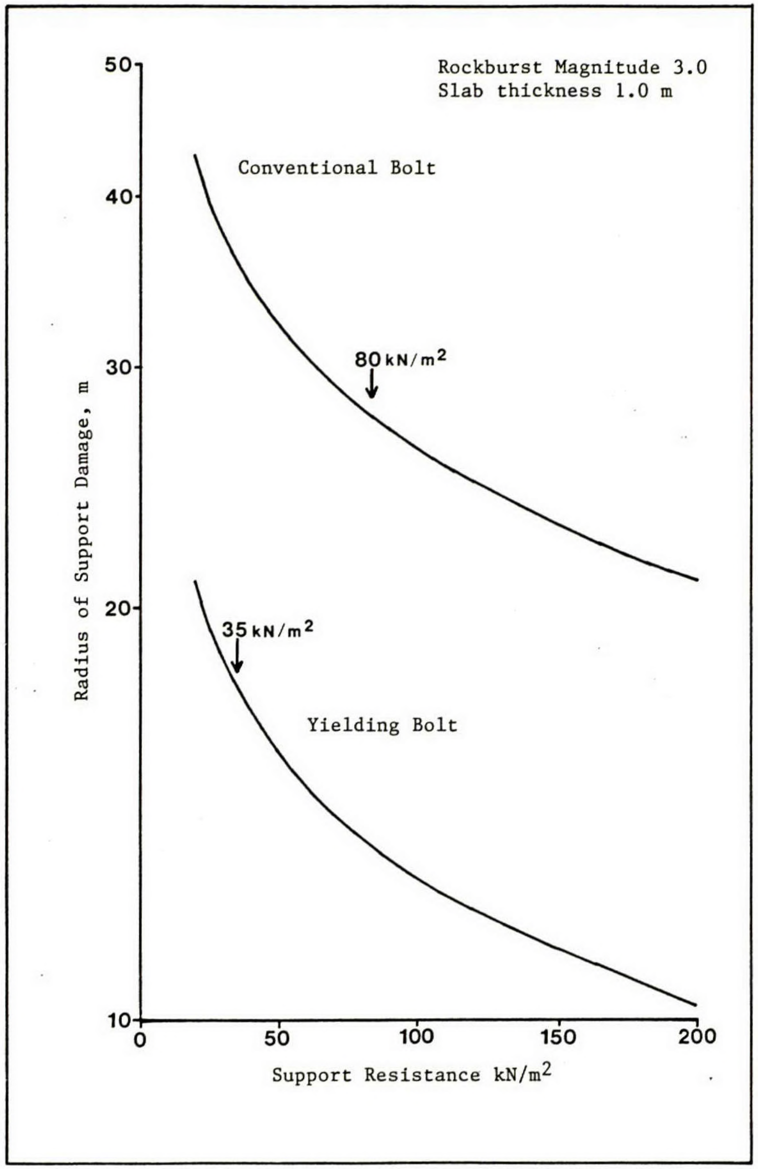


Fig. 6 - Radius of support damage as a function of support resistance for conventional and yielding bolts.

For slabs in the roof, subject to gravitational forces, Equation 4 becomes:

$$Mr = \frac{Fs\ c}{2\ (\frac{V^2}{2} + gc)} \qquad \text{Eq 5}$$

For slabs in the side walls, not affected by gravity:

$$Mr = \frac{Fs\ c}{V^2} \qquad \text{Eq 6}$$

Equations 5 and 6 apply to support systems with a predominantly linear load-deformation characteristic (e.g., rock bolts). If the support has a constant yielding load, Fy, (e.g., friction type support) then,

$$Mr = \frac{2\ Fy\ C}{V^2} \qquad \text{Eq 7}$$

The mass of the slab supported by the bolts can be expressed by:

$$Mr = \frac{\rho t}{A} \qquad \text{Eq 8}$$

where, $\rho$ = rock density

A = support density in bolts/$m^2$

t = slab thickness.

These equations can be used to compare different types of support systems under rockburst conditions. Suppose a rock slab 1 m thick with a density of 2700 kg/$m^3$ is being pinned on the side of a drift. Assuming that conventional mechanical rockbolts have a failure load, under dynamic loading conditions, of 120 kN, and a maximum elongation of 20 mm, whereas yielding friction type supports slip at a constant 50 kN over 100 mm. Rearranging Equations 6, 7 and 8 gives:

$$V = \sqrt{\frac{Fs\ Ac}{\rho t}} \qquad \text{Eq 9}$$

For conventional bolts, and

$$V = \sqrt{\frac{2Fy\ Ac}{\rho t}} \qquad \text{Eq 10}$$

for yielding bolts.

The terms Fs A and Fy A are the support resistances expressed in $kN/m^2$. Substituting the relevant support characteristics into Equations 9 and 10 gives the maximum particle velocity the support system can withstand. Equation 1 can then be used to calculate the distance from the source over which the support system would be severely damaged for different rockburst magnitudes.

Figure 6 shows the radius of support damage as a function of support resistance for a rockburst magnitude of 3.0. The area under the curves represents the conditions where the support system will suffer damage. The radius of support damage is much less for the yielding bolts due to their ability to absorb more energy. Increasing the support resistance decreases the extent of the damage, but with reduced effectiveness at higher resistance. A better improvement is obtained by increasing the yielding characteristics even at a lower failure or slippage load.

In Ontario, bolts are usually installed at 1.2 m (4 ft) centres. For the above examples, this gives a support resistance of 35 $kN/m^2$ for the yielding bolts, and 80 $kN/m^2$ for the conventional bolts, as indicated by the arrows in Figure 6. Although the yielding bolts have less than half the resistance, their radius of damage is 18 m compared with 28 m for the conventional rockbolts.

Wagner (1984), and later Roberts and Brummer (1988), put forward a set of conditions that support systems have to satisfy under dynamic loading. Those relevant to Ontario hardrock mines are as follows:

1. The support elements must be capable of yielding at 2 m/s and preferably 3 m/s.
2. The support system must be capable of accommodating wall displacements of not less than 60 mm in drifts.
3. The support resistance should not be less than 60 $kN/m^2$ in drifts.

4. The ability of support systems to do work against the dynamic rock movement is as important as its load-bearing capacity.
5. The support system must be able to maintain the integrity of the rock mass surrounding the excavation during the entire yield process.

One of the greatest uncertainties in support design is the peak particle velocity near the source. Kirsten and Stacey (1988) suggested using a value of 10 m/s to provide a safe design criterion. Brune (1970) calculated the particle velocity, at the source, for fault-slip type earthquakes, where,

$$V\ \text{source} = \frac{\Delta\tau\ \beta}{G} \qquad \text{Eq 11}$$

where, $\Delta\tau$ = drop in shear stress along the fault

$\beta$ = shear wave velocity $\simeq$ 3600 m/s

G = shear modulus $\simeq$ 30,000 MPa.

Stress drops for earthquakes rarely exceed 10 MPa which gives a particle velocity at source of 1.2 m/s. For pillar rockbursts the perpendicular stress on the pillar, typically 120 MPa, can be reduced to zero instantaneously. Assuming a shear stress of half this value produces a particle velocity at source of 7.2 m/s. It is not surprising that in Ontario mines the damage associated with pillar bursts is much more severe than fault-slip bursts of comparable magnitude.

## LABORATORY AND UNDERGROUND TESTS

Slow and rapid loading tests on rock bolts have been done in both Canada (Hedley and Whitton, 1983), and South Africa (Hepworth and Heins, 1983). An underground trial using conventional and yielding rock bolts, subject to impulse loading from explosives has been described by Ortlepp (1969).

Some broken mechanical rockbolts from the rockburst area at Quirke Mine in Elliot Lake indicated only about half the stretch normally obtained in laboratory testing. In was thought that the rockbursts were subjecting the bolts to impulse loading which resulted in a lower failure load with less stretch. Tests were done in the laboratory on 0.76 m long bolts, 15 mm in diameter. The maximum displacement velocity achieved with the testing machine was 60 mm/sec. It was found that the failure load of 129 kN was independent of the loading rate and the stretch at failure was also constant at about 61 mm. Fatigue tests on the same type of bolts indicated that the failure load decreases with the number of loading cycles and a reduction to 102 kN was achieved. However, the bolt stretch at failure decreased marginally to 57 mm. None of these tests duplicated the lack of bolt stretch observed underground. Ortlepp (19969) has suggested that in post-yield, plastic deformation of steel cannot occur at very high displacement velocities and failure will be abrupt with reduced stretch.

Tests were done by the South African Chamber of Mines on two types of high tensile steel bolts and one low tensile steel rebar. Also a yielding device, attached at the bolt head, was tested. Displacement velocities of 0.5 mm/s (slow), and up to 1.9 m/s (fast), were used. A summary of the results is shown in Table 1.

The effect of a high displacement velocity was to slightly increase the failure load on all the bolts and the stretch remained more or less the same. Bolts with a yielding collar accommodated about 100 mm more displacement without affecting the failure characteristics. Low tensile steel rebar, with a larger diameter, had a greater failure load and stretched about 40% more than the high tensile steel bolts.

It appears that a velocity of 1.9 m/s is still not high enough to affect the yield characteristics of the bolts during plastic deformation.

Table 1 - Bolt characteristics at slow and fast loading rates (after Hepworth and Heins, 1983).

| Displacement Velocity | Failure Load | | Stretch | |
|---|---|---|---|---|
| | Slow | Fast | Slow | Fast |
| H.T. steel bolt 13.5 mm dia. | 108 kN | 115 kN | 33 mm | 32 mm |
| H.T. steel bolt 13.5 mm dia. with yielding collar | 109 kN | 114 kN | 131 mm | 138 mm |
| H.T. steel bolt 18.5 mm dia. with yielding collar | - | 198 kN | - | 146 mm |
| L.T. steel rebar 16 mm dia. | 129 kN | 137 kN | 46 mm | 47 mm |

H.T. - High tensile; L.T. - Low Tensile; Bolts 1.2 m long.

A comprehensive underground trial using conventional and yielding rock bolts was done at the ERPM Mine in South Africa. The yielding bolts had an extra 225 mm threaded portion at the anchor end. A smooth-bored die of internal diameter less than the threads was fitted to the bolts. Slippage occurred at a constant load as illustrated in Figure 7.

An isolated drift 2.7 m high by 3 m wide was bolted with 1.2 m long conventional and yielding bolts at 0.76 m centres, on opposite sides of the drift. A double layer of 8 gauge linked, 50 mm wire mesh was also installed under the rockbolt plates. A 3 m length of drift was supported in this manner. Holes for the explosives were drilled 3 m deep at 430 mm centres, parallel to the drift axis and about 600 mm from the perimeter of the drift. These holes were loaded with 100 mm by 22 mm cartridges of 40% dynamite, uniformly spaced to fill 15% of each borehole (i.e., 0.60 kg/hole, 24 holes, total charge 14 kg). This instantaneous explosive charge completely destroyed both the conventional and yielding support systems.

In a second test, the length of the yielding bolts was increased to 1.5 m and the explosive charge reduced to 8% of each borehole (i.e., 0.32 kg/hole,

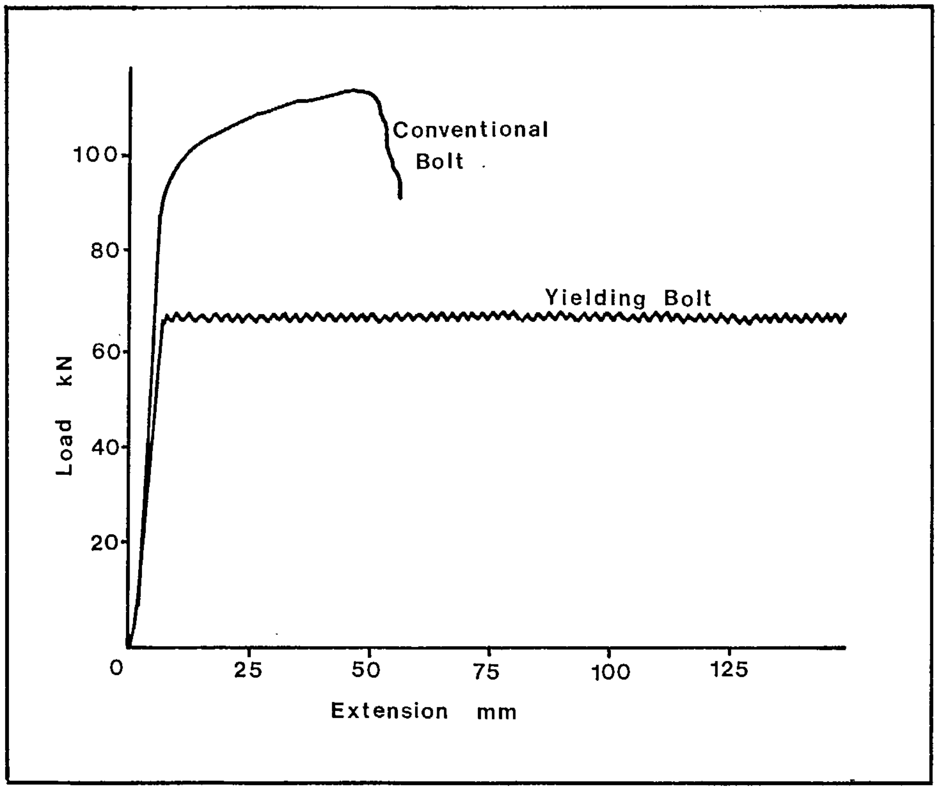


Fig. 7 - Load-extension characteristics of conventional and yielding bolts (after Ortlepp, 1969).

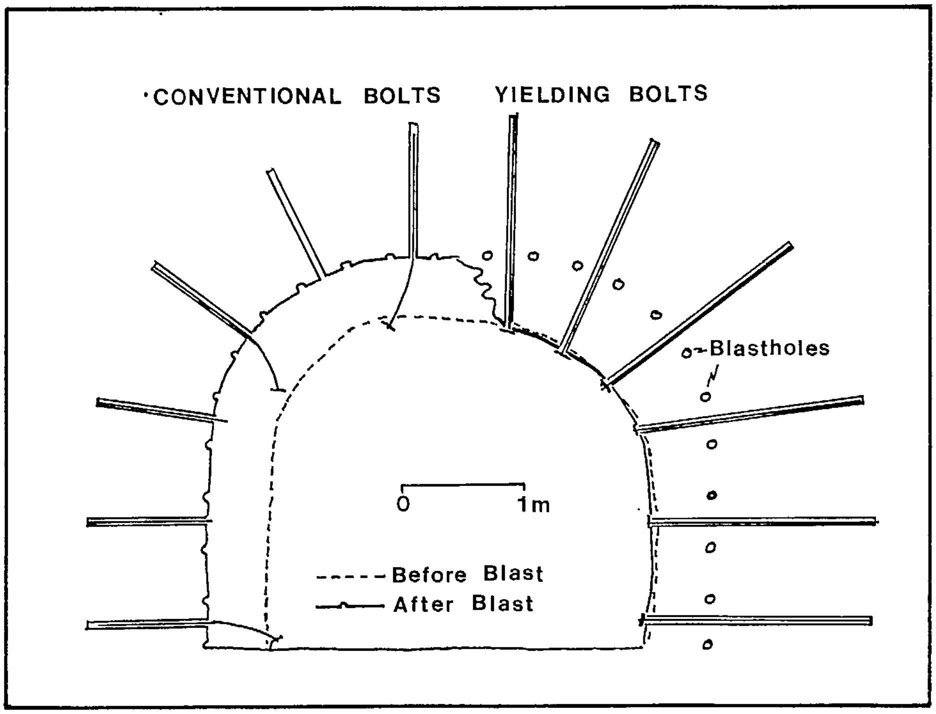


Fig. 8 - Layout of supports and drift profile after a blasting trial (after Ortlepp, 1969).

24 holes, total charge 7.6 kg). Figure 8 shows the layout of the bolts and explosive holes, and a profile of the drift after the blast. In this case the side of the drift supported by conventional rockbolts was destroyed, whereas the yielding bolts survived with minor cracking of the rock.

Unfortunately, no peak particle velocity measurements were taken in these trials.

## UNDERGROUND OBSERVATIONS ON SUPPORT SYSTEMS

The reaction of various support systems to rockbursts, over the last 50 years, has been fairly well documented in Ontario mines. The following sub-sections illustrate the failures and successes of different support systems.

### TIMBER POSTS AND BEAMS

Although wood is a relatively soft material, posts and beams do not stand up well to rockbursts and they readily buckle. It is not known whether this buckling is due to impulse loading or the closure of the drift following a rockburst. Wooden cribs, on the other hand, stand up fairly well to rockbursts, perhaps because their aspect ratio and load-bearing capacity is much greater.

### STEEL SETS

Steel sets, either of circular or eliptical shape, were used in some hardrock mines in the 1940's and 50's. Generally, wood lagging was placed behind them and any space with the wall rock was backfilled with sand. Their use comes from coal mining, with the concept of evening out the pressure over the whole support system rather than a point load. This type of support was effective in controlling damage from small seismic events, but did not prevent extensive damage from large rockbursts.

## TENDON SUPPORTS

Support tendons include mechanical bolts, rebar and cable bolts. They typically have support capacities of 110 kN, 150 kN and 260 kN, respectively.

Mechanical bolts are used extensively in Ontario mines with either a forged or nut head. Also failure of these bolts is significant in rockburst areas, especially in the mines at Elliot Lake. Three types of failure are observed: tensile failure at the threads on the anchor end; shear failure due to lateral displacement of the borehole; and tensile failure about 2 cm behind the forged head. One common observation in the Elliot Lake mines is that failure at the forged head occurs before there is any major damage to the drift or stope. In other words, bolt failure is being caused by small seismic events. Metallurgical investigations on these bolts suggest that they are failing by fatigue.

Grouted rebar, using resin or cement, is a rigid support system and would be expected to behave like mechanical bolts when subjected to impulse loading. However, very few rebar failures have been observed. Rebar has a larger diameter and hence is stronger. Also failures may be concealed by the resin or cement retaining the broken rebar in the borehole.

Grouted cable bolts are used in the Sudbury mines. Failures have occurred, but it is difficult to tell whether they are caused by structure/gravity factors or rockbursts. However, the vibrations from seismic events could be a contributing factor, even for gravity falls. Generally failure occurs at the cement/cable bond which allows the rock to unravel. Only rarely has breakage of the cables been observed. It has been found that the use of high pulp density grout improves the effectiveness of the support system (MacDonald, 1988).

Mechanical bolts, rebar and cable bolts are more effective against rockbursts when used with steel wire mesh, although some failures still occur.

At one mine a drift had one wall supported by wire mesh as well as 2.4 m long mechanical bolts. The wall with mesh survived a rockburst of magnitude 2.4, although the rock behind the mesh was extensively fractured. The wall without mesh failed completely.

## FRICTION SUPPORTS

Two types of friction supports are used in Ontario mines, a steel tube with a longitudinal slit which is hammered into a borehole (trade name, Split Set) and a sealed steel pipe which is pressurized against the borehole (trade name, Swellex). The Split Sets typically yield at about 50 kN and can slip in excess of 100 mm. Swellex bolts have a typical load capacity of 130 kN, but their slippage characteristics are unknown since, during pull tests, failure occurs at the collar prior to slippage in the borehole.

Friction supports installed by themselves do not prevent spalling of a drift when subject to rockbursts. In conjunction with wire mesh they are much more effective. At one mine a haulage drift was rehabilitated with Split Sets and wire mesh. Subsequent rockbursts extensively fractured the rock behind the mesh and slippage of at least 100 mm was observed on some Split Sets.

## LACING

Lacing is a three-tiered support system with grouted rebar inside a borehole, wire mesh and steel cable between rebars in a diamond pattern. The steel cable acts like an automobile seat belt and absorbs a considerable amount of energy radiating from a rockburst. Present day lacing is a South African gold mining innovation, however, a forerunner of lacing was used in the Lakeshore Mine at Kirkland Lake in the 1940's. In drifts opposite highly stressed pillars two steel cables were run down the roof and anchored at 5 m intervals with 2 m long rock bolts. Round wood lagging was placed between the cable and the roof. The wooden mat/steel cable formed a yielding membrane which cushioned the impact of a rockburst. Sometimes this technique was also used in the walls.

In South African gold mines mild steel rebar, 13 mm in diameter, is used as the support tendon. Wire mesh is generally chain-linked and the steel cables are 13 mm diameter slusher cables, tensioned to about 40 kN. This type of support system has survived a rockburst of magnitude 4.0 a few metres away.

Falconbridge's Strathcona Mine has also installed the same type of support system (Davidge et al. 1988). A footwall development drift and accesses to the overcut of a blasthole stope were laced. Subsequently, a rockburst of magnitude 3.0 occurred in the area. The conventionally supported overcut (grouted rebar and wire mesh) which was about 25 m away was severely damaged. The nearest lacing was about 40 m from the burst and suffered no damage. The same area was subjected to a series of rockbursts in June 1988 up to a magnitude of 2.7. This time, in places, the rock fractured and had 'bagged' behind the mesh, also, one broken rebar was observed.

At INCO's Copper Cliff North Mine a different design of lacing has been installed. In this case, welded wire screen is held against the rock using 2 m long Split Sets. Steel cable in a square pattern is tensioned against the wire screen by driving a short 45 cm long Split Set inside the longer Split Set.

## DISCUSSION

From theoretical considerations and underground observations, the effectiveness of various support systems to rockburst conditions can be evaluated, in at least a qualitative sense. Mechanical bolts are the most prone to failure and lacing is the most effective. In between these two extremes, different types of supports can be used, based on the likely maximum rockburst magnitude they are subjected to, and the distance from source.

This concept is shown in Figure 9, with the ranking of the support

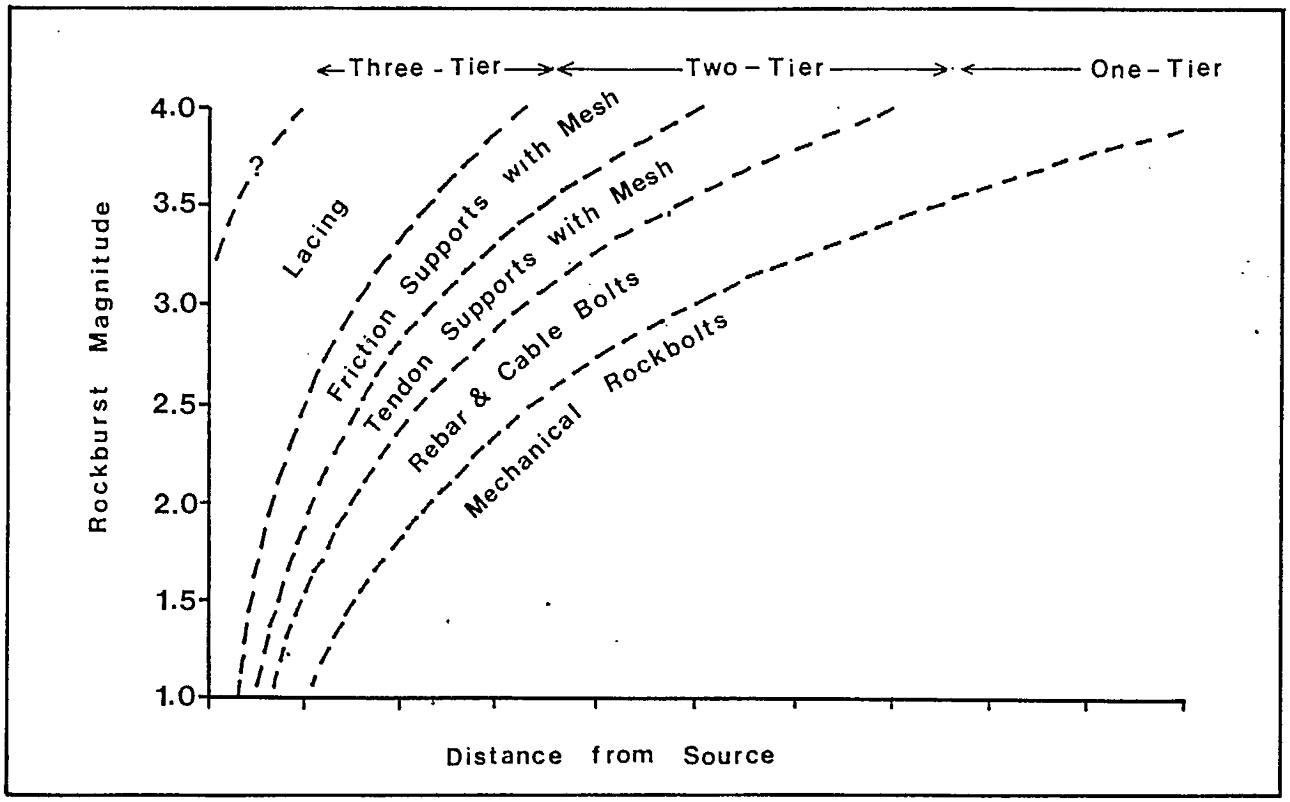


Fig. 9 - Conceptual support requirements for rockburst conditions.

techniques mainly based on underground observations. The ranking is not exclusive. For instance, a mechanical bolt with a yielding mechanism could have the same yielding characteristics as a friction support. The curves are actually the peak particle velocity that each support system is considered capable of withstanding. At present, numerical values cannot be placed on the axes of the graph. Back analysis of examples of support failures and successes are required. Also in situ trials, with explosives generating sufficiently high peak particle velocities, would provide additional information.

## ACKNOWLEDGEMENTS

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