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Evaluation of Kiruna mine drifting data using the NIOSH design approach

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ABSTRACT: The National Institute of Occupational Safety and Health (NIOSH) has developed a new drift round blast design approach as part of a comprehensive program to improve underground mine safety through the introduction and implementation of better blasting practices in drifting. The key to better perimeter control blasting has been found to lie in the proper design of the buffer/helper row. Although in retrospect this finding is not particularly surprising, it goes against conventional wisdom in which special design efforts are spent on perimeter hole charge, spacing and burden design and rather little on the buffer row design. The paper begins with a brief introduction to the NIOSH buffer row design approach. This is followed by the presentation of a comprehensive set of data from 9 different drift rounds in iron ore and 3 rounds in waste rock in which the buffer row design remained constant but significantly different wall control designs were employed. Because the buffer row design was well-chosen, the extent of the over-break is shown to be essentially independent of perimeter charge concentration over a very wide range of values. This finding has a significant impact on future drift design concepts. The paper concludes with some ideas regarding future developments.

1 INTRODUCTION

The NIOSH drift round design approach incorporating perimeter control blasting described in the recent paper by Hustrulid & Johnson (2008) consists of the following steps:

- Step 1: Design the buffer/helper row
- Step 2: Add the contour holes
- Step 3: Design the lifters
- Step 4: Add the cut
- Step 5: Add fill-in holes as required

The basic premise is that the buffer/helper row design holds the key to successful perimeter control blasting. If the buffer/helper row has been properly designed, the effective damage produced by this row should extend out to but not beyond the as-designed perimeter. The perimeter row of holes should then have little, if any, undamaged rock remaining to be broken (the effective burden) and the primary action of the perimeter row then becomes one of smoothing rather than of virgin breakage. The limiting factor is the effective burden on the perimeter holes. At a limit of zero, the perimeter charges can be visualized as simply resting on or very close to a free surface. In this case, most of the explosive energy in the perimeter charge would be directed inward toward the drift and not outward into the remaining rock mass. This leads to the hypothesis that if the buffer row of holes is properly designed, the extent of damage to the rock outside of the perimeter row of holes should be largely independent of the perimeter row charge.

Today, a significant amount of effort is being expended on developing perimeter row charges, associated designs, and firing techniques but relatively little on the buffer row (Mandal et al. 2008). It seems that maybe the level of effort should be reversed, with the primary emphasis on the buffer row design.

As part of the NIOSH careful blasting research and development program, practical drift driving designs and results have been collected and evaluated. This paper looks beyond those evaluations and presents a re-assessment of the Swedish results of from a drift driving program conducted by LKAB at the Kiruna mine in the early 1990's (Hustrulid 1994) based upon the NIOSH approach. In this case, the basic drift design was maintained constant with only the charging of the wall perimeter holes being varied. The amount of over break was monitored. The completeness and quality of these data provide an opportunity to evaluate (1) the "goodness" of the buffer row design and (2) the hypothesis that over break is largely independent of the perimeter charge if the buffer row design is correct.

2 THE NIOSH BUFFER ROW DESIGN APPROACH

The key to the NIOSH approach is the ability to assign a practical radius of damage (R_d) to each blasthole and explosive combination being considered for use in the particular rock mass. By "practical" radius of damage, it is meant that if the rock mass lying outside of this ring were removed, the rock remaining within the ring would easily break apart. This practical damage radius is illustrated in Figure 1.

The technique used for calculating R_d is presented in section 2.2.

2.1 *Example design*

Consider the 7 m wide by 5m high drift with arched roof shown in cross-section in Figure 2. The perimeter (walls and roof) are to be excavated using smoothwall blasting techniques.

For illustration purposes, it will be assumed that $R_d = 0.8$ m for the fully-coupled buffer row holes. The first step in the design is to draw parallel shells located at distances of R_d and $2R_d$ inside the desired contour as shown in Figure 3.

Next, circles of radius R_d are added. The center of the circle corresponds to a future buffer row hole location. Figure 4 shows the placement of the buffer row holes in the "just-overlapping" scenario.

As can be seen, there is a considerable amount of "un-touched" rock or cusps between the as-designed coverage and the perimeter. This is overcome by translating the holes along the design line so that they more fully overlap. Figure 5 shows one possible arrangement.

In this particular case, the distance between the roof buffer row holes is 1.35 R_d or 1.08 m. In Figure 6, the buffer row wall holes have been added.

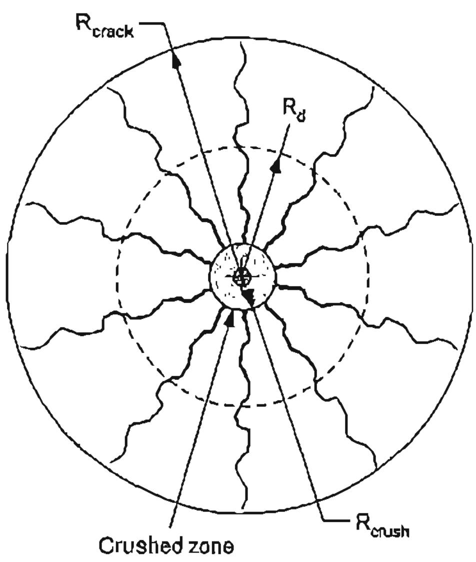

Figure 1. Diagrammatic representation of the crush, crack, and practical damage zones surrounding a blast hole.

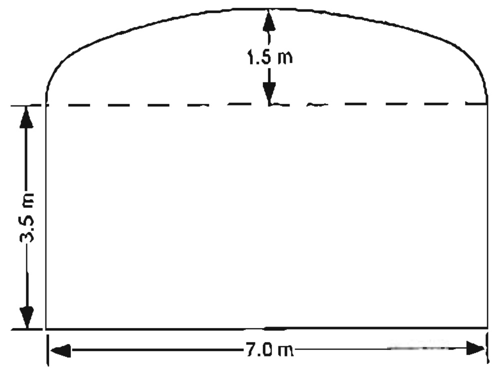

Figure 2. Example drift shape.

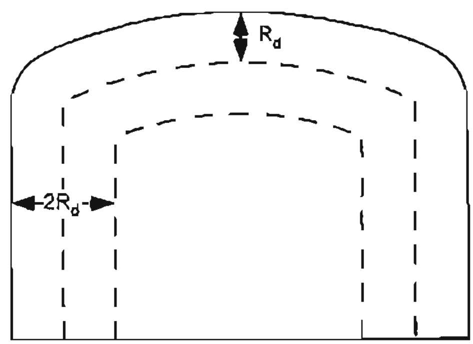

Figure 3. First step in the buffer row design.

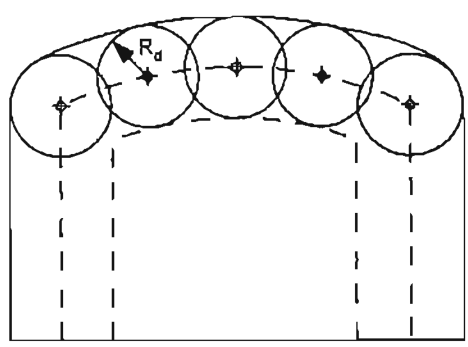

Figure 4. Initial placement of the buffer row roof holes.

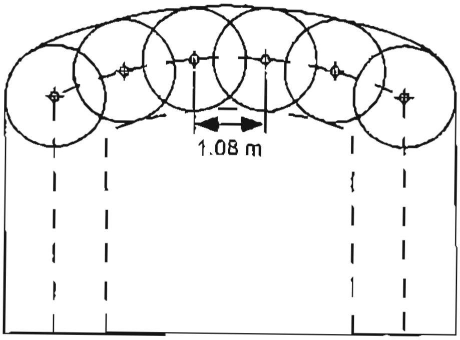

Figure 5. Final placement of the buffer row roof holes.

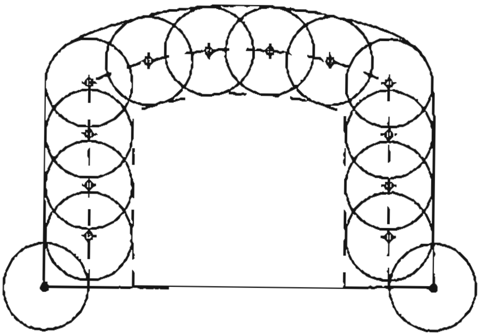

Figure 6. Addition of the buffer row wall holes.

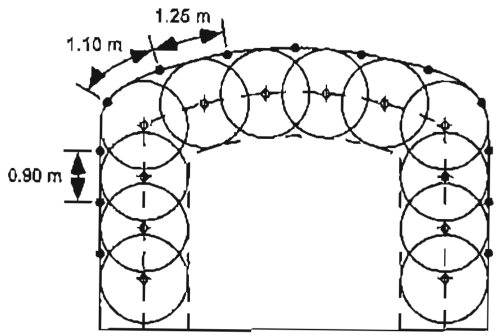

Figure 7. Addition of the contour holes.

In step 3, the contour row holes are positioned to "smooth out" the surface created by the buffer row holes. The first holes placed are in the drift corners. They have the required look-out and look-up angle to provide the space needed for drilling the next round. These holes also ensure proper extraction of rock from the corners. The remaining holes along the roof are placed to remove rock cusps between adjacent damage circles. As can be seen, the amount of rock associated with each hole (the burden) is rather small (Figure 7).

2.2 *Estimation of the damage radius*

NIOSH is exploring several different approaches for assigning a damage radius to different explosive—rock combinations. These are the subject of another paper presented at this conference. Pending the final results of these studies, NIOSH is currently using two different estimators which are loosely based upon results originally obtained by Ash (1963) in quarry and open pit blasting applications. He showed that, for fully coupled charges, the burden is directly related to the blasthole diameter. The required constant depends on the type of explosive used. The first estimator is based on explosive energy. The practical damage radius is

$$R_d/r_h = 25\sqrt{\frac{\rho_e s_{ANFO}}{\rho_{ANFO}}}\sqrt{\frac{2.65}{\rho_{rock}}} \qquad (1)$$

where R_d = damage radius (m), r_h = hole radius (m), ρ_e = explosive density (g/cm³), ρ_{rock} = rock density (g/cm³), s_{ANFO} = weight strength with respect to ANFO, ρ_{ANFO} = ANFO density (g/cm³), and 2.65 = density of the rock for which the factor 25 applies.

Equation 1 can be simplified to

$$R_d/r_h = 25\sqrt{RBS}\sqrt{\frac{2.65}{\rho_{rock}}} \qquad (2)$$

where RBS = relative explosive bulk strength with respect to ANFO expressed as a ratio.

Equations 1 and 2 apply to fully coupled charges and were adapted (Hustrulid 1999) based on the use of energy concepts from the original Ash (1963) work.

The second estimator is based on borehole wall pressure (Hustrulid & Johnson 2008). For fully coupled charges, the damage radius is:

$$R_d/r_h = 25\sqrt{\frac{P_{e\,Exp}}{P_{e\,ANFO}}}\sqrt{\frac{2.65}{\rho_{rock}}} \qquad (3)$$

where $P_{e\,Exp}$ = explosion pressure for the explosive and $P_{e\,ANFO}$ = explosion pressure for ANFO initiated in a hole of blasthole diameter.

For de-coupled charges, Equation 3 is modified to:

$$R_d/r_h = 25\sqrt{\frac{P_{w\,Exp}}{P_{e\,ANFO}}}\sqrt{\frac{2.65}{\rho_{rock}}} \qquad (4)$$

where $P_{w\,Exp}$ = the wall pressure for the explosive being used.

The wall pressure ($P_{w,Exp}$) is calculated using the co-volume approach described by Hustrulid (2007) and Hustrulid & Johnson (2008).

3 LKAB DRIFTING DATA

3.1 *Introduction*

Over the period 1991–1992, LKAB converted from the use of Kimanol (ANFO) to Kimulux R (a re-pumpable emulsion) in their drifting operations. As part of the change-over process, a number of different blasting tests were conducted to assist in the development of a standard drifting design which could be used both in ore (magnetite) and waste rock (syenite porphyry) with minimum over-break. The rock properties are summarized in Table 1.

The location of the test area was on Level 686 m near the south end of the orebody. Although the ore and footwall rock were "relatively free" from strong structural control, structures were still present and had an affect on the degree of over-break. As shown in Figure 8, four headings (400, 400.2, 400.1 and 402) were selected for inclusion in the test program.

Table 1. Rock properties.

Rock type	Density (g/cm^3)	P-wave velocity (m/s)
Syenite porphyry (type 5)	2.9	3856
Syenite porphyry (type 3b)	2.7	4196
Magnetite ore (type B)	4.8	3950

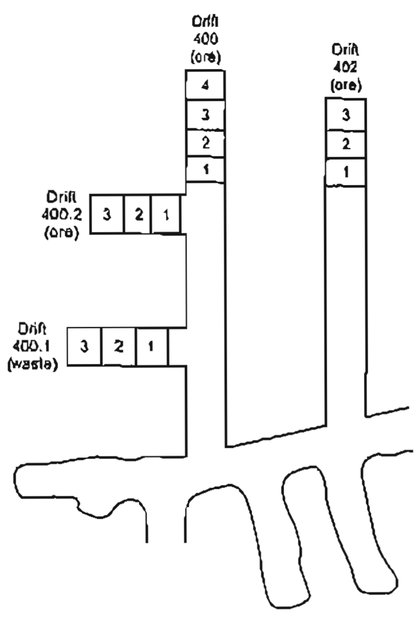

Figure 8. Location of the headings.

Two of the ore headings (400 and 402) were oriented from footwall to hangingwall (basically E-W) while the third heading (400.2), was oriented along the orebody (basically N–S). Heading 400.1 located in the footwall waste rock was driven parallel to drift 400.2. Since the maximum principal stress direction is oriented E–W and is essentially sub-horizontal, one could expect some stress-related contribution to over break in the N–S oriented drifts as compared to those oriented E–W. At the start of the test series, Drift 402 was considerably oversize. The floor was nearly 1 m too low and adverse structures in the roof had contributed to some unwanted caving. As a result, design modifications, particularly to the floor elevation, were made to correct the situation. All of the headings were driven upward at a grade of 2.5%.

3.2 *Blast design and execution*

The design denoted as 123/3 shown in Figure 9 was chosen for the tests.

There are a total of 59 holes consisting of 13–64 mm diameter holes in the cut and 46–48 mm diameter holes in the rest of the round. The nominal hole length is 5 m. The distance from the buffer row of holes to the as-designed roof and wall locations is 80 cm. The numbers shown on Figure 9 indicate the initiation plan.

LKAB drifting practice at the time of the test was (1) to carefully draft the desired hole pattern, (2) to transfer the design to a 35 mm slide, and (3) to have the surveyors project the pattern onto the face using a slide projector. The position of the projector is moved until the orientation, the vertical position and the size of the heading are correct. The hole locations are then marked with paint. Using this procedure, any desired design.

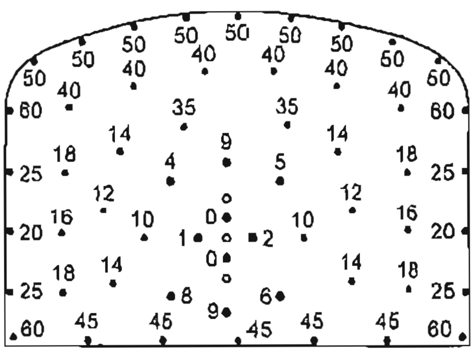

Figure 9. Blast design 123/3 showing the location of the holes and the initiation plan employed in the tests.

even non-rectangular hole grids can be easily implemented. Once the hole locations have been accurately marked, it is up to the driller to position the bit at the mark. Typically, hole collaring errors can be held to a few centimeters. The desired amount of look-out at the hole bottom is 200 mm for the wall and roof holes and 300 mm for the floor holes. If the drifts were to be driven horizontal, this corresponds to a lookout angle of 2.3° for the wall and the roof holes and 3.4° for the floor. However, since the drifts were to be driven at a 2.5% grade, the desired inclination angles are:

- wall holes: 2.3° outward
- roof holes: 3.7° upward
- floor holes: 2.0° downward

The remaining holes in the round were drilled horizontal and oriented parallel to the drift axis.

After drilling, the surveyors measured the collar position and the lookout angle of each of the wall holes. Using the wall hole collar locations and orientations, the position of the hole at any depth could be calculated. Although the actual collar locations of the roof and lifter holes were not surveyed, their lookout angles were measured using a digital level.

All of the holes (including the buffer row holes) with the exception of the wall holes, the roof holes and the three uncharged holes in the cut, were charged with Kimulux R, a re-pumpable emulsion explosive. The properties of the explosives used in the test are given in Tables 2 through 4.

For the buffer holes, the Kimulux charge extended to within about 0.5 m of the hole collar. If this was not done, rock which eventually had to be scaled down often remained between the buffer holes and the contour holes in the uncharged region.

The perimeter design consisted of 4 holes along each wall and 9 roof holes. The roof holes were charged with detonating cord at 40 g/m for all 13 tests. The cord with detonator attached was pushed to the hole bottom with the emulsion hose and a small amount of Kimulux was injected to hold the cord in place. This charging procedure was maintained constant for all of the tests.

Table 2. Properties of the Kimit explosives (Kimit AB 1992).

	Kimulux R	Kimanol	Kimulux 42
Diameter, mm	48	48	22
Density, g/cm³	1.2	0.85	1.1
VOD, m/sec	5500	3500*	5000
Energy, MJ/kg	2.94	3.8	3.43
Gas volume, l/kg	906	964	903
Det. pressure, MPa	9100	2600	6800

*for diameter $d = 50$ mm.

Table 3. Explosive properties used in the damage radius calculations (Kimit AB 1992).

	Kimulux R	Kimanol	Kimulux 42
RWS*, pct	81	100	90
RBS*, pct	115	100	114
Expl. pressure, MPa	4550	1300	3400

* Based upon ANFO = 0.85 g/cm³.

Table 4. Properties for detonating cord (40 g/m).

Core density, g/cm³	1.4
Core diameter, mm	6
Velocity of detonation, m/s	7,180
Energy, MJ/kg	6.21
Gas volume, l/kg	780
Detonating pressure, MPa	18,000
Explosion pressure, MPa	9,000
RWS*, %	139
RBS*, %	229

*Based upon ANFO = 0.85 g/cm³.

Four different explosives were used for charging the wall holes:

- Kimulux R (KXR)—pumped emulsion
- Kimanol (KIM)—blown ANFO
- Kimulux 42 (KX42)—emulsion in 22 mm diameter tubes
- Detonating cord (PS)—40 g/m

Because of possibility of differing rock conditions being present in the various headings, the explosive used in charging the wall holes was varied from round to round. In this way, the results obtained using the different explosives could be directly compared in basically the same rock. A total of 13 rounds were shot of which the results from 12 were evaluated.

3.3 *Blast follow-up*

After blasting and very thorough scaling involving the use of a mechanical scaling machine, the surveyors returned to survey the drift profile at the mid-length position of the round. This was done using a total station set up at the profile location and then reading the vertical angle and distance to a series of cap lamp light points. The results are shown in Figures 10 through 13.

For the wall holes, the designed hole collar location is denoted by the symbol "•" and the actual collar position is denoted by an "X". Knowing the

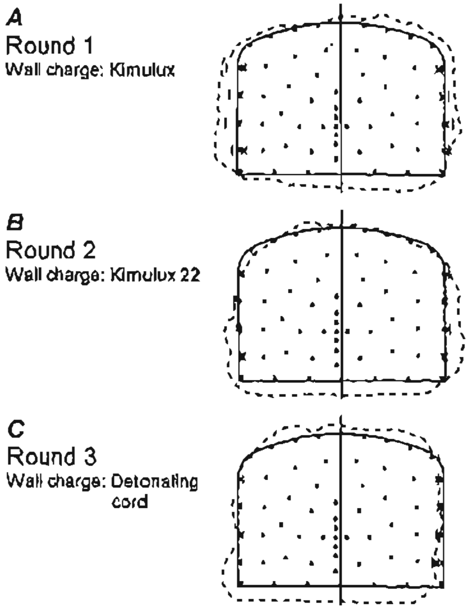

Figure 10. Profile mapping results from Drift 400 in ore.

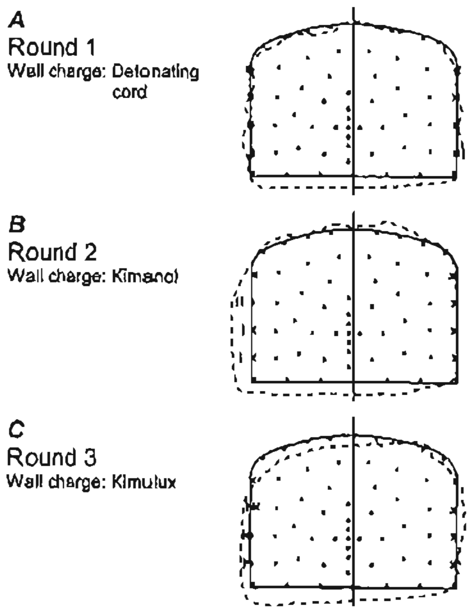

Figure 11. Profile mapping results from Drift 400.2 in ore.

look out angle and the hole collar position, the position of the hole on the profile can be calculated. This position is denoted by an "|". The distance between the "|" location and the profile line is then measured. The results for the wall holes are

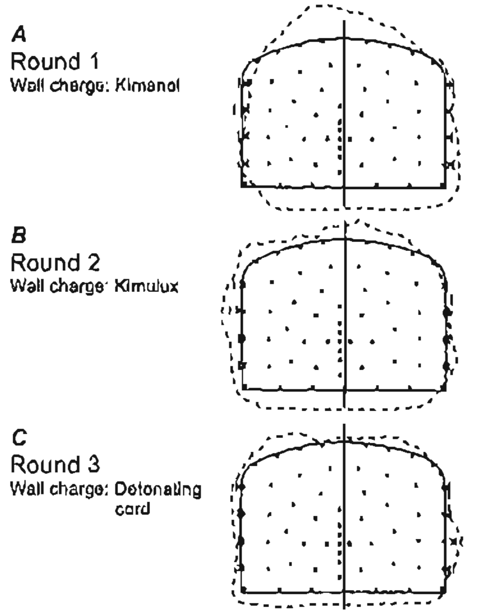

Figure 12. Profile mapping results from Drift 402 in ore.

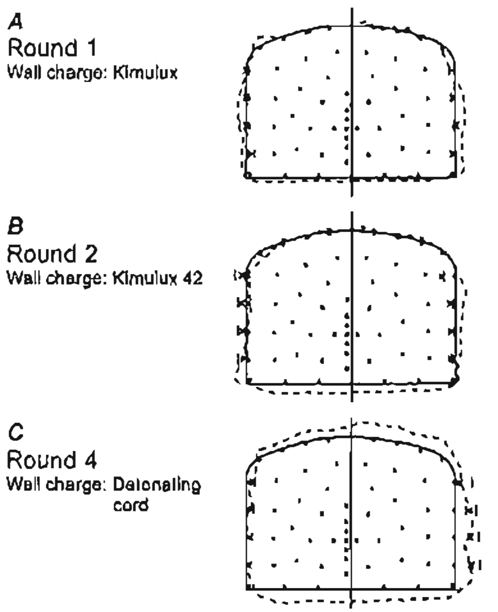

Figure 13. Profile mapping results from Drift 400.1 in waste.

summarized in Tables 5 and 6. The (+) sign indicates that the final scaled profile lies outside of the hole location "|" and represents over-break. The (–) sign indicates that the scaled profile lies inside the hole location "|" and represents under-break. The

Table 5. Mid round over-break/under-break distances at the projected left wall hole positions for all 12 tests, listed by location and explosive type.

Drift	Material	Round	Explosive	Over-break/under-break, cm				Average, cm
400	Ore	1	KXR	5	27	3	21	14
		2	KX42	–47	–45	–30	–10	–33
		4	PS	7	6	13	10	9
402	Ore	1	PS	–11	0	7	21	4
		2	KIM	23	16	37	25	25
		3	KXR	–30	13	3	10	–1
400.2	Ore	1	KIM	–36	–40	–8	6	–20
		2	KXR	13	22	33	25	23
		3	PS	12	21	10	13	14
400.1	Waste	1	KXR	17	36	25	57	34
		2	KX42	–13	–7	50	37	17
		3	PS	–10	16	13	40	15

Table 6. Mid round over-break/under-break distances at the projected right wall hole positions for all 12 tests, listed by location and explosive type.

Drift	Material	Round	Explosive	Over-break/under-break, cm				Average, cm
400	Ore	1	KXR	7	37	35	23	26
		2	KX42	53	35	20	23	33
		4	PS	7	–25	–14	–27	–24
402	Ore	1	PS	–9	3	0	–15	–5
		2	KIM	0	–4	–9	–14	–7
		3	KXR	0	29	26	16	18
400.2	Ore	1	KIM	0	–5	16	17	7
		2	KXR	–7	12	7	21	8
		3	PS	–12	–7	–3	0	–6
400.1	Waste	1	KXR	39	25	25	0	22
		2	KX42	9	45	40	50	36
		3	PS	4	–6	15	–10	1

number associated with the (+) and (–) sign is the distance in centimeters.

The roof profile measurements were also evaluated although some caution is required since roof structure and gravity forces play a much more significant role than for the wall holes. As indicated, for each and every round, the roof perimeter holes were charged with detonating cord. Two sets of measurements were made:

- the distance from the as-designed collar position of the perimeter holes to the measured profile
- the distance from the as-designed collar position of the buffer row holes to the measured profile

It will be assumed that:

- The roof holes are collared at their designed locations and drilled with the designed look-out angles
- The buffer holes are collared at their designed positions and are drilled horizontal (parallel to the drift axis).

Under these assumptions it is expected that with no over-break or under-break, the distance should be 16 cm for the perimeter holes and 96 cm for the buffer holes. The results are summarized in Tables 7 and 8.

It is observed that although there is considerable variation as might be expected, the average results are not too different from those corresponding to a no over-break/under-break condition. This will be discussed in more detail later. The Drift 402 over-break results are considerably greater than the others. As indicated earlier, Drift 402 started oversize with roof over-break due to the structures present and measures were taken to bring the drift geometry back to the desired dimensions. There was an improvement in the amount of over-break as the heading was advanced.

Table 7. Distance (cm) from the roof hole collar positions to the measured mid-round profile.

Drift	Material	Round	Distance, cm									Ave., cm
400	Ore	1	30	10	−7	10	0	0	5	−5	−7	4
		2	−20	15	5	10	12	10	10	17	0	7
		4	15	25	35	25	35	50	50	45	45	36
402	Ore	1	0	40	90	115	100	125	60	30	15	64
		2	60	35	35	40	45	55	42	15	7	37
		3	25	50	50	10	15	17	40	30	0	26
400.2	Ore	1	−10	0	−10	−15	15	−10	0	10	−10	−3
		2	15	20	10	25	10	25	30	20	10	18
		3	−50	−33	−30	−35	−30	−20	0	15	−15	−22
400.1	Waste	1	25	25	15	0	27	20	40	45	37	26
		2	−15	−5	22	20	0	0	−5	0	0	2
		3	5	33	50	20	33	20	33	40	15	28

Table 8. Distance (cm) from the buffer row hole collar positions to the measured mid-round profile.

Drift	Material	Round	Distance, cm						Ave., cm
400	Ore	1	115	88	90	98	100	95	98
		2	75	95	87	100	105	95	93
		4	110	125	120	135	135	145	128
402	Ore	1	95	165	200	170	140	95	144
		2	135	120	130	150	135	100	128
		3	125	145	100	100	130	105	118
400.2	Ore	1	70	80	75	85	95	90	83
		2	90	90	105	100	120	105	102
		3	45	55	50	60	95	70	63
400.1	Waste	1	105	105	90	95	125	125	108
		2	70	100	95	90	90	95	90
		3	100	135	115	105	130	120	118

Table 9. Estimated wall pressures for the different explosives.

Explosive	Hole diameter, mm	Charge diameter, mm	Explosion pressure, MPa	Wall pressure, MPa
Kimulux R	48	48	4550	4550
Kimanol	48	48	1300	1300
Kimulux 42	48	22	3400	262
Det. cord	48	6	9000	28

4 ANALYSIS OF THE DAMAGE RADIUS

To assist in the interpretation of the field results, calculations of the expected damage radii were performed based upon the estimator expressions presented earlier in the paper.

When applying the pressure-based approach Equations 3 and 4, one must first calculate the blasthole wall pressures. These values are given in Table 9.

Table 10. Damage radius (cm) for the different explosives, materials and damage predictor approaches.

Explosive	Material	Coupling	R_d (energy)	R_d (pressure)
Kimulux R	Ore	Fully	48	83
	Waste	Fully	63	109
Kimanol	Ore	Fully	45	45
	Waste	Fully	58	58
Kimulux 42	Ore	De-coupled	NA	20
	Waste	De-coupled	NA	26
Det. Cord	Ore	De-coupled	NA	7
	Waste	De-coupled	NA	9

Practical damage radii obtained from the energy-based and pressure-based estimator expressions are shown in Table 10.

5 DISCUSSION OF THE RESULTS

The predictors apply for charges detonated in holes in an infinite medium. In practice this means that the burden is sufficiently large that there is no influence from a free surface. If, for example, the free face is very close, the explosive gases would easily escape and the damage pattern would be affected.

The discussion will focus primary on the fully coupled charges. Table 11 summarizes the average drift wall over-break (OB) and under-break (UB) measurements when using fully coupled charges in both ore and waste. Irrespective of the predictor used, the measured damage zone is much smaller than that predicted. The predicted damage radii in ore:

- Kimulux R: 45–83 cm
- Kimanol: 45 cm

The predicted damage radius in waste:

- Kimulux R: 58–109 cm
- Kimanol: 58 cm.

There are several possible reasons for this, two of which are:

- The predictors are fundamentally not correct
- The amount of burden affects the extent of the damage zone

With regard to the first point, other NIOSH experience has suggested that the two predictors provide results in line with other techniques and with field observations. With regard to the second point, it is logical that if the buffer/helper holes are properly designed, the measured damage should be less than that predicted for fully confined charges. As can be seen from the blast design shown in Figure 9, the buffer row of holes both along the roof and along the wall is quite well-confined. Thus it is expected that the predicted damage radius could fully develop.

Table 11. Summary of the measured over-break/under-break.

Explosive	Material	Drift	Round	Wall	Average OB/UB
Kimulux R	Ore	400	1	Left	14
				Right	26
		402	3	Left	-1
				Right	18
		400.2	2	Left	23
				Right	8
	Waste	400.1	1	Left	34
				Right	22
Kimanol	Ore	402	2	Left	25
				Right	-7
		400.2	1	Left	-20
				Right	7
	Waste	NA	NA	NA	NA

Turning now to the roof holes which were charged the full length with detonating cord, it was observed that if the buffer holes were not charged nearly to the collar, the inter-lying rock remained soundly in place. The buffer row charge was required to break the rock to the perimeter, largely by itself. Since the distance from the buffer row to the perimeter is nominally 80 cm, this suggests that the damage radius must at least be this order of magnitude. In reviewing the predictions for the Kimulux R used in the helper holes, one finds that:

- Energy based damage radius for ore, $R_d = 48$ cm and the damage radius for waste, $R_d = 63$ cm
- The pressure based predictions for ore, $R_d = 83$ cm and waste, $R_d = 109$ cm

Based upon these numbers, it would appear that the pressure-based estimator is the most appropriate. The damage radius (based on pressure considerations) associated with the detonating cord is 7 cm in ore and 9 cm in waste rock. As seen in Figure 7, the maximum predicted extent of the damage with respect to the buffer hole position is obtained by summing the contributions from the buffer and perimeter hole rows. In the case of ore, one finds

Damage = 83 cm + 7 cm = 90 cm

The average measured distance is 106 cm and the range is 63 cm to 144 cm (Table 8). Considering the presence of adverse structural features in the roof, particularly in drift 402, and the effect of gravity, this is considered quite good agreement. In waste, the buffer hole damage radius is

Damage = 109 cm + 9 cm = 118 cm

which is to be compared with the average of 105 cm and the range of 90 cm to 118 cm (Table 8). The agreement is good.

Based upon the roof measurements, it appears that the wall buffer row holes should be able to produce damage to the desired perimeter in ore and possibly somewhat beyond in waste. The conclusion is that the perimeter row of holes should have little work to do and thus, the over-break/under-break should be largely independent of the wall loading.

The average results for the tests in ore were summarized in Table 11. The amount of over-break is greatest for the fully coupled emulsion explosive (KXR) and the least for the highly de-coupled detonating cord but the difference is much less than one might expect. The results for the waste rock as given in Table 11. The results are the same as found with the ore.

It appears that the pressure-based approach is the most appropriate, at least based on the data presented.

6 CONCLUSIONS AND FUTURE WORK

Underground excavation with contour control has become increasingly important to some mining operations and it is expected that the trend will continue. For this to happen, practical design guidelines are necessary. Although considerable attention has been placed on the development of special perimeter explosives, timing techniques, etc, over the years, it appears that the real key to successful perimeter blasting lies in the design of the buffer and helper row holes. Unfortunately, relatively little information is available in the literature concerning their design. NIOSH has recently developed a drift round blast design approach which begins with the design of the buffer/helper row. The charges in this row are designed to break to the designed perimeter so that the contour row of holes has little remaining rock to remove. If this is achieved, then, in principle at least, it makes no difference how the perimeter row is charged. Several different design approaches based on estimating the damage radius (R_d) associated with different explosives in different rock types are being investigated. This paper has presented two estimators based loosely on the work of Ash (1963). Both estimators were applied in this paper to the rather complete data set collected at LKAB's Kiruna mine. It was found that:

- The estimator based on borehole wall pressure appeared to provide a better representation of the field data than the energy-based estimator.
- The buffer row design used in the tests provided quite satisfactory results in both ore and waste.
- Although the observed over-break was somewhat dependent on the charging of the perimeter holes, it was much less than one would have expected. This would tend to support the hypothesis that over-break is largely independent of the perimeter blasting technique if the buffer/helper holes are well designed.
- Although the "as-presented" estimators of damage radius provide values which are the right order of magnitude, it is clear that more work is required in this regard.
- As has been shown somewhat indirectly in these tests, confinement appears to be an important factor in determining the amount of damage behind the hole. This is quite logical. Further work is required to quantify the effect of confinement (burden) on the extent of damage.

NIOSH is currently aggressively pursuing a research and development program to provide a sound base for perimeter-control drift round blast design. The results of these efforts are being incorporated into its user-friendly drift design software "Drift."

In closing, the authors would like to invite you to share with us (1) your thoughts/approaches regarding the definition of the practical damage zone and (2) your approach to the design of the helper/buffer row in different rock types with different explosives. Perhaps, in this way, overall progress toward full acceptance of perimeter control blasting will be accelerated.

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