SECTION 8a

THE USE OF GEOMECHANICAL INSTRUMENTATION IN COST CONTROL IN UNDERGROUND MINING

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ABSTRACT

In deep underground mining, ground support represents a significant component of the total mining cost. Serious failure of the ground support system in such high stress areas, and particularly in burst prone ground, can result in production delays, loss of reserves, damage to equipment, injuries, etc., the cost of which greatly exceeds the original ground support cost. Historically, the major problem has been the inability of engineering and operations staff to evaluate the "as placed" status of a ground support system, with any degree of confidence, as the mining front progresses. The development of "direct ground support instrumentation" [e.g. SMART cable bolts; SMART contractometers] combined with conventional geomechanical instrumentation [e.g. microseismic monitoring systems; Multiple Position Borehole Extensometers; stress cells; etc.] provides the ability to evaluate ground support behaviour and available support capacity in real time. Such data is invaluable in assessing critical operational factors such as safe access to mine openings, the need for and timing of rehabilitation, the performance and safety of pillars, and backfill behaviour, etc. The use of instrumentation systems and their impact on the direct ground support and mining costs are discussed in this paper.

8b.1 INTRODUCTION

Figure #8b1 shows the three primary functions of ground support elements. The role of each element is described fully in the Canadian Rockburst Research Program, Chapter 2 [1996] and are repeated here briefly for completeness.

The goal of *reinforcing* the rockmass is to strengthen it and to prevent loss of strength, thus enabling the rockmass to support itself. Even when fracture initiation is not prevented, the reinforcing mechanisms help to restrict and control bulking of the

rockmass, thus ensuring that inter-block friction and rockmass cohesion are fully exploited. Hence reinforcement is beneficial even when the rockmass eventually breaks up.

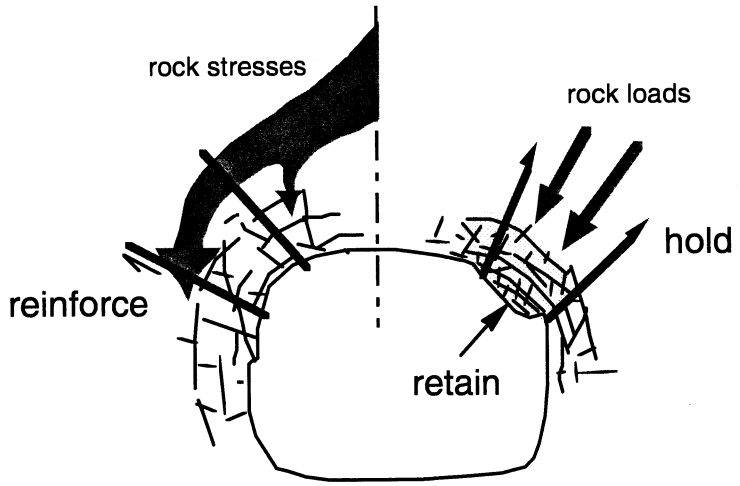

Figure #8b1: Three primary functions of support elements [After Canadian Rockburst Research program, 1996].

While *retaining* broken rock may be required for safety reasons it becomes essential under high stress conditions to prevent progressive failure processes leading to unraveling of the rockmass. Field observations indicate that full areal coverage by retaining elements becomes increasingly important as the level of rockburst severity increases [McReath and Kaiser, 1992].

The *holding* function is needed to tie the retaining elements of the support system back to stable ground. Under certain conditions this can be accomplished using high strength anchorage into deeper-lying stable rock. However, under conditions of large imposed (relative) displacements between the anchor and the head of the holding element, a yielding support element is required.

Under many conditions, and particularly under deep and high stress mining, an *'integrated support system'* comprised of combined retention, reinforcing and holding elements is required. While some design tools exist [e.g. the Stability Graph for cable bolt design; numerical packages such as Unwedge for gravity wedge support design; etc.] ground support design remains highly empirical in nature. The design of 'integrated support systems', particularly under high stress ground conditions, thus remains more engineering art than engineering science. Ground support comprises a significant component of overall underground mining cost. Instrumentation is essential in order to minimize and control ground support costs while providing a safe work environment and maintaining production.

8b2 GENERAL USE OF INSTRUMENTATION FOR GROUND SUPPORT COST CONTROL IN UNDERGROUND MINING

Instrumentation is key element to any integrated support design. Cablebolts represent the only support element that, at present, can be directly and routinely instrumented. Instrumented SMART cable technology was introduced in 1997 [Hyett et al, 1997] and is patented in Canada, the USA, Australia, South Africa and Europe. Bawden [2002] describes the development of this technology which has achieved increasing application in mining and civil engineering projects. SMART cable bolt instruments have two major impacts on mining operations:

1. safety, and
2. support and operational cost control.

Instrumentation, including SMART cables and associated conventional and microseismic instruments, are critical to understanding the function of integrated support systems and to optimizing integrated ground support design. In this way such instrumentation directly impacts ground support and mine production costs.

8b2.1 Cable bolt design optimization

Optimization of the cable bolt array, both in terms of support density [i.e. pattern] and support length can result in a decrease in the number of cables used in an array. More significantly, cable bolts are often placed to a much greater depth than actually required since the optimum cable length is impossible to determine theoretically. SMART cable instruments indicate appropriate cable lengths for particular mining situations. In critical areas longer [and less expensive] MPBX instruments can be used to test for possible ground movement beyond the cable bolt support. A case study of this aspect presented by Gauthier, 2000, is discussed later in this paper.

Instrumented cables can be used to evaluate the utility of cable bolt support in critical areas such as brows and drawpoints where the rockmass is often highly relaxed, [i.e. areas where poor cable bolt performance would be expected]. An example of this is presented by Nelson (et al), 2000. *Where cable performance is shown to be very poor, the cable support should be abandoned in favor of alternate support techniques*.

With bulb cables, cable stiffness can be engineered by varying the cable bulb spacing [Moosavi, 1997]. SMART cables, in combination with SMART MPBX's, can be used to evaluate the advantage/disadvantage of varying cable stiffness. For example, the optimum cable bulb spacing can be determined by comparative instrumented tests of different bulb configurations in-situ.

Confusion in the industry continues concerning the need for addition of surface fixtures [i.e. plates] on cables. The requirement for plates on plain strand cable should go without saying since under many common ground conditions plain strand cable will not mobilize sufficient bond strength to resist stripping failure. Such failure is commonly accompanied by untwisting of the cable through the grout. Plates on plain strand cable help to resist stripping failure. In the case of bulb strand cable however the attachment of surface fixtures in many cases is of little value. De Graff [1998] in an experiment at the Hudson Bay Mining and Smelting Callinan Operation showed that plated and non-plated bulb cable performed identically. Even in areas of personnel exposure, since cable bolts are

secondary support and loose surficial broken rock should be retained by the bolt and screen primary retention support, there is no pragmatic benefit to plating bulb cables. SMART cable instruments can be used to demonstrate this fact to crews, mines inspectorate personnel, management, etc.

8b2.2 Controlling support rehabilitation issues

Rehabilitation of mine access infrastructure, generally resulting from support failure due to over-stress from the advancing mining front, represents one of the most expensive issues in ground support practice. Not only is the rehabilitation work itself costly, it poses higher risk to the work force and often delays production. Rehabilitation often requires the application of an integrated support design. SMART cables can be used to help evaluate the interaction of cable bolt support with other support types [e.g. shotcrete] in order to estimate the remaining capacity of the combined support system as the mining front advances. Such knowledge can then be used to evaluate rehabilitation requirements for affected areas. Timing of rehabilitation, often critical to mine production resource allocation, can be 'engineered' by working with the known properties of the cable bolt support. Figure 8b2 shows a load – strain curve for a typical 15.2mm seven strand steel cable. Working with this basic data and using instrumented cables, standard rehabilitation procedures can be established. For example, at one operation in Canada for non-dynamic loading conditions, the following cable bolt rehabilitation control standard is applied:

- No rehabilitation required until cables reach ~ 1% strain or a load of ~ 20 tonnes/strand.
- After 1% strain if access to the area must be maintained cable replacement is required. Between 1% and 2.5 % strain cables can be replaced by drilling all holes in the pattern, stuffing all cables and finally grouting and plating [if required].
- After 2.5% strain cables must be replaced using complete cable installation [i.e. drill, stuff and grout] two rings at a time. Depending on ground conditions the area may require a shotcrete spray using a remote arm prior to drilling.

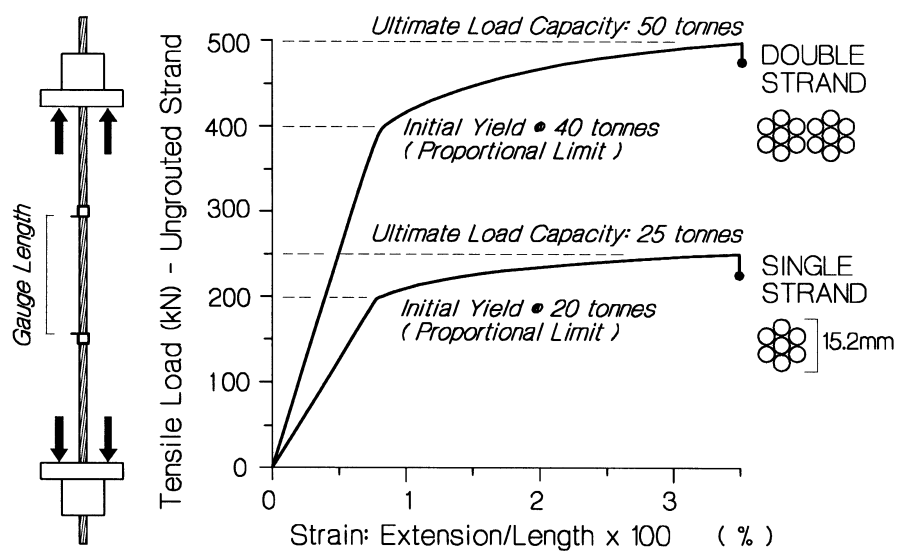

Figure 8b2: Minimum strand performance specifications for cablebolting applications [modified after Hutchinson and Diederichs, 1996].

8b2.3 In-stope cable instrumentation

Instrumented ground support 'in-stope' can provide tremendous advantage to mine operators. Some of the advantages gained from such instrumentation include:

- Optimization of stope cable bolt support arrays.
- Provide protection for LHD equipment being used for remote mucking.
- Provide real time data for management concerning the timing of backfill. For example, should a stope be fully cleaned or should some ore be left such that filling can be done prior to potential back failure of an overlying stope?

While 'in-stope' instrumentation can provide significant advantages and return on investment, the risk of complete instrument failure due to severing of lead wires during blasting today presents a serious obstacle to the use of conventional instrumentation under such conditions. Successful 'in-stope' instrumentation programs can and have been accomplished [e.g. Golden Giant, Bousquet II, and Callinan mines]. This requires that instrument leads be placed in protective steel or heavy rubber conduit. The conduit should then be attached to the screen along the center of the back of the stope access and should be covered with several centimeters of shotcrete. Such efforts require a significant commitment on the part of the operator. Achieving this under normal production constraints is extremely difficult and high instrument failure rates should be anticipated, even with full protection.

8b2.4 Instrumentation under rockburst conditions

Safety issues can easily mean the difference between successful ongoing operations and premature mine closure. The impact of instrumented cable bolt support on mine safety can therefore be very dramatic. The importance of this aspect increases exponentially with increasing mine extraction [and therefore with increasing mine induced stress] and for operations susceptible to rockburst conditions. Increasing extraction ratios can lead to significant mine induced stress damage in the backs and walls of critical long term infrastructure [e.g. main haulage drive intersections, ramps, crusher stations, etc.], key areas where personnel are exposed. This can result in the loss of effectiveness of the primary support and the subsequent need for longer and higher capacity cable bolt support. SMART cables are critical in determining the ongoing support capacity [and hence factor of safety] for such critical infrastructure as mining advances. In potential rockburst areas it is imperative to demonstrate that gradual loading of the support due to increasing mine induced stress damage has not consumed support capacity to such a degree that the support cannot withstand additional dynamic loading due to a rockburst event [Bawden et al, 2002]. SMART cable technology has been applied to help manage risk in strongly rockburst prone areas in a number of mines in Canada. Cost implications from one such application are discussed later in this paper.

8b3 EXAMPLES OF INSTRUMENTATION BASED GROUND SUPPORT COST CONTROL

Ground support instrumentation programs, when properly designed and executed, can result in extremely attractive return on investment [ROI's] to operations. The statement

'when properly designed and executed' in this case is critical. Instrumentation should not be installed underground without the following:

- a well formulated model of what is to be measured and why,
- a clear understanding of how the proposed instrumentation will provide the required measurements, and
- a well formulated model on how the required data will be used.

The above is particularly important in areas of direct mine personnel exposure. For example, if instrumentation is to be used to restrict personnel access to an area, what criteria will be applied to permit re-entry?

Several case studies are presented in the remainder of this paper indicating the type of ROI that can be achieved using well designed and executed ground support instrumentation programs.

8b3.1 Cable bolt support optimization – Callinan Mine [After deGraff, 1998]

An extensive ground instrumentation program was undertaken at the Hudson Bay Mining and Smelting Co. Ltd. (HBMS) Callinan mine. This 600,000t/yr base metal mine is located in western Manitoba. Sublevel open stoping, by longitudinal retreat, is the most commonly used stoping method. The instrumentation included; two Multiple Point Borehole Extensometers (MPBX's) and nine instrumented cables, using the SMART technology [Figure 8b3]. The field trial was designed to allow *in situ* performance of different cable types and configurations to be compared and quantified. The experiment also aimed to provide a better understanding into cable-rock mass interaction, which in turn would allow cable support design to be optimized.

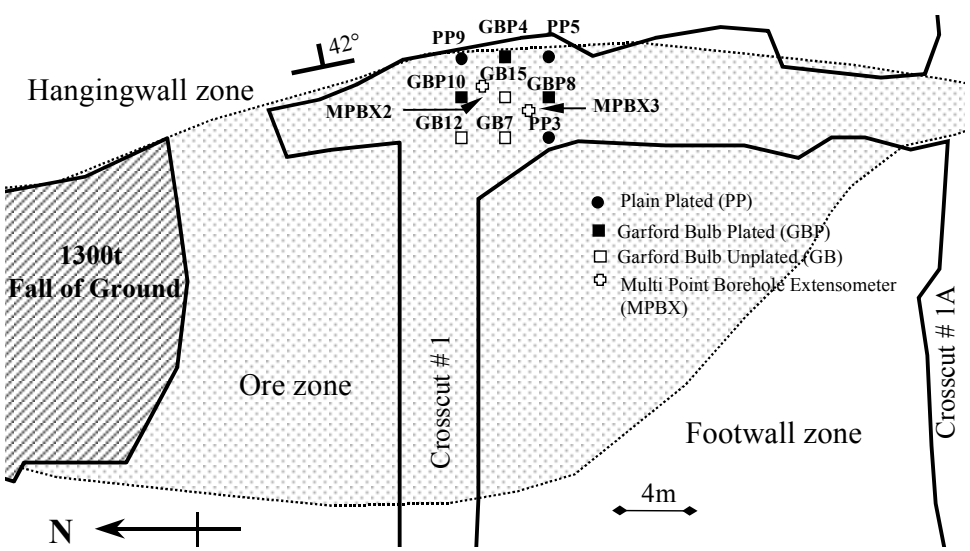

Fig.8b3: Schematic plan view of the Callinan test stope, East Zone, 1050m level, showing the geotechnical zones and the SMART instrumentation locations.

The ground conditions in the test stope at Callinan may be simplified into three geotechnical zones: hangingwall, ore and footwall zones. Rock mass properties of the

footwall, hangingwall and ore zones were assessed to be '*good rock*' according to both the RMR'$_{76}$ (GSI) and Q' classification systems. A dual stress driven and structurally controlled deformation mechanism was recognized. The stress driven component was evident since rebar faceplates were observed to have failed and some sidewall slabbing was apparent. Structurally controlled failure had been identified in adjacent stope backs, and was confirmed visually when the test stope back failed.

The Callinan stope represents a late stage, sill pillar or remnant stage of mining [Figure 8b4]. The excavation sequence of mining in the East Zone, subsequent to installation of the SMART instrumentation, is illustrated in Fig. 8b5.

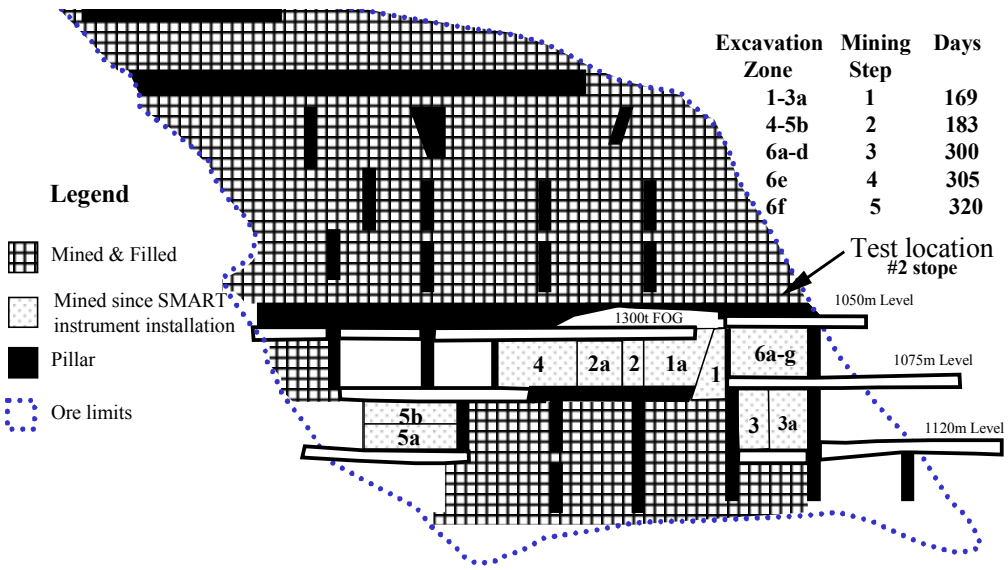

Fig. 8b4: Longitudinal section of the Callinan East Zone showing the excavation zones in the mining sequence.

The instruments were installed in the #2 stope on the 1050m level. Conventional cables within the existing support design were replaced with SMART cables. The instrument array comprised two different cable configurations (plated and unplated) and two cable types (Garford and plain strand) which were combined to make up three SMART cable instrument types; plated plain strand (PP), plated Garford (GBP), and unplated Garford (GB) cables [Figure 8b3]. This allowed load distributions along conventional and modified geometry cables to be quantified and compared, as the rock mass responded to mining induced stress changes. Standard ground support had been comprised of single 7.5m long plain plated cables on a 1.8 by 1.8m pattern, and 2.25m (22mm) resin rebar on a 1.25 by 1.25m spacing. Fig. 8b5 shows the detailed mining sequence of the development of the slot and excavation of the test stope.

The result of this instrumentation program was the complete redesign of cable bolt support at the Callinan mine. The new support design comprised the following:

- High stiffness, high capacity, no faceplates,
- 7.5m long, twin unplated GB cables on 2.0 x 2.0m, [with the potential to expand up to 2.2 x 2.2m with further testing], and

- Significantly improved support design performance.

The immediate economic return on this instrumented ground support experiment included material and drilling cost savings of 9.5% and 31% respectively.

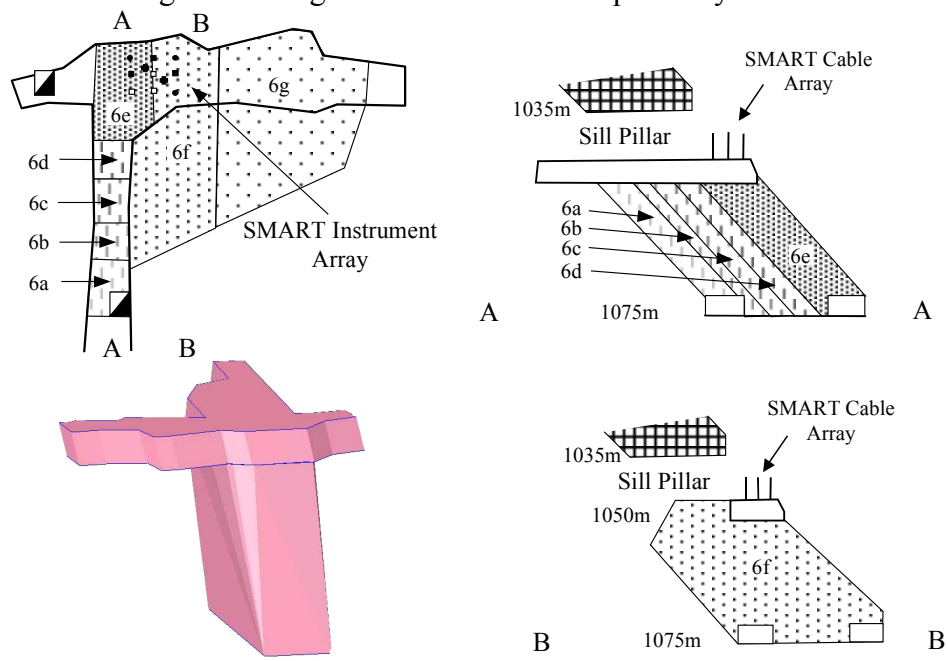

Fig. 8b5: Views showing the sequence of mining (6a-g) within the test stope. Clockwise: Plan, cross-sections, and perspective view of the test stope at the completion of the 6f excavation zone.

8b3.2 Cable bolt pattern optimization – Bousquet II mine - an in-stope experiment [After Gauthier et al, 2000]

The Barrick Gold Bousquet II mine in Northwestern Quebec, Canada, utilized a primary/secondary long hole stope extraction method to produce 2200 tpd of ore. The 3-1 Zone was a new mining zone located approximately 4500 ft below surface. For primary stopes the mine was using approximately 530 meters of single strand Garford bulb cables per stope overcut. At a cable cost of $35.00/meter this resulted in an average cable support cost of $18,550 per stope. The Mine was spending a total of $1.52 Million a year on cable bolting (43,500m) in stopes.

An instrumentation program was designed to test existing cable bolting patterns in order to maximize the cost benefit of the cable bolt support. Typical primary [Figure 8b6] and secondary [Figure 8b7] stopes were instrumented. The primary stope had 8 SMART cables and 1 MPBX while the secondary Stope had 5 SMART cables and 1 MPBX. The 3500 level #32 stope was selected as the primary test stope. Six SMART cables were installed in the hangingwall and 2 in the stope back. Lead wires were brought out of the stope in a 2” ABS pipe that was shotcreted in place. Readings were recorded with a STARS data logger and retrieved remotely on surface via phone modem connection.

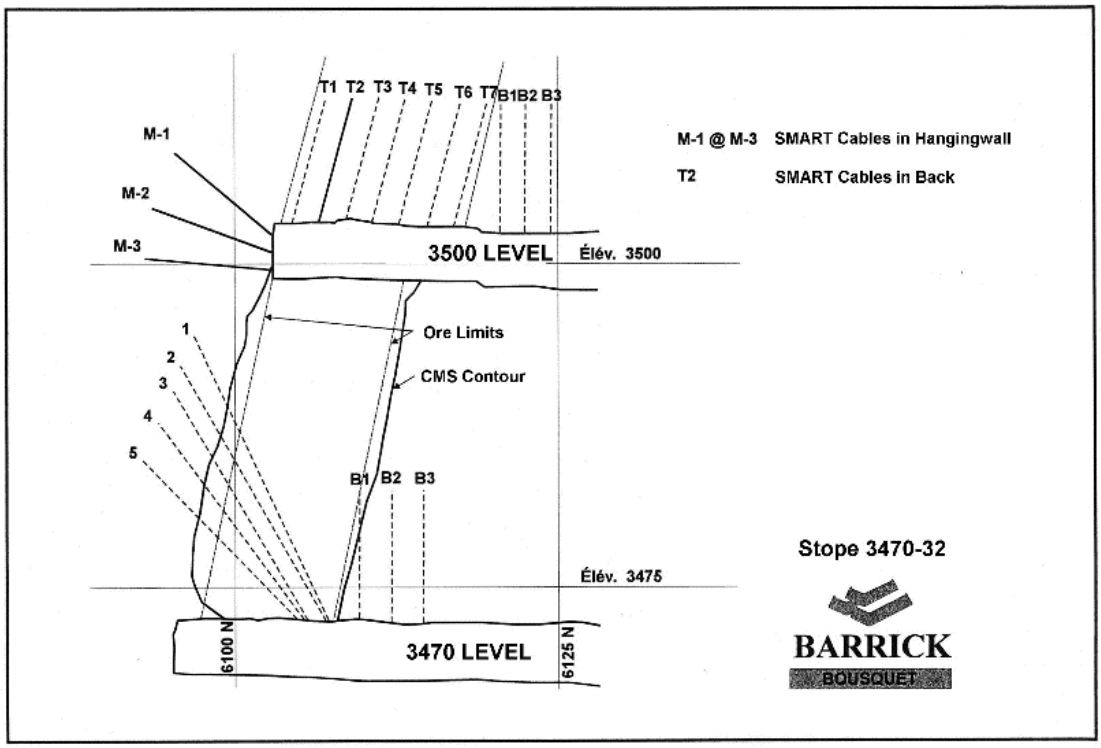

Figure 8b6 Location of SMART instruments in primary stope [After Gauthier, 2000].

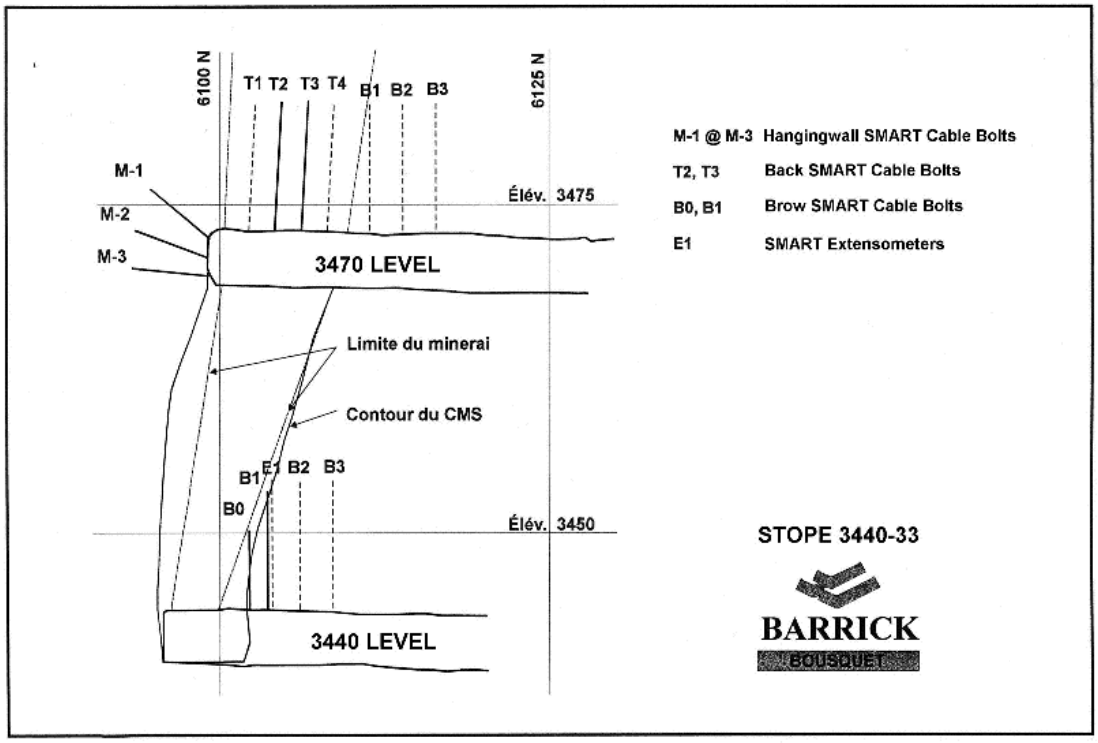

Figure 8b7 Location of SMART instruments in secondary stope [After Gauthier, 2000].

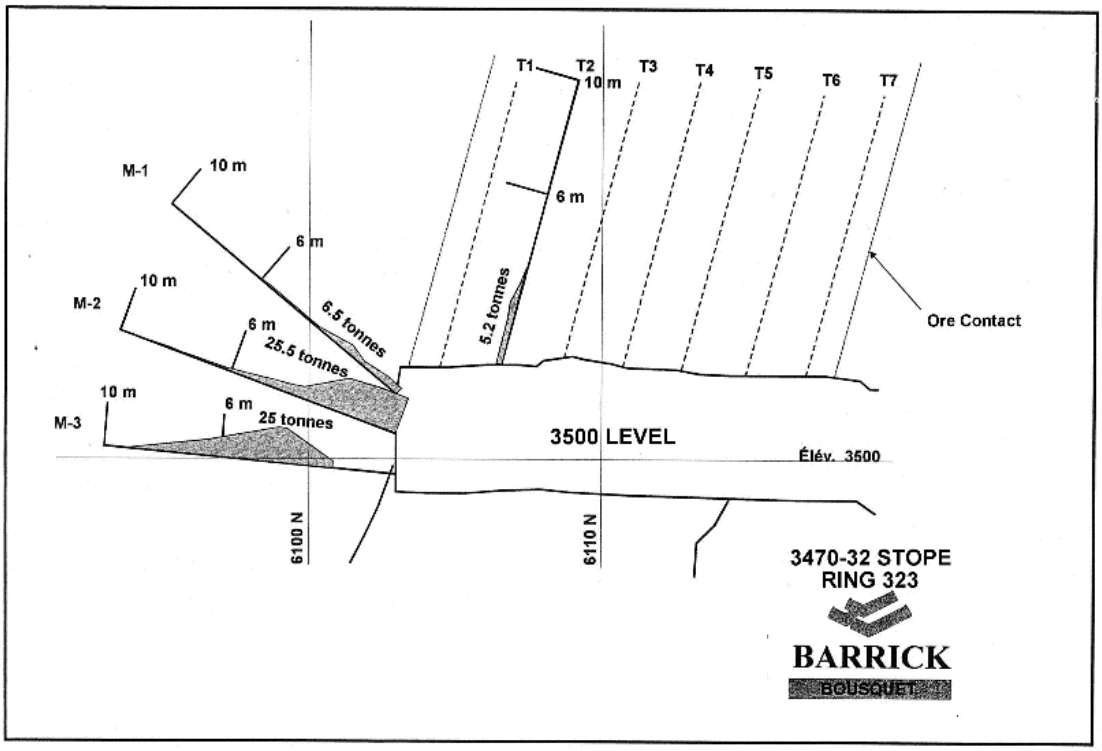

Figure 8b8: Load distribution on primary stope SMART cables following final stope blast [After Gauthier, 2000].

Two cable rings in the primary stope were instrumented in the manner shown in Figure 8b6. The instrumentation results, as typified in Figure 8b8, indicated that holes M-2 and M-3 had all yielded. The movement recorded in the hole 3 instruments exceeded 100mm. SMART cables and the SMART MPBX located in the stope back however recorded less than 6mm of movement and no movement past 2m above the back. The stope suffered significant hangingwall dilution, with 3-4 m of over break. The majority of the load on the hangingwall cables was observed to occur within 6m of the collar, and back cable loads were observed to be insignificant [i.e. easily handled by primary rebar support].

SMART cable results from the secondary stope showed similar results [Figure 8b9] with the exception that back loading was observed to occur above the 2.1 m depth limit of the primary rebar support.
Based on the results of the in-stope SMART instrument study, the cable bolt standards for primary and secondary stopes were altered as shown in Figures 8b10 and 8b11. The economic impact of this change in cable support design was very dramatic, as shown in tables 8b1 and 8b2. The mine wide economic impact of this simple instrumentation study for the 3-1 zone was a reduction of 9300 meters of cable bolting, resulting in annual savings of $325,000 (projected over five years of reserves approximately $1.5M in savings). The instrumentation cost was $11,150 for the primary stope and $7,250 for the secondary stope, resulting in a 1 year return on investment of 94%.

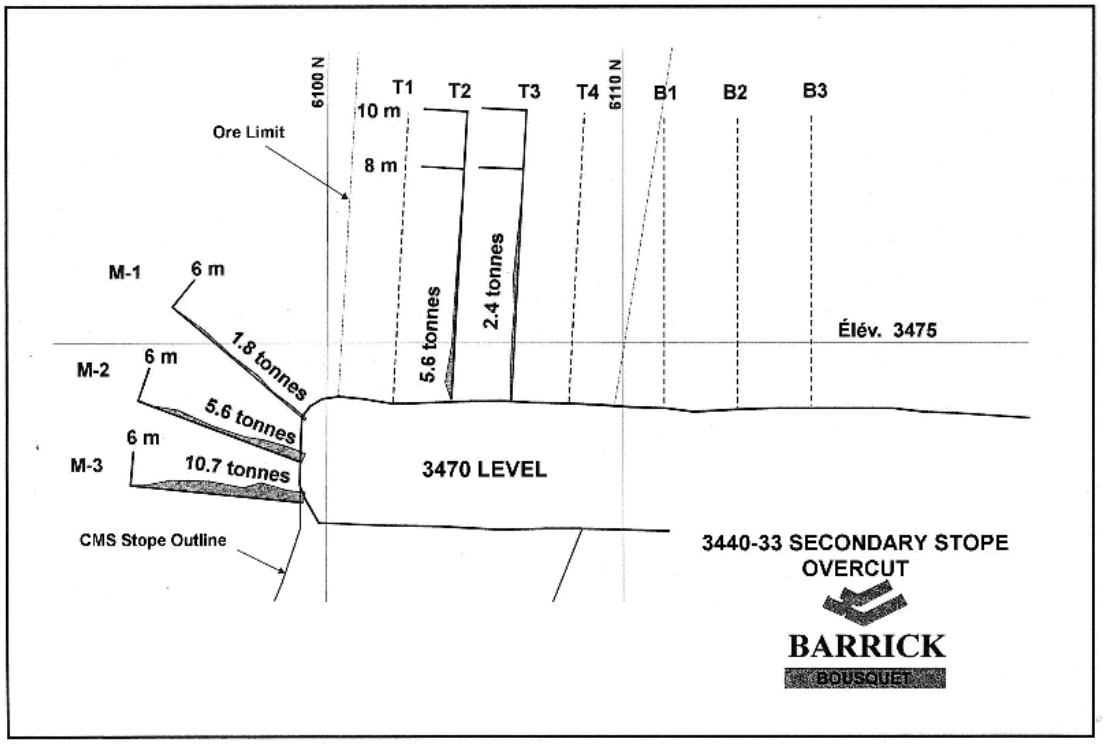

Figure 8b9: Load distribution on secondary stope SMART cables following final stope blast [After Gauthier, 2000].

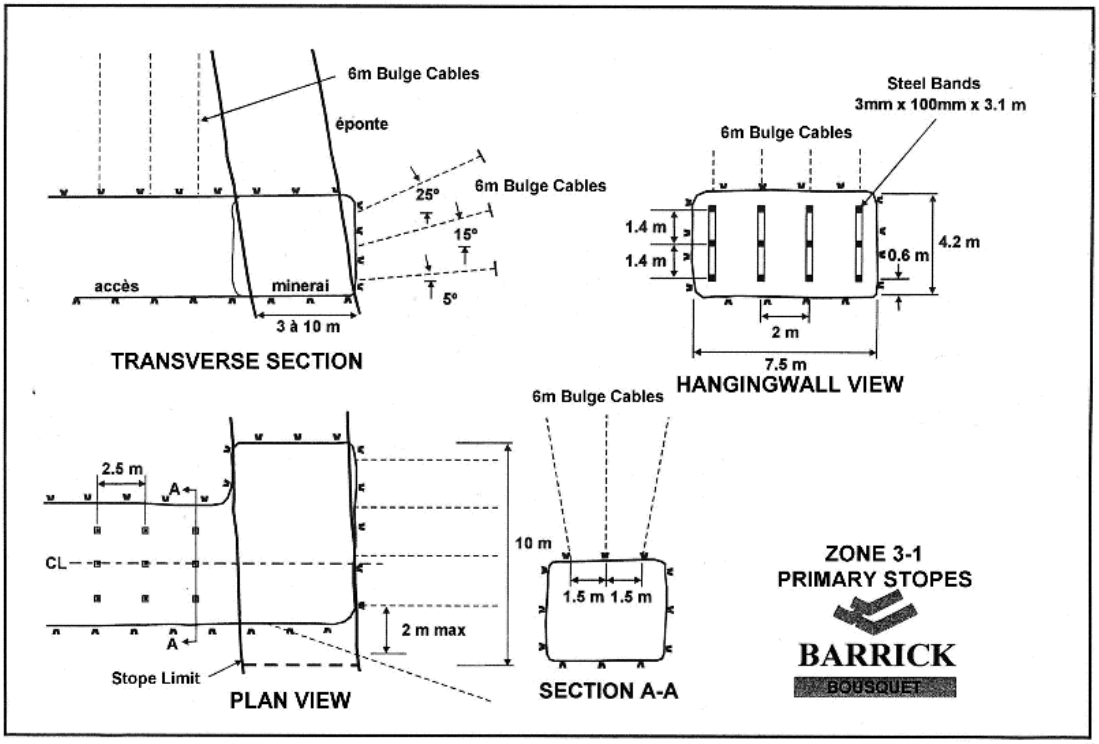

Figure 8b10: Revised cable layout for primary stopes [After Gauthier, 2000].

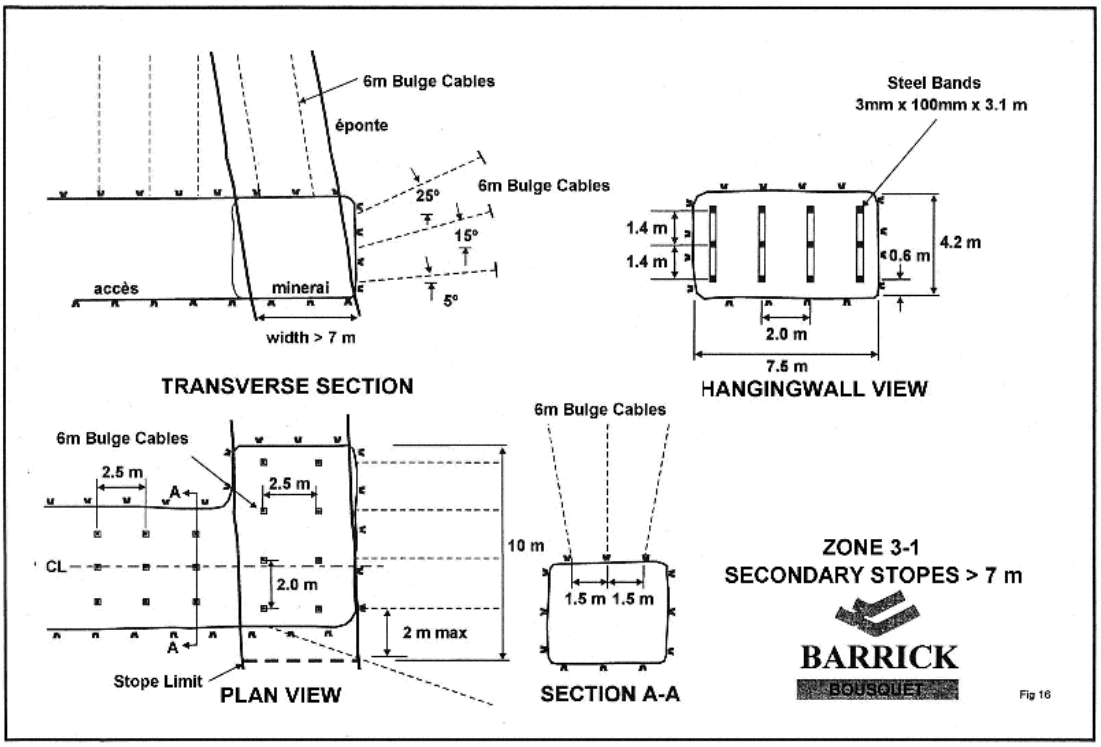

Figure 8b11 Revised cable layout for secondary stopes [After Gauthier, 2000]

Cable Bolting Requirements in a Typical Primary Stope in the 3.1 Zone

Description	Old Standard	New Standard	% Reduction
Stope Back	150 m	0 m	100 %
Stope Hangingwall	200 m	90 m	55 %
Stope Overcut	180 m	108 m	40 %
Total	530 m	198 m	63 %

Total Cost Savings of $11,620 per primary stope on average

Table 8b1: cost savings for primary stopes [After Gauthier, 2000].

Cable Bolting Requirements in a Typical Primary Stope in the 3-1 Zone

Description	Old Standard	New Standard	% Reduction
Stope Back	150 m	90 m	40 %
Stope Hangingwall	200 m	90 m	55 %
Stope Overcut	180 m	108 m	40 %
Total	530 m	288 m	46 %

Total Cost Savings of $8,470 per secondary stope on average

Table 8b2: cost savings for secondary stopes [After Gauthier, 2000].

8b3.3: Instrumented shotcrete pillars – Newmont Golden Giant Mine [After Chew, 2000].

Longitudinal retreat mining of deep, narrow vein lenses under sand fill at the Golden Giant mine resulted in shear failure parallel to the ore - HW contact [Figure 8b12]. Cable bolt support was applied to control this failure. However the cable bolt support was very rapidly taken to yield and it was concluded that, due to geometric constraints in the relatively small sill drive, adequate cable coverage could not be achieved to control the ongoing failure process and to assure safety of personnel. Shotcrete pillars were then installed to support the back. The ultimate behaviour of the shotcrete pillars under these operating conditions was uncertain. Newly developed instruments, SMART contractometers, were installed in several of the shotcrete pillars to monitor pillar behaviour as the stope was retreated. The purpose of the instrumentation program was twofold:

- To determine loads developing in shotcrete pillars, and
- To optimize the design and construction of shotcrete pillars.

Figure 8b13 shows the contractometer installation and the final shotcrete pillar. Several important lessons were learned during the shotcrete pillar design and construction:

Sono-tubes are not strong enough for forming shotcrete pillars:
  – Half-culvert bolted together much stronger

Blowing shotcrete into pillar resulted in too many voids:
  – Better to pump shotcrete and vibrate to remove voids

Design procedure used was effective:
  – pillars were not over designed

Pillar failure occurs at about 1% strain:
  – For 3m pillars, this is about 30 mm of deformation

Contractometer movements:
  – coincide with movements of other instrumentation
  – match visual damage assessments in pillars.

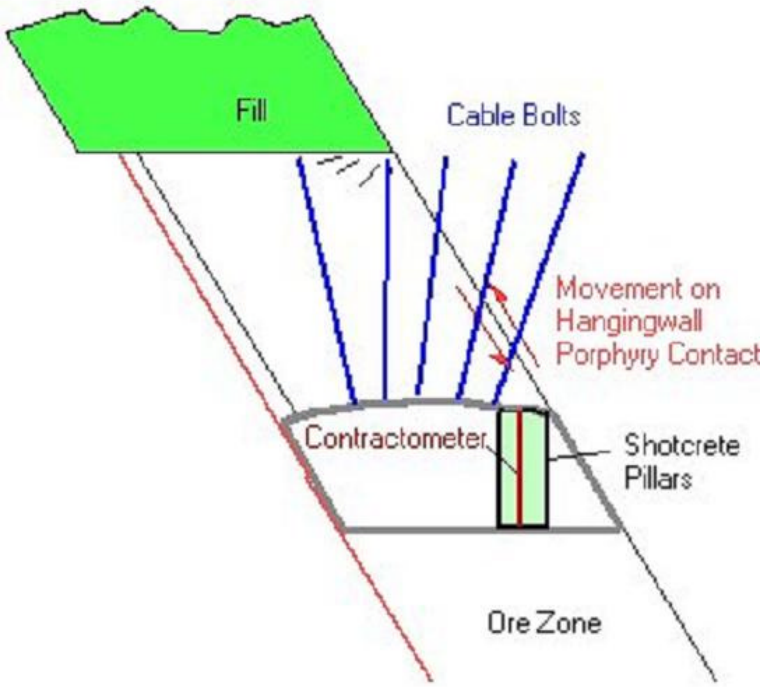

Figure 8b12: Schematic section showing shotcrete pillar installation.

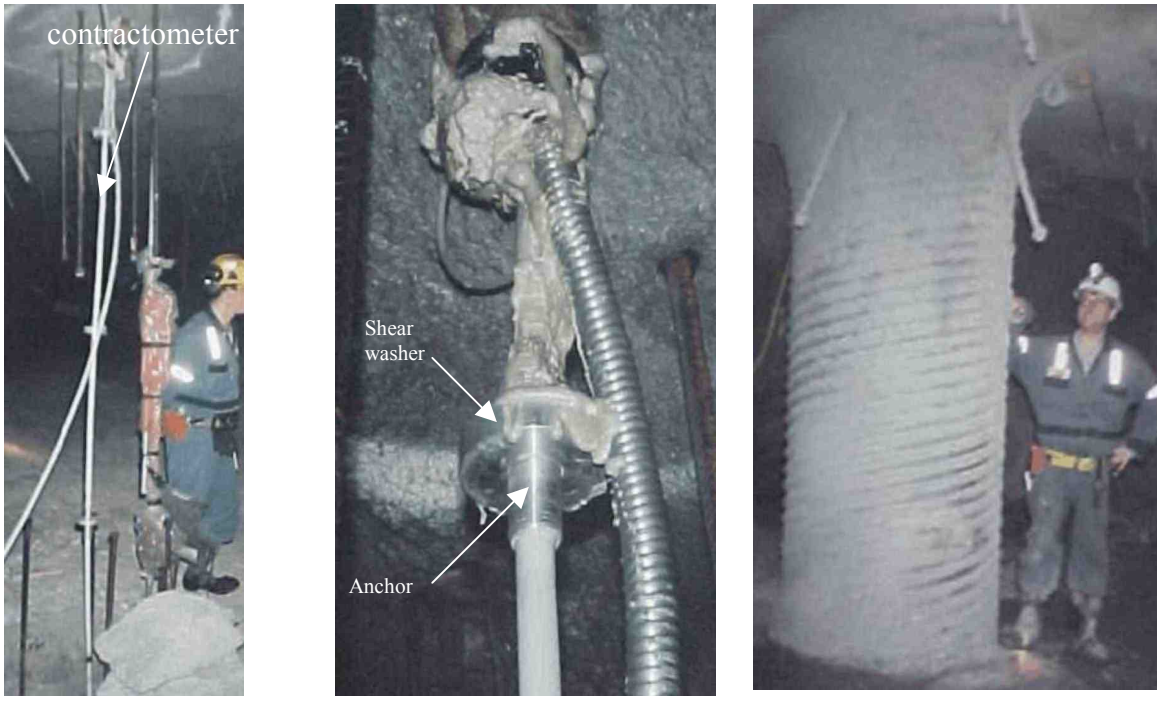

Figure 8b13: Instrumented shotcrete pillar.

It is difficult to assess the load condition of shotcrete pillars visually. The contractometer instrumentation program allowed internal quantitative assessment of pillar damage as the mining front approached. Operators gained confidence in the contractometer data and this critical longitudinal retreat zone was successfully extracted without incident. The cost of the entire instrumentation program was under $10,000 CDN.

8b3.4: Cost control using instrumentation under strong rockburst conditions [After LeBlanc et al, 2000, Bawden et al, 2000 and Bawden, 2002].

Williams Mine, located in the Hemlo region of Northwestern Ontario, is the largest underground gold mining operation in North America. Annual production is currently 2.1 million tonnes from underground and 400,000 tonnes from a surface pit operation, generating approximately 400,000 ounces of gold. The uniformity of the orebody, with its steeply dipping orientation, lends itself well to longhole open stope mining with delayed backfill. The two main mining areas in the B Zone are Block 3 and Block 4, which are separated by a sill pillar. Block 3 has been in production since mid 1987 and the mining configuration is a chevron shape. [Figure #8b14]. Initial indications of problems in the sill pillar began shortly after removal of the first stope under backfill in 1994. As mining progressed several sidewall failures occurred, along with the first significant back failures. In November 1996, the first major ground failure occurred, which affected the mine's ability to produce from this area. As mining continued the frequency of ground falls increased. In all, from November 1996 to October 1997 there were four major ground occurrences in the Block 4 sill pillar area. This delayed the mining of approximately 1,000,000 tonnes containing some 300,000 ounces and seriously hampered production from the mine.

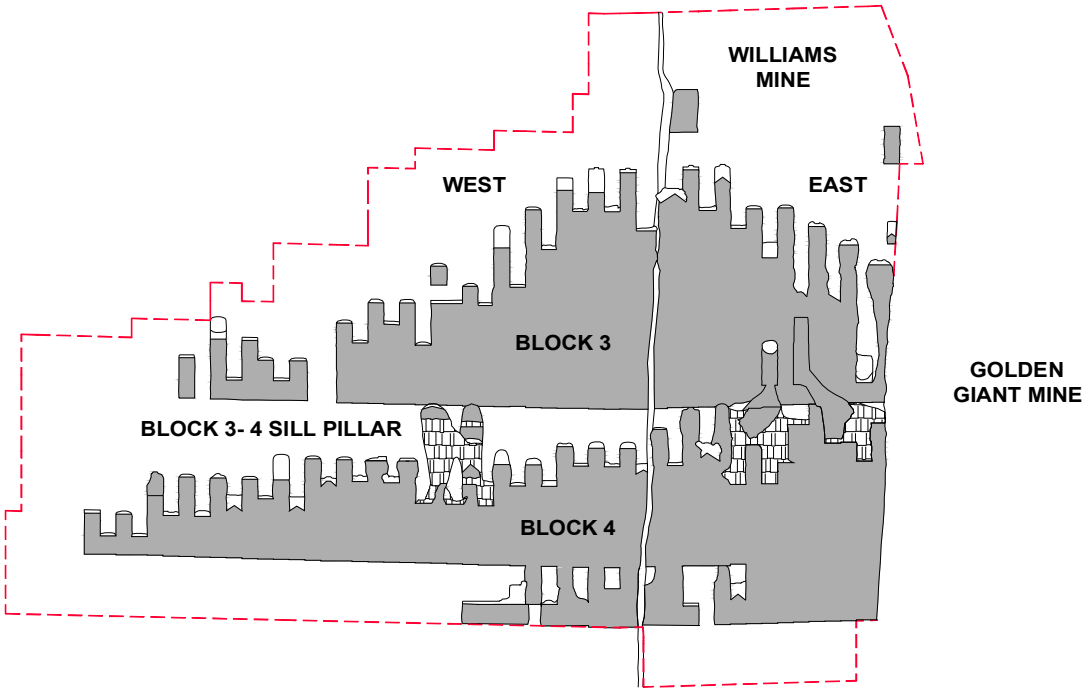

Figure 8b14: Longitudinal of B-Zone looking north. [After LeBlanc and Murdoch, 2000].

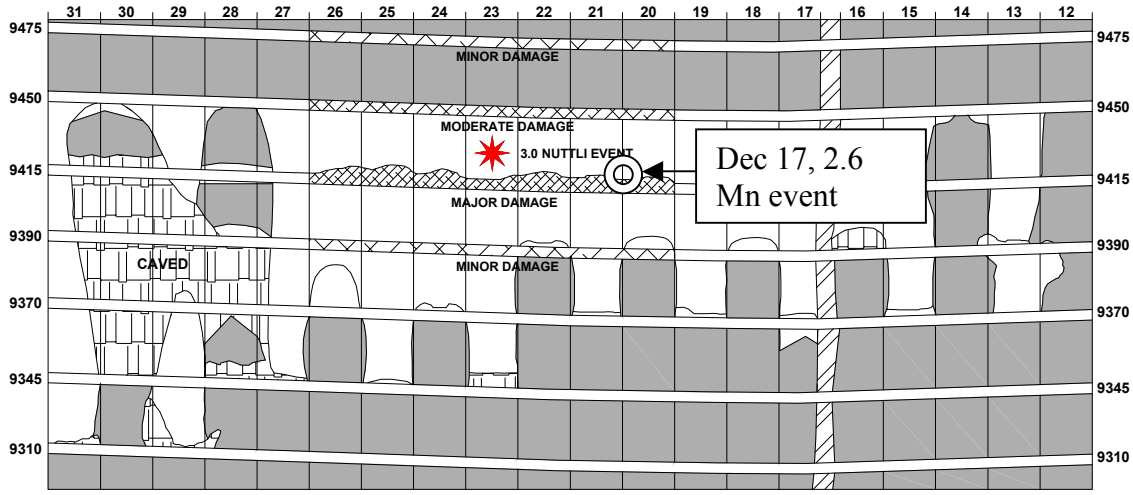

Figure 8b15: Longitudinal looking north of area affected by March 29th rockburst [Modified after LeBlanc and Murdoch, 2000].

On March 29, 1999 a magnitude 3.0 Nuttli rockburst occurred in the Block 4 sill pillar [Figure 8b15]. The event was felt on surface and was picked up by the Geological Survey of Canada at several sites in Ontario. Previously, no event larger than an estimated 1.0 Nuttli had been experienced in the Hemlo camp. The location of the event was a major concern as previously all ground fall and seismic activity had taken place within the ore zone. In this case, the main damage zone was located in the footwall drift, centered between 18 x/c and 26 x/c on the 9415 level, one level below the cemented sill of Block 3. Massive failures occurred in the back of the footwall drift at the cross cut intersections from 20 x/c to 26 x/c on 9415. Floor heave, buckling of the lower south wall and spalling of the upper north corner of the foot wall drift occurred on the 9450 level throughout the same area. The center of the damaged area was located in the shadow of the #3 ore pass system. Only minor damage occurred on levels above 9450 and below 9415. The only active mining ongoing in the area was 26-9370 stope, where the first lift had been blasted and removed one week earlier.

Approximately 160m of extensive rehab work was required to reopen the 9450 level. The rehab consisted of scaling and debagging loose, rebolting with 5' resin grouted rebar on a 1.1 m x 1.1 m pattern and screening sill to sill with 6 gauge screen. A 3" layer of shotcrete was then applied from floor to floor. The drift back and south wall were then bolted with specially designed 7m cables, installed 5 to a ring on 2.5 m spacing. The cables were double 7m cables with the first 5m plain strand, to act as a "shock absorber", and the bottom 2m having bulges to firmly anchor the cable. The cables were grouted, plated with 12" x 12" plates to increase the effectiveness of the cable, and then tensioned to 2,500 psi. Instrumented SMART cables were installed in the footwall opposite each cross cut. The cost of the rehab on 9450 totaled approximately $500,000.

On the 9415 level, rehab costing $250,000 was be necessary to reopen the area from 17 x/c to 19 x/c but the caved area beyond 19 x/c was abandoned. The damage in this area was considered too extensive to rehab as the drift back had failed 4 to 5m high at the cross cut intersections. A ramp was driven from the 9370 level to the 9415 level to re-establish access to the west end of the level. The cost was in the order of $ 1.1 million for some 535 m of ramp and access drifting. Alternative mining methods were looked at to recover the ore in the area of 20 to 27 stope. The most cost effective option, judged to

have the best chance of success, was to develop a hanging wall drift through 27 x/c, drifting eastward to access the existing stope crosscuts as required. The cost of rehabbing 27 x/c, developing and supporting this hanging wall drift was approximately $1,300,000.

The 9390 level received little damage during the rockburst; however, as the upper levels had already experienced extensive damage to the rockmass it was judged to be the area of greatest risk should another rockburst occur. A support system was designed to withstand a seismic event of similar magnitude to the 3.0 Nuttli event, which caused the extensive damage to the levels above. Minor debagging and scaling was done and the level was then rebolted, screened with 6 gauge mesh and a 3" layer of shotcrete was applied. Rings of the same "soft" support cable bolts used on 9450 were installed on a 3.0 m spacing from 17 x/c to 28 x/c. Both walls, as well as the back, were cabled and the back was instrumented with SMART cables at the cross cut intersections. In addition, a yielding support of rings of 6m super swellex was installed between the rings of cable bolts. The estimated cost of the rehab and support on this level, when completed, was ~ $600,000.

While the capacity and ductility of the support system was believed adequate at the time of installation it was recognized that due to the high stress nature of the sill area, combined with continued stress transfer from surrounding active mining areas, the support system capacity could gradually be consumed which would adversely impact the factor of safety over time. A comprehensive monitoring program was established in order to evaluate when and if additional support would be required in order to maintain an adequate safety standard in this high risk area. The monitoring system consisted of three components:

(i) SMART cables were installed as part of the cable design in each intersection on the 9450, 9390 and parts of the 9370 and 9415 levels. Occasional SMART MPBX instruments were installed to depths exceeding the depth of the cable support on each level to check for possible deformations beyond the maximum depth of ground support installed.

(ii) A temporary 8 channel portable microseismic system was installed in the Block 4 sill area in the summer of 1999. This was replaced with a 64 channel mine wide microseismic monitoring system in the fall of 2000. This system has its highest event resolution in the sill area.

(iii) Regular visual inspections of ground conditions are conducted through the sill area.

A second major 2.6 Mn seismic event occurred in this area on December 17th , 1999 [Figure 8b15]. The ground support design withstood this seismic event very well; no direct damage was observed to the shotcrete support. Direct rehab costs due to this event were approximately $200K [versus in excess of $4M from the first Mn 3.0 event]. In this case it was only necessary to replace local cable bolt capacity consumed by the 2.6 Mn rockburst. The workforce and management have subsequently gained confidence in the support design and in the post-burst support capacity.

Cost savings from rehabilitation following the 2.6 Mn event were estimated at ~ $1 million [Reference: Brian LeBlanc, Underground Superintendent, personal

communication]. Instrumentation cost [exclusive of microseismic system costs] for this program was in the order of $50,000. Additional large seismic events have occurred in this area requiring upgrading of the instrumentation and additional 'surgical' rehabilitation [Bawden et al, 2002].**8b4: DISCUSSION AND CONCLUSIONS**
Underground instrumentation programs can provide exceptional economic rates of return to mine operators, as illustrated in the case studies discussed in this paper. In late stage, high extraction and deep, high stress mining environments, ongoing production can often be sustained only when an acceptable risk assessment can be provided. Complex instrumentation systems combining conventional and microseismic instrumentation are often the controlling factor permitting continued production under these circumstances. Significant economic advantage can also be obtained through the judicious use of instrumentation in early mine life. The key to realizing these high ROI's rests in high quality engineering of the instrumentation program design followed by very high quality control in its execution and in interpretation of results.

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