Sublevel Stoping at a large underground quarry: studies for the improvement of the production blasting

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In a limestone underground quarry, whose exploitation method and general features are exposed in another paper presented at this Congress, important problems are represented by minimisation of the development/production ratio and by the increase of the productivity, and both call for an increase of the size of the production blasts.

After a synthetic account on possible solutions known from published cases and a detailed description of the current practice, the principles and the first results of a test campaign, aimed to the definition of the maximum safe volume of the blasting charge are presented. The study implies mainly a careful, close proximity seismic survey of the actual production blasts, the geomechanical evaluation of the rock mass strength and structure, and the analysis of the historical seismic data routinely collected by the quarry.

Criteria for an improved blast design and an improved development scheme are presented in the conclusions.

Keywords: optimisation of mining operations; sublevel stoping; blasting pattern.

1. Foreword

The case of a large underground quarry exploiting a limestone deposit for the production of quicklime is discussed also in another paper presented at this Congress, to which it can be made reference as regards exploitation method and general features.
Underground exploitation took the place of surface exploitation, no longer possible mainly for “environmental” reasons, and was accompanied by an increase of the unit production cost, as expected. A part of the cost increase is to be credited to large underground infrastructures (hauling, ventilation and access ways), representing a long term investment, and a part to the drilling tunnels, to the opening of the slot, to the preparation of the draw-points and other works whose cost is sustained by the single stopes (Mancini et al. 2003).

As to the production blasts, specific drilling and powder factor are higher in underground practice, but not dramatically higher than in surface blasts (however, drilling and charging require more sophisticated means). In figure 1 and figure 2 are shown the blasting scheme usually adopted by the surface exploitation, presently discontinued, and the scheme adopted in the exploitation, presently accomplished, of the first underground stope.

| vertical holes | bench height | 15 m |
|               | inclination  | 20°  |
|               | diameter     | 95 mm|
|               | length       | 16.5 m|
|               | spacing      | 5 m  |
|               | burden       | 4 m  |
|               | explosive    | 54 kg TUTAGEX - A |
| horizontal holes | diameter | 76 mm |
|                 | length      | 4 m  |
|                 | spacing     | 5 m  |
|                 | burden      | 4 m  |
|                 | explosive   | 10 kg TUTAGEX |
|                 | Powder factor | 0.231 kg/m³ |
|                 | Specific drilling | 0.068 m/m³ |
|                 | Detonating cord | 0.055 m/m³ |

Figure 1. Blasting scheme usually adopted by the surface exploitation.
2. Reasons for increasing the size of production blasts

The part of the cost increase (with respect to surface operation) due to the stope exploitation can be reduced by increasing the size of the production blasts, and the trend towards larger blasts is commonplace in sublevel stoping exploitation (figure 3).

The cost reduction is attained through the reduction of the number of drilling tunnels, of the number of holes to be drilled and charged, of the interruptions of other works due to blasting; a more coarse fragmentation can be a consequence of the increase of the side of the drilling mesh, but in the case examined is not a drawback, being usually accompanied by a reduction of the unwanted primary fines.
Figure 3. Examples of the evolution of the blasting technique towards simpler blasting pattern and large hole diameters at the mines of Kidd Creek, Canada (after Blakey et al. 1976).

The initially adopted scheme for the stope 1 (figure 4) with three drilling sublevels and one tunnel per sublevel is good as far as the incidence of the drilling tunnels is concerned, but not satisfactory under the aspect of pillar wall regularity. The presently adopted scheme for the stope 2 has two drilling tunnels per sublevel, and makes possible a good control of the pillar walls, being the contour marked by parallel holes, but doubles the incidence of the drilling tunnels (figure 5). To lower the cost of the drilling tunnels, the number of drilling sublevels could be reduced to two, as shown in the same figure, or even to one (a single drilling level at the top of the chamber): in these cases the maximum charge per hole should be noticeably increased, with respect to present practice.
In the quarry considered some features of the production blasting system are due to remain unchanged for some years, for practical reasons: the drilling diameter (76 mm), the initiation system (redundant, with two NONEL detonators, and dynamite primer, one at the bottom and one at the collar of the charged hole, or simple, with one NONEL detonator at the bottom, depending on the length of the hole and on the type of blast: redundant priming is preferably adopted in the rounds aimed to open the basal trough); the type of charge (a continuous column of ANFO or similar, blown by compressed air charger); burden (approx. 3 m); powder factor (approx. 0.3 kg/m³). Hence an increase of the drilling length leads to unavoidable increase of the maximum charge per delay.

3. The research on safe charge limits

In order to safely design blasts larger than the ones presently used, the problem arises of defining a maximum safe value for the charge per delay. By “safe” we mean:
warranting long term stability of the pillars of the stopes exploited or under exploitation and of the tunnels (drilling, haulage, access, ventilation and so on) and other underground voids (filling of the voids upon exploitation can not be seriously considered, and caving is forbidden for environmental reasons).

To establish a criterion for the calculation of the maximum allowable charge per delay, suitable to the design of large blasts, an experimental research, here reviewed, has been carried out, based on the “Blast Damage Indicator” concept. Indeed, existing guidelines aimed to keep under control the seismic effects of blasts, being mainly concerned with the protection of buildings from blast damage, do not provide useful suggestions for the case here dealt with.

The feared side effects of the blasts, as far as the stability of the underground structures are concerned are:

- the rock crushing in the immediate proximity of the detonated charge, with tensile cracks radiating from the charge axis, possibly by 10-20 diameters: against this damage we are protected by the assumption, in the static calculation, of a rock layer adjacent to the wall contributing load (weight) but not bearing power;

- the cracks and possibly detachments, due to tensile stresses arising from the reflection by free surfaces of the pressure pulses radiating from the detonating charge: against this possible damage we must find a protection by respecting a charge-free surface distance relationship, to be found through a purposely planned research.

To this aim, we decided to rely on the “Blast Damage Indicator” (BDI) concept, as exposed by Yu T.R. and Vongpaisal S. (1996), because it is simple and theoretically well founded: the intensity of the transient tensile stress is indicated by the product
\(v \cdot c \cdot \delta\), \(v\) being the peak particle velocity of the rock, \(c\) the propagation speed of the P wave in the rock, \(\delta\) the volumic mass of the rock (the \(c \cdot \delta\) product is usually termed “acoustic impedance”); the strength of the rock mass by a dynamic tensile strength value \((T_d)\), that can be inferred from laboratory tests and from geostructural surveys; the dimensionless ratio of the expected maximum stress to the expected strength \((v \cdot c \cdot \delta / T_d)\), practically the reciprocal of a safety factor, labelled as BDI, is assumed as indicator of the safety level (as far as stability is concerned) of the blasting operation in the considered case.

The development of the research implies the following steps:

- to obtain reliable data on the rock properties: to this aim the data from geological and structural surveys, geomechanical and geophysical tests, lab. tests on cores and so on, already employed for the static design of the quarry, have been utilised. The dynamic tensile strength has been indirectly estimated, as suggested by the quoted Authors;

- to obtain a reliable (and conservative) relationship linking the mass of the charge detonated to the distance and to the peak particle velocity. To this aim, the seismic effects of a number of production rounds detonated in the stope 1, have been recorded by sensors placed in the drilling tunnels of the same stope, of the stope 2, still to be opened, and in the service tunnels (an example of the typical arrangement is shown in figure 6). At the relatively short distance investigated, it has been observed that each detonation of the round gives a distinct recognizable peak in the vibrogram: superposition is exceptional (anyhow an increase of the time-interval is advisable). Therefore it has been possible to measure the charge, the distance from
the centre of the charge and sensor, and the PPV for each hole of the rounds, in most cases.

Figure 6. Above, plan of stope 1 on the left and of the drilling tunnel of stope 2; the sensor is placed in the drilling tunnel. Below, section of the drilling tunnels, with the triangle made by sensor position, drilling centre and charge centre.

- to establish conservative values of BDI to be respected. This has been done on the basis of literature data, being the exploitation at the very beginning.
- to draw the charge/distance relationship to be respected in order to stay within the selected BDI limit values; these relationships represent the practical means to design “safe” blasts larger than the presently employed (present practice seems over-conservative, under this aspect).
3.1 Rock properties

Specific gravity averages 2.670 kg/m³, the P wave propagation velocity averages 6300 m/s (in the direction of the bedding) and 5700 m/s (in the transversal direction), hence for the acoustic impedance we assume 16.82·10⁶ kg/m² s for the direction of the bedding and 15.25·10⁶ kg/m² s for the direction crossing the bedding.

As to the dynamic tensile strength, only data from static tests (C₀ and Brazilian indirect tensile strength) were available; tests had been performed on cores, taken in different directions in the preliminary studies carried out to design the exploitation. We decided to use estimated values, calculated from the static strength values by applying adjustment coefficient as suggested by the quoted Authors. Dynamic tensile strength values of 36 MPa, parallel to the layers, and 25 MPa, orthogonal to the layer (roughly six times larger than the Brazilian strength, and one fourth of the C₀). The dynamic tensile strength should be multiplied by a reducing factor (site quality constant) to obtain a dynamic strength value appropriate to the particular rock mass. As suggested by the quoted Authors, the RMR value known from previous surveys has been adopted: 81% for the surfaces orthogonal to the bedding (favourable orientation) and 69% for the surfaces parallel to bedding (unfavourable orientation – figure 7).

![Figure 7. RMR values depending on the direction. Reported values do not consider the presence of supporting systems.](image-url)
Summing up, the rock is described by the set of data reported in table 1.

### Table 1. Set of data describing the rock mass.

<p>| | | |</p>
<table>
<thead>
<tr>
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<tr>
<td>Specific gravity:</td>
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<tr>
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<tr>
<td>transverse to bedding</td>
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<tr>
<td>Acoustic impedance</td>
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<td></td>
</tr>
<tr>
<td>parallel to bedding</td>
<td>$16.82 \times 10^6$ km/m² s</td>
<td></td>
</tr>
<tr>
<td>transverse to bedding</td>
<td>$15.25 \times 10^6$ km/m² s</td>
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<tr>
<td>Dynamic tensile strength</td>
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<tr>
<td>transverse to bedding</td>
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</tr>
<tr>
<td>prop. transverse to bedding</td>
<td>0.69</td>
<td></td>
</tr>
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</table>

#### 3.2 PPV / charge / distance relationship

The square root scaled distance formula has been adopted, not because of some particular merit, but because is simple and “statistically” the most commonly used; other mathematical expressions do not provide sizable advantages; the expression used is: $PPV = k \cdot (R/W^{1/2})^n$ where K and n are constants which characterise the site, R is the distance and W the charge per delay.

The dispersion of the experimental data with respect to the interpolating line is expected to be very great (as usually happens in test campaigns aimed to define the low scaled distance side of the relationship, as in this case), therefore, being our aim to define a conservative relationship, the exponent has been obtained from the data by the regression analysis, and the coefficient has been increased as much as needed to have only 10% of the experimental points above the line representing the expected values. As indicator of the PPV, the peak of the vector sum of the three components of the particle velocity has been assumed; the distance considered is measured from the sensor to the midpoint of the charged explosive column. The results of the vibration measurements and the relationships that will be used to evaluate the BDI are showed in figure 8 and figure 9.
Figure 8. PPV / scaled distance (DS) relationship, in the case of direction parallel to the axis of the stope, and transverse with respect to bedding. Triangles represent experimental points; dotted line the average value; continuous line the upper value, adopted as site law.

Figure 9. PPV / scaled distance (DS) relationship, in the case of direction orthogonal to the axis of the stope, and parallel to bedding. Diamonds represent experimental points; dotted line the average value; continuous line the upper value, adopted as site law.
To be noticed, a marked difference in the attenuation is observed between seismic path parallel to bedding (that means, measuring points located in tunnel parallel to the axis of the stope) and transversal (that means, measuring points in the drilling tunnels and in the access tunnels); though being the joints closed, the effect of the joints on the propagation of the chock wave is important.

3.3 Safe values of BDI to be respected

The most feared event is the development of cracks in the pillars due to production blasts, in particular when one stope is already exploited and the next is under exploitation: even cracks parallel to the wall and not emerging from the wall surface are dangerous, because can transform a solid pillar in a packet of mechanically separated slender pillars, of much lower strength.

Damage to the drilling tunnels, apart from back-break that should be avoided by an appropriate charging scheme, is less likely to occur because the chock wave does not hit normally the free surface. Other tunnels are at greater distance from the blast than the free wall of the pillar.

For the most critical condition, above mentioned, we assume conservatively the BDI value 0.125, suggested by the quoted Authors as perfectly safe (having in mind the physical meaning of BDI, it can be said that the safety factor suggested is 8). For less critical conditions a BDI of 0.25 can be adopted.
3.4 Safe charge limits in production blasts

From the rock data quoted in 3.1 a BDI / PPV relationship is easily obtained, for the parallel and transverse direction of the seismic paths (figure 10).

![Graph showing BDI / PPV relationship](image)

**Figure 10. BDI / PPV relationship for the parallel and transverse direction of the seismic paths. The equation BDI = 0.557·PPV refers to the parallel direction, the equation BDI = 0.882·PPV refers to the transverse direction.**

It is therefore easy to find the PPV value corresponding to a given BDI. The PPV /charge / distance relationship can be converted to a BDI /charge / distance relationship, and, for a given BDI value, to a “safe charge” / distance relationship. Calculations have been developed for the BDI values of 0.125 and 0.25. The result is shown in figure 11.
Figure 11. “Safe charge” / distance relationship in function of BDI values. Continuous line refers to a direction parallel to bedding; dotted line to an orthogonal one.

Being the thickness of the pillar at least 30 m, production charges can not be closer than 30 m (28 m, if allowance is made for a couple of metres of incompetent rock adjacent to the wall, as conservatively has been done in the statical calculations) to the opposite face of the pillar separating the already exploited stope from the one under exploitation.

For a distance (R) of 28 m, the charge giving $BDI = 0.125$ is more than 950 kg, which greatly exceeds the current practice, even in the case of simultaneously firing of several holes.
On the other side, a hole with a diameter of 76 mm can contain no more than 4.5 kg/m of explosive of the ANFO type, hence even a blasting scheme with a single drilling level, requiring holes perhaps 70 to 80 m long and single charges of 300-400 kg could be designed (however, it is not advisable to use a so small diameter for so long holes).

Timing does not represent a problem, being possible to have as many delays as needed with the NONEL System.

The underground structures (access, haulage and ventilation tunnels running transversal with respect to the axis of the production stope) can pose some restriction in the slotting (anyhow, it is not realistic to employ larger charges in slotting) and in the last rounds, completing the exploitation of the stope (see dotted lines of figure 11).

4. Conclusions

The analysis has shown that the increase of the size of the production blasts does not meet serious restrictions. Obviously the change will be gradual, because the BDI limits considered in the analysis came from literature rather than from observation, due to the comparatively recent introduction of the underground exploitation method in the quarry, and the effects of changes will be kept under careful control. With no changes of drilling diameter and powder factor the maximum volume blasted per round can be raised, from the present value (approx. 3300 m³) to 5000 m³ (with two drilling levels) and even to 10000 m³ (with a single drilling level). The latter achievement however probably requires also an increase of the drilling diameter (that should entrain a further increase of the blasted volume, through an increase of the burden).
The general exploitation plan is elastic enough to accommodate for further changes that may be suggested by technical progress in the 20 years to come.

References

