A Practical, Yet Technically Sound, Design Procedure for Pre-Split Blasts

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ABSTRACT

Pre-split blasting is a primary technique for creating slopes with a minimum amount of unwanted damage to the remaining rock mass. Rock slopes created in such a way have both safety-related and economic advantages. The applications extend from road cuts to rock quarries to large open pit mines. For example, as deep open pit mines approach the end of their operating lives, slope steepening is one option to be considered. To be able to safely achieve and maintain steepened final slopes, good blast design techniques must be available and applied together with close operational control. Pre-split blasting is an important element in this slope creation process.

The effective use of de-coupled charges in the perimeter blast holes is an important ingredient for achieving the desired result. The present procedure for selecting the hole size, hole spacing and explosive combination used for pre-split blasting is largely based on work done more than 30 years ago by Sanden (1974). It assumes that the explosive gases expand adiabatically within the annulus separating the charge and the hole wall. Recent work by Meng (2005) has shown that the expansion in the annulus occurs isothermally rather than adiabatically. This affirms the writings of Hino (1959) and Johansson and Persson (1970). The result is that the basis for the currently used pre-split design procedure must be re-assessed and appropriately modified.

The paper examines in some detail hole spacing, explosion pressure, borehole wall pressure and rock properties with regard to pre-split design. A practical isothermal expansion-based approach for calculating the pressure on the borehole wall is presented.

A simple approach to pre-split design is then presented based upon using commonly available explosive and rock properties. The practical application of the new technique is demonstrated through an example.

1. INTRODUCTION

Pre-split blasting is the primary technique for creating slopes with a minimum amount of unwanted damage to the remaining rock mass. Rock slopes created in such a way have often have both safety-related and economic advantages. The applications extend from road cuts to rock quarries to large open pit mines. For example, as deep open pit mines approach the end of their operating lives, slope steepening is one option to be considered. To be able to safely achieve and maintain steepened final slopes, good blast design techniques must be available and applied together with close operational control. Pre-split blasting is an important element in this slope creation process.

Perhaps the earliest and most famous application of pre-split blasting was as part of the Niagara Power Project in New York (1961, 1962, 1973). It involved the use of carefully drilled, closely spaced holes, de-coupled charges and detonating cord initiation. Although the actual designs appear to have been developed using the cut-and-try approach, a quantitative analysis based on the explosively produced waves was also presented. The required borehole wall pressure was calculated assuming isothermal expansion in the annulus between explosive and the wall. The results, as can be seen in Figure 1, were very, very impressive. In 1974, Sanden (1974) presented the design procedure which is that generally in use today. Based upon the action of the gas pressure rather than the shock wave, this procedure assumes that the explosive gases within the annulus separating the charge and the hole wall expand adiabatically. Recent experimental work by Meng (2005) has affirmed the ideas of Hino (1959) and Johansson and Persson (1970) who suggested that the expansion in the annulus occurs isothermally rather than adiabatically. The result is that the basis for the currently used pre-split design procedure must be re-assessed and appropriately modified. This paper presents a practical, but soundly based, pre-split design procedure based on the isothermal expansion of the explosive gases in the annulus. A practical example is included to guide the user through the steps.

2. OVERVIEW OF THE PRE-SPLITTING CONCEPT

Figure 2 shows plan and section views from the Niagara Power Project (Paine et al, 1961, 1962), one of the first large scale applications of pre-splitting. In this case, the holes forming the pre-split row are 2-1/2" in diameter and are spaced on 18" to 24" centers depending on the conditions. The holes were charged using 1-1/4" diameter by 4" long sections of 40% gelatin extra cartridges on 12" centers taped to a length of reinforced detonating cord. Each hole was stemmed completely with minus 3/8" clean stone chips. The pre-split rows were blasted prior to drilling the pattern. Since those early successful beginnings, the techniques have changed somewhat, however the basic principles remain the same.

Figure 3 summarizes the design parameters. For the hole, one needs to specify the diameter, length, whether stemming is to be used, and if so, the type and location. For the explosive, one needs to specify the type, the volume, the distribution in the hole, and the initiation. In practice, the explosive characteristics commonly given are the diameter, density and detonation velocity. It is of importance to know the actual charge diameter rather than the diameter of the product. This is particularly true for detonating cord. Various techniques are used to produce borehole wall pressures that are less than the explosion pressure. These include de-coupled charges and air-decking. It is of importance to have the holes to initiate as closely to instantaneously as possible to maximize interaction both of the stress waves and the gas pressure.

Figure 4 is a diagrammatic representation of pre-splitting design performance. It is desired to prevent crushing around the hole. This means that the wall pressure should be less than the compressive strength of the rock. For a crack to be generated between adjacent holes, the wall pressure must be greater than the tensile strength of the rock. Since the borehole pressure is greater considerably greater than the tensile strength, short radial cracks will form around the boreholes. By increasing borehole pressure, the spacing between adjacent can generally be increased. This leads to reduced drilling costs but to the increased development of the radial cracks. Superior drilling procedures and techniques must be applied if pre-splitting is to be successful.

The primary functions of the pre-split crack are first to clearly define the excavation boundary, and second, to isolate the rock wall from certain adverse effects of the main blast. The benefits are:

1. The presence of the pre-split limits crack extension into the wall from the main blast
2. The presence of the pre-split limits to some degree the passage of the stress wave from the main blast into the wall rock
3. The pre-split provides an escape way for explosive gases from the main blast preventing them from entering and extending existing cracks in the rock wall.

In the future, there will be an ever-increasing emphasis on careful rock excavation for a variety of reasons and this technique should be carefully considered.

3. THE SPACING FORMULA

3.1 The Niagara Power Project Formula

The Niagara spacing equation (Paine, et al (1961,1962), and Griffin (1973))is based upon the shock wave generated by the blast. The peak radial stress (P) of the cylindrical stress wave generated by the explosive may be expressed as (Jeffreys, 1950)

$$P = \frac{1}{2} P_w \sqrt{\frac{a}{r}} \tag{1}$$

where

P = peak amplitude of the radial stress wave component
P_w = pressure on the borehole wall
a = hole radius
r = distance measured from the hole center

The $(a/r)^{1/2}$ term represents the geometric spreading effect. To account for inelastic effects, an exponential decay factor was added

$$P = \frac{1}{2} P_w e^{k_1 (r/a)} \sqrt{\frac{a}{r}} \tag{2}$$

where

k_1 = attenuation constant

At the midpoint along the line joining the holes, the radial pressure would

$$P = \frac{1}{2} P_w e^{k_1 (S/d_h)} \sqrt{\frac{d_h}{S}} \tag{3}$$

where

S = hole spacing

d_h = hole diameter

Assuming that adjacent holes initiate at exactly the same time, the total radial pressure at the midpoint would be

$$P = P_w \, e^{k_1 (S/d_h)} \sqrt{\frac{d_h}{S}} \tag{4}$$

By applying Poisson's ratio (υ_1) the stress (S_T) at the midpoint oriented in the tangential direction was given as

$$S_T = - \nu_1 \, P_w \, e^{k_1 (S/d_h)} \sqrt{\frac{d_h}{S}} \tag{5}$$

For a pre-split to occur, the value of S_T must be greater than or equal to the tensile strength

$$S_T \geq -T \tag{6}$$

where

T = tensile strength

One can then write

$$\nu_1 \, P_w \, e^{k_1 (S/d_h)} \sqrt{\frac{d_h}{S}} \geq T \tag{7}$$

Knowing the values of P_w ,υ_1, k_1, and the hole diameter d_h, one can solve for the spacing S.

3.2 The Canadian Approach

3.2.1 Introduction

The basic development of the formula was done by Sanden (1974 and reported in his M.S. thesis prepared under the direction of Dr. Peter Calder at Queen's University. As is often the case for such theses, the results have not been widely disseminated. Fortunately, the theoretical portion of his thesis did appear in somewhat modified form in a paper prepared by Chiappetta (1982) "Pre-splitting Theory and Applications" and in "Perimeter Blasting" which formed Chapter 7 of the CANMET Pit Slope Manual (CANMET, 1977). Chapter 7, written by staff members of the Mining Engineering Department of Queen's University, was prepared under the direction of Dr. Calder. In this paper, it will be referred to as the Canadian Approach since it originated in Canada. Perhaps it should be more properly called the "Current Approach" or the "World Approach" due to its widespread use. The two approaches used, both of which yield essentially the same final result, will be described here.

3.2.2 Pressured Circular Hole in a Thick-Walled Cylinder

Sanden (1974) assumed that the pre-splitting was caused by the gas pressure rather than the shock wave. He performed two different analyses. The first was based upon the elastic solution for the stresses developed in a thick-walled cylinder under internal pressure. The second based on force equilibrium is the subject of the following section.

The general equations for the radial stress, σ_r, and tangential stress, σ_t, developed in a thick-walled cylinder with inner radius 'a' and outer radius 'b' when a radial pressure, P_i, is applied to the inner wall and a radial pressure, P_o, is applied to the outer wall are given below (Obert and Duvall, 1967):

$$\sigma_r = \frac{a^2b^2\,(P_o - P_i)}{(b^2 - a^2)\,r^2} + \frac{P_i\,a^2 - P_o\,b^2}{b^2 - a^2} \tag{8}$$

$$\sigma_t = -\frac{a^2b^2\,(P_o - P_i)}{(b^2 - a^2)\,r^2} + \frac{P_i\,a^2 - P_o\,b^2}{b^2 - a^2} \tag{9}$$

The sign convention used for the stresses is tension is positive and compression is negative. For the case when $P_o = 0$, the equations become

$$\sigma_r = -\frac{a^2\,b^2\,P_i}{(b^2 - a^2)\,r^2} + \frac{P_i\,a^2}{b^2 - a^2} \tag{10}$$

$$\sigma_t = \frac{a^2\,b^2\,P_i)}{(b^2 - a^2)\,r^2} + \frac{P_i\,a^2}{b^2 - a^2} \tag{11}$$

Letting the outer diameter, b, become very large (approach infinity), equations (10) and (11) reduce to

$$\sigma_r = -\frac{a^2\,P_i}{r^2} \tag{12}$$

$$\sigma_t = \frac{a^2\,P_i}{r^2} \tag{13}$$

When the radius, r, is equal to 'a', the radial stress is

$$\sigma_r = -P_i \tag{14}$$

and the tangential stress becomes

$$\sigma_t = P_i \tag{15}$$

Figure 5 is a diagrammatic representation of the tangential stress distribution along a line passing through the center of the cylinder. At the boundary of the hole, the stress magnitude is P_i and at infinity it is 0. The total force F_t acting in the direction normal to this line can be obtained by integrating equation (13) with respect to distance r and then evaluating the integral for r extending from radius 'a' to infinity.

$$F_t = a^2\,P_t \int_a^{\infty} \frac{dr}{r^2} = -a^2\,\pi\,\left[\frac{1}{r}\right]_{r=a}^{r\to\infty} \tag{16}$$

The result is

$$F_t = -a^2\,P_i\,[0 - 1/a] = a\,P_i \tag{17}$$

In pre-splitting, a series of closely spaced holes are drilled along the pre-splitting line. The center-to-center separation of the holes (the hole spacing) is S. The initiation of the explosives is planned so that all holes detonate as closely as possible to the same instant in time. This means that all of the holes would be concurrently pressurized to P_i. The result would be that the tangential stress fields induced by adjacent holes would overlap. This is shown diagrammatically in Figure 6.

Sanden (1974) has approached this situation by integrating equation (13) over the limits of 'a' to S.

$$F_t = a^2 P_i \int_a^S \frac{dr}{r^2} \tag{18}$$

He obtained

$$F_t = - a^2 P_i [1/S - 1/a] = a P_i [1 - a/S] \tag{19}$$

Since the spacing S is normally many times the hole diameter, equation (19) may be approximated by

$$F_t \cong a P_i \tag{20}$$

Due to the fact that there are two holes influencing the rock web, the total tensile force, F_D, is given by

$$F_D \cong 2 a P_i \tag{21}$$

The total resisting force, F_R, is given by

$$F_R = (S - 2 a) T \tag{22}$$

where

T = tensile strength
S - 2a = length of the rock web separating the holes

At equilibrium, the driving force will just be matched by the resisting force. Hence,

$$(S - 2a) T \cong 2 a P_i \tag{23}$$

Replacing the symbols by those used in blasting one finds that

$$S \cong 2 r_h \left(\frac{P_w + T}{T}\right) \tag{24}$$

This is the equation commonly found in pre-splitting references.

3.2.3 Force Equilibrium Approach

Although not widely known, Sanden(1974) also applied the force-equilibrium approach in developing a hole spacing relationship. Consider the radial stress acting on the boundary of the hole as shown in Figure 7. For consistency with the tangential force evaluation, only the right hand side of the hole will be considered. The incremental force (dF_i) in the 'r' direction produced by the pressure P_i acting over a small incremental area $ad\theta$ on the circumference of the hole is given by

$$dF_i = - P_i \, a \, d\theta \tag{25}$$

Here it is of interest to find the component of the force acting in y-direction, normal to the line connecting the hole center lines. This is given by

$$dF_i = - P_i \, a \, \sin\theta \, d\theta \tag{26}$$

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

$$F_y = -\int_0^{\pi/2} a \, P_i \, \sin\theta \, d\theta \tag{27}$$

The result is

$$F_y = a \, P_i \tag{28}$$

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

$$F_D = 2 \, a \, P_i \tag{29}$$

The resisting force is the same that was found in the previous section. Equating the driving and resisting forces and replacing the symbols by those used in blasting one finds that

$$S = 2 \, r_h \frac{(P_w + T)}{T} \tag{30}$$

The approximation symbol is no longer required. Furthermore, the force equilibrium approach does not require elastic conditions to prevail in the pre-splitting situation. This is the probable general situation. If the wall pressure is designed to be the compressive strength C, then equation (30) becomes

$$S = d_h \frac{C + T)}{T} = d_h \left(\frac{C}{T} + 1\right) \tag{30a}$$

By knowing or estimating the compressive strength/tensile strength ratio one can obtain a first estimate of the hole spacing.

Spacing equation (30) based on force equilibrium is that used in the Utah/NIOSH approach described later. To be able to the spacing equation, one must know the borehole wall pressure and the rock strength. These factors will now be examined.

4. THE EXPLOSION PRESSURE

For practical purposes it has been found that the explosion pressure (P_e) can be approximated using the expression

$$P_e = 1/8\ \rho_e\ D^2 \qquad (31)$$

where:

P_e = explosion pressure (MPa)
ρ_e = explosive density (kg/m^3)
D = detonation velocity (km/sec)

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

For ANFO with a density ρ_e = 800 kg/m^3 and detonation velocity D = 4530 m/sec, the explosion pressure is

$$P_e = 800\ (4.530)^2/8 = 2052 \text{ MPa} = 297635 \text{ psi}$$

This pressure would be 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 the hole spacing equation.

There is a problem, however, when trying to use equation (31) to determine the explosion pressure for detonating cord, of the most commonly used pre-splitting explosives. Although detonating cord has been in use for many, many years, it is impossible to obtain the required PETN density and core diameter data from the manufacturers. Instead, they provide the cord strength expressed in terms of grains/ft or g/m and a nominal detonation velocity. Thus one is unable to apply equation (31) and calculate the explosion pressure. Without knowing the equivalent core diameter, one has a problem in the next step of the process, calculating the wall pressure.

One solution, although not a very satisfactory solution, is to disassemble cords of different strengths and determine the volume and weight of the collected PETN. This was done in recent studies conducted at the University of Utah (Meng, 2004). Using this crude approach, the density was determined to be approximately 1.4 g/cm^3 independent of the cord strength. This agrees with a value provided by Sanchidrian and Patino (2002). Using the cord strength provided by the manufacturer, for example 50 grains/ft, and the determined density, the average core diameter can be determined. In the case of a 50 grain/ft cord, using the conversion factors of 1 grain = 0.0648 grams and 3.28 ft/m, one finds that the cord strength metric equivalent is

$$50 \text{ grains/ft} = 50(0.0648)\ (3.281) = 10.63 \text{ g/m}$$

Assuming the core density to be ρ_e = 1.4 g/cm^3, the diameter of the core d_c is calculated using

$$\pi/4\ d_c^{\ 2}\ \rho_e\ (100) = 10.63$$

In this case

$$d_c = 0.3109 \text{ cm} = 0.122 \text{ ins.}$$

The outer diameter of such a cord is about 0.200 ins. Detonation velocity tests performed at the University of Utah (Meng, 2004) on cords of nominal strength 25 grain/ft and 18 grain/ft yielded an average value of D = 7036 m/sec. For these particular cords, the manufacturer (Dyno Nobel, 2005) indicated that the detonation velocity was approximately 7000 m/sec (23000 ft/sec). Applying equation (31), the explosion pressure for the PETN, would be

$$P_e = 1/8\ (1400)\ (7.036)^2 = 8660 \text{ MPa} = 1{,}256{,}000 \text{ psi}$$

Johansson and Persson (1970) have provided the detonation velocity versus density curve for PETN shown in Figure 8 which is based upon the experimental work of Friedrich (1933). Figure 9 presents a similar curve constructed based upon calculations of Cook (1958). If one knows the detonation velocity for a particular cord, something which can easily be supplied by the manufacturers, one can estimate the core density using these curves. Knowing the core density and the detonation velocity, one can calculate the explosion pressure. The core diameter can then be estimated knowing the density and the cord strength.

The University of Utah determined data point of 1.4 g/cm^3 and 7036 m/s for PETN falls nearly exactly on the curves presented in Figures 8 and 9. This indirect approach to determining the PETN density in the detonating cord appears promising. It obviously should be further checked. Although this seems to be a very laborious and perhaps over-complicated process, one will see PETN density values given in the literature ranging from 0.6 to nearly 2.0. To advance pre-split designs based upon the use of detonating cords, there must be a well-accepted way of determining the core density and the diameter. Use of these curves, perhaps, might serve that purpose.

5. THE PRESSURE ON THE BOREHOLE WALL FOR DE-COUPLED HOLES

5.1 Background

Generally, if explosives would be fully coupled (completely fill the hole), the explosion pressures generated would be far too high for pre-splitting applications. As a result, pre-splitting holes are generally not completely filled with explosive. The remaining hole volume, generally in the form of an annulus between the explosive and the hole wall is air-filled. It may, however, be filled with other materials as well. In this paper, the annulus will be considered to be air-filled.

For de-coupled charges, the high pressure, high temperature explosive gases must first expand to contact the hole wall. For ideal gases, the standard expression relating pressure, volume and temperature is

$$P\upsilon = nRT \qquad (32)$$

where

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

For the case of nitroglycerine with a density of 1.6 g/cm^3

$n = 31.95$ moles/kg
$T = 4850$ °K
$R = 0.08207$ l-atm/(mol – °K)

If the temperature of the gas remains constant during the expansion process, one can write

$$P\upsilon = 12717 = K_o = \text{constant} \qquad (33)$$

From this, one can easily develop the pressure-volume curve by choosing a value for υ, for example, and calculating the corresponding value of P. The resulting is shown in Figure 10. If ideal gas laws apply, by knowing the specific volume of the explosive, υ_e, and the volume of the hole υ_h expressed in terms of the specific volume of the explosive, one could use this curve to determine the wall pressure for de-coupled charges. As will be seen, this is the procedure used to calculate wall pressure at the Niagara Power Project.

In actuality, ideal gas laws do not apply and a more complex procedure must be used to obtain the pressure-volume curve. Johansson and Persson (1970) have provided the volume function

$$f(\upsilon) = \frac{\upsilon}{1 + 0.834/\upsilon + 0.435/\upsilon^2 + 0.167/\upsilon^3 + 0.093/\upsilon^4} \tag{34}$$

required for determining the isothermal pressure-volume curve for nitroglycerine. The results of evaluating this curve have been superimposed on Figure 10. The major differences in the curves clearly demonstrates that the simple ideal gas law approach does not apply under high pressure, high temperature conditions. The two curves do agree, as would be expected, for low pressure, low temperature conditions.

Cook (1958) has shown that through the introduction of a co-volume factor, α, one can apply the same ideal gas law approach to high temperature and pressure conditions. The modified ideal gas law equation is written as

$$P\,(\upsilon-\alpha) = nRT \tag{35}$$

Where

P = pressure (atm)
υ= specific volume (l/kg)
α = co-volume (l/kg)
n = moles/kg
R = universal gas constant = 0.08207 l-atm/(mole – ^{o}K)
T = temperature (^{o}K)

It is common to represent the term υ-α by the symbol 'a'.

$$a = \upsilon-\alpha \tag{36}$$

Thus,

$$Pa = nRT \tag{37}$$

Cook (1956) presented the curve shown in Figure 11 relating the co-volume to the volume. Its practical application will be demonstrated in this section.

5.2 The Hino (1959) Approach

Hino (1959) indicates that

> "The effective pressure which can be transmitted to the rock is reduced if an air space exists around the explosive charge."

To account for this "cushion effect" when calculating the wall pressure, Hino applied the co-volume concept introduced by Cook. His approach is described below.
The explosion pressure may be written as

$$P_e\ (\nu_e\ -\ \alpha_e)\ =\ nRT_e \tag{38}$$

where

P_e = explosion pressure
υ_e = specific volume of the explosive = V_e/M
n = number of moles of gas/unit mass
R = Universal Gas Constant
α_e = co-volume
T_e = explosion temperature
M = explosive mass
V_e = explosive volume

Similarly, the borehole wall pressure can be written as

$$P_w\ (\nu_h\ -\ \alpha_h)\ =\ nRT \tag{39}$$

where

P_w = wall pressure
υ_h = volume of the hole expressed in terms of υ_e
T = gas temperature

The expansion is assumed to take place isothermally

$$T = T_e$$

and hence

$$P_w\ (\nu_h\ -\ \alpha_h)\ =\ P_e\ (\nu_e\ -\ \alpha_e) \tag{40}$$

Equation (40) can be re-arranged to yield

$$P_w\ =\ P_e\ (\frac{\nu_e\ -\ \alpha_e}{\nu_h\ -\ \alpha_h}) \tag{41}$$

If the co-volume correction is neglected, one obtains

$$P_w\ =\ P_e\ (\frac{V_e}{V_h}) \tag{42}$$

By this simplification, one obtains the result which would be obtained by applying the ideal gas law.

To facilitate use of equation (41), Hino (1959) fitted an equation to the empirical curve shown in Figure 11. The result was

$$\alpha\ =\ 0.92\ (1\ -\ 1.07\ e^{-1.39\ \nu}) \tag{43}$$

From this, he calculated values for α for the different values of υ given in Table 1.

Table 1. Co-volume α and the specific volume υ. After Hino (1959).

υ (l/kg)	α (l/kg)	υ (l/kg)	α (l/kg)
0.0484	0	1.4	0.779
0.2	0.175	1.6	0.814
0.3	0.271	1.8	0.829
0.4	0.355	2	0.859
0.5	0.429	3	0.904
0.6	0.492	4	0.916
0.7	0.547	5	0.919
0.8	0.596	6	0.92
0.9	0.638	10	0.92
1	0.675	100	0.92
1.2	0.754		

Using his equation he extended his table values far beyond the limits of the data.

Hino (1959) demonstrated the use of the approach by assuming a hole diameter of 42mm and an explosive diameter of 32mm. The given density ρ_e of the explosive was 1g/cm^3 and hence the specific volume of the explosive υ_e is

$$\nu_e = 1/\rho_e = 1$$

The hole/explosive volume ratio, V_R, is

$$V_R = (\frac{42}{30})^2 = 1.96 \cong 2$$

The value of υ_h then becomes

$$\nu_h = V_R \, \nu_e = (2)(1) = 2$$

From Table 1, one finds that

$$\alpha_e = 0.675$$

$$\alpha_h = 0.859$$

Substituting the values into equation (41) yields

$$P_w = P_e\left(\frac{\nu_e - \alpha_e}{\nu_h - \alpha_h}\right) = P_e\left(\frac{1-0.675}{2-0.859}\right) = 0.28P_e$$

If the co-volume correction is neglected, one obtains

$$P_w = P_e\,(\frac{V_e}{V_h}) = P_e\,(\frac{30}{42})^2 = 0.51\ P_e$$

The uncorrected pressure is higher than the co-volume-corrected pressure by nearly a factor of 2. It is unfortunate that the Hino (1959) approach has largely been lost in time due to the fact of its being included in an obscure publication and rather poorly described. The Utah/NIOSH approach will build on the Hino approach.

5.3 The Niagara Power Project Approach

In their quantitative analysis, the Project chose to use Hino's (1959) simplified equation (42) relating the borehole pressure to the explosion pressure.

$$P_w = P_e \left(\frac{V_e}{V_c} \right) \quad (42)$$

By making this simplification, they accepted the Ideal Gas Law approach.

5.4 The Canadian Approach

In his thesis, Sanden (1974) writes

"If the explosive does not completely fill the borehole, the effect of decoupling will bring this borehole pressure down (from the fully coupled state). The condition of gaseous expansion in a borehole from a decoupled charge can be assumed by the ideal gas law as adiabatic."

The basic adiabatic expansion equation is

$$PV^\gamma = \text{constant} \quad (44)$$

where

γ = ratio of the specific heats

Another form of equation (44) which is more convenient to use in this case is

$$P_e V_e^\gamma = P_h V_h^\gamma \quad (45)$$

where

e = subscript referring to the explosive
h = subscript referring to the hole

Equation (45) can be re-written as

$$P_h/P_e = [V_e V_h]^\gamma \quad (46)$$

For a continuous de-coupled charge, equation (46) can be written as

$$P_h/P_e = [r_e/r_h]^{2\gamma} \quad (47)$$

Combining equation (31) and equation (47) one finds that

$$P_h = 1/8\ \rho_e\ D^2\ [r_e/r_h]^{2\gamma} \quad (48)$$

It is possible to reduce the borehole pressure through the use of air-decking as well as through de-coupling. In this case, air-filled spaces are left between the individual de-coupled charges. The hole volume can be written as

$$V_h = \pi r_h^2 \text{ x } L_h \tag{49}$$

and the corresponding explosive volume written as

$$V_e = \pi r_e^2 \text{ x } L_e \tag{50}$$

The coupling ratio (C_r) then becomes

$$C_r = V_e/V_h = (r_e/r_h)^2 (L_e/L_h) \tag{51}$$

Letting

$$w = L_e/L_h \tag{52}$$

one obtains

$$V_e/V_h = (r_e/r_h)^2 (w) = (r_e w^{0.5}/r_h)^2 \tag{53}$$

Substitution of equation (53) into equation (46) one obtains

$$P_h/P_e = [r_e w^{0.5}/r_h)]^{2\gamma} \tag{54}$$

Sanden (1974) assigned the exponent, 2γ, a value of 2.4. This was based on the experimental results obtained by Bauer (1969) and reported in his paper entitled "The Status of Rock Mechanics in Blasting" which was included in the proceedings of the 9th U.S. Symposium on Rock Mechanics. The figure from which he extracted this result is shown as Figure 12. The slope of the pressure/diameter ratio line is indicated as 2.4 and it would therefore seem logical to accept this value for 2γ. That is, if the reported results applied for cylindrical charges. However, as is indicated in the caption, the experiments were conducted using pentolite <u>spheres</u> and not cylindrical charges. Therefore, the assignment of the value of 2.4 for 2γ on this basis is incorrect. In support of his choice of 2.4 for 2γ, Sanden (1974) also referred to typical values of (applying to ideal gases (Cole (1948) and Crawford (1963)). These values which apply for gases at 1 atm and 15°C ranged from about 1.2 to 1.4 and the values of 2γ would range from about 2.4 to 2.8. Today, the Sanden-assigned value of 2.4 for 2γ, or something similar, remains in effect. As indicated, this is not correct.

5.5 The Utah/NIOSH Approach

5.5.1 Background

The Utah/NIOSH approach to pre-split design follows very closely the work of Hino (1959) which, unfortunately, was discovered only recently. Based upon the co-volume concept introduced by Cook(1958),

$$P (\upsilon - \alpha) = nRT \tag{35}$$

Where

P = pressure (atm)
υ = specific volume (l/kg)

α = co-volume (l/kg)
n = moles/kg
R = universal gas constant = 0.08207 l-atm/(mole – °K)
T = temperature (°K)

the present author developed the expression

$$\alpha = 1.1\ e^{-0.473/\upsilon} \tag{55}$$

relating the co-volume and the specific volume. As shown in Figure 13, it describes the empirica curve quite well over the specific volume range of 0.4 l/kg through 1.5 l/kg which covers the currently available commercial explosives. Substituting equation (55) into equation (35), one find that

$$P\ a = P\ (\upsilon - 1.1\ e^{-0.473/\upsilon}) = nRT \tag{56}$$

where

$$a = (\upsilon - 1.1\ e^{-0.473/\upsilon}) \tag{57}$$

For the isothermal (iso-energy) conditions assumed to apply as the explosive gases expand to fill the hole cavity, one can write

$$P\ a = nRT_E = K_o\ \text{(constant)} \tag{58}$$

There are two ways of using this equation to develop the desired isothermal expansion pressure-volume curve.

<u>Technique 1</u>

If, for the given explosive, one knows the number of moles (n) and the explosion temperature (T_E) one can calculate the value of the constant K_o directly. Knowing K_o, one can easily develop the pressure-volume curve by choosing a value for υ, for example, calculating the corresponding values of α and 'a' and then calculating the corresponding value of P (expressed in atmospheres).

<u>Technique 2</u>

It is not very often that the practitioner has access to the values of n and T_E for the explosive used. In this case, the explosion pressure P_e (expressed in MPa) is calculated from the known explosive density and detonation velocity.

5.5.2 The Isothermal Expansion Expression

One begins with the isothermal expansion equation

$$P_e\ a_e = K_o\ \text{(constant)} \tag{59}$$

Where:

P_e = explosion pressure (MPa) calculated using equation (31)
a_e = specific volume corrected by the co-volume term(l/kg)

The term a_e is obtained using

$$a_e = \upsilon_e - 1.1\ e^{-0.473/\upsilon e} \tag{60}$$

where

$$\upsilon_e = 1/\rho_e = \text{specific volume (l/kg)} \quad (61)$$

Knowing the values for P_e and a_e one can calculate the constant K_o.

$$K_o = P_e\, a_e \quad (62)$$

The pressure (P) for any other volume (υ) can be calculated using

$$P = K_o/a \quad (63)$$

Where

$$a = \upsilon - 1.1\, e^{-0.473/\upsilon} \quad (64)$$

Assuming a continuous de-coupled charge, the volume of explosive per unit length of borehole is given by

$$V_e = \pi/4\ d_c^2 \quad (65)$$

Where

d_c = charge diameter

The volume of hole per unit length of borehole is given by

$$V_h = \pi/4\ d_h^2 \quad (66)$$

Hence, the ratio of the volumes (V_R) is

$$V_R = (d_h/d_c)^2 \quad (67)$$

To obtain the pressure acting on the hole wall one expresses υ in terms of υ_e.

$$\upsilon = V_R\, \upsilon_e \quad (68)$$

This value is then substituted into equation (64) and the value of 'a' calculated.

$$a = \upsilon - 1.1\, e^{-0.473/\upsilon} \quad (69)$$

One can now determine the wall pressure (P_w) using

$$P_w = K_o/a \quad (70)$$

There are instances when the explosive is not continuous along the hole length but rather is separated by air-gaps or air-decks. This situation can also be handled by the proposed approach. The explosive volume per unit length of hole now becomes

$$V_e = \pi/4\ d_c^2\, w \quad (71)$$

Where

w = explosive length (m) per unit length of hole

The ratio of the hole volume to the explosive volume becomes

$$V_R = (1/w)\ (d_h/d_e)^2 \qquad (72)$$

The process of calculating the borehole wall pressure continues as before.

5.5.3. Confirmation of the Approach

It is important to demonstrate the correctness of the approach. This will be done using the nitroglycerine values provided by Johansson and Persson (1970). For nitroglycerine with a density of 1.6 g/cm^3

$n = 31.95$ moles/kg
$T_E = 4850\ ^oK$
$R = 0.08207$ l-atm/(mol – ^oK)
$D = 7800$ m/s

Using Technique 1, the constant K_o is

$$P_e\ a_e = 12717 = K_o$$

Since

$$v_e = \frac{1}{\rho_e} = \frac{1}{1.6} = 0.625$$

and

$$a_e = v_e - 1.1\ e^{\frac{-0.473}{v_e}} = 0.625 - 1.1\ e^{\frac{-0.473}{0.625}} = 0.1089$$

one obtains

$$P_e = \frac{12717}{0.1089} = 116{,}780\ atm = 118324\ bars = 118.3\ kbars$$

Applying Technique 2, where the explosion pressure is calculated using

$$P_e = 1/8\ \rho_e\ D^2$$

one finds that

$$P_e = 1/8\ (1600)\ (7.8)^2 = 12168\ MPa = 121.68\ kbars$$

As can be seen there is a slight difference which is of no practical consequence.

The pressure obtained by Johansson and Persson (1970) was 97.76 kbars so the explosion pressure values calculated using these two approximate techniques are somewhat high. The procedure is now continued for other values of υ. Table 2 presents the overall results.

Table 2. Comparison of the Johansson-Persson and the Utah/NIOSH Results

	Pressure (kbars)		
		Utah/NIOSH	
V_h/V_e	Johansson/Persson	Technique 1	Technique 2
1	97.76	118.33	121.28
2	21.33	25.96	26.68
4	7.30	8.10	8.34
8	3.06	3.22	3.34
16	1.40	1.44	1.49
32	0.67	0.68	0.70
64	0.33	0.33	0.34
128	0.16	0.16	0.16
256	0.08	0.08	0.08

These are plotted in Figure 14. Although discrepancies are noted between the results at high pressures, they are very similar for volume ratios normally applicable for de-coupled charges and pre-splitting ($V_h/V_e > 4$). It must be emphasized that expressions of the type presented by Johansson and Persson (1970) are seldom available to practitioners.

6. ROCK STRENGTH

To apply the pre-splitting approaches described in this paper, one needs to have an estimate, at least, of the compressive and tensile strength. It is desired to keep the wall pressure below the compressive strength and above the tensile strength. Over the years, there has been considerable discussion regarding the appropriate strength values which should be used. Sanden(1974), for example, indicates that dynamic strength values should be used in the evaluations since the values determined from his back-analyses were many times the static values. Various researchers have suggested major strain rate effects on strength.

In a small attempt to address the question as to the proper strength values which should be used, the experimental results collected by Sanden (1974) have been re-analyzed using the approach described in this paper. Here, his results from a series of pre-splitting tests performed on individual blocks of Queenston limestone will be re-examined. The hole diameter was 1.75 ins and the explosive used was 50 grain/ft detonating cord with a given detonation velocity of D = 21,000 ft/sec (6400 m/sec). Based on Figures 8 and 9, the PETN density corresponding to this detonation velocity is about 1.25 g/cm^3. The given static tensile strength was 820 psi. Although the static compressive strength was not provided, the strength of a similar limestone coming from the same general area was found to be 60 MPa (8700 psi).

Sanden's experimental results are summarized in Table 3. They will be analyzed with respect to the apparent rock strength.

Table 3. Queenston Block Test Parameters and Results. After Sanden (1974).

Factor	Shot No. 1	No. 2	No. 3	No. 4	No. 5
Explosive charge (strands)	4	5	4	5	6
Hole spacing (ins)	25	25	15	15	20
Pre-split	No	Yes	No	Yes	Yes

The first step in the analysis is to determine equivalent PETN core diameters for the 4, 5 and 6 strand bundles. The equivalent metric charge for the 50 grain/ft cord is 10.63 g/m. Using this value, one obtains:

4 – strands: 42.52 g/m
5 – strands: 53.15 g/m
6 – strands: 63.78 g/m

Given the PETN density of 1.25 g/cm^3, the equivalent core diameter (d_{eq}) for the 4-strand bundle can be calculated using

$$(\pi /4)\ d_{eq}^2\ (1.25)(100) = 42.52$$

Solving for d_{eq} one finds that

$$d_{eq} = 0.658 \text{ cm} = 0.259 \text{ ins}$$

The values for the 5 and 6-strand bundles are:

<u>5 – strands</u>
$d_{eq} = 0.736$ cm = 0.290 ins
<u>6 - strands</u>
$d_{eq} = 0.806$ cm = 0.317 ins

The blasthole diameter is

$$d_h = 1.75 \text{ ins} = 4.445 \text{ ins}$$

Since the charge is continuous, the volume ratios are:

<u>4-strands</u>
$V_R = (4.445/0.658)^2 = 45.63$
<u>5- strands</u>
$V_R = 36.47$
<u>6- strands</u>
$V_R = 30.41$

The explosion pressure for the PETN is calculated using equation (31)

$$P_e = 1/8\ \rho_e\ D^2 = 1/8\ (1250)\ (6.4)^2 = 6400 \text{ MPa}$$

The values of a_e are calculated using equation (61).

$$a_e = \upsilon_e - 1.1\ e^{-0.473/\upsilon e} \quad (61)$$

Since

$$\upsilon_e = 1/\rho_e = 1/1.25 = 0.80\ \ l/kg$$

The value of a_e becomes

$$a_e = 0.800 - 1.1\ e^{-0.473/0.800} = 0.191$$

Using equation (63), the value of the constant K_o becomes

$$K_o = P_e a_e = 6400\ (0.191) = 1222.4$$

The expansion volumes for the three different charge configurations are calculated using:

$$\upsilon = V_r\ \upsilon_e \quad (69)$$

For the 4-strand configuration one finds that

$$\upsilon = 45.63\ (0.80) = 36.50\ l/kg$$

This value is now substituted into equation (65) to yield 'a'

$$a = \upsilon - 1.1\ e^{-0.473/\upsilon} = 36.50 - 1.1\ e^{-0.473/36.50} = 35.41$$

The wall pressure (P_w) is determined using equation (64)

$$P_w = K_o/a = 1222.4/35.41 = 34.52\ MPa = 5005\ psi$$

This applies for both of the 4-strand charge configurations tests. In block test No. 1, the hole spacing was S = 25 ins. Using equation (30), the minimum required tensile strength so that the block will not split is

$$T = P_w/[(S/2r_h - 1) = 34.52/[25/1.75 - 1] = 2.58\ MPa = 374\ psi$$

Since the block did not split, the actual tensile strength must be greater than this value. For block test No. 3, the hole spacing was reduced to 15 ins. Substituting the appropriate values into equation (30), one finds that the minimum required tensile strength is

$$T = 4.56\ MPa = 661\ psi$$

Since the block did not split, the tensile strength must be greater than this value.

The process is now repeated for the other charge configurations. The wall pressures for the 5-strand and 6-strand configurations are:

5 – strand

$P_w = 43.27$ MPa = 6273 psi

6 - strand

$P_w = 52.45$ MPa = 7606 psi

The 5-strand configuration was used in block tests No. 2 and No. 4. The calculated tensile strengths based on the appropriate hole spacing are:

Block test No. 2

T = 3.26 MPa = 473 psi

Block test No. 4

T = 5.71 MPa = 829 psi

In both cases, the block split.

The 6-strand configuration was used in block test No. 5 and the spacing was 20 inches. Using the calculated borehole wall pressure, the calculated tensile stress in the web would be

T = 5.03 MPa = 729 psi

The block split.

In summary,

Block Test	T(MPa)	T(psi)	Pre-split
No. 1	2.60	377	No
No. 2	3.26	472	Yes
No. 3	4.56	661	No
No. 4	5.71	829	Yes
No. 5	5.03	729	Yes

With the exception of block test No. 2, it would appear that the tensile strength of the limestone is

$$4.6 \text{ MPa} < T < 5.0 \text{ MPa}$$
$$661 \text{ psi} < T < 729 \text{ psi}$$

Since the static tensile strength of Queenston limestone has been reported to be 820 psi (5.66 MPa), these values would suggest no rate of loading effect on the tensile strength.

Sanden(1974) did not observe crushing around any of the holes. As is observed, the maximum borehole wall pressure was calculated to be 52.45 MPa (7606 psi) which is below the estimated compressive strength of 8700 psi. Since no tests in this series were conducted at wall pressures above 8700 psi, no definite conclusions can be made regarding possible "rate of loading" effects on the compressive strength. However, other experience by the author using this approach, suggests that statically determined strength values are appropriate for use with pre-splitting.

7 PRACTICAL APPLICATION OF THE UTAH/NIOSH APPROACH

7.1 Background

In this section, the application of the Utah/NIOSH approach to the design of a pre-split blast will be demonstrated. The example is loosely based on one originally presented by Chiappetta (1982). The example will be worked both in English and SI units.

7.2 Road construction application (English units)

Problem Statement

A construction crew is nearing the final road cut limit and is planning to use the Atlas Powder pre-splitting product "Kleen Cut" in 2" diameter holes. The material to be blasted is granite with an unconfined compressive strength of 25,000 psi and a tensile strength (direct pull) of 600 psi. Three different "Kleen Cut" products could be selected.

Explosive	D (ft/sec)	ρ_e (g/cm^3)	Diameter (ins)
Kleen Cut C	14,000	0.92	1.13
Kleen Cut E	9,200	0.58	1.13
Kleen Cut U	9,200	0.76	1.13

The requirement is to recommend an explosive and spacing which will minimize the number of holes to be drilled and yet preserve the final wall.

Solution

1. Calculate the borehole wall pressure for each of the three explosive-hole combinations. Only the calculations for Kleen Cut C will be included here.

$$P_e = 1.69 \times 10^{-3}\ \rho_e D^2 = 1.69 \times 10^{-3}\ (0.92)\ (14000)^2 = 304{,}740 \text{ psi}$$
$$\upsilon_e = 1/\rho_e = 1/0.92 = 1.087$$
$$a_e = \upsilon_e - 1.1\ e^{-0.473/\upsilon e} = 1.087 - 1.1\ e^{-0.473/1.087} = 0.375$$
$$K_o = P_e\ a_e = 304{,}740\ (0.375) = 114300$$
$$V_R = (2/1.13)^2 = 3.133$$
$$\upsilon = V_R\ \upsilon_e = 3.133\ (1.087) = 3.406$$
$$a = \upsilon - 1.1\ e^{-0.473\upsilon} = 3.406 - 1.1\ e^{-0.473/3.406} = 2.449$$
$$P = K_o/a = 114300/2.449 = 46680 \text{ psi}$$

Summary

Explosive	Wall Pressure (psi)
Kleen Cut C	46,680
Kleen Cut E	16,770
Kleen Cut U	18,960

2. Select the best combination in comparison to the compressive strength (25,000 psi). Both Kleen Cut E or Kleen Cut U have wall pressures which are below the compressive strength of the rock and are candidates for selection. Kleen Cut C is rejected.

3. Calculate the maximum borehole spacing based upon the tensile strength (600 psi)

   a. Kleen Cut E
   $$S = \frac{2r_h\ (P_w + T)}{T} = \frac{2(16770 + 600)}{600} = 58 \text{ ins.}$$
   b. Kleen Cut U
   $$S = \frac{2r_h\ (P_w + T)}{T} = \frac{2(18960 + 600)}{600} = 65 \text{ ins.}$$

4. Select a borehole spacing which is less than calculated in step 3 based upon assuring

successful pre-splitting. The decision basis would include rock structural and operational factors.

In this case, $S_{selected}$ = 4 ft.

5. Since both explosives are expected to yield about the same result, selection is based on cost and availability.

7.3 Road construction application (SI units)

Problem Statement

A construction crew is nearing the final road cut limit and is planning to use the Atlas Powder pre-splitting product "Kleen Cut" in 50.8 mm diameter holes. The material to be blasted is granite with an unconfined compressive strength of 172 MPa and a tensile strength (direct pull) of 4.1 MPa. Three different "Kleen Cut" products could be selected.

Explosive	D (m/sec)	ρ_e (g/cm^3)	Diameter (mm)
Kleen Cut C	4268	0.92	28.7
Kleen Cut E	2805	0.58	28.7
Kleen Cut U	2805	0.76	28.7

The requirement is to recommend an explosive and spacing which will minimize the number of holes to be drilled and yet preserve the final wall.

Solution

1. Calculate the borehole wall pressure for each of the three explosive-hole combinations. Only the calculations for Kleen Cut C will be included here.

$P_e = 1/8\ \rho_e D^2 = 1/8\ (920)\ (4.268)^2 = 2095$ MPa
$\upsilon_e = 1/\rho_e = 1/0.92 = 1.087$
$a_e = \upsilon_e - 1.1\ e^{-0.473/\upsilon e} = 1.087 - 1.1\ e^{-0.473/1.087} = 0.375$
$K_o = P_e\ a_e = 2095\ (0.375) = 786$
$V_R = (50.8/28.7)^2 = 3.133$
$\upsilon = V_R\ \upsilon_e = 3.133\ (1.087) = 3.406$
$a = \upsilon - 1.1\ e^{-0.473\upsilon} = 3.406 - 1.1\ e^{-0.473/3.406} = 2.449$
$P = K_o/a = 786/2.449 = 321$ MPa

Summary

Explosive	Wall Pressure (MPa)
Kleen Cut C	321
Kleen Cut E	115
Kleen Cut U	130

2. Select the best combination in comparison to the compressive strength (172 MPa) Both Kleen Cut E or Kleen Cut U have wall pressures which are below the compressive strength of the rock and are candidates for selection. Kleen Cut C is rejected.

3. Calculate the maximum borehole spacing based upon the tensile strength (4.1 MPa).

a. Kleen Cut E

$$S = \frac{2r_h (P + T)}{T} = 50.8 \frac{(115 + 4.1)}{4.1} = 1480 \text{ mm} = 1.48\text{m}$$

b. Kleen Cut U

$$S = \frac{2r_h (P + T)}{T} = 50.8 \frac{(130 + 4.1)}{4.1} = 162 \text{ mm} = 1.62\text{m}$$

4. Select a borehole spacing which is less than calculated in step 3 based upon assuring successful pre-splitting. The decision basis would include rock structural and operational factors.

   In this case, $S_{selected} = 1.2$ m

5. Since both explosives are expected to yield about the same result, selection is based on cost and availability.

8. SUMMARY AND CONCLUSIONS

In the future, there will be an increased emphasis on careful rock excavation for both safety and economic reasons. Pre-split blasting is one of the important techniques to be considered in this regard. Most often, the use of de-coupled explosive charges are involved in these applications. The pre-split design equations in common use today are based on an assumption that the expansion of the explosive gases in the annulus between the charge and the hole wall takes place adiabatically. This is incorrect. The actual expansion takes place isothermally or perhaps more correctly, without doing work. The Utah/NIOSH approach provides a practical, yet technically sound, procedure for designing pre-split blasts based on iso-thermal expansion of the explosive gases. The hole spacing equation used is based on the force equilibrium approach described by Sanden (1974). This requires use of the wall pressure and knowledge of the tensile strength of the rock. Over the years, much discussion has revolved around whether static or dynamic rock properties should be used. Application of the Utah/NIOSH approach to back-analyze pre-splitting data developed by Sanden (1974) suggests that static strength values can be applied. This agrees with other experience using the Utah/NIOSH approach. Detonating cord is often used in pre-splitting applications. To be able to apply the Utah/NIOSH approach, data concerning the density and the diameter of the PETN core are required. Unfortunately, these values are not provided by explosive manufacturers today. A technique to overcome this problem is suggested. It is hoped that the industry will carefully review the material contained in the paper and contribute their experience regarding its applicability and /or the required corrections.

DISCLAIMER

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

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