Evaluating the Effects of Non-coaxial Charges for Contour Blasting

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ABSTRACT

Contour blasting is commonly performed by employing linear charges decoupled from the boreholes. This method is common in surface and underground excavations, either for civil or mining purposes. To achieve the best results in terms of rock breakage and respect of the excavation profile, blasting theory suggests that charges should be inserted coaxial to the holes to grant uniform distribution of the explosive energy, therefore obtaining a uniform radius of damage. Nonetheless, due to readiness of operations or lack of availability of specific products on the market, non-coaxial charges are often employed in blasting practice. Non-coaxial charging methods include the employ of high-power detonating cord (40 to 100 g/m), low-power detonating cord connecting small-diameter cartridges (commonly 10 g/m detonating cord priming 1″ cartridges) or string loading (a thin layer of bulk emulsion pumped with controlled flow and controlled extraction of the injecting rod). This research focuses on evaluating the effects of the first two charging methods on the quality of final walls in open pit and underground operations. Different drilling geometries and charging configurations were applied to both quarrying and tunnelling blasts. The half-cast factor, the overbreak and the underbreak were evaluated as control indicators. Rock quality designation and rock mass rating were used to classify the rock mass. The research was aimed to push contour blasts to their limits, observing for which geometry and charge configuration the blast lost its design threshold with respect to the final wall for every given rock mass. Results show the operational limits of non-coaxial charges encountered in the rock masses object of this study. In good-quality rock, smooth blasting with decoupled linear charge of 40 g/m can be extended to a spacing $S = 22\varnothing_f$ with little or no detectable drawbacks in terms of final wall quality, in contrast with theoretical formulae for the determination of the radius of damage. On the other hand, when the rock is poor, any quality of the final wall is hardly achieved at all, in spite of any care in the details of execution of smooth blasting. It is concluded that any design criterion and theoretical approach modelling the effects of contour blasting cannot ignore the features of the rock mass.

INTRODUCTION

This paper focuses on evaluating the effects of different contour blasting methods on the quality of final walls in open pit and underground operations. The main purpose of this research was to understand the limits of application of contour blasting depending on variations in drilling, charging and rock mass features.

Contour blasting is used for removing material along the final slope face. In some cases, it's also used before production blasting to create an artificial fracture along the final cut slope, which will prevent the radial cracks caused by production blasting from penetrating back into the finished face (Konya and Walter, 2006).

Contour blasting can also be used alone without production blasting: it creates less back-break than production blasting because it removes less burden and uses more tightly spaced drill holes with lighter charges. The concept of controlled blasting is motivated by the goal of directing the fracture along the desired surface. Thinking in 'quasi-static' terms (an approach that seems preferable at low charge's density and high decoupling), different conditions must be respected based on elementary considerations:

- the distance of the holes from the free wall must be great enough (ie greater than the distance between the holes), to avoid that some fracture develop towards the free wall

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- tensile stresses in radial planes induced by the gas pressure around the single hole should not be able to open fractures (otherwise random fractures would occur), but
- in the plane of superposition of the effects (the plane in which the holes lie), the sum of the stresses induced by the pressure acting in the contiguous holes must be enough to induce failures.

This 'static' point of view, definitely questionable and inaccurate, gives account of the method adopted for currently obtaining the guidance, which consists of not falling below a certain value of decoupling (usually two) when loading, and in preserving the ratio diameter/spacing above a certain limit (typical range 1/6 to 1/12). Decoupled charges are recommended to be inserted coaxially in the hole to grant uniform decoupling.

An extreme application of the concept of contour blasting is the dynamic splitting. The process consists in creating one (or more) separation fractures, which isolate a predetermined volume of rock, to be blasted subsequently or, more frequently, to be squared to produce dimension stones. This latter type of blasting adopts decoupled strands of det cord inserted in the holes and usually water as a coupling medium. There are several types of contour blasting; they vary most importantly in the amount of burden they remove and the type of powder they use. Contour blasting techniques that best minimise the visual impacts of the blasting process are presplitting and smooth blasting (Mandal, Singh and Dasgupta, 2008).

The prevailing method for underground perimeter control is smooth blasting. Simultaneously with the development of smooth blasting explosives, research was going on to define the damage zone for different explosives to the contour. It was found that ANFO in a 45 mm blasthole could cause cracks in the surrounding rock up to 1.5 m from the hole and 80 per cent nitroglycerine dynamite in 25 mm cartridges had a crack extension of close to 1 m from the hole. For the specially designed smooth blasting explosive Gurit (to be inserted coaxial within the hole), the crack extension was limited to 0.3 m (Olofsson and Frändberg, 1993). Increased mechanisation of the construction and mining industry demanded for faster and simpler methods for the charging of the contour holes with a light and well-balanced explosive. The whole working underground cycle, drilling, charging the blastholes, blasting, excavation and loading, ventilation and reinforcement became faster, but the charging of the contour holes was still made in an old-fashioned way. Sometimes tubular plastic cartridges were connected together, but often were separated dynamite cartridges taped to a wooden bar and initiated with a det cord: a time-consuming method, which also added more carbon monoxide into the blasting fumes. Trial blasts started with emulsions in the buffer holes closest to the contour and one string of 40 g/m detonating cord were employed in the contour holes. The result of the trials was excellent in the weathered rock (Olofsson and Frändberg, 1993).

Other research, conducted by Iverson, Hustrulid and Johnson (2013) proposes an easy-to-use blast design method that includes improvements for determining the perimeter burden based on the effect of damage from buffer holes. This means that the distance or burden between the perimeter holes and the next line of blastholes defined as the buffer holes is determined by the damage caused by the buffer holes detonation. The research and development of the new design was aimed at:

- identifying the effectiveness of perimeter control in current drift designs
- conducting experiments to determine the blast damage extent and factors influencing damage
- studying blast damage models
- packaging the blast damage models into an engineer and miner friendly design concept.

This paper focuses on evaluating the effects of different contour blasting methods on the quality of final walls in open pit and underground operations. The main purpose of this research was to understand the limits of application of contour blasting depending on variations in drilling, charging and rock mass features.

Different drilling geometries and charging configurations were applied to both quarrying and tunnelling blasts. The half-cast factor (HCF), the overbreak (OB) and the underbreak (UB) were evaluated as control indicators. Rock quality designation (RQD) and rock mass rating (RMR) were used to classify the rock mass. The research was aimed to push contour blasts to their limits, observing for which geometry and charge configuration the blast lost its design threshold with respect to the final wall for every given rock mass. Final results emphasised the role of geological and structural features affecting the quality of the contour blasting.

Rock mass features

Rock mass features cannot be changed but their knowledge can help to properly select the explosive characteristics and the blast design parameters to obtain optimum results (Singh and Xavier, 2005; Innaurato, Mancini and Cardu, 1998). The RQD is a practical method to evaluate the natural fracturing of the rock mass. It was developed by Deere (1963) to provide a quantitative estimate of rock mass quality from drill core logs. It is defined as 'the percentage of intact core pieces longer than 100 mm in the total length of core'. The core should be at least 54.7 mm in diameter and should be drilled with a double-tube core barrel. The RQD is an easy and quick measurement as only certain core pieces (longer than 10 cm) are included. It is, therefore, frequently applied in core logging and is often the only method used for measuring the degree of jointing along the core drill hole. The most important use of RQD is as a component of the RMR and Q rock mass classifications. RQD gives an average measurement of the degree of jointing along the actual section (core run); measured along several sections, the RQD has, of course, a variation (Palmstrom, 2005).

As mentioned by several authors (Bieniawski, 1973, 1984; Edelbro, 2003), the RQD has several limits. Similar to all types of one-dimensional measurements, RQD is directional, but due to its definition it is more sensitive to the hole or line direction than joint spacing or fracture frequency measurements. This has been shown by Choi and Park (2004) for Korean conditions. Simulations of directional errors of RQD using computer spreadsheets have been performed by Palmstrom (1995). RQD has been chosen as a valid parameter for the present research, being estimated by Palmstrom's indirect method (1995) along the axis of the contour holes.

Wall damage

The damage process in rock blasting has gained extensive attention over the years and its mechanism seems to be well understood (Khandelwal and Monjezi, 2013; Onederra *et al*, 2013; Hamdi, Romdhane and Le Cléac'h, 2011; Malmgren *et al*, 2007; Saiang and Nordlund, 2009; Ambrosini *et al*, 2002). The processes of blast-induced damage and fragmentations around a borehole are strongly dependent on the parameters of the explosive and the dynamic response of rock (Bohloli and Hovén, 2007; Hudson *et al*, 2009; Tripathy and Gupta, 2002; Wang *et al*, 2007; Zhu, Mohanty and Xie, 2007). Damage is required to be minimised even in conventional blasting in several cases, such as pit wall blasting, tunnel and underground chambers excavations, and so on.

For the past many years, considerable efforts have been made to study the effect of blasting on rock damage and also to minimise damage resulting from blasting (Rathore and Bhandari, 2007; Li *et al*, 2011; Netherton and Stewart, 2009). The common practice shows that a reasonable excavation sequence and the contour blasting method have a direct effect on the excavation quality, construction schedule, and excavation economy (Lu *et al*, 2011; Mandal, Singh and Dasgupta, 2008; Sheng *et al*, 2002). To reduce the extent of damage and guarantee the quality of the contour surface, smooth blasting and presplitting methods are widely used.

A contour blast should, as known, leave in place smooth walls, corresponding to the geometry of the project. This perfection is hardly ever reached, but it can be used as a goal to evaluate the quality of the result, which is much better as it approaches the ideal result. It is necessary to employ quantitative indicators representing the distance from the 'perfect result', and also to establish criteria to distinguish what part of imperfection is due to the rock and what is due to defects of the project or execution, which can consequently be improved by the operator. The ideal outcome would consist in the perfect coincidence of the surface in which the contour holes lie with the residual wall; however, this rarely occurs (Holmberg and Persson, 1978). Apart from errors in the positioning and targeting of the holes (inaccuracies that are not due to the blast), the most frequent difference that can be observed between the actual result and the ideal one is the overbreak: it not only endangers the safety but also increases the cost and the completion time of the work. To achieve a drill and blast cost-effective operation for excavation of any underground structure it should strictly adhere to specific controlled blasting pattern, to minimise the unacceptable impact on peripheral *in situ* rock mass. In addition to geotechnical properties of rock mass, *in situ* stress condition plays an important role in enhancing the magnitude of overbreak (Mandal and Singh, 2009).

In tunnelling and other underground operations, overbreak is recognised as the principal cause of hazards and deterioration costs in management, where numerous related research projects have been conducted. Many of them have been devoted to clarifying the overbreak phenomenon, but they are still unable to explain the exact occurrence process. Factors causing overbreak can be classified into two groups: geological and blasting (Mahtab *et al*, 1997; Mandal, Singh and Dasgupta, 2008). Perimeter damage was assessed by researchers from the National Institute for Occupational Safety and Health (NIOSH) during field investigations of blasting practices at mines in the United States (Iverson *et al*, 2007; Warneke, Dwyer and Orr, 2007; McHugh, Warneke and Caceres, 2008a, 2008b). There were found many cases where drilling blastholes were performed without precision, and at best were only able to maintain parallel blasthole orientations. Further, the blast designs were based primarily on miners' experience and capabilities. Blasting parameters are changeable factors. Excavation conditions affect the overbreak occurrence. Along with the advanced blasting methods, final wall customised explosives and computer base drilling systems significantly reduce the possible failures on blasting operations. Geological factors, however, are unchangeable and they have a significant influence on the overbreak phenomenon: if the rock is not strong enough to support itself, possibly no blasting techniques can stop the occurrence of overbreak. Jang and Topal (2013) conducted a study focused on the effects of geological parameters to the overbreak phenomenon in a tunnel, where the blasting design was fixed as a standard blasting pattern. RMR parameters were collected through 49 blasting sections as geological data and overbreak data were individually investigated. The optimum model was selected by comparing measured and predicted overbreak where the correlation coefficient of each proposed model was found. The authors showed that the causing factors of overbreak are complex and mutually correlated, so it is important to consider all RMR parameters together.

A useful indicator is the HCF, ie the ratio (percentage) between the total length of 'half holes' observable on the wall after the blast and the total length of contour holes that were drilled and blasted. It is an indirect indicator of the damage caused by the contour blastholes, as directly indicates if detachment was 'well-led'. But since it is known that a good guided fracture occurs only if the pressure in the holes not far exceeds the minimum necessary to obtain the detachment, this can be taken as an indicator of a serious (HCF nil), moderate (HCF medium) or low (HCF high) mechanical damage of the residual wall (Figure 1).

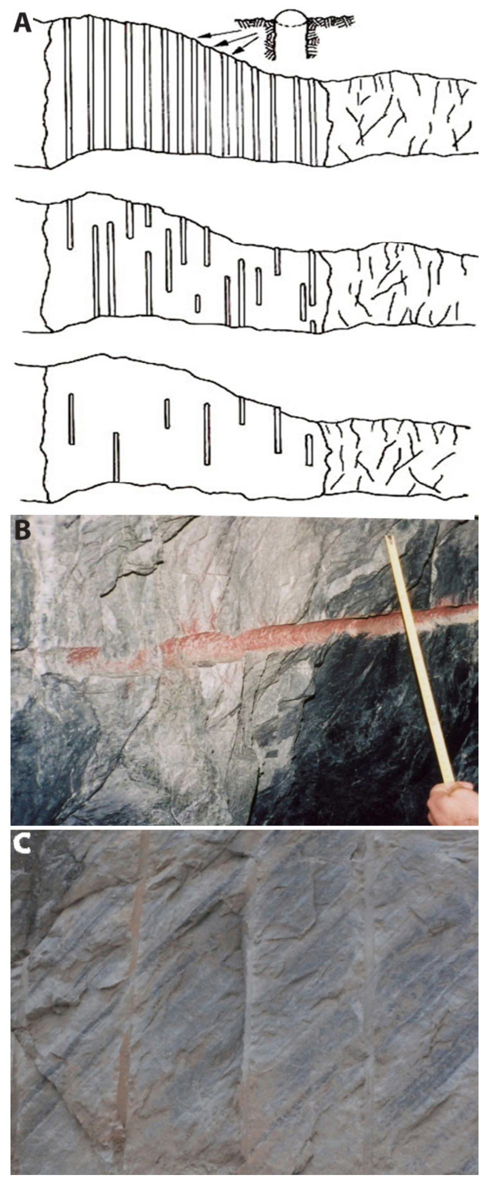

FIG 1 – (A) Schematic representation of low (half-cast factor high), moderate (half-cast factor medium) and serious (half-cast factor nil), mechanical damage to the residual wall (Mancini and Cardu, 2001). (B) Half cast clearly observable after a production blast in tunnel driving and (C) after a contour blast in open pit mining.

CONTOUR BLASTING TECHNIQUES

Presplitting

The fracture has the purpose of preliminary separating the volume of rock intended to be fragmented with ordinary blasts and removed. The isolation may be required to prevent damage to the rock which must remain in place, to limit the transmission of vibrations to the rock mass, or for both reasons. Presplitting in rock mass creates a continuous fracture all along the accurately charged and appropriately spaced boreholes away from the free face. The stable fracture resulting after presplitting reflects the elastic stress waves and uses the explosive energy from being wasted beyond the fracture zone (Gupta, Singh and Singh, 1987). The blast is performed by an array of parallel holes with small and regular spacing (usually from six to 25 times the diameter), weakly loaded (with strongly decoupled charges or explosive with very low power and pressure), lying strictly in the desired plane of detachment, which are detonated simultaneously. The presplitting blastholes may be included as contour charges in an ordinary blast.

The charges can be arranged referring to the principles of the so-called static dimensioning and the most important factors for obtaining the wanted result are the unitary linear charge (the ratio between charge and length of the hole), the diameter of the hole and the ratio between spacing and diameter; 'burden' and 'specific consumption' concepts are not useful.

The explosive is used with strong decoupling and blasting should be simultaneous. The hole diameter is generally small, but in special cases diameters larger than 100 mm can also be used: in such cases, the linear charge was not made up of tubular cartridges expressly designed but with rows of ordinary cartridges secured to a detonating cord, or with bulk explosive poured into tubes with a diameter much smaller than the hole or with bulk diluted explosive, etc.

In typical presplitting, manufacturers generally provide the necessary data (hole diameter, spacing); a summary is provided in Sandvik Tamrock Corp (1999). If such charges are not available, the dimensioning can be done with empirical formulas, among which one based on the principle of static dimensioning (Del Greco *et al*, 1983) is mentioned:

$$\left(\frac{\Phi_c}{\Phi_f}\right)^2 \cdot \frac{L_c}{L} \cdot \frac{\delta_c}{1000} \cdot P_s \cdot \left(\frac{\Phi_f}{E - \Phi_f}\right) = T$$

where:

$ø_c$ is the diameter of cartridges (m)
$ø_f$ is the diameter of holes (m)
δ_c is the density of cartridged explosive (kg/m^3)
P_s is the specific pressure of explosive (MPa)
E is the spacing (m)
L_c is the charged length (m)
L is the hole length (m)
T is the tensile strength of rock (MPa)

If data on the latter are not available it can be assumed, for safety reasons, the value of 20 MPa. As to the specific pressure, there are different values for different explosives, and in particular it can be assumed: 1200 MPa for gelatine-dynamites, 1000 MPa for the ordinary dynamites, 900 MPa for dry AN bulk explosives and water gels, 800 MPa for ANFO.

Another empirical formula, extensively tested and able to provide reproducible results, was suggested by Morin (2000):

$$W = 0.036 \cdot (D)^2$$

where:

W is the weight of explosive per foot of borehole (lb)
D is the diameter of borehole (in)

Experimental tests performed by the same author provided the results shown in Table 1.

Smooth blasting

Also called contour blasting or perimeter blasting, smooth blasting can be used before production blasting as an alternative to presplitting; however, it is mainly used after production blasting, either as an entirely different event or as the last delay of the production blast. Smooth blasting uses drill holes with roughly the same diameter and depth as those used in presplitting, slightly more spaced and loaded with a somewhat larger charge density. If the burden is adequately reduced, smooth blasting produces a quite regular bench face with minimal back-break. Some radial fractures can occur from both the controlled blasting and production blasting. Although smooth blasting gives rise to high values of HCF, they are generally less than the half casts left by presplitting. If drill hole traces are not acceptable, smooth blasting may be suitable only if the drilling length is small.

Smooth blasting is best preformed in hard, competent rock, although it can be used in soft or highly fractured rock by increasing the spacing of the drill holes and/or adding uncharged guide holes to the pattern. Smooth blasting can be used on a variety of benches with different inclination. Sandvik Tamrock Corp (1999) show the most commonly recommended borehole diameters, burden, spacing and explosive charges for smooth blasting. Smooth blasting holes are smaller than production holes in order to limit fracturing around the drill hole. The burden-to-spacing ratio for smooth blasting is approximately 1.5 to 1. Hole spacing is about 14 to 20 times the hole diameter. Wider spacing for hard rock and closer spacing for weak rock are used. Unloaded guide holes (drilled between normally spaced blastholes) are generally employed for weak and soft formations or for blasting corners. In underground operations, the perimeter holes of the backs (roof) of headings and tunnels are drilled along the design profile parallel to the direction of the

TABLE 1

Experimental results performed by Morin (2000). All holes were loaded with 0.37 kg/m (0.25 lb/ft) with a 0.9 kg (2.0 lb) toe load for a total of 4.6 kg (10.2 lb). All holes were stemmed 1.2 m (4 ft).

Hole diameter, mm (in)	Spacing, mm (in)	Results
63.5 (2.5)	762.0 (30)	Well-defined crack, rock breakage at the collar, shatter beyond perimeter, apparent disturbance of the final wall
76.2 (3.0)	762.0 (30)	Well-defined crack, little breakage at the collar, little shatter beyond perimeter, little or no disturbance of the final wall
101.6 (4.0)	762.0 (30)	Crack starting from each hole extended completely toward adjacent holes in hard rock, but failed to reach adjacent holes in soft rock
127.0 (5.0)	762.0 (30)	Peripheral cracking around the holes
152.4 (6.0)	762.0 (30)	Limited peripheral cracking around the holes

excavation. Generally, the spacing between the final lines of holes is less than 1.5 times the burden. The borehole should be sealed with a tamping plug, clay plug or other type of stemming to prevent the charge from being extruded from the hole by charges on earlier delays. Stemming also prevents excessive rifting (splitting) of the rock and permits the use of lighter charges because blast energy is better contained and therefore better distributed. A variant of the smooth blasting is called cushion blasting: it is applicable in surface mining when the goal is to trim the excess of material from the final highwall to improve stability. A single row of holes is drilled along the perimeter of the excavation. The diameter of the drill holes varies between 50 mm to 164 mm. Cushion blastholes are charged with small, well-distributed charges in completely stemmed holes, which are fired after the main blast is excavated. The charges are fired with no delay, or minimum delay between boles.

Many studies were performed to examine the whole damage process of the smooth blasting and presplit blasting excavation method (Yingguo *et al*, 2014). The results demonstrate that, in the case of contour blasting with the method of smooth blasting, the total damage of rock slope is a result of cumulated damage induced by the production holes, buffering holes and smooth holes. Among the total damage, the blasting of the production holes is the more relevant, followed by the smooth and buffering holes. For the presplitting, the final damage of rock slope is mainly induced by presplitting blasting itself.

EXPERIMENTAL CASES

Two experimental campaigns were carried out in opencast and underground environments. The first campaign was conducted at the Experimental Mine of the Research Centre for Responsible Mining of the University of São Paulo, Brazil; the second campaign was conducted along the construction of a civil hydraulic tunnel in Northern Brazil.

Experimental quarry

Experimental tests were carried out with the aim of evaluating the minimum charge/maximum spacing ratio which guarantees a good control of the walls. Smooth blasting was employed adopting different configurations of non-coaxial charges following different drilling geometries with:

- constant linear charge along the holes and varying the spacing between contour holes
- constant spacing between contour holes and varying the linear charge along the holes.

This was achieved adopting four charging configurations:

1. low-grain detonating cord (10 g/m) with small-diameter (1″ × 12″) emulsion cartridges taped to it at a distance equal to the length of the cartridge (12″)
2. two strands of high-grain (40 g/m) detonating cord along the hole
3. a single strand of high-grain (40 g/m) detonating cord along the hole
4. low-grain detonating cord (10 g/m) with small-diameter (1″ × 12″) cartridges taped to it at distances varying from hole to hole (in functional groups of four holes with the same linear charge).

Three experimental blasts were performed:

1. Half of the perimeter of the final wall was charged with Charge 1; the other half with Charge 2. Constant spacing was maintained.
2. For every two adjacent holes, one was charged with Charge 1; the other hole with Charge 3. Constant spacing was maintained.
3. All the holes were Charge 3. Spacing between the holes was regularly increased in functional groups of four holes with constant spacing.

All holes were drilled with diameter $Ø_f$ = 56.4 mm (2.5″). In every test the holes were filled with water before blasting, to act as a coupling medium. In some cases, when the rock was naturally fractured, water could not be retained and some holes were detonated only partially or not at all filled with water. The initiation was simultaneous for all the holes in every blast, given by a main line of 10 g/m detonating cord. The details of the blasts are given in Figure 2 and Table 2. Figure 3 represents the result of Blast 1 and Figure 4 represents the results of Blast 2. In both cases, HCF approaches 100 per cent and no damages or back-break problems were noticed. Figure 5 shows the results of Blast 3: HCF approaches 100 per cent in the good quality portion of the bench (lower) and is equal to zero in the poor quality portion (upper).

Discussion of the results

Hustrulid (1999) has assumed that the damage radius R_d is equal to the radius of influence; he reported that is important to have an expression for R_d that can be applied to different explosive – rock combinations. Based upon energy considerations, he wrote the following equation:

$$\frac{R_d}{r_h} = 25\sqrt{\frac{P_{eExp}}{P_{eANFO}}}\sqrt{\frac{2.65}{\rho rock}}$$

where:

r_h is the radius of the hole
P_{eExp} is the explosion pressure for the explosive
P_{eANFO} is the explosion pressure for ANFO (1550 MPa)

It was moreover assumed:

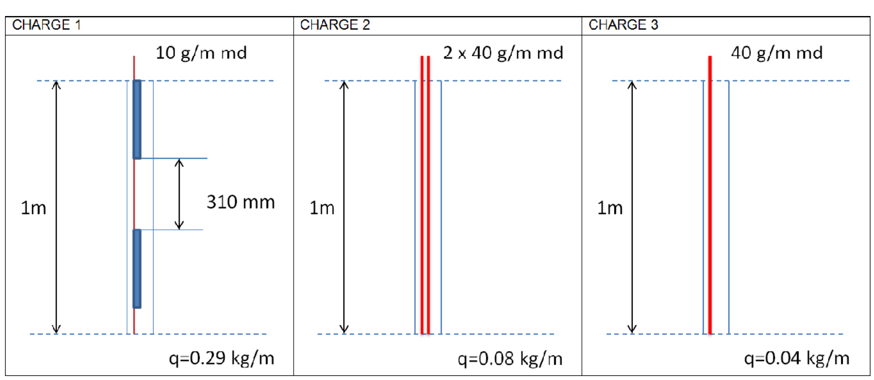

FIG 2 – Details on loading adopted for experimental blasts 1, 2 and 3.

TABLE 2
Details of the blasts performed at the quarry.

Features	Blast 1	Blast 2	Blast 3
Bench height (m)	6.5	6.5	6.0
Drill hole length (m)	7.0	7.0	6.5
Drill hole diameter (mm/inch)	63.5/2.5	63.5/2.5	63.5/2.5
Spacing (m)	0.75	0.75	0.8; 1.0; 1.1; 1.2; 1.3; 1.4
Burden (m)	1	1	1
Detonating cord DC (g/m)	10	40	40
Ø Column charge (mm/inch)	25.4/1	/	/
L Column charge (mm/inch)	304/12	/	/
Gap between Q_c (mm/inch)	304/12	/	/
Strands of DC/hole	1	2	1
Cartridges/hole (n)	11	/	/
Holes/blast (-)	13	12	24 (four for each scheme)
Charge/hole + DC (kg)	1.95 + 0.07 = 2.02	0.56	0.26
Charge/blast (kg)	26.26	6.72	6.24
Linear charge (kg/m)	0.29	0.08	0.04
Delays (n)	none	none	none
Results	• Smooth wall • HCF = 100% • No visible difference in the results between the charging methods	• Smooth wall • HCF = 100% • No visible difference in the results between the charging methods	• HCF 100% for RQD 100% • HCF = 0 for RQD < 40% • Smooth walls up to S = 130 cm • Significant underbreak with S = 140 cm

DC – detonating cord; HCF – half-cast factor; RQD – rock quality designation; S – spacing

$$P_d = \frac{\rho_e D^2}{4} \text{ and } P_e = 0{,}5\, P_d$$

where:

P_d is the detonation pressure (MPa)

ρ_e is the explosive density (kg/m^3)

D is the detonation velocity (km/s)

Table 3 contains the theoretical results of the calculation of the radius of damage obtained applying this equations to different charging configurations. Coupling ratio = 1.25 considers a 2.5″ hole charged with a 2″ cartridge of emulsion (typical of production blasting); coupling ratio = 7.5 considers a 2.5″ hole charged with Charge 2 (the diameter of the explosive is considered as the equivalent diameter of the two penthrite cores of the two strands of detonating cord); coupling ratio = 2.5 considers Charge 1, considering the diameter of a 1″ cartridge and ignoring the low-grain det cord.

The experimental tests of Blast 1 and Blast 2 show that smooth walls and uniform HCF were obtained applying side by side Charge 1 and Charge 2, characterised by drastically different features (comparable to the last two cases considered in Table 3). Apparently, there is not a correlation between the linear charge employed and the occurrence of damage to the wall: different linear charge values and different geometrical shape of the charges give identical results. This could be due to the fact that the order of magnitude of the stresses produced by these charges is way superior to the order of magnitude of the failure stress of the rock: in this case any charge does the job. As for Blast 3, which was performed by progressively increasing the spacing between the holes and maintaining constant the linear charge, the results show that the HCF is almost 100 per cent for every spacing (from 0.8 m to 1.40 m) in RQD = 100 per cent, and almost 0 in RQD <40 per cent, see Figure 5. This can be attributed to two main reasons:

1. the poor quality of the rock with low RQD
2. the absence of the coupling medium (water) to effectively transfer the detonation wave to the hole of the wall (as mentioned, fractured portions of the mass water could not be retained in the hole and filtered away).

Underbreak was observed at a spacing of 1.4 m. The term, according to Rustan (1998), refers to the rock that remains unbroken inside the theoretical contour line in a tunnel drift, stope, bench etc after firing a round. Since the quality of the rock did not change along the same bench, it seems that within the conditions of the test a spacing S = 1.4 m (S = 22Øf) is the upper limit of effectiveness of a proper smooth blasting.

These results suggest that by increasing the spacing while maintaining the same charge, a very little or no influence on the results is noticed when the rock is competent: the progressive decrease of the specific consumption (g/m^2) did not lead to important differences in the quality of the walls: at the limit of 1.4 m some underbreak was noticed but still HCF was 100 per cent and the rock effectively detached between the holes. The 'minimum specific consumption', the one that is no longer capable of allowing the detachment along the inter-axis of adjacent holes, was not reached yet. These results contrast with the theoretical values of the radius of damage that each hole should develop. According to Hustrulid's equations, the results obtained on field should have not been achieved (see the second line in Table 3 compared to the results of Blast 1). The formulations available nowadays to evaluate the radius of damage present ambiguous issues on which it would be worth to further investigate. Two main aspects deserve to be considered regarding the theoretical formulae:

1. the rock mass features are not taken into account
2. they are based on a 2D ideal visualisation, considering a cross-section of the hole, where the parameter characterising the explosive is its diameter in the cross-section and the pressure applied radially from the centre of the hole outwards.

It is evident that the rock mass features must by all means be taken into account. Experimental results show this clearly. Additionally, the concept of 2D cross-section visualisation does not allow the inclusion of the distribution of the charge along the hole in the third dimension. Future research will aim to include rock mass indicators in the design methods and to integrate the linear charge along the axis of the hole and its continuity.

Underground operations

The research was carried out during the excavation of the tunnel for the transposition of the São Francisco River, in Northern Brazil. The length of the tunnel is 3.08 km. The rock is an altered gneiss; the excavation was performed close to the surface (average overburden is about 30–40 m); RMR was

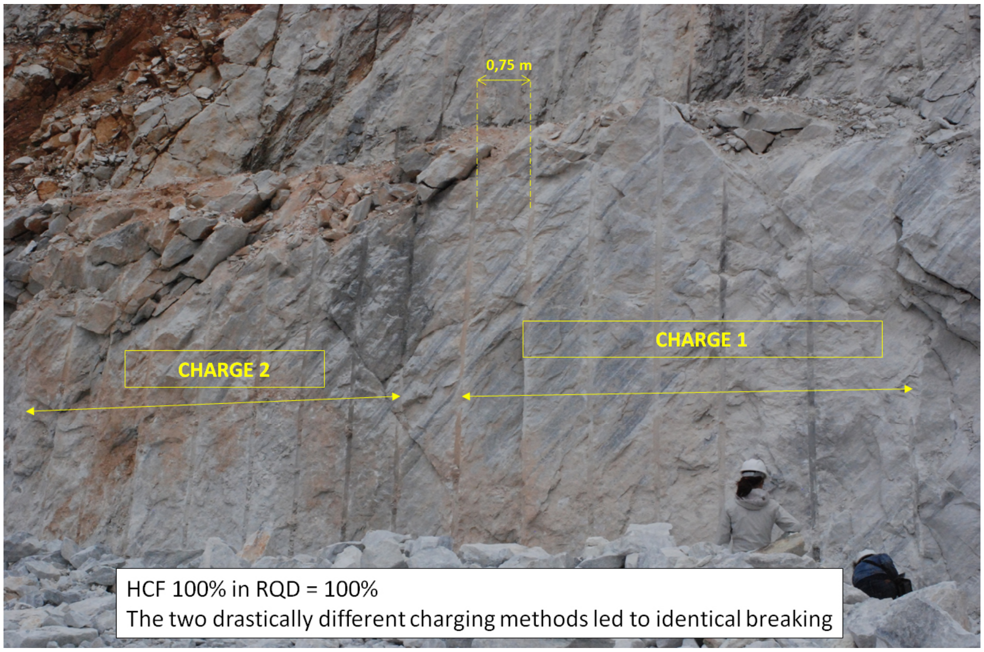

FIG 3 – Results of Blast 1. On the remaining wall, the presence of the contour holes with a half-cast factor that approaches 100 per cent can be observed.

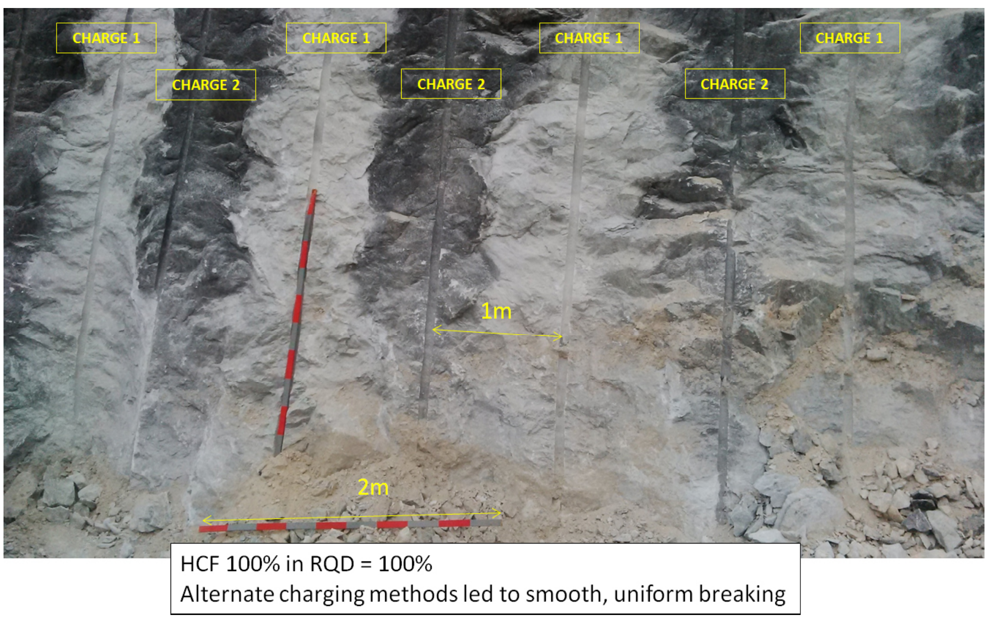

FIG 4 – Results of Blast 2. On the remaining wall, the presence of the contour holes with a half-cast factor that approaches 100 per cent can be observed. Holes charged with detonating cord can be recognised by the black mark left by the decomposition of the cord jacket.

estimated through 32 measurements and a mean value of 47 was found (class III, according to Bieniawski, 1973), with only two exceptions in Class IV. Both parallel holes and V-cuts were used for the tunnel excavation; the schemes of the blasts are given in Figures 6 and 7. For the smooth blasting holes were drilled with diameter $ø_f$ = 45 mm and the spacing was set as S = 510 mm, being S = 11$ø_f$. The contour holes were weakly loaded, as shown in Figure 8. Figure 9 shows the poor quality of the profile obtained (HCF lower than 30 per cent), although contour blastholes were charged with 40 g/m detonating cord and buffer blastholes were employed to improve the result. The blasts were very carefully carried out, respecting the drilling design, loading and timing; buffer holes were performed to increase the chance of success of the smooth blasting, but in almost all cases HCF was not satisfactory. In this case, the explosive appears not to be the most suitable design choice, due to the persistency of rock discontinuities which affect the success of the blasts. It must be noted that employing explosives in RMR class IV can be less productive and more damaging than other methods; in fact, the energy released by the explosive to the outside of the volume to be blasted not only can create an extra profile, but also diminishes the stability of the rock mass, as it causes fracturing around the void (Innaurato, Mancini and Cardu, 1998). According to Page (1987), the damage can be correlated with the detectable peak particle velocity in the rock. The potential instability caused by the blast should be taken into consideration during rock characterisation for the calculation of the support structures. The importance of minimising the fracturing around the tunnel by correctly choosing the excavation technique as a

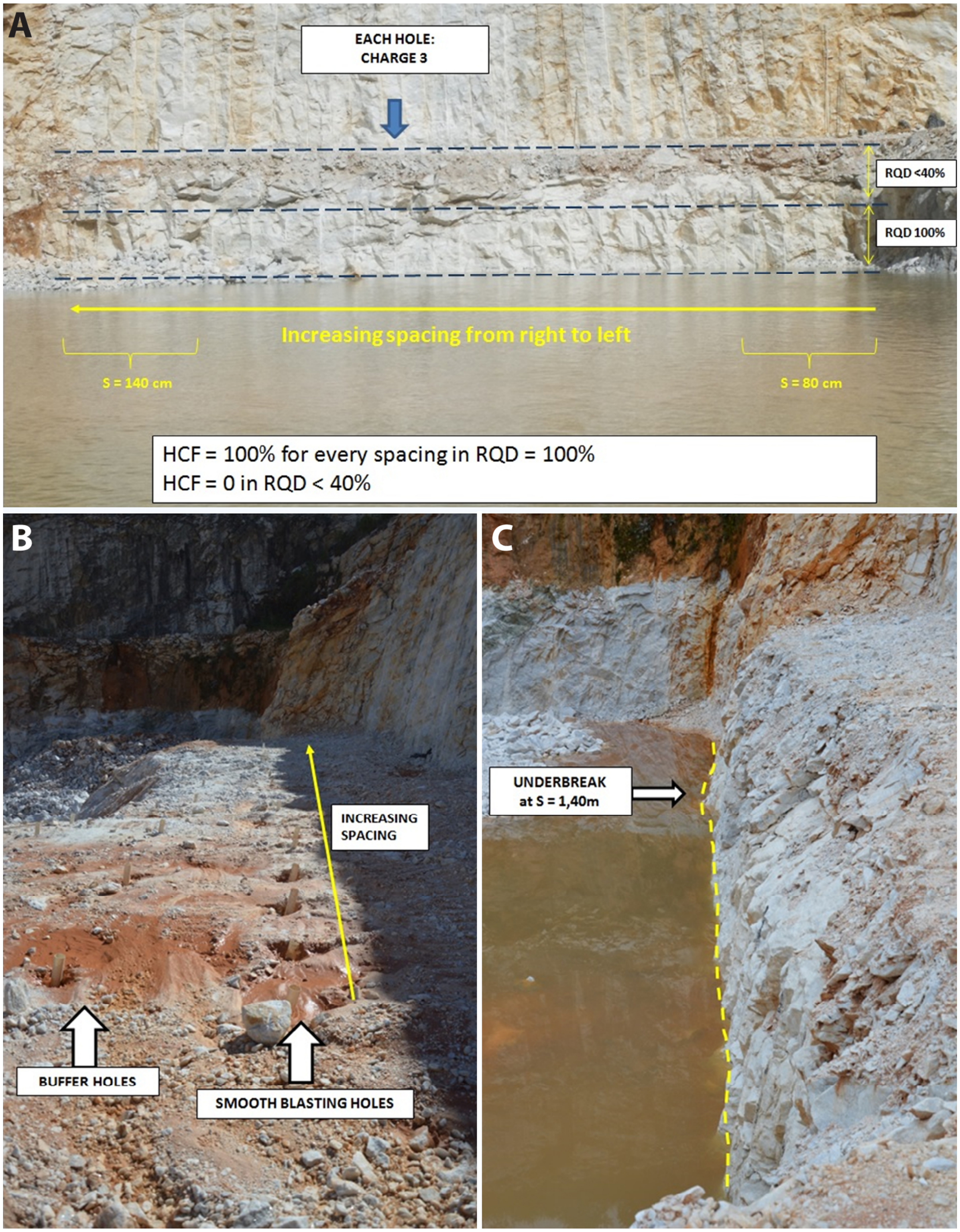

FIG 5 – Results of Blast 3, which was performed by progressively increasing the spacing between the holes: (A) front view of the final wall, (B) side view of the blast along the contour line – execution and (C) final wall.

function of RMR is also emphasised by Holmberg, Larsson and Sjoberg (1984). Research into the quality of blasting then should not only involve the technological problems, but also a series of other problems that refer to the mechanics of the rock that showed that the quality of the blasting and the quality of the rock are strictly connected.

CONCLUSIONS

Two experimental campaigns were carried out in opencast and underground environments with the aim of understanding the limits of application of contour blasting depending on variations in drilling, charging and rock mass features. The results obtained led to the following conclusions:

- When the rock is little fractured (RQD = 100 per cent), open pit smooth blasting with decoupled charges and linear charge of 40 g/m can be extended to a spacing S = 22$\varnothing_f$ (a proportion falling in the range of production blasting) maintaining HCF = 100 per cent and with little or no detectable drawbacks in terms of final wall quality.
- When the rock is highly fractured (RQD <40 per cent) the quality of the final wall is heavily affected: almost no half-

TABLE 3
Theoretical results expected from a contour blast, applying classical equations of radius of damage.

Decoupling	Ø		$Ø_{exp}$	VOD	ρ_e	ρ_r	P_{eExp}	Rd/rh	Rd (VOD)
	"	mm	mm	km/s	kg/dm³	kg/dm³	MPa	-	mm
ø_Hole⁄ø_Exp = 1.25	2.5	63.5	57.15	4.5	1.15	2.6	2213	32	1915
ø_Hole⁄ø_Exp = 7.5 (Charge 1)	2.5	63.5	8.5	6.8	0.9	2.6	28	0	215
ø_Hole⁄ø_Exp = 2.5 (Charge 3)	2.5	63.5	25.4	4.5	1.15	2.6	269	32	667

VOD – velocity of detonation; Rd/rh – radius of damage/radius of the hole.

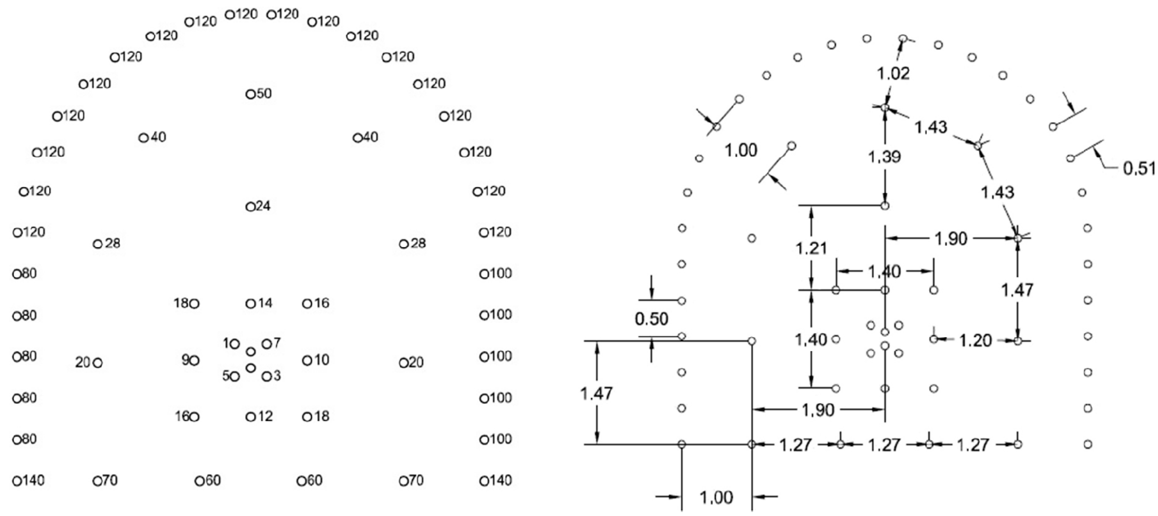

FIG 6 – Scheme of parallel holes cut. Pull – 2.5 m; surface – 30 m²; total charge – 119 kg; detonating cord 40 g/m – 90 m; powder factor – 1.32 kg/m³; holes diameter – 45 mm; delay time – 25 ms. Numbers refer to the delay sequence. Shock tube initiation for the whole blast.

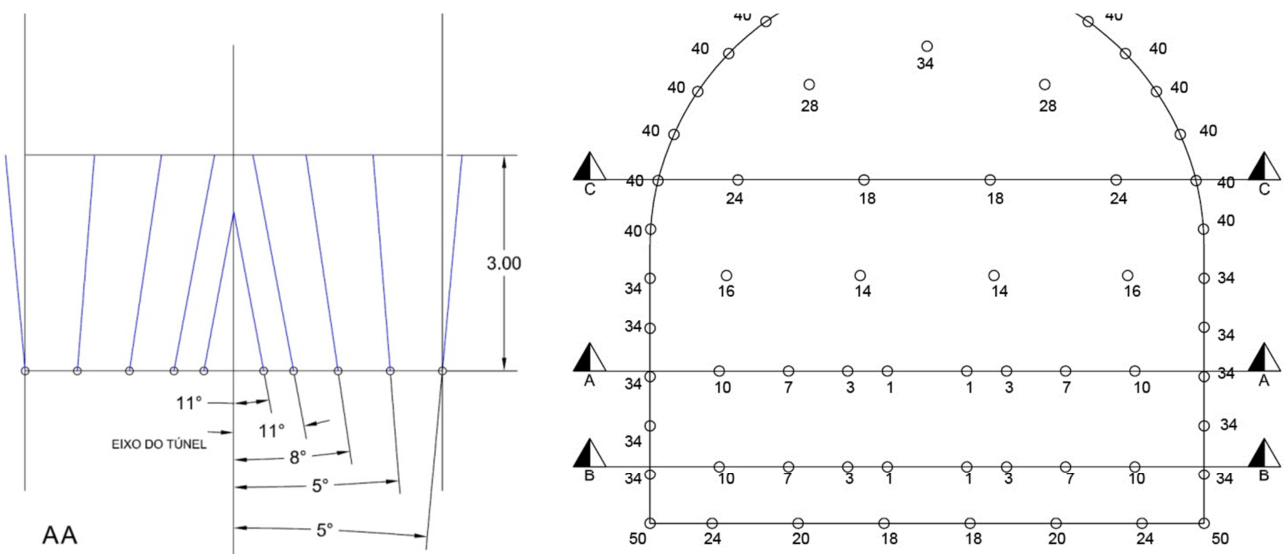

FIG 7 – Scheme of a V-cut. Pull – 3.0 m; surface – 30 m²; total charge – 119 kg; detonating cord 40 g/m – 90 m; powder factor – 1.32 kg/m³; holes diameter – 45 mm; delay time – 25 ms. Numbers refer to the delay sequence.

cast remains visible and the contour is heavily affected by overbreak.

- In underground, when the rock mass can be classified with low RMR classes (poorly competent) and drilling and blasting should not probably have been chosen as an excavation technique on the first place, any quality of the final walls is hardly achieved at all, in spite of any care in the details of execution of smooth blasting.
- Experimental results contrast with theoretical formulae for the determination of the radius of damage. Results obtained on field show smooth detachments that are not contemplated, simulating the effects of the charges with

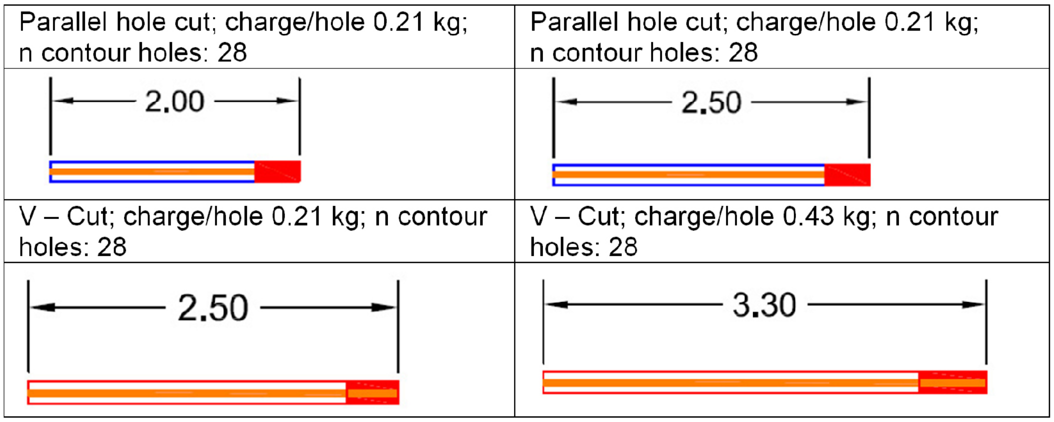

FIG 8 – Details of the charges employed in contour holes. Tunnel face is on the left of the picture.

FIG 9 – The poor quality of the contour obtained is noticeable.

theoretical models. This suggests that models based on the pressure of detonation to determine the radius of damage are still to be improved.

- All results demonstrate that the rock mass features must be included in any design criterion and theoretical approach modelling the effects of contour blasting.

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