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Evolution of Blasting Practices at the Ekati™ Diamond Mine

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ABSTRACT: A review of blast designs and improved blasting practices at the Ekati Diamond Mine is presented along with the results of three blast-monitoring experiments. Blast monitoring was undertaken to investigate blast damage mechanisms in the mine's well jointed rock mass. Rock mass damage caused by production, pre-shear and wall control blasts was measured. Ekati uses 270mm holes for production blasting, and 165mm holes loaded with a decoupled charge for pre-shearing. The production blasts are loaded with bulk emulsion / ANFO blends. The mine plan involves using 30m double benches. A 30m high pre-shear is drilled and blasted prior to the production blasting that İs done with sequential 15m benches. This paper summarizes the monitoring equipment and data gathered to date.

1 INTRODUCTION

The Ekati™ Diamond Mine is located about 300km northeast of Yellowknife, NWT, Canada. The mine is a joint venture of BHP Diamonds Inc. (51%), Dia Met Minerals Ltd. (29%) and geologists Charles Fipke and Stewart Blusson (10% each). The Ekati mine is Canada's first diamond mine and BHP is the operator. The mine is accessed by air and by a winter ice road.

The first kimberlite pipe mined at Ekati is known as the Panda pipe. The Panda open pit mine will have a total depth of 315m (Figure 1), width of about 600m, and a 50° overall slope. Initial ore production is 9,000 tonnes per day.

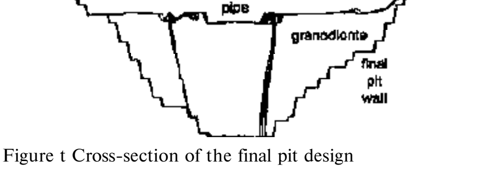

Figure t Cross-section of the final pit design

This paper summarizes the pre-strip blasting and the evolution of the production and wall control blast designs from 1997 to 2000. The initial blast designs and the rational behind subsequent changes that took place are presented along with the techniques currently m use.

The paper also briefly presents an experiment done to measure the rock mass response to blasting. Three blasts were instrumented with geophones, gas pressure sensors and time domain reflectometry cables. A brief summary of the instrumentation and data gathered is included.

2 BLAST DESIGNS

2.1 *Pre-strip Blasting*

During 1997 the Panda kimberlite pipe was prestripped to prepare for mining. The pit was initially stripped in 10m benches. Drilling was carried out with an Ingersoll-Rand DM-45 that drilled 165mm holes and an Ingersoll-Rand DM-M2 that drilled 270mm holes. The smaller unit was used for pioneering work due to the rough terrain encountered. The larger rig was used once level benches had been established. The blast patterns used were a 4m by 6m staggered pattern for 165mm holes and a 6m by 7m staggered pattern for the 270mm holes.

In the early stages of pre-stnpping it was found that the majority of the drilled holes contained water. This resulted in all of the holes being loaded with DYNOFLO Lite, a 70% emulsion / 30% Ammonium Nitrate and Fuel Oil (ANFO) chemicaUy gassed bulk explosive. The cup density of the product was 1.2g/cc. The product was manufactured on site at a temporary plant during the pre-stripping phase.

All of the blasts were tied in using detonating cord and millisecond connectors. This system was chosen because of the reliability of a dual path tie-in. The blast patterns were designed to have hole-by-hole initiation to ensure the optimum fragmentation and displacement in the muck pile. Typically the inter-hole delays were from 17 to 50 milliseconds with the inter-row delays 100 to 135 milliseconds. Blasts were shot in V-patteras or en-echelon depending on the shot geometry.

The blast results were generally very good. The blasts were mucked with smaller equipment and finer material was required for road construction so the blasted material was smaller then would be typically expected İn a large mine.

2.2 *Production Blasting*

In the summer of 1998, BHP commissioned two D90KS drill rigs equipped for drilling 311 mm holes. At the same time, the mine plan increased to 15m bench heights. Between the summer of 1998 and spring of 1999 several blast designs were tested before arriving at the current design.

The first blast design with 31 lmm holes was a 7.5m by 7.5m square partem. The holes were loaded with 750 kg per hole of 70% emulsion / 30% ANFO blend at a density of I.2g/cc. The sub-drill on the pattern was 1.5m giving a 16.5m total depth. This design resulted in 8m of stemming in each hole and no explosives in the upper half of the bench. The resulting muck piles were poorly fragmented and very tight in die upper half of the bench and the shovel faces were standing close to vertical, resulting in lower productivity and increased shovel wear. A fragmentation study done on this material showed that the material was 90% passing 0.5m (Peterson 1998).

Several methods were tried to improve the blast performance. These included:

- dividing the explosive load into two decks to improve explosive distribution,
- loading the holes with lower density explosives to improve explosive distribution,
- increasing the total load per hole,
- adjustments to timing sequence, and
- switching to a 7m by 8m staggered pattern.

There were varying degrees of success in each method tried. The use of two decks noticeably improved the digging conditions, however the blast crew productivity and the accessory costs suffered as a result. During the winter of 1998 several blasts were loaded with ANFO in the top portion of the holes over a toe load of 70/30 emulsion/ANFO blend. By loading the upper part of the hole with ANFO at a density of 0.83g/cc, the kilograms per metre of borehole was lowered by close to 45%. This improved the explosive distribution and reduced the stemming length and gave excellent results, however water conditions did not allow for this practice to continue.

The blasthole layout was also switched from the 7.5m square pattern to a 7m by 8m staggered pattern to gave a better distribution of explosive for the same amount of drilling.

Problems with drill productivity as well as the desire to improve fragmentation and diggability led the operation to try using 270mm holes in the production patterns. Several patterns were tried on a 6m by 7m staggered pattern. The good results and increased drilling productivity led the mine to using 270mm holes on all production shots. The pattern was also slightly expanded to a 6.5m by 7.5m equilateral pattern.

The holes are presently loaded with 70% emulsion / 30% ANFO chemically gassed to a density of 1.15g/cc. Each production hole is loaded with 775kg of explosive and stemmed with I0-20mm crushed rock. The holes are toe primed with a 454-gram pentolite cast booster on a 17m long 500 millisecond non-electnc detonator.

The blast initiation sequence and timing was also adjusted before the current design was adopted. Earlier blasts were shot faster using 35 to 50 millisecond delays along the rows and 117 milliseconds between rows. The present design uses 65 millisecond delays between holes and 340 milliseconds between rows. It was found that die blast results improved when increasing the delay times.

2.3 *Wall Control Blasting*

Open pit development commenced at the 458 to 465m elevations above sea level, and will conclude at the 150m elevation giving a total pit depth of 315m. Contractors were used to excavate down to the 435m elevation. The pit design consists of 30m high double benches mined in 15m increments. Desired bench face angles range from 75° to 85°. Mine inspectors approved a minimum catch bench width of 11m.

The steep slopes in the pit require the final wall to be left as undisturbed as possible. Therefore, the blast patterns must be significantly altered when blasting against the final wall. During the pre-stripping phase there was little attention paid to wall control blasting. As the depth increased and the bench size increased to 30m it was imperative that the wall control blasting practices be implemented.

The first attempts at wall control blasting were done on the 435 bench as a test away from the final wall and involved the use of large diameter 31 lmm holes. The initial blasts were modified production blasts (7.5m by 7.5m, square pattern, 311 mm holes, 750kg/hole) with a reduced load in the final row of holes (460kg/hole). The final row of holes was offset from the desired wall by several metres. It was not possible to dig to the limits and there was blocky material in the buffer row. The next modification in the design was to place a trim row of unstemmed holes along the design toe with a toe load of 100kg on 5m spacing. These holes were 5.5m from the buffer row. This helped reduce the toe along the base of the final wall, however mere was still significant damage at the crest. Several minor modifications were made before arriving at the following design that was used on the 420 bench:

- trim row = 4m spacing, 5.5m burden, 70kg/hole - no stemming,
- buffer row = 7.5m spacing, 7.5m burden 400kg/hole with air-deck,
- delays = inter-hole = 35ms, inter-row = 167ms.

This design resulted in a low powder factor in the area of the buffer row, which caused excess confinement, greater vertical movement, and less forward movement during the blast as well as poor fragmentation and significant damage to the final wall. To improve the design, the burden was reduced on the buffer row.

The next evolution in the design was to drill and blast a row of pre-shear holes prior to drilling and blasting the wall control blast. This was done in an attempt to eliminate penetration of the gases from the production holes into the rock mass of the final wall and to reduce the vibration levels beyond the final wall. The first pre-shear blasts were drilled with 311mm holes on 4m spacing and loaded with 70kg per hole. A crack did not form between all of the holes, therefore die spacing was reduced to 3m for the pre-shear holes.

The results from pre-shearing still did not provide the quality of wall that was desired. In many cases the buffer row was damaging the crest behind the pre-shear and loosening wedges (Figure 2). This resulted in a reduction of the catch bench width.

As the design continued to evolve, more changes were implemented. In several cases there were large toes left at the base of the final wall. These required secondary blasting for removal to allow drilling of the mid-bench pre-shear. Placing the buffer row 3m from the toe and reducing the spacing to 4m significantly reduced the incidence of these toes.

The second issue that faced the operation was being able to attain the toe of me final wall in the design location. It soon became apparent that the configuration of the DK90S drills did not allow for the final row of holes on the second pass of the double bench to be drilled in the proper location (Figure 3). The drill could not get close enough to the wall to collar the final hole of the second bench. This resulted in the loss of 2 to 3 metres of the catch bench in order to maintain the overall pit design. By doing two separate pre-shears there was also a lip mat developed at the middle of the bench face. This lip was a source of loose material, and could potentially deflect rocks falling from above over the catch bench below.

Figure 2. Blast damage to the bench walls, (a) increased fracturing near crest of bench and (b) loosening of rock wedges and loss of bench width.

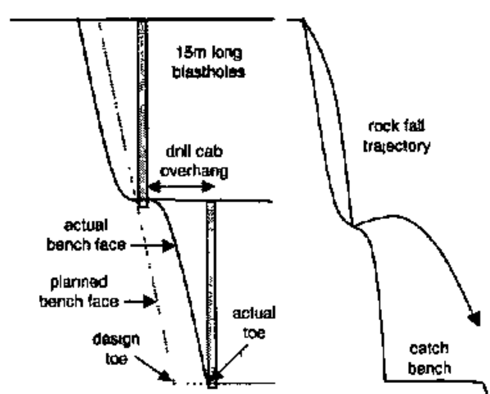

Figure 3 Small lip created by offset of 15m pre-shear holes used for 30m high benches.

In late 1999, a pre-sbear was attempted using 165mm holes on 2m spacing. The holes were loaded with 44mm Dynosplit C, a continuous water-gel explosive. The buffer design was the same as for the large hole pre-shear. The wall conditions improved drastically. The time spent scaling was also reduced resulting in increased productivity. Success with the small hole pre-shear led the operation to try pre-shear blasting for the entire 30m bench using 165mm holes. This eliminated the small lip at mid-bench.

The current wall control blast design involves pre-shearing the entire 30m-bench height with smaller diameter (165mm) holes prior to drilling 15m high trim blasts. The trim blast is drilled with the production drills (270mm). The pre-shear holes are drilled 30m deep on 2m spacing. The holes are then loaded with a radially de-coupled charge. The product loaded is 44mm diameter Dynosplit C. This product is a continuous watergel explosive containing a 25 grain detonating cord running the length of the product. The toe of the hole is loaded with two 75mm chubbs of the packaged emulsion Blastex. The hole is loaded to within 3m of the collar and not stemmed to allow for further decoupling.

The trim or wall control blast fired next to the pre-shear has the closest two rows loaded lighter with reduced burden and spacing compared to the production rows (Figure 4). The buffer row closest to the final wall is 3m from the pre-shear line. The holes are drilled on 4m spacing. These holes are loaded with two 150kg decks. The second row of holes is drilled on 5m spacing with a 5m burden and loaded with 200kg and 250kg decks. The third row is a laid out the same as a standard production row.

The use of a smaller diameter hole for the pre-shear resulted in better quality walls, reduced time scaling the final wall, and a better catch bench (Figure 5).

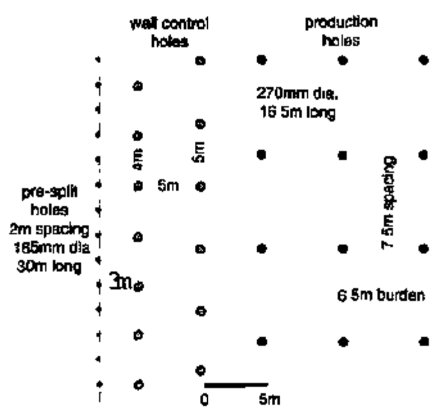

Figure 4 Layout of blastholes for the pre-shear, wall control, and production blasts.

Figure S. Successful wall control blasts showing half barrels from pre-spht holes and stable benches

3 BLAST MONITORING EXPERIMENTS

Three blasts were monitored at the Ekati mine during August to October 2000. The blasts were the 345-38 production blast, 345-40 wall control blast, and the 330-45PS pre-shear. The 345-38 blast was the 38th blast on the 345 bench, and was drilled from the 360 bench. The 345-40 blast was also drilled from the 360 bench and was the 40 blast on the bench. The pre-shear blasts are drilled every second bench to accommodate the 30m double bench.

3.1 *Vibration Monitoring*

The peak particle velocity, PPV, is often related to a blast's ability to fracture rock, through the relationships between PPV and dynamic stress or strain. McKenzie et al (1992) identified two mechanisms by which blast vibrations can cause damage:

- generation of fresh fractures in intact rock, and
- promoting slip along unfavorably oriented joint and fracture surfaces.

The first is a near field effect, and the second can occur up to hundreds of metres from a blast.

The vibration amplitude is a function of the charge weight per delay, rock type, scaled distance and charge geometry. The vibration data can be used to develop a scaled distance law that relates the PPV to the charge distance (Dowding 1985).

Over the past ten years of blast monitoring, the consensus among the leading practitioners is that geophones are the best choice for vibration monitoring. Geophones that are robust and have a suitable dynamic range and are suitably grouted in place provide the best means of collecting vibration information. The vibration monitoring was carried out with uniaxial geophones (OYO 101LT 900ß 14Hz) grouted into boreholes at mid-level of the benches. The geophones were oriented with placing rods in the boreholes to point toward the blast before grouting.

3.2 *Gas Pressure Monitoring*

One of the objectives of the field tests was to assess whether explosive gases penetrated along induced or existing fractures to a significant distance beyond the blast perimeter. Research on gas penetration monitoring has been published by Preston & Tienkamp (1984), Williamson & Armstrong (1986), Lilly (1987), LeJuge et al. (1994), Bulow & Chapman (1994), Forsyth et al. (1997), Ouchterlony et al. (1996), and Brent & Smith (1996, 1999). It is difficult to collect reliable and reproducible gas penetration data. The difficulties involve the instrumentation design and the location of the instrument and the local geology.

The instrumentation used at Ekati was a sensitive Honeywell 18615PCDT pressure sensor installed in a sealed borehole a given distance from the blast. The borehole had a diameter of 100 or 165mm and was drilled to depth of 15m below the bench. A 2m long section of 50mm diameter ABS pipe with a threaded end cap was sealed by grout and cuttings over the upper 2m of the borehole. The pressure sensor, which is capable of reading 137kPa over-pressures to 137kPa under-pressures, was installed inside the cap threaded onto the end of the ABS pipe and thus exposed to the pressure changes occurring in the borehole. The 18615PCDT Honeywell pressure sensor is designed to read relative pressure changes and a short tube was vented to the atmosphere through the top of the threaded cap for this purpose.

The pressure sensor has good dynamic response <1ms and gives direct output voltage in the 7 to 16 volt range. An Instantel Minimate logger was used to record the pressure signals at a sampling rate of 16kHz. When other researchers conducted borehole pressure monitoring near blasts to detect penetration of high-pressure explosive gasses, it was noted that that under-pressures or negative pressures with respect to atmospheric pressure often occurred. Work by Brent & Smith (1996, 1999) suggests that this phenomenon is due to volume increase caused by crack formation and overall rock mass dilation. Their data are based on twelve free-face blasts where there were no instances of high-pressure gas penetration. Pressures were monitored at distances less than one burden. This was also supported in work done by Ouchterlony et al. (1996) where 10 of 13 blasts showed under-pressures, the three over-pressures coming from pre-split blasts.

4 RESULTS

4.1 *Visual Observations*

The damage behind the production blasts typically follows this pattern: 5 to 7m back break from last row of holes, large cracks opened up to the 10m range and fine cracks as far as 25m behind the last row blasted. In all cases the cracking appears to be related to the jointing (Figure 6). There is often vertical offset on these cracks as well.

The wall control blasts typically resulted in small structurally controlled failures along the crest and some opening of horizontal joints is visible near the crest.

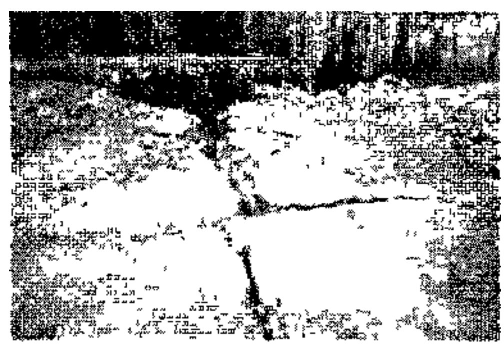

Figure 6. Cracking behind production blast at Ekati.

4.2 *PPV- Scaled Distance Relationships*

Two PPV - scaled distance relationships were used to process the recorded vibration data from the 345-38 production blast and the *345-40* wall control blast. A traditional square root scaling relationship was used. A square-root relationship assumes the explosive charge acts at a point and is best for condition involving near spherical concentrations of explosive or when monitoring relative far from the explosive column. In this approach, the square root of the charge weight is used to fit an equation of the form:

$$PPV = K\left(\frac{R}{\sqrt{W}}\right)^{-\beta} \tag{1}$$

where:

PPV = peak particle velocity (mm/sec)
R = distance from charge (m)
W= charge weight (kg)
K, ß = site specific constants.

The scaled distance is given *by R/sJ W*.

A second method to analyze the vibration data was proposed by Holmberg and Persson (1979). This method works best for near-field prediction of vibration amplitude since the method essentially integrates the effects of vibration generated over the length of the borehole. The PPV data were fit to an equation of the form:

$$PPV = K\left[\frac{l}{R_o}\right]^{\beta/2}\left[\phi - \arctan\left(\frac{R_o Tan\phi - H}{R_o}\right)\right]^{\beta/2} \quad (2)$$

with the Holmberg term defined as:

$$HolmbergTerm = \left[\frac{l}{R_o}\right]\left[\phi - \arctan\left(\frac{R_o Tan\phi - H}{R_o}\right)\right] \quad (3)$$

where:

PPV = peak particle velocity (mm/sec)
/ = linear charge density (kg/m)
K, a, ß = site specific constants
R_o, ϕ, x, H are defined in Figure 7.

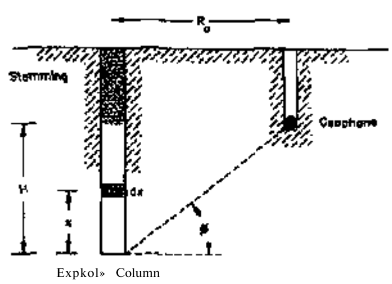

Figure 7. Definition of terms used in the Holmberg term.

The field data were plotted in two forms: log (PPV) vs. log (scaled distance) and log (PPV) vs. log (Holmberg term). On each plot linear regression lines were added. From each equation the site specific constants K and ß were calculated. The plots for each are shown in Figure 8 and Figure 9.

The square-root scaling relationship gave the constants K = 332 and ß = 1.53 for the production blast and K= 206 and ß = 1.14 for the wall control blast. The Holmberg relationship gave the constants K= 1650 and ß = 1.58 for the production blast and K = 7644 and ß = 1.20 for the wall control blast. In bom cases, the degree of fit on the production data was very good (R = 0.85), while the data from the wall control blast had a poor fit (R^2 = 0.26 or 0.29).

A value of K = 1686 was used by Holmberg & Persson (1979) for a large open pit scenario. This agrees well with the K value of 1650 from the

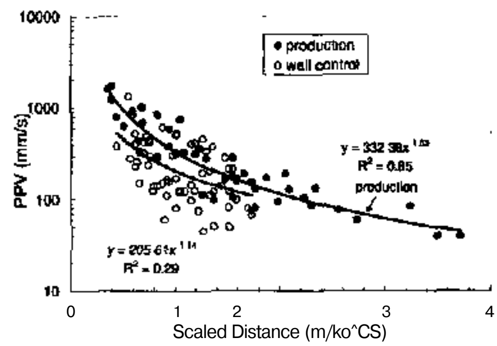

Figure 8. PPV versus scaled distance for the 345-38 production blast and the 345-40 wall control blast.

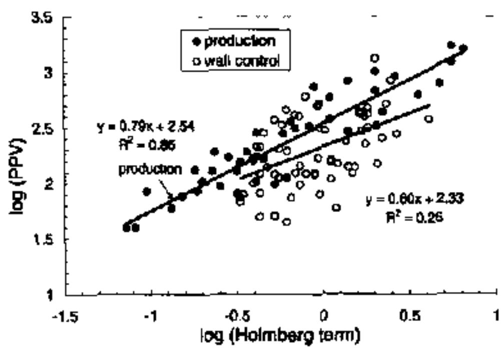

Figure 9. PPV versus Holmberg term for the 345-38 production blast and the 345-40 wall control blast.

production blasting. The ß value for the production shot monitored at Ekati was 1.58. The ß value in the study by Holmberg and Persson (1979) was 1.78. Both values are within the range of 1 to 2 that is commonly encountered. Due to die effects of the pre-shear and choked face (Figure 10) it Ìs difficult to compare the constants from the wall control blast to values from literature.

The increased scatter in the data from the wall control blast may be attributed to one or a combination of the following issues:

- me blast was choked on one free-face,
- large amounts of water were draining into the pre-shear fracture,
- larger variation in the charge weight per delay than the production blast, and
- the geophones were located behind a previously blasted pre-shear.

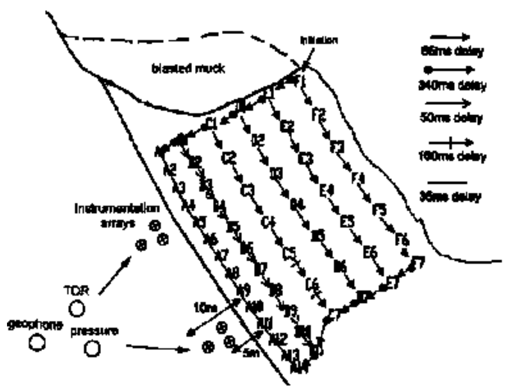

Figure 10. Layout of blastholes and instrumentation holes for the 345-40 wall control blast.

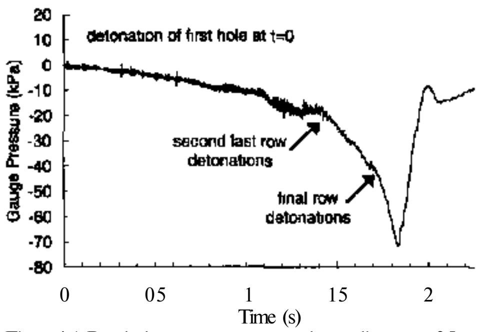

Figure ! 1 Borehole pressures measured at a distance of 5m from the boundary of the 345-40 wall control blast

The choked free face on the wall control blast had a noticeable effect on the vibration levels in the final wall. The geophone located closest to the pre-existing pile of blasted muck recorded a PPV of 1300mm/s while the other location, which was near a better free face, only recorded a PPV of 600mm/s. This clearly illustrates the effect that a choked face can have on the damage to the final wall.

For the conditions monitored *at* Ekati Mine both the square-root scaled-distance and Holmberg methods gave similar degrees of fit to the data. Based on this observation it is recommended that the square-root scaled distance relationship be used for simplicity. It is difficult to back calculate distances for the Hohnberg equation, and no significant improvement in quality of PPV prediction İs realized.

4.3 *Borehole Pressures During the Production and Wall Control Blasts*

The gas pressures measured in empty sealed boreholes near the blasts followed the trend of other researchers in that no explosive gas penetration was observed from the production blast or the wall control blast. The wall control blast resulted in an under-pressure of 70kPa below gauge pressure (Figure 11) and the production blast resulted in an under-pressure of 67kPa.

The time of the greatest pressure drop in the wall control blast corresponds to the detonation of the second last row and final row of blastholes. These detonations generated a PPV of 850mm/s and 1300mm/s, 5m behind the pressure monitoring hole The second last row of blastholes was 8m from the instrument array. The rock mass likely dilated along a combination of pre-existing discontinuities that opened as the stress wave passed as well as new fractures. There was also likely to have been some heaving from the blast that would also cause dilation.

The production blast had a similar trace to the wall-control blast. The pressure drop was slightly larger (70kPa) than the production shot. By comparing vibration signatures from individual holes on the vibration trace to die pressure trace, it is clear that the pressure drop did not occur until the second last row was detonating. The blast was choked on the free face closest to the pressure sensor (Figure 10). As a result, the PPV values at that monitoring location were double the values at the second location.

Penetration of explosive gases was not observed for the production or wall control blasts because the gases appear to preferentially vent towards the free face where the resistance to flow is the smallest.

4.4 *Borehole Pressures During the Pre-Shear Blast*

Borehole pressure monitoring was also carried out on the 330-45PS pre-shear blast. For this blast, explosive gas penetration was observed as indicated by an increase in borehole pressure in the monitoring hole that was 5m from the nearest blastholes. The pressure trace is shown in Figure 12. The pre-shear holes were all detonated at the same time (at t=0s on the trace).

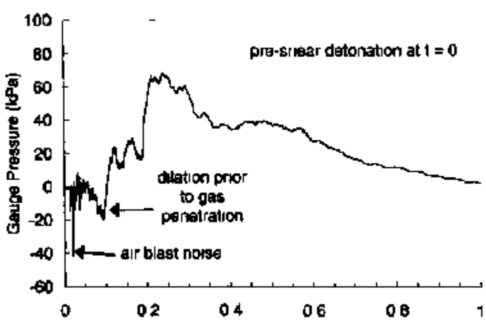

Figure 12. Borehole pressures measured at a distance of 5m from the 330-45PS pre-shear blast

The PPV recorded from the pre-shear blast was 685mm/s. There was no change in borehole pressure as the stress wave passed the monitoring hole. There is some high frequency noise at the beginning of the trace that is likely caused by an air blast from the unstemmed pre-shear holes. The air blast traveled at a velocity of =330m/s arriving at the monitoring hole 15ms after detonation.

Following the air blast noise and prior to the increase in borehole pressure there is a pressure drop of 18kPa. This phenomenon has been observed by others and according to McKenzie et al. (1992), it results from rock mass dilation occurring prior the inflow of gases. It Is assumed that the penetrating gases act like a wedge driven into the rock mass causing tensile stresses and fracturing ahead of the gas front. The rock dilation manifests itself as a pressure drop immediately preceding the arrival of the gases themselves in the borehole.

By comparing the vibration and pressure traces it is possible to calculate the average velocity for the explosive gases from the blastholes to the pressure monitoring hole. The gases took 100ms to travel a distance of 5m, giving an average velocity of 50m/s.

LeJuge et al. (1994) published a similar trace to Figure 12 recorded behind a pre-shear blast. This study also monitored pressures at greater distances from a pre-shear. While all holes showed an initial dilation, it was only the closest hole that recorded an overpressure. They concluded that bench swell can extend significantly further than gas penetration. This also appears to be the case at Ekati Mine.

5 CONCLUSIONS

Ekati Mine has implemented a number of modifications to their blast designs to improve fragmentation and diggability as well as to minimize blast damage to the final bench walls. Blast monitoring showed that explosive gases from the wall control blast do not penetrate past the row of previously fired pre-split holes. Blast vibrations measured in the final pit wall from production or wall control blasts are also reduced because of the presence of the holes and fractures from the pre-split blast. The use of 30m long pre-split holes allowed creation of stable double lift benches without any lips.

It appears that there are two main blast damage mechanisms. The first is from the penetration of explosive gases from the pre-shear blasts. The second mechanism, heaving, results from the production and wall control blasts. This is caused by the vertical movement from the blast, and can extend tens of metres from die blast perimeter. This mechanism causes the greatest damage to the final wall.

While the pre-shear appears to cause some damage to the wall, it also helps to reduce the subsequent damage occurring from heaving. Reducing the damage to the final wall would best be accomplished by minimizing the confinement of the last two or three rows of the wall control blasts, and therefore reducing the resulting heave. Reducing or eliminating the stemming in these rows could achieve this. LeJuge et al. (1994) saw improved results after eliminating decking and stemming of buffer holes in wall control blasts.

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