A gas pressure-based drift round blast design methodology

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Abstract

The National Institute for Occupational Safety and Health (NIOSH), as part of a comprehensive program to improve mine safety through the widespread acceptance of careful excavation principles in drifting, have revisited standard drift round design concepts. Although the initial emphasis was on contour row design and providing improved design tools for blasting with de-coupled charges, the focus has broadened into the development of a general gas pressure-based drift round design approach. The concept of a damage radius (R_d) for a given explosive-hole-rock mass combination is introduced. With the damage radius as the basic building block, the blast holes are positioned on the face, beginning with the buffer row, to achieve the desired excavation size, shape and smoothness. The design of the contour row of holes is also performed using a pressure-based approach.

The paper presents in some detail the overall approach and the required gas pressure-based design equations.

1 Introduction

The development of all mass mining systems relies heavily on drifting. In general, it is important that these drifts remain stable over long periods of time both for safety and economic reasons. Hence, care must be exercised in the excavation process. Today, it is possible to rapidly drill the required blast holes with good precision using modern drill jumbos. A wide variety of explosive products are available to charge the holes and with the use of electronic delays the holes can be properly sequenced. The remaining ingredient is the availability of a practical perimeter control blast design methodology.

Holmberg (1982) presented a very useful approach to drift blast design in his paper "Charge Calculations for Tunneling" which is based on the early work of Langefors and Kihlström (1963). In this approach, the face is divided into cut, contour, lifter, and stoping sectors. The required equations providing burden and spacing dimensions as a function of hole size, charge concentration, etc. are developed for application in each sector. For perimeter charge design, an approach based on a relationship between peak particle velocity, linear charge concentration, and distance has been recommended. This has become known as the Holmberg-Persson (1978, 1979) approach.

As part of a comprehensive program on improving mine safety through the application of careful excavation techniques in drift driving, the National Institute for Occupational Safety and Health (NIOSH) revisited traditional drift round design concepts. The initial focus was more narrowly aimed at improving contour row design. However, as the study proceeded, it became quite obvious that the key to successful perimeter control was, first and foremost, the proper design of the buffer row of holes. Rather than the four design sectors identified by Holmberg (1982), there are actually five: cut, buffer, contour, lifter and stoping. Except for cut design, which is based largely on geometrical considerations, it appeared that a gas pressure-based approach could be logically applied to improved blast round design both with and without perimeter control. To accomplish this, the concept of a damage radius (R_d) for a given explosive-hole-rock mass combination was introduced. With the damage radius as the basic building block, the holes are positioned on the face, beginning with the buffer row, to achieve the desired excavation size, shape and smoothness. Whereas the burden and spacing dimensions are the building blocks in a standard blast round, in this new, pressure-based approach the burden-spacing dimensions can be calculated, if desired, but they are an output of the design approach and not an input.

A gas pressure-based approach is applied to the contour row as well. This is quite logical since, when using de-coupled charges in the contour holes, the borehole wall pressure is strongly dependent on the charge diameter-hole diameter ratio. The simple use of linear charge concentration is not enough to predict the damage extent.

In perimeter control blasting, if the buffer row of holes has been properly designed, the primary function of the contour row is to smooth the final excavation surface and not to fragment and remove significant quantities of rock. With this "smoothwall" approach, much of the "burden" lying between the contour and buffer rows has already been fragmented and/or removed by the buffer row holes.

The paper begins with a description of the gas pressure-based design approach and then provides a simple technique for estimating the damage radius associated with the buffer row of holes. It concludes with a discussion of smoothwall design for the contour row of holes.

2 An overview of the gas pressure-based design approach

In the way of introduction, consider the 4.5m wide by 4m high drift with arched roof shown in cross-section in Figure 1. The perimeter (walls and roof) is to be excavated using smoothwall blasting techniques. The following steps are used:

Step 1: Design the buffer 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 key to the approach is the assignment of a "practical" radius of damage (R_d) to each blasthole/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.

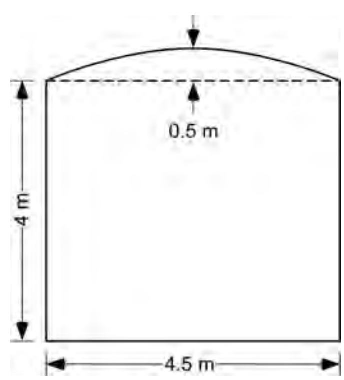

Figure 1. Example drift shape

As can be seen in Figure 2, the practical damage zone consists of both crushing and cracking components.

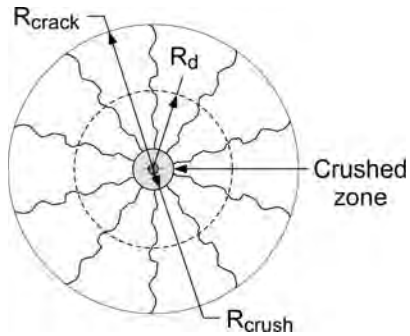

Figure 2. Diagrammatic representation of the crushed, cracked and damaged zones surrounding a blast hole

At this point in the discussion, it will be assumed that R_d = 0.5m for the fully-coupled buffer row holes. The technique used for calculating R_d will be presented later in the paper. To start the design, parallel shells located at distances of R_d and $2R_d$ inside the desired contour are drawn as shown in Figure 3.

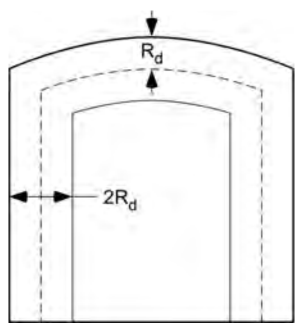

Figure 3. First step in the buffer row design

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.

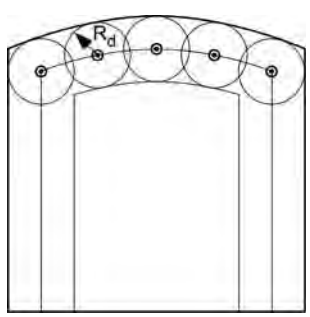

Figure 4. Initial placement of the buffer row roof holes

As can be seen, there is a considerable amount of “un-touched” rock 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.

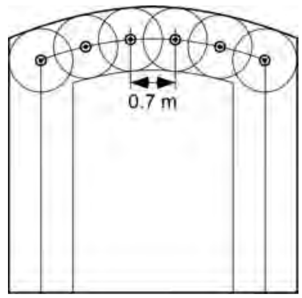

Figure 5. Final placement of the buffer row roof holes

In this particular case, the distance between the buffer row holes is 1.4 R_d or 0.70m. In Figure 6, the buffer row has been added to the walls.

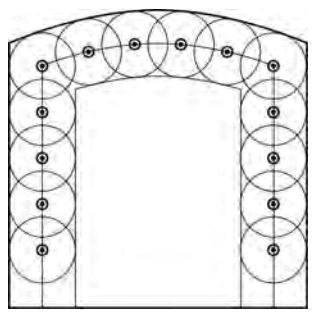

Figure 6. Addition of the buffer row wall holes

In step 2, 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. The remaining holes along the roof are placed to remove the remaining rock cusp between adjacent damage circles. The design basis for the contour row is presented later in the paper. As can be seen, the amount of rock associated with each hole (the burden) is rather small (Figure 7).

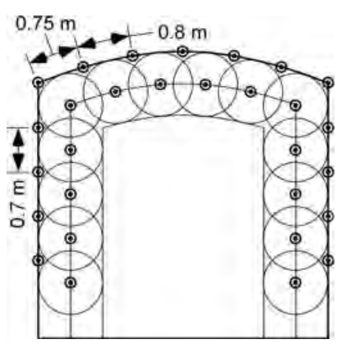

Figure 7. Addition of the contour holes

In step 3, the lifters are added. They have extra work to do both working against gravity and against the weight of the overlying pile, so, even if they are the same diameter as the other holes in the round, they are often charged with a more energetic explosive. In this particular case, it is assumed that the associated damage radius is 0.7m. A lifter hole is placed in each corner and the remaining holes are positioned to cover the remaining distance. To improve floor evenness, the circles are overlapped (see Figure 8).

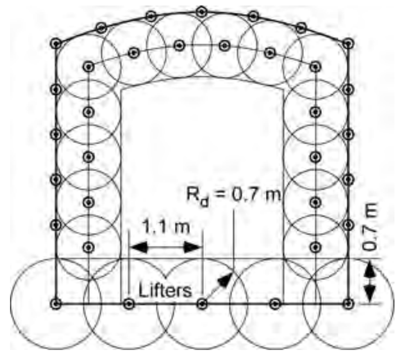

Figure 8. Lifters added to the design

In step 4, a four-quadrangle cut with a final side dimension of 1.4m has been selected for use. A single, large diameter, uncharged hole provides the initial free surface. The numbers refer to the number of the half second delay (No. 1 is 0.5 s delay, No. 2 is 1.0 s delay, etc.) being used. The cut has been superimposed on the design in Figure 9.

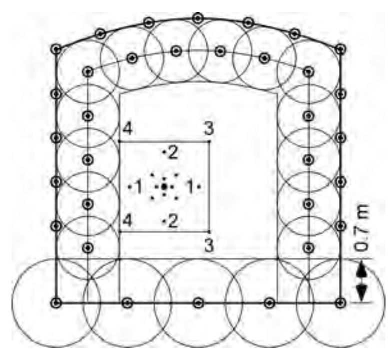

Figure 9. Addition of the cut to the design

Finally, additional “stoping” holes are added to fully cover the face. The final result is shown in Figure 10.

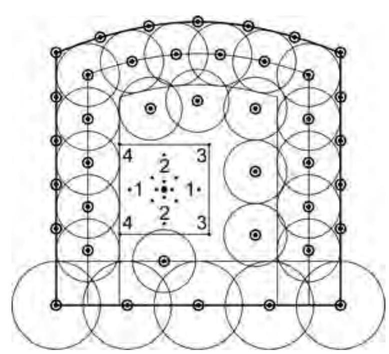

Figure 10. Completion of the design with the addition of stoping holes

Figure 11 shows one possible design for the 4.5m x 4.0m drift provided by Holmberg (1982) using the burden-spacing equations developed by Langefors and Kihlström (1963).

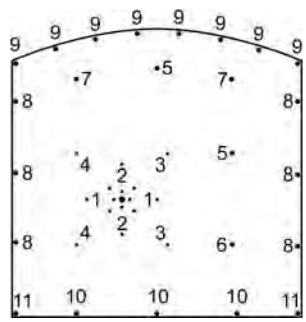

Figure 11. A design provided by Holmberg (1982)

In Figure 12, the damage radius circles have been superimposed on several of the holes. The gaps in the coverage are clearly seen.

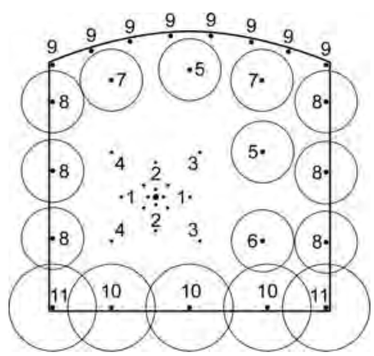

Figure 12. Superposition of influence circles on the Holmberg (1982) design

The total number of holes in the Holmberg (1982) design (excluding the cut) is 24 whereas in the gas pressure-based design it is 40. The latter will clearly require more time to drill and to charge although with the application of modern technology this forms a relatively small part of the total drifting cycle. It should be pointed out that the designs shown in Figures 10 and 12 are not directly comparable since Holmberg (1982) did not include "smoothwalling" of the walls.

The gas pressure-based approach is very logical and easy to apply, providing one has (1) a reasonable technique for estimating the damage radius associated with a particular hole – charge combination and (2) a "smoothwall" design procedure.

The remainder of this paper will focus on one gas pressure-based approach for selecting the required values of R_d and some design guidelines for the "smoothwall" row of holes. Before this can be done, there must be a procedure for pressure calculation. This is the subject of the next section.

3 Calculation of blasthole wall pressure

3.1 Introduction

An important assumption in this approach is that the damage zone radius is dependent upon the blasthole wall pressure. The first step is the calculation of the explosion pressure. Once this has been determined, one needs to obtain the pressure applied to the wall of the borehole. For fully coupled charges (the explosive entirely fills the hole cross-section), the wall pressure is just the explosion pressure. When using de-coupled charges, the explosive gases must expand to fill the cross-section with an accompanying decrease in pressure. The calculation of the borehole wall pressure for both cases is described in this section.

3.2 Explosion pressure

There are several techniques for obtaining the explosion pressure for the explosive(s) of interest. The simplest of these is to obtain the value directly from the explosive manufacturers. They often provide the detonation pressure on the specification sheets. This generally has been calculated using the relationship

$$P_d = \frac{\rho_e D^2}{4} \tag{1}$$

Where:

P_d = detonation pressure (MPa)
ρ_e = explosive density (kg/m^3)
D = detonation velocity (km/s)

It is not known why the manufacturers provide the detonation pressure since it is not the same as the explosion pressure (P_e) required in blast design calculations. For practical purposes, it has been found that P_e can be approximated using the expression

$$P_e = \frac{1}{2} P_d \tag{2}$$

Or

$$P_e = \frac{\rho_e D^2}{8} \tag{3}$$

The required explosive density and detonation velocity parameters are normally supplied by explosive manufacturers.

For ANFO with a density ρ_e = 820 kg/m^3 and detonation velocity D = 3900 m/s, the explosion pressure is

$$P_e = \frac{820\,(3.9)^2}{8} = 1560\ MPa$$

This pressure is oriented radially outward from the wall of the explosive charge. If the explosive charge was in intimate contact with the hole wall (fully coupled conditions), this would be the wall pressure P_w used in further calculations.

As is often pointed out by explosive specialists, this approach to calculating the explosion pressure is very simplified and other factors need to be taken into account (confinement, diameter, ideal vs non-ideal explosives, etc). That is true, but then someone must supply the information in a form readily available and applicable by users of explosives. Until that occurs, the above approach is recommended for use.

3.3 The pressure on the borehole wall for de-coupled charges

Generally, the explosion pressures as calculated in the previous section are much higher than the compressive strength of the rock being blasted. Although this is desired when fracturing the rock in the interior part of the drift round, it is not true for the perimeter holes when perimeter control blasting is to be used.

The first design requirement for the perimeter holes is to keep the borehole wall pressure less than or equal to the compressive strength. This is normally accomplished by using de-coupled charges. The explosion pressure calculated in the previous section applies at the outer boundary of the charge. To reach the borehole wall, the explosive gases must expand and, in the process, the pressure decreases.

For ideal gases (gases at atmospheric pressure and room temperature), the standard expression relating pressure, volume and temperature is

$$P\nu = nRT \qquad (4)$$

Where

P = pressure
υ = specific volume
n = number of moles of gas present
T = absolute temperature
R = the Universal Gas Constant

Assuming isothermal expansion, one writes

$$P_w \nu_h = P_e \nu_e \qquad (5)$$

Where

P_e = explosion pressure
υ_e = specific volume of the explosive
P_w = wall pressure
υ_h = specific volume of the explosive gasses filling the hole

Assuming that

$$\rho_e = 0.82 \text{ g/cm}^3$$

the specific volume of the explosive would be

$$\upsilon_e = 1/\rho_e = 1/0.82 = 1.22 \text{ cm}^3\text{/g} \qquad (6)$$

For the case when

d_h = hole diameter = 54 mm
d_e = explosive diameter = 30 mm

the specific volume of the gasses filling the hole is given by

$$\nu_h = \left(\frac{d_h}{d_e}\right)^2 \nu_e = \left(\frac{54}{30}\right)^2 1.22 = 3.95\ cm^3/g \qquad (7)$$

Assuming the explosive to be ANFO (P_e = 1560 MPa), the wall pressure calculated using equation (5) is

$$P_w = P_e\left(\frac{\nu_e}{\nu_h}\right) = 1560\left(\frac{1.22}{3.95}\right) = 482\ MPa \qquad (8)$$

However, one cannot apply this approach for the very high pressure, high temperature explosive gas conditions involved here. To account for non-ideal gas behavior, the co-volume correction term introduced by Cook (1956, 1958) and first applied by Hino (1959) for perimeter control applications will be used. It forms part of the Utah/NIOSH pre-splitting approach described in a recent paper by Hustrulid (2007). The relationship relating pressure, volume and temperature in a consistent set of units is

$$P\ (\nu - \alpha) = nRT \qquad (9)$$

Where

P = pressure (MPa)
υ = specific volume (cm^3/g)
α = co-volume (cm^3/g)
n = moles/g
R = universal gas constant = 8.314474 cm^3 – MPa / (mole – ^oK)
T = temperature (^oK)

Assuming, as before, that the expansion of the gases in the borehole occurs isothermally, one can write

$$P_w(\nu_h - \alpha_h) = P_e(\nu_e - \alpha_e) \qquad (10)$$

Hustrulid (2007) has shown that the expression

$$\alpha = 1.1\ e^{-0.473/\upsilon} \qquad (11)$$

may be used to relate the co-volume and the specific volume. Substituting the appropriate values into equations (12) and (13)

$$\alpha_h = 1.1e^{-0.473\nu_h} \qquad (12)$$

$$\alpha_e = 1.1e^{-0.473\nu_e} \qquad (13)$$

one finds that

$$\alpha_h = 1.1\ e^{-0.473(3.95)} = 0.17$$

$$\alpha_e = 1.1e^{-0.473(1.22)} = 0.56$$

The wall pressure with the co-volume correction becomes

$$P_w = P_e\left(\frac{(\nu_e - \alpha_e)}{(\nu_h - \alpha_h)}\right) = 1560\left(\frac{1.22-0.56}{3.95-0.17}\right) = 272\ MPa$$

As can be seen, the co-volume correction has a major effect on the calculated wall pressure. If the compressive strength (σ_c) of the rock mass is, for example,

$$\sigma_c = 200 \text{ MPa}$$

one would expect to see crushing around the hole. If this is not permissible, one would consider changing the explosive, changing the hole diameter or changing the charge diameter. The calculation procedure described above would be repeated until the desired wall pressure is achieved.

4 Buffer row design based on damage radius

4.1 Introduction

With the detonation of an explosive charge in a borehole, a shock wave is generated in the surrounding rock mass. Depending upon the explosive and the rock mass, somewhere in the range of 5-15% of the total explosive energy goes into shock energy. In spite of the relatively low amount of energy involved, the shock wave is thought to be responsible for most, if not all, of the new crack generation. The remaining energy is contained in the high pressure gases. Upon expanding, these gases produce extension in the old and new cracks and eventual displacement of the burden. The overall damage to the rock surrounding the borehole involves both of these effects. Previous investigators have generally focused on one or the other of these producers of rock damage. Since both contribute to the overall damage, both must be included. This section presents a pragmatic first approach to predicting the damage radius based on an integration of the two effects.

4.2 The modified Ash approach

For open pit mining applications, Ash (1963) has suggested the following relationship between the burden (*B*) and the hole diameter (d_h) for fully-coupled explosives.

$$B = K_B\, d_h \tag{14}$$

Where

K_B = constant

Ash (1963) found that when using ANFO (with a density of 0.82 g/cm^3) to blast average rock (density of 2.65 g/cm^3), the use of

$$K_B = 25$$

provided very satisfactory results. By way of an example, if the hole diameter (d_h) is 0.10m, then the appropriate burden would be

$$B = 25\ (0.10) = 2.5m$$

When using explosives of greater specific energy (energy/volume) than ANFO to blast the average rock, one would use

$$K_B = 30$$

Or even

$$K_B = 35$$

For surface blast design, Hustrulid (1999) recommended that designers think in terms of cylindrical fragmented plugs of rock surrounding each hole. For the "just-touching" scenario, the radius of influence (R) of the plug is equal to B/2. Equation (14) can be written as

$$2R = K_B\, d_h$$

But since

$$d_h = 2\, r_h$$

Then

$$R = K_B\, r_h \tag{15}$$

For the present application, it will be <u>assumed</u> that the damage radius is equal to the radius of influence.

$$R_d = R \tag{16}$$

And hence

$$R_d = K_B\, r_h \tag{17}$$

Or

$$R_d / r_h = K_B \tag{18}$$

It is important to have an expression for R_d that can be applied to different explosive – rock combinations. Based upon energy considerations, Hustrulid (1999) has shown that

$$R_d / r_h = 25 \sqrt{\frac{\rho_e\ s_{ANFO}}{\rho_{ANFO}}}\ \sqrt{\frac{2.65}{\rho_{rock}}} \tag{19}$$

Where

ρ_e = explosive density (g/cm^3)
ρ_{rock} = rock density (g/cm^3)
s_{ANFO} = weight strength with respect to ANFO
ρ_{ANFO} = ANFO density (g/cm^3)

If ANFO of density 0.82 g/cm^3 (s_{ANFO} = 1) is used in 38mm diameter holes in granite of density 2.65 g/cm^3, one finds that the damage radius is

$$R_d = 25\ (0.019) = 0.48\text{m}$$

If an emulsion with the following properties

$$\rho_e = 1.15 \text{ g/cm}^3$$
$$s_{ANFO} = 0.88$$

is used instead, then

$$R_d / r_h = 25 \sqrt{\frac{\rho_e \; s_{ANFO}}{\rho_{ANFO}}} \sqrt{\frac{2.65}{\rho_{rock}}} = 25 \sqrt{\frac{1.15(0.88)}{0.82}} \sqrt{\frac{2.65}{2.65}} = 27.8$$

The corresponding damage radius would be

$$R_d = 27.8 \, (0.019) = 0.53 \text{ m}$$

If the comparison basis is the explosion pressure rather than the explosive energy, one can write

$$R_d \; / \, r_h = 25 \sqrt{\frac{P_{e\,Exp}}{P_{e\,ANFO}}} \sqrt{\frac{2.65}{\rho_{rock}}} \tag{20}$$

Where

$P_{e\,Exp}$ = explosion pressure for the explosive
$P_{e\,ANFO}$ = explosion pressure for ANFO

The specification sheets provided by an explosive manufacturer indicate that

$$P_{e\,ANFO} = 1550 \text{ MPa}$$
$$P_{emulsion} = P_{e\,EXP} = 3150 \text{ MPa}$$

One finds that

$$R_d \; / \, r_h = 25 \sqrt{\frac{P_{e\,Exp}}{P_{e\,ANFO}}} \sqrt{\frac{2.65}{\rho_{rock}}} = 25 \sqrt{\frac{3150}{1550}} \sqrt{\frac{2.65}{2.65}} = 35.6$$

For the 38 mm diameter hole filled with emulsion, the damage radius would be

$$R_d = 35.6 \, (0.019) = 0.68 \text{ m}$$

This pressure-based approach appears to provide results more in keeping with those of Ash (1963) and thus is to be recommended for use in underground blast design. As indicated, this is a simple and easy to understand technique to estimate the damage radius. It involves the use of readily available explosive and material properties.

4.7 Preliminary recommendations for buffer row design

Although several other approaches are available for estimating the damage radius R_d associated with fully coupled charges, it is recommended that the modified Ash approach based on explosion pressure (equation (20)) be applied.

5 Design of the contour row

5.1 Introduction

As was indicated earlier, the primary task of the contour holes is simply to smooth the surface produced by the buffer row of holes. The actual “burden” is small. It is not, as is often stated, the distance to the buffer row since the fragmentation of the inter-lying rock is largely the responsibility of the buffer holes. The spacing-burden ratio for the contour row of holes is high. In this section, a rule for the spacing of the contour line of holes will be given.

5.2 Spacing based on the force equilibrium approach

Sanden (1974) applied the force-equilibrium approach in developing a hole spacing (S) relationship for pre-splitting. The same approach will be applied to this contour blasting application. Consider the radial stress acting on the boundary of hole of radius ‘r_h’ as shown in Figure 13.

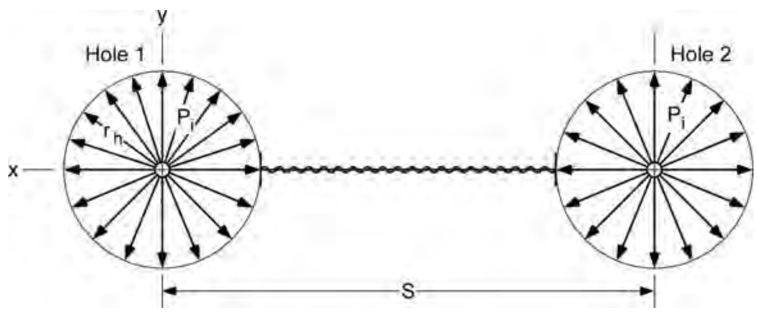

Figure 13. Diagrammatic representation of two holes in the contour row

Only the right hand side of the left hole will be considered. The incremental force (dF_i) in the radial direction produced by the pressure P_w acting over a small incremental area $r_h\, d\theta_i$ on the circumference of a hole of unit length is given by

$$dF_i = -P_w\ r_h\ d\theta_i \tag{21}$$

The component of the force acting in y - direction, normal to the line connecting the hole center lines is given by

$$dF_{yi} = -P_w\ r_h\ \sin\theta_i\ d\theta_i \tag{22}$$

The total force in the y-direction is obtained by adding up (integrating) these contributions. This may be expressed as

$$F_y = -\int_0^{\pi/2} r_h\ P_w \sin\theta\ d\theta \tag{23}$$

The result is

$$F_y = r_h\ P_w \tag{24}$$

Since there are two contributing holes, the total driving force is

$$F_D = 2\, r_h\, P_w \tag{25}$$

The resisting force, F_R, is

$$F_R = \sigma_t \left(S - 2\, r_h\right) \tag{26}$$

Where:

S = hole spacing
σ_t = tensile strength of the rock mass

Equating the driving and resisting forces one finds that

$$S = 2r_h \left(\frac{P_w + \sigma_t}{\sigma_t}\right) \tag{27}$$

If the wall pressure is designed to be equal to the compressive strength (σ_c), equation (27) then becomes

$$S = d_h \left(\frac{\sigma_c + \sigma_t}{\sigma_t}\right) = d_h \left(\frac{\sigma_c}{\sigma_t} + 1\right) \cong d_h \left(\frac{\sigma_c}{\sigma_t}\right) \tag{28}$$

By knowing or estimating the compressive strength/tensile strength ratio one can obtain a maximum value for the perimeter row hole spacing. As was indicated earlier, the spacing of the perimeter holes is coordinated with the buffer row spacing. In actual practice, one would compare the latter value with that given by equation (28). If it is greater, than design adjustments would need to be made.

5.4 Preliminary contour row design recommendations

The following steps are followed in the contour row design:

1. The compressive strength of the rock is determined. It provides an upper limit for the borehole wall pressure.
2. For the chosen perimeter hole diameter, borehole wall pressures are calculated using the co-volume approach described in this paper for the candidate explosive products. The best product with respect to the pressure limit is selected. Alternatively, for a particular explosive product, one can select the required borehole diameter.
3. The maximum borehole spacing is calculated using equation (28). This value is compared to the spacing determined by the buffer row design. If the latter value is greater than that determined from equation (28), design adjustments are made.

6. Concluding remarks

This paper has presented a gas pressure-based approach to drift round design. It is logical and easy to apply. The key to the practical application of this design approach is the ability to estimate the radius of damage surrounding fully-coupled buffer row blast holes. The recommended approach is based on the early work of Ash (1963). Once the buffer row holes have been designed, the contour holes are placed to smooth the excavation surface. The contour row charge concentration is selected based upon keeping the borehole wall pressure at or below the compressive strength of the rock. Contour row hole spacing is based upon the guidelines provided by Sanden (1974) as well as practical considerations (removing the rock cusps left in the buffer row design). This damage radius approach may be applied both to the lifter row design and to the stoping hole design. The cut design is largely based on geometrical considerations and has not been addressed in this paper.

Disclaimer

The findings and conclusions in this paper are those of the authors and do not necessarily represent the views of the National Institute for Occupational Safety and Health.

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