

## In Situ Measurements in a Paste Backfill: Backfill and Rock Mass Response in the Context of Rockburst

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**ABSTRACT:** Rockburst can be a major problem in deep underground mines and has the potential for causing injury to mine operators as well as damage to underground workings. An extensive investigation was conducted to study in situ behaviour of mine backfill, in response to mining induced stresses with special reference to the phenomenon of rockburst. The field program included various in situ instrumentations, laboratory testing, and a detailed geomechanical study. Within the time frame of this study a rockburst event occurred within the mine that was monitored for this investigation. When the field observations are analysed a definite stress trends is detected prior to this event. The potential of in situ instrumentation will be presented and the role of die paste backfill will be discussed in relation to the case described, from die point of view of ground support.

### 1 INTRODUCTION

From die standpoint of operations, backfilling of mined out openings serves multiple functions such as working surface, roof, secondary pillar or ground support by maintaining the stability of the openings. It also impacts on the overall stability of the mine by limiting convergence interacting with die rock and can also reduce the impact of rockburst.

The backfill/rock interaction is complex. In ground control we will be concerned with the rock mass behavior around openings (convergence, induced stresses, seismicity) but also on the loads supported by the backfill and its stress-strain response to the rock mass deformations. To achieve an understanding of this complex interaction a good knowledge of both rock and backfill behavior is required (Gürtunca et al, 1989, Gürtunca et Adams, 1990, Hassani et al., 1998).

In order to better understand the role of mine backfill a monitoring of both the backfill and rock mass is required. Obtaining field data on both will allow a quantitative appreciation of the backfill role as a ground support element. To this end, more specifically in a mining operation having rockburst incidents, it will be shown how die use of in situ instruments can contribute to better understand the backfill impact on the rock response to mining. A brief review will be given on the possibilities of in situ instrumentation in backfills and the various recent experiments published so far. For the case

study presented here from mine Chimo, it will be shown how in situ instrumentation could help forecast a rockburst event by monitoring stresses and strains. The combination of field measurements with geomechanical and geophysical characterisation of the mine site gives the possibility of comparing numerical analysis results with actual field performance. This in turn gives the possibility to validate die numerical models and dispose of a valuable tool to understand the rock mass response to mining and predict eventually how mining operations will impact on the ground stability.

Numerical software and powerful computers available today allow the engineers to make very complex analysis. These tools can be very useful to perform back-analysis or be used as predictive tools to foresee ground response to mining operations. But to get meaningful results from these numerical tools good field data are essential. This paper presents a review of the means used in a mine to obtain in situ data required to understand the in situ response of a paste backfill to the rock mass in the context of rockburst and to better understand its role and impact as a ground support element.

### 2 BACKFILL INSTRUMENTATION

There is limited documentation about backfill instrumentation in underground mines. A research program was started by the US Bureau of Mines to

monitor the in situ backfill behaviour (McNay and Corson, 1961). The in situ monitoring work included recording the pressure build-up in the backfill and measurement of vein wall deformation in the mines of the Coeur d'Alene district in Idaho, USA. This research was carried out to assess the backfill pressure caused by deformation of the stope's sidewalls. The stability of stopes was made mainly by the intact rock pillars left between mined-out stopes as well as ore body crown pillars. Having an idea of backfill performance in mining stability would mean a reduction in the pillar size.

They reported that backfill pressure was measured directly by hydraulic pressure cells which were placed in the backfill (McNay and Corson, 1961). Laboratory tests were conducted to interpret in situ data at maximum and minimum backfill densities. The working capacity of the pressure cells were in the range of 3 MPa to 6 MPa. The recorded data was reliable in the range of 2.5 MPa to 4 MPa. The rock pressure changes in areas around the backfilled stopes was measured using borehole pressure cells. The deformation of stope wall was measured between rockbolts installed in opposite walls. The closure was recorded using two techniques : a) a dial extensometer, b) a measuring stainless steel wire attached to rockbolts; total stope wall closure was measured by a hydraulic cylinder which displaced fluid into a container.

Corson and Wayment (1967) reported a monitoring program in an area 60m x 40m in a near vertical vein at the Star Mine in Idaho, USA. The stress versus displacement measurement in a backfill stope of 3 meters wide at a depth of 2000 m were described. The data was collected using 22 total pressure cells and six closure measurement instruments. Stresses in the backfill initially build up hydrostatically but men pressures across the vein increased more rapidly than in the plane of the ore. The maximum pressure measured across the vein was 3.5 MPa, while the pressure in the plane of the vein in the horizontal and vertical directions only reached 1 MPa, then the cells broke down rendering further measurements impossible. The amount of closure recorded in the stope was about 10 cm.

A monitoring of backfill stresses and strains at the Lucky Friday silver Mine at Mullen, Idaho, USA, was recorded by McNay and Corson (1961). Five backfilled areas were instrumented at a depth of 1300 m at this level. Very little increase in backfill pressure was observed at the beginning due to a production delay because of a miners strike. However, when mining recommenced, rockbursts were experienced and the pressure in the backfill increased. The rate of strain increased following the rockbursts. A maximum pressure of 5 MPa in the backfill was reported.

A field evaluation of backfill compaction at the Luck Friday Mine, Idaho, USA was also reported by Corson (1971). This monitoring was conducted in two backfilled stopes separated by 30 m. on the same elevation at a depth of 1000 m. Both sites were filled with cemented fills but at the ore site backfill was placed normally while the other was vibrated. Each stope was instrumented with six pressure cells and a mechanical closure device to measure the vein wall movements. The density and moisture of the backfill was measured at three elevations using an access pipe.

Thibodeau (1989) reported an extensive field instrumentation at Levack Mine in Canada. The instrumentation included 36 electronic total earth pressure cells, three electronic piezometers and seven convergence monitors. The mechanical behaviour of the fill under load was studied. It was found that arching and the elastic beam behaviour could not be applied since the principal stresses in the fill were much larger than the maximum fiber stresses calculated using arching equations. The principal stresses measured in the backfill were then attributed to pillar convergence.

The stresses measured at backfill site in deep levels South African gold mines are much higher than the results from other mines. The first known measurements of stresses and displacements in backfill in South African mines were reported by Gay et al (1986). They discussed the development of many instruments suggested by Greig et al. (1979) of COMRO (Chamber of Mine, Research Organization) for recording stresses and strains in harsh underground environment. Gay et al. (1986) reported backfill pressure in two ranges: a) backfill acting as a local support would have a pressure less than 10 MPa. b) backfill acting as a regional support with a pressure exceed up to 100 MPa. at 4-5 km depth.

Bruce and Klokow (1988) explained a backfill instrumentation program at West Driefontein Mine in South Africa. A closure meter was placed in backfill underground, some 13m. from the backfill edge. A Glötzl pressure cell was installed alongside the closure meter. The stress-strain curve from the field data and laboratory was compared. The stress generation at a particular load in underground was lower than the results from similar material tested in the laboratory.

Clark et al. (1988) reported the development of instruments for backfill by COMRO. These included stress meters to record stresses up to 100 MPa and closure meters to monitor up to 0.5 m of closure inside the backfill. They presented closures and stresses data in three orthogonal directions in classified tailing backfill. The highest vertical stress recorded in backfill was almost 30 MPa. They showed that the ratio of the average of the two

horizontal stresses to the vertical stress remained fairly constant above a vertical stress of about 3 MPa. They also showed a good agreement between the in situ measurements and a laboratory confined compression test on similar material.

A backfill monitoring program was carried out by Gürtunca et al. (1989) to investigate the behaviour of three types of backfills, comminution waste, classified tailings and dewatered tailing backfill. A good agreement between the in situ and laboratory data in confined compression was reported. They suggested that the lower the starting or placement porosity of any particular backfill, the suffer that backfill will be. The ratio of dip or strike stresses in backfill, to the stresses acting at right angles to the roof plane, was showed to be between 0.3 to 0.6.

Using the data available to them, Gürtunca et al. (1989) present simple three dimensional models for the closure and stress profiles across complete backfill ribs. The closure profiles shows the maximum closures occurring near the edge of the backfill with lesser closure occurring near the backfill center. The three dimensional stress profile model suggests that stresses in the backfill increase with increasing distance from the stope face. On any section parallel to the stope face, the stresses build up slowly from the edge of the backfill and reach a plateau of maximum stress in a central section of the backfill rib. This is in contrast to the stress profile across a rock pillar which has the highest stresses at or close to the edge of the pillar, with the stresses reaching a plateau of lowest values near the pillar center (Hoek and Brown, 1986).

A backfill instrumentation result including the stresses in three orthogonal directions as well as closure at two sites was reported by Clark (1989). He showed that the backfill appeared to go through phases of stiffness related to the ratio of the horizontal to vertical orthogonal stresses. It was suggested that the backfill experienced failure at certain times when the deviatoric stresses ( $\sigma_1 - \sigma_3$ ) dropped. The aim of work conducted by Clark (1989) was to obtain a constitutive model of backfill behavior for use in numerical modelling.

Squelch (1990) explained a field measurement in a classified tailings backfill at 2030 m. underground mine with a dip angle of 10° for reef. Three triaxial stressmeters with mechanical closure meter were installed in a backfilled stope with an average width of 1.22 m. A closure-ride station was also placed outside the backfill for comparison with that inside the backfill. The maximum recorded pressure and closure was 4.5 MPa and 130  $\mu\text{m}$  respectively.

Gürtunca and Adams (1991) reported a field instrumentation in classified and dewatered tailing backfills at West Driefontein gold mine. They proposed an explanation of the effect of backfill in reducing the closure of stopes. They also stated that

the behaviour of backfill is determined to a large degree by whether the material is partially confined or fully confined. The field results presented by Gürtunca and Adams (1991) showed that at constant strain curves, the highest stresses are developed at or near the center of backfill rib, and stresses decreased towards the edge of the backfill rib. They also showed that the maximum closure recorded at the edge of the backfill rib with the minimum closure at the center.

A backfill instrumentation result by laboratory and numerical modelling result was reported by Adams et al. (1991). He presented the performance of the three dimensional behaviour of three different types of backfill; a) de-watered tailings, b) full plant classified tailings and c) comminution waste. This included measurement of stresses and displacement inside backfill in three directions. These results show that the highest stresses are expected near but not at the edge of the backfill rib. The stresses drop slightly toward the center of the rib and fall significantly towards the edge of the rib. He also showed that stress cells may be inclined up to 25° to the principal stress plane and measure a decrease in stress from the principal stress of only 5%.

### 3 ROCKBURST

A seismic event is a transient earth motion caused by a sudden release of potential or stored strain energy in the rock. Rockburst is a seismic event which can cause injury to mine operators as well as damage to underground workings. The general and essential feature of rockburst is its sudden and violent nature. It is sudden, failure occurs within a very short period of time, and violent in the sense that it can result in significant damage around mine openings. The various types of rockburst have been classified as follows:

a) *Strain bursts*: These are caused by high stress concentrations at the edge of mine openings.

b) *Pillar bursts*: These are usually associated with the sudden failure of complete pillars when they become relatively small and/or overloaded.

c) *Crush bursts*. These are commonly observed in room-and-pillar mining where sudden, multiple pillar failure occurs.

d) *Fault-slip bursts*: These take the form of sudden slippage along a geological plane of weakness.

The above rockburst types can generally be grouped into two categories:

i) *Inherent bursts* (replaced strain bursts) : occurs where the pre-mining stresses are high enough to cause failure when the initial development openings are driven.

ii) *Induced bursts* (replaced pillar bursts and crush bursts): are caused by mining operations transferring and concentrating stress on the remaining structures such as pillars.

Several techniques are currently available for the estimating of rockburst potential of underground excavations. From the review conducted, it appears that the "Stress" and "Strain Energy" approaches tend themselves easily to numerical modelling. Both methods were used in this project as the main analytical tools for the evaluation of rockburst potential.

### 3.1 *Effect of Backfill on Rockburst Alleviation*

There is very little information available on the effect of backfill on either the number of rockbursts or the damage caused. The use of backfill for reducing rock wall closure and hence the incidence of rockbursts in the Coeur D'Alène mining district of the north-western United States of America is reported in papers by Pariseau and Kealy (1972), Board and Voegelé (1983) and Corson et al. (1983). Comments are made by these authors about the benefits of using backfill and that higher density backfill improved the degree of control provided. However, no quantitative results, which could assist towards understanding backfill impact on rockburst, were documented.

Dhar et al. (1983) reported on a mine in the Kolar Gold Fields of India, with depths exceeding three kilometers, where a modified cut-and-fill method was developed and reduced the occurrence of rockbursts. Quesnel et al. (1989) reported on the use of a stiff consolidated rock fill for the alleviation of rockbursts at a mine in Ontario, Canada had exceeded all expectations. They stated that the use of the rock fill appeared to have achieved significant reduction of the frequency of rockbursts. The use of semi-stiff (uncemented rock fill) backfill resulted in minimizing the damage caused by rockbursts but appeared to have little effect on the frequency of rockbursts.

Close and Klokow (1986) reported that a mined out panel which had been backfilled for two years was less severely damaged than adjacent unfilled panels after a major rockburst. Bruce and Klokow (1986) reported a reduction in rockburst damage due to backfilling but indicated that it was not possible to identify a reduction in the number of seismic events or radiated energy because of the limited extent of backfilling which had taken place at that time. Gay et al. (1986) reported on visits made to inspect rockburst damage at a number of mines using backfill. In all cases, they reported that the use of backfill had been beneficial in that damage at the slope face had been restricted compared to unfilled areas, and gullies had remained open.

Recorded information on the effect of backfill on the number and magnitude of rockbursts is limited. In a paper by Gay et al. (1986) seismicity is compared for a backfilled and an unfilled area. The data suggested that, although seismic events occurred more frequently in the backfilled areas than in the conventionally supported area, the magnitude of the events in the backfilled area were smaller and generally more uniform.

When Cook et al. (1966) first proposed a method for evaluating the rockburst potential of a mining layout, they used it to quantify the effect of waste-filling and partial extraction (using regularly spaced stabilizing pillars) on the rockburst hazard. No direct comparisons between the two regional support systems were made by the authors but they concluded that the rockburst hazard can be reduced very effectively by partial extraction. The authors added that, on the basis of the criterion used, waste-filling is capable of reducing the hazard by an estimated 50 per cent in deep mines. However, it will only influence the hazard in shallower mines if the span of the excavation is exceptionally large. In addition, they commented that the waste-filling should be carried out continuously if the maximum benefits, in terms of the rockburst hazard, were to be achieved.

DeJongh (1986) conducted a comparison of stabilizing pillars and backfill for regional support. The study stated that, for the mining layout modelled, backfill could achieve approximately the same reduction in the rockburst hazard as stabilizing pillars covering 15 percent of the area at an average depth of approximately 3300 m. A criterion known as energy release rate was used by all the above authors to estimate the level of the rockburst hazard from their calculations. Although these studies provide an indication of the possibility of replacing stabilizing pillars with backfill as a means of regional support, none of them have determined the influence of backfill quality or depth below surface on stabilizing pillar requirements, both of which are important parameters in establishing the efficiency of backfilling.

Undoubtedly, the most comprehensive analysis of backfill on the rockburst hazard was carried out by Ryder and Wagner (1978). First, they developed a more accurate and direct method of modelling the effect of backfill on energy release rates after establishing that the methods used previously underestimated the effects of backfill considerably. They then conducted numerous simulations of mining layouts, primarily to establish the effect of backfilling parameters on energy release rates, and hence the rockburst hazard. They concluded that high quality backfill can provide up to a six-fold reduction in the energy release rate, compared to a layout with no backfill, and that the distance at

which backfill is placed from the face is a critical parameter in achieving energy release rate reductions. In a report written by Lloyd (1978), mention is made that backfill should be placed within 5 metres of the face and that high quality backfill can reduce the energy release rate from 80 MJ/m<sup>2</sup> to 20 MJ/m<sup>2</sup> at a depth of 4 kilometres. Lloyd suggested that the rockburst hazard would be reduced from 0.7 to 0.1 seismic events per 1000 square metres mined. Adams et al. (1989) evaluated the use of concrete pillars as a possible replacement for stabilizing pillars in deep mines. They concluded that the substitution is possible, from a rock mechanics point of view, with energy release rate and average pillar stress values being similar at a depth below surface of 3800 metres.

Pariseau and Kealy (1972) modelled the effect of backfill on the closure in a deep isolated vertical stope. Their results showed that relatively low quality backfill can reduce stope closure by approximately 50 per cent. When high quality backfill was used in the model, the stope closure was only one-third of that obtained with a lower quality backfill and one-fifth of the closure for the same stope with no backfill. A reduction in stope closure has been related to the number and magnitude of rockbursts by Cook et al. (1966). From this relationship, it can be assumed that the reduction in closure reported by Pariseau and Kealy will result in a reduction in the rockburst hazard. Whyatt et al. (1989) determined that, for the mining layout modelled, backfill reduced energy release rates by 42 percent. They also established that a one per cent increase in the density of their backfill reduces the energy release rate by approximately one per cent and that an additional 28 percent reduction in energy release rate can be achieved by using a high quality backfill. They reported that the addition of cement in small quantities does not appear to affect energy release rates.

The following can be concluded from the literature review of the effect of backfill in rockburst alleviation: First, it appears that backfill has the potential to provide important regional support benefits in addition to substantial local support benefits which have already been recorded. Second, despite many decades of backfilling, there are no quantitative results from underground measurements which conclusively demonstrate the effect of backfill and backfilling parameters on the number and magnitude of rockbursts. The quantitative results which have emerged have been derived from the application of analytical solutions and numerical models. Third, the theoretical investigations which have been conducted to date have all used the concept of energy release rate as a mean of relating the analytical results to the rockburst hazard.

#### 4 CASE STUDY OF MINE CHIMO

Chimo Mine is located 50 km east of Val D'Or, near Louvicourt, Quebec. Native gold is found associated to quartz and sulfides (arsenopyrite and pyrrhotite). The deposit is part of a sequence of massive volcanic rocks, surrounded in the north by a sequence of metasedimentary and volcanic rocks, in the south by sedimentary rocks, grauwakes and pelitic schists.

About six ore bearing zones have been identified in Chimo Mine. These ore zones are generally striking east-west and dip steeply (around 70°) to the north. The ore body between levels 16 and 17 included zones B, F and A with 4 meters wide and an average of 90 m long. Mining method is open stoping longhole with 70 m high of stopes. Hanging wall and foot wall are generally strong except in some situations where they contain a graphite bearing schist material. Blasting holes are 4 inches diameter.

##### 4.1 Field instrumentation

The instrumentation at Chimo Mine consisted of rock mass instrumentation and backfill instrumentation. In the rock mass extensometers and vibrating wire Stressmeters were used to measure the displacements and change of stresses respectively.

In the backfill pressure cells were installed in the stopes prior to backfilling to measure the actual pressure within backfill. In order to choose the instrumentation location in the mine, a preliminary numerical modelling was conducted to study the displacement and mining induced stresses in the rock mass and backfill during the mine-and-fill sequence. A 3-dimensional finite elements program was used to create 10 models presenting the mine-and-fill sequence of the mine. The results of numerical modelling were used to determine the stopes where significant build up of backfill pressure could be expected. The mine-and-fill sequence were also studied to determine when instruments had to be installed in order to respect the mining schedule.

A total of 6 hydraulic pressure cells were installed successfully in two backfilled stopes as follows:

- Three pressure cells were installed in 20 meters depth at middle of the stope 17-5B-5 in level 17.
- Three pressure cells were installed in 20 meters depth at 1/4 width from the east side wall at the stope 18-5F-8 in level 18.

A special procedure was used to install the pressure cells from the top of stope. This was the first experience for installation of pressure cells from the top of an open stope in a Canadian underground mine. The procedure was as follows:

- A cubic frame of elbow steel with dimension of (2" x 2" x 2") w/s prepared.
- 3 hydraulic pressure cells were fixed in x, y and z planes of the cubic frame.
- A cable was passed by the top of the stope using bow, arrow and fishing line. The arrow was shot to an access on the other side of the stope from the top access. The fishing lines was then used to pass cables from the top and other side of the stope. These cables were then fixed at both sides of the stope.

- The cubic frame together with 3 fixed pressure cells was lowered to the 20 meters depth from the top of the stope using cables and pulleys. Reflectors positioned on the corners of the box were used to target the box and align it properly with surveying equipment once the target level in the stope was reached

Backfilling of the stope started after hanging the frame inside the stope. The pressure cells in North-South (across ore), East-West (along ore) and vertical directions were monitored and data registered once they were covered with backfill. The installation procedure of the pressure cells with frames proved to be very successful.

#### 4.2 Pastefill Performance

Field data from the pressure cells in two stopes were recorded over 200 days and excellent results were obtained. These data showed that a significant amount of energy was absorbed by paste backfill materials. The energy absorbed by backfill during the mining operation as well as rockburst activity was related to the geological condition of surrounding rock mass, mining activities around the backfilled stope and Young's modulus of the backfill. A performance analysis of these data is presented in below.

#### 4.3 Backfilled Stope 17-5B-5

Three pressure cells were installed in the center of the backfilled stope 17-5B-5 at a depth of 20 meters, on September 13 1995. Backfill operation started on October 20 1995 and finished on November 18 1995 (7 meters of backfill on top was left for later operation). Figure 1 presents the recorded backfill pressure in 3 directions versus the number of days since installation.

#### 4.4 Pressure build up in backfill

When backfill operation was finished, the recorded backfill pressures were as follows:

$$\begin{aligned} \sigma_{(N-S)} \text{ (across ore body)} &= 245 \text{ kPa} \\ \sigma_v \text{ (vertical)} &= 180 \text{ kPa} \end{aligned}$$

$$\sigma_{(E-W)} \text{ (along ore body)} = 149 \text{ kPa}$$

In last day of the operation, backfill was in a liquid situation with hydraulic pressure. Considering the 7 meters left on top, the vertical pressure on top of the pressure cell should be:

$$\begin{aligned} h &= 20 - 7 = 13 \text{ m.} \\ \sigma_v &= \gamma * h = 0.0145 \text{ MN/m}^3 * 13 \text{ m.} \\ &= 0.189 \text{ MPa} = 189 \text{ KPa} \end{aligned}$$

This is almost the same value that was collected from the field (180 kPa). In general, after termination of the backfill operation in stope 17-5B-5, vertical pressure in the backfill was equal to gravity loading.

The ratio of the build up pressure in different directions inside the backfill was compared with in situ stresses and presented in Table 1. This comparison shows that whereas the ratio of  $\sigma_{(N-S)}/\sigma_{(E-W)}$  for backfill pressure was in close agreement with in situ stresses, the ratio of  $\sigma_{(N-S)}/\sigma_v$  was different. This comparison tends to indicate that the ratio of build up pressure in the horizontal section (or plane) is related to the ratio of the in situ stress in the horizontal section. Because, if only gravity loading was involved, the stress ratio in the horizontal plane should have been 1.

Table 1 Stress ratio rock/backfill (after termination of the backfill operation).

	$\sigma_{(N-S)} / \sigma_{(E-W)}$	$\sigma_{(N-S)} / \sigma_v$
Backfill	1.64	1.36
In situ Stress	1.52	2

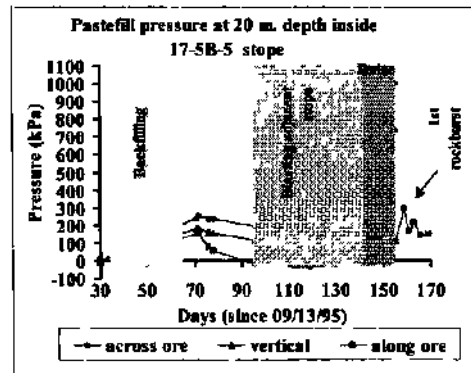


Figure 1 Pressure inside the backfilled stope 17-5B-5 since the day of installation.

#### 4.5 Curing Period of Backfill

During the curing period, the temperature inside the backfill increased. The backfill temperature at the beginning of the backfill operation was 18° C. During the next 60 days, backfill temperature increased to the 38° C and then started to decrease very smoothly.

During the curing period (from the termination of the backfill operation and over the next 40 days), the backfill pressure decreased in 3 directions. Decreasing of the backfill pressure in N-S and vertical directions was very smooth, keeping a ratio between 1.3 to 1.7. But, the backfill pressure in E-W direction dropped significantly to a nominal zero pressure. This was due to the opening of the adjacent stope (17-5B-3) on December 16 1995.

#### 4.6 Opening the Adjacent Stope

During the mining advance of the adjacent stope 17-5B-3, the backfill pressure in N-S and vertical directions increased, but in E-W the pressure remain close to zero. After blasting the last layer of the ore body in stope 17-5B-3, backfill pressure increased by 400 and 275 kPa in N-S and E-W directions respectively. The pressure cell in the vertical direction was damaged. It seems that the backfill pressure in vertical direction continued to keep the same ratio as the N-S direction. The backfill pressures on day 155 (before rockburst) in 3 directions were as follows:

$$\begin{aligned} \sigma_{(N-S)} \text{ (across ore body)} &= 1.0 \text{ MPa} \\ \sigma_v \text{ (vertical)} &= 0.7 \text{ MPa} \\ \sigma_{(E-W)} \text{ (along ore body)} &= 0.1 \text{ MPa} \end{aligned}$$

During the 20 days (since day 140 to 159 in Figure 1) preceding the rockburst activities, the backfill pressure in N-S direction increased from 500 to 1000 kPa. This might be used as an indicator for the potential of rockburst. The backfill pressure increased by a rate of 32.5 kPa/day.

#### 4.7 Backfill Pressure vs Stresses in Hanging Wall

The backfill pressure can be used as an indicator to check the movement of the surrounding rock mass toward the backfill. Figure 2 presents a vertical section of the backfilled stope 17-5B-5. The hanging wall of the stope was instrumented using one extensometer and 2 VBS to measure the displacement and change of the stresses respectively. Figure 2 shows that the extensometer 17-3 intersected with a thick graphite zone. This graphite zone had a major effect on stability of the hanging wall.

Field data recorded from extensometer 17-3 is presented in Figure 3. Anchors 1 and 2 were located above the graphite zone and anchors 3, 4 and 5 between graphite zone and stope wall. Figure 3 shows that anchors 3, 4 and 5 had moved up to 4 times more than anchors 1 and 2 prior to the rockburst. This was due to the fault separation that caused a significant pressure increase on backfill.

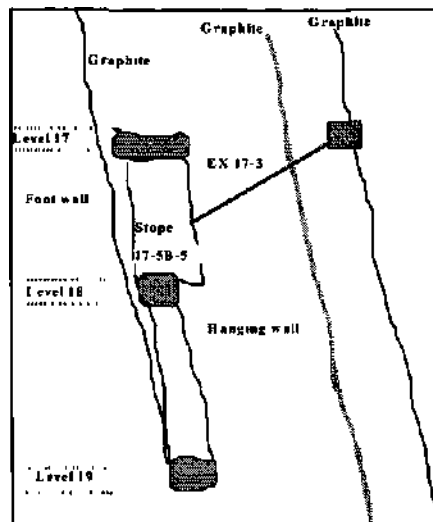


Figure 2 Vertical section of the backfilled stope 17-5B-5

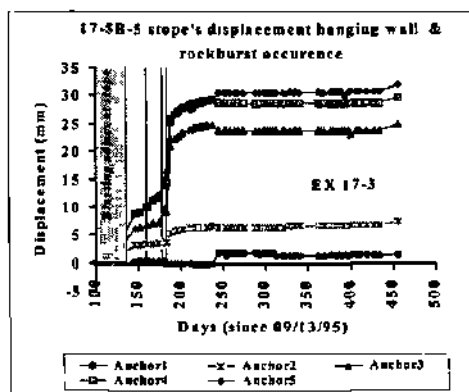


Figure 3 Displacement at the hanging wall of the stope 17-5B-5.

The maximum measured displacement of the hanging wall of backfilled stope 17-5B-5 was 32 cm. The hanging wall of the adjacent backfilled stope (17-5B-6) had a 8 cm displacement. At the hanging wall of the other adjacent open stope (17-5B-3), closest anchor to the opening was lost, this would

indicate a displacement of more than 1 meter. A comparison among the displacement of these three stopes showed that the backfill had prevented a major collapse of the hanging wall.

Figure 4 shows a comparison between displacement of hanging wall in N-S direction and build up pressure in backfill since pressure cell installation. The backfill pressure grew due to the displacement of the hanging wall. Figure 5 shows a comparison between the change of vertical stress in hanging wall and vertical backfill pressure. It can be observed that the reduction of the vertical stress in the hanging wall was parallel to the increasing vertical stress in the backfill. This indicates that, through convergence, the backfill was being solchited increasingly.

Figure 5 shows a comparison between the change of horizontal stress in the hanging wall and the horizontal pressures measured in the backfill. It can be observed that changes in the backfill stresses were synchronized with stress changes in the surrounding rock mass.

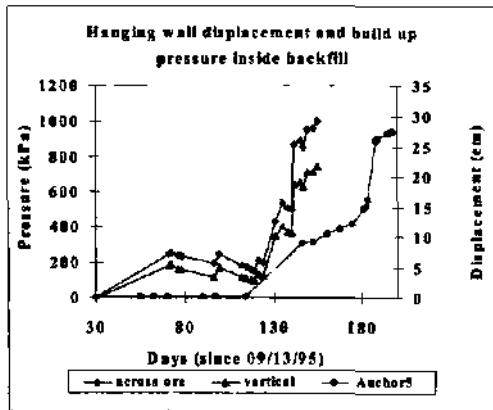


Figure 4. Comparison between displacement of hanging wall in N-S direction and build up backfill pressure.

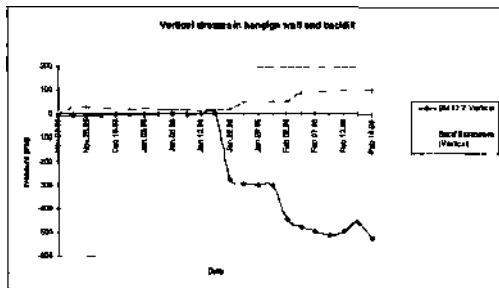


Figure 5. Comparison between change of vertical stress in hanging wall and backfill pressure

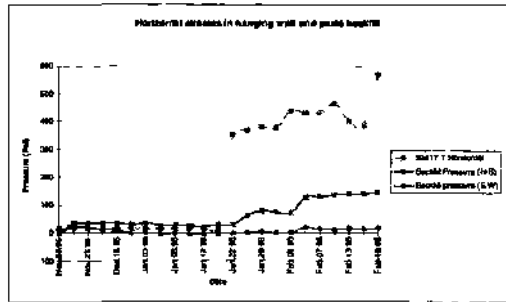


Figure 6. Comparison between change of horizontal stress in hanging wall and build up backfill pressure

A comparison between Figures 5 and 6 shows that in general, the vertical stress in the hanging wall was decreasing as follows; a) a rotation of stresses inducing higher horizontal stresses within the hanging wall, b) a certain amount of stress transfer from the rock to the backfill. It is apparent that there was intense stresses redistributions in the rock around these stopes and from the rock to the backfill.

#### 4.8 Stress Calculation Inside the Backfill

The stress in N-S direction inside the backfill can be calculated as follows:

$$\sigma = E * \epsilon$$

where E is the Young's Modulus of the backfill (E nominal = 22 MPa) and  $\epsilon$  is the strain of the backfill due to the displacement of hanging wall in N-S direction. Assuming that the major part of backfill strain was related to the hanging wall displacement then, considering the displacement of the hanging wall prior to the rockburst 10 cm and the width (vertical distance between two side walls) of the stope 4 meters:

$$dX = 10 \text{ cm}$$

$$\epsilon = dX / L = 10 \text{ cm} / 400 \text{ cm} = 0.025$$

$$\sigma = 22 \text{ MPa} * 0.025 = 550 \text{ kPa}$$

This value is within the range of data collected in the field in N-S direction right before the rockburst.

These results explain that the pressure applied inside the backfilled stope 17-5B-5 was related to both the displacement of the hanging wall and the Young's Modulus of the backfill. The fault movement towards the backfill causes higher backfill strain and pressure.



#### 4.9 In Situ Paste Backfill Behavior (Stress Versus Strain)

The paste backfill inside the stope with a size of 4m.x 15m.x70m. can be assumed as a large specimen that is tested by a testing machine in laboratory. The major source of the force was the displacement and bending of the hanging wall due to the fault separation. The value of strain in the middle of the backfill was obtained by dividing the displacement of the anchor 5 by the stope wide:

$$\epsilon = dX/L = dX_{\text{anchor 5}} / 400 \text{ cm}$$

Considering that the angle between the extensometer and stope wall is 70° then, the value of the stress vertical to the stope side (Force applies in the direction of extensometer) was calculated as follows:

$$\sigma = \sigma_{(N-S)} \times \cos 70^\circ + \sigma_{(V)} \times \sin 70^\circ$$

A graph of the backfill stress in direction of extensometer versus displacement of anchor 5 was prepared and presented in Figure 7.

#### 4.10 Strain Energy Density

Having the stress and strain inside the backfill, the strain energy density was calculated as follows:

$$\text{Strain Energy Density} = 1/2 \times E \times a$$

Figure 8 presents the strain energy in backfilled stope 17-5B-5 in a period of one month before starting the rockburst activities. This graph shows that the strain energy density increased significantly prior to the rockburst activity. The rate of absorption of the strain energy into the backfill might be used as an indicator to study the potential of rockburst. This method needs further investigation in field sites.

#### 4.11 Energy Absorbed by Pastefill

The field data of pressure cells show that backfill absorbed a large quantity of the energy from the surrounding rock mass. This energy transfers to the backfill through the work done by displacement and build up pressure.

$$\text{Work} = \text{Displacement} \times \text{Force} \\ (W = D \times F)$$

$$\text{Force} = \text{Stress} \times \text{Area} \\ (F = \sigma \times S)$$

To calculate the energy absorbed by pastefill in stope 17-5B-5, with 70 m. height and 15 m. width:

$$S = 15 \times 70 = 1,050 \text{ m}^2$$

The recorded backfill pressure in N-S direction before the rockburst was 1 MPa, therefore:

$$\text{Force} = 1 \text{ MPa} \times 1,050 \text{ m}^2 = 10^6 \times 1.05$$

$$\text{Pascal} \times \text{m}^2 = 10^6 \times 1.05 \text{ Newton}$$

$$\text{Force} = 10^6 \times 1.05 \text{ Kgf}$$

Considering the 10 cm displacement in hanging wall at the end of rockburst period:

$$\text{Work} = (10^6 \times 1.05 \text{ Kgf}) \times (10 \times 10^{-2}) \\ = 32.5 \times 10^3 \text{ Joule} = 105 \text{ kJ}$$

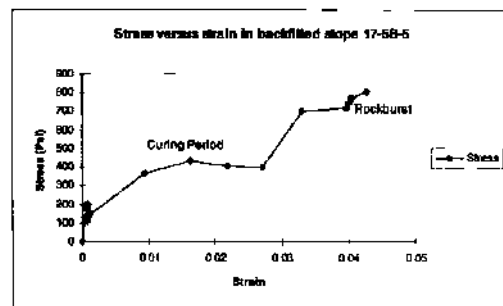


Figure 7 Stress versus strain at backfilled stope 17-5B-5

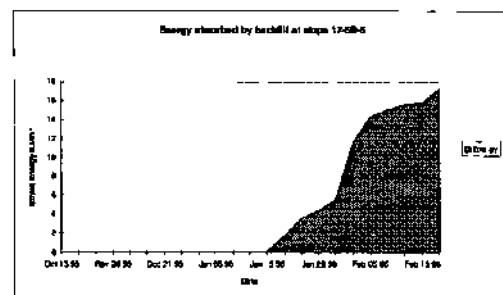


Figure 8 Energy absorbed by backfill in a period of one month prior to rockburst

## 5 CONCLUSION

The performance of backfill in the alleviation of rockburst can be concluded as follows:

The build up vertical pressure after termination of backfill operation was equal to hydraulic pressure in the case of no mining activity around the stope. In the case of mining activities around the backfilled stope, the vertical backfill pressure was 3 times more than hydraulic pressure.

The ratio of build up backfill pressure in different direction was related to the in situ stress in the case of no mining activity around the stope. In the case of mining activities around the stope, the convergence had a major role in the magnitude of the backfill pressure in each direction.

During the curing period, the backfill pressure and temperature decreased in the case of no mining activity around the stope. In the case of mining activities around the stope, the temperature still dropped but the backfill pressure increased and the some stress transfer took place from the rock to the backfill.

Significant stress redistributions took place due to the mining activities and stress transfer occurred from the rock toward the backfill.

The pressure inside the backfill was related to both the displacement of the side walls of the stope and the Young's Modulus of the backfill. The fault movement in side walls of the stope caused higher backfill strain and pressure.

The strain energy increased significantly in a period of one month before the start of rockburst activities. The rate that strain energy was absorbed into the backfill could be used as an indicator to study the potential for rockburst. This method needs further investigation in field sites. The rate of stress increase in the backfill might be used as an indicator to predict the potential of rockburst.

Backfill protected the stope walls from sudden movement. During the rockburst period, the hanging wall displacement for two backfilled stopes were 8 and 20 cm. At the same period, anchor 1 (the closest anchor to the stope) in the adjacent open stope was lost (indicating a potential displacement of more than 1 meter). The presence of backfill inside the stope prevented these large movements.

Backfill absorbed energy through the compressive strain. The movement of the side walls compressing the backfill, increased the backfill strain as well as the pressure inside the backfill. This deformation work done on the backfill by the convergence reduced the concentration of energy in surrounding rock mass. By decreasing the

concentration of stress in surrounding rock mass, the rockburst intensity probably reduces.

The change of strain energy density of backfill, in a period of one month prior to the rockburst activity, was presented. This graph showed that the strain energy density increased significantly prior to the rockburst. The rate that strain energy was absorbed into the backfill might be used as an indicator to examine the potential of rockburst. This method needs further investigation in field sites.

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## REFERENCES

- Adams, D.J., Gürtunca, R.G., Jager, A.J and Gay, N.C., 1989. "Assessment of a new mine layout incorporating concrete pillars as regional support", *Proceedings of the 4th International Symposium on Mining with Backfill*, Montreal, October 1989. pp. 199-208.
- Adams, D.J., Gürtunca, R.G. and Squelech, A.P., 1991. "The 3-dimensional in situ behaviour of backfill materials". *Proceedings of the 7th International Congress on Rock Mechanics*, Witke(ed.), Balcema, Rotterdam.
- Board, M.P. and Voegele, M.D., 1983. "Examination and demonstration of undercut and fill sloping for ground control in deep vein mining", *Proceedings of the 3rd International Symposium on Mining with Backfill*, Lulea, June 1983. pp. 300-306.
- Brace, M.F.C. and KJokow, J.W., 1986. "A follow-up paper on the development of the West Driefontein tailings backfill project". *Proceedings of a Symposium on Backfill in South Africa Mines*, South African Institute of Mining and Metallurgy, Johannesburg, November 1986.
- Clark, I.H., 1989. "The strength and deformation behaviour of backfill in tabular deep-level mining excavations. PhD Thesis, University of the Witwatersrand.
- Clark, I.H., Gürtunca, G.R. and Piper, P.S., 1988. "Predicting and monitoring stress and deformation behaviour of backfill in deep-level mining excavations. *Proc.5<sup>th</sup> Aust New-Zealand Conference on Geomechanics*, pp 214-218.
- Close, A.J. and Klokow, J.W., 1986. "The development of the West Driefontein tailings backfill project", *Association of Mine Managers of South Africa, Papers and Discussions*, vol.3/85, November 1985.
- Cook, N.G.W., Hoek, E., Pretorius, J.P.G., Ortlepp, W.D. and Salamon, M.D.G., 1966. "Rock mechanics applied to the

- study of rockbursts". *Journal of the South African Institute of Mining and Metallurgy*, Johannesburg, vol.66, May 1966, pp.436-528.
- Corson, D.R., Dorman, K.R. and Sprute, R.H., 1983. "Improving the support characteristics of hydraulic fill". *Proceedings of the 3rd International Symposium on Mining with Backfill*, Lulea, June 1983, pp.93-99.
- Corson, D. R. and Wayment, 1967. "Load-displacement Measurements in a Backfilled slope of a Deep Vein Mine". Report of Investigation 7038. U.S. Bureau of Mines, Washington, D.C.
- Corson, D.R., 1971. "Field evaluation of hydraulic backfill compaction at the Lucky Friday Mine, Mullen, Idaho". Report of investigation 7546. U S Bureau of Mines, Washington, D. C.
- De Jongh, C.L., 1986. "The potential of backfill as a stope support in deep mines", *GOLD 100 - Proceedings of an International Conference on GOLD*, South African Institute of Mining and Metallurgy, vol.1, pp.271-297, Johannesburg, 1986.
- Dhar, B.B, Shankar, K.V. and Sastry, V.R., 1983. "Hydraulic filling - an effective way of ground control", *Proceedings of the 3rd International Symposium on Mining with Backfill*, Lulea, June 1983, pp.113-120.
- Gay, N.C, Jager, A.J. and Piper, P.S., 1986. "A quantitative evaluation of fill performance in South African gold mines", *Proceedings of a Symposium on Backfill in South African Mines*, South African Institute of Mining and Metallurgy, Randburg, June 1986.
- Greig, J. D., Lloyd, P. J. D., Quinn, M. and Blight, G. E., 1979. "Instrumentation to measure various parameters of fill in deep-level excavations and a preliminary assessment of the performance of a cemented slimes fill".
- Gürtunca, R.G. and Adams, D.I., 1991. "Results of a rock engineering monitoring programme at West Driefontein Gold Mine". *Journal of SAIMM*, Vol 91, No 12, December 1991.
- Gürtunca, R.G., Jager, A., Adams, D.J. and Gonlag, M., 1989. "The in situ behaviour of backfill materials and the surrounding rockmass in South African gold mines". *4th Int. Symp. on Innovations in Mining Backfill Technology*, Montreal, Canada.
- Gürtunca, R.G and Squelch, A.J., 1990. "Benefits of backfilling". Unpublished research report *Chamber of Mines of South Africa*. Johannesburg, November 1990.
- Hedley, D.G.F., 1992. "Rockburst handbook for Ontario hard rock mines", CANMET special report SP92-1E.
- Hoek, E. and Brown, E. 1986. "Underground excavations in rock". IMM pp 123.
- Lloyd, P.J.D., 1978. "The potential of integrated mining and extraction systems on the Witwatersrand", Unpublished research report. *Chamber of Mines of South Africa*, Johannesburg, 1978.
- McNay and Corson, 1961. "Hydraulic sandfill in deep metal mines". Bureau of mines, Spokane, Washington.
- Pariseau, W.G. and Kealy, C.D., 1972, "Support potential of hydraulic backfill", *Proceedings of the 14th Symposium on Rock Mechanics*, Pennsylvania, June 1972, pp.501-526.
- Ryder, J.A. and Wagner, H., 1978. "2D Analysis of backfill as a means of reducing energy release rates at depth". Unpublished research report, *Chamber of Mines of South Africa*, Johannesburg, 1978.
- Squelch, A. P., 1990. "Results of a monitoring program at vaal Reefs, 5 Shaft". COMRO Reference Report 24/90,1990.
- Thibodeau, D., 1989. "In situ determination of high density alluvial sand fill". *Innovations in mining backfill technology: 4th international symposium*, Montreal.
- Quesnel, W.J.F., De Ruiter, H. and Pervik, A., 1989. "The assessment of cemented rockfill for regional and local support in a rockburst environment", *Proceedings of the 4th International Symposium on Mining with Backfill*, Montreal, October 1989, pp.217-224.
- Whyatt, J.K., Board, M.P. and William T.J., 1989. "Examination of the support potential of cemented fills for rockburst control". *Proceedings of the 4th International Symposium on Mining with Backfill*, Montreal, October 1989, pp.217-224.

