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Going Underground in Quarrying: Technical Perspectives for Marble in Portugal

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ABSTRACT: This paper examines the application of the criteria for the alternative choice of underground quarrying in ornamental stone. In these quarries, the rock mass quality is usually good, and in most cases exploitation is carried out by means of open pit methods. Underground quarrying in ornamental stones is possible when certain prerequisites are satisfied: geostructural features, stability contraints, land planning, and when the recovery and safety goals are achievable. After a general discussion of these items, the paper analyzes the case of the pink marble of the Alentejo basin in Portugal, in which a first tentative design method and monitoring approach has been proposed. The development of open pit exploitation in Alentejo has determined the creation of very deep pits from the flat ground surface: in some cases, a depth of one hundred meters has been reached with vertical lateral surfaces

1 GENERAL FEATURES

The production of ornamental stones takes on noticeable importance in the sector of natural building materials, both from the economic and technological point of view.

The quarries are exploited by open pit methods, but there are several cases in which exploitation is actually carried out underground: Italy, Croatia, Portugal, Greece, and France are example of countries where these methods are applied.

Underground exploitation of dimension stones will be proposed increasingly as production methods, not only in cases where the underground option is imposed by the features of the rock mass as in the past, but also for a number of reasons that have gained weight in recent years, make underground exploitation preferable in terms of economy and in cases where surface operations are technically feasible.

The underground option can be necessary or preferable when: 1) the surface is very dipping and irregular (mountain areas); 2) me external surface is regular but the overburden is thick; 3) land costs and reclamation taxes are too high; 4) the selected rock mass is confined in a well-defined ore body; and 5) safety requirements and stability features are no longer suited to open pit methods.

Five elements should be evaluated when the underground choice has to be considered: 1) good structural conditions of the rock mass (in ornamental stone quarries this aspect is generally satisfied, even though there are very particular rock mass conditions, such as the case of stratified rocks with very thin clay filling); 2) technology of the excavation (mainly mechanical cutting to separate the blocks from the mass); 3) commercial features (some properties, for example colour and grain size, are not always the same, due to the limited homogeneity of the rock mass, where change of colour, and stains and inclusions can be encountered); 4) economic profitability compared to the costs of open pit exploitation, taking into account the probable lower recovery due to underground support structures such as pillars, but also the savings on overburden removal and muck disposal; and 5) safety and environmental reclamation.

The stability features in this particular type of void should be ensured on a long-term basis, without significant contribution from artificial supports.

The geostructural conditions determine both the design of the excavations and the methods that should be used to separate the blocks from the faces. The availability of structural data for rock masses where some underground quarries have been developed and technical results after some years of exploitation can allow one to consider the above-mentioned elements in a critical way, with an emphasis on the first two elements in particular. The collection of statistical data on rock mass structure and block recovery should become a regular procedure in ornamental quarries.

2 THE MARBLE QUARRIES IN ALENTEJO

The pit quarries in the Alentejo region have two types of problems. The first is a geomechanical problem, and it is linked to the redistribution of stresses under the new geometrical configuration, where a lateral confinement is no longer applied and therefore the instability of lateral walls is possible when discontinuities occur in unfavourable orientation.

The second is a technical and economic problem when changing the excavation method to go underground. In this case, the underground adits are excavated directly in the productive and massive rocks, avoiding the removal of ovreburden. It is necessary to maintain stable structures in the rock (rib pillars and eventually thick horizontal beams) without excessive loss in block recovery .

This study, on the basis of geostructural surveys and geomechanical tests, describes the modelling of the rock excavations, as the first answer to support the industrial decisions for the development of the exploitation, taking into account the particular conditions of the area. The study has the purpose of suggesting a possible formulation of a tunnel site for the feasibility of successive underground mining.

The Estremoz-Borba-Vila Viçosa basin is situated about 100 km East of Lisbon. The anticline in general presents a northwest/southeast immersion direction, with inclinations that vary from 15° to 75° on the southeast side to vertical on the northwest side.

The lithologies involved in the mining are different-coloured marbles: blue marble, cream marble, pink marble, etc., but also primary dolomites which are locally called Cascalva rock.

The area appears, from me orographic point of view, to be made up of slight and gradual land reliefs which are frequently sub-level. The quarries are practically all located in plain areas and are exploited by means of a pit whose vertical walls reach an elevation of several tens of metres. The mining involves rock mass portons with altered coverings of soil and rock of variable thickness, but which are of the order of 5-15 m. The transition between covering and rock in some places is not complete because of the presence of karst voids later filled with soil material that goes deep into me substratum.

The examined quarry takes up an area of about 21,250 m^2, and 5,000 m^2 of this is subject to mining.

The quarry has been mined down to eight levels; the base yard at the moment reaches a depth of 35 m. The presence of joints and karst, sometimes even at a depth, creates a rather important static and mining problem to which block recovery is connected. Furthermore, the destressing phenomenon due to excavation and the final removal of the blocks from their position further increases the negative effect of the joint systems.

Block recovery, which has so far been carried out through open pit mining, is, however, satisfactory. The rock overall appears to be quite compact and in particular at least one homogeneous and accessible area can be recognised, so that the formulation of underground quarrying can be considered.

In the northeast area of the quarry, at about 58 m in depth, the exploitation intercepts the primary dolomite layer. As this layer is not encountered in the southwest area, it is assumed that it runs at a deeper level, perhaps at 70 m or even more. As the pink marble is above the primary dolomite, the southwest area would seem more suitable for the experimentation of underground mining, also because the stone presents better characteristics for ornamental use: reduced jointing and probably greater cubature.

2.1 *Geomechanical features*

The geomechanical characterisation of the studied material was performed at the I.S.T. laboratories in Lisbon. Different tests were carried out on the numerous samples that were collected (85) and then a statistical elaboration was performed and me geomechanical parameters of the rock were defined on the basis of these results.

The tests were performed on four denominated lithological classes: azure marble (MA), clear marble without veins (MSV) and dolomite (D), obtaining the results given in Table 1.

The characterisation of me rock also involved in-situ tests that were carried out to establish the acting stress state.

This was determined through die reset of the deformations using a flat jack technique.

This involves the measurement of a cortical stress state which is influenced by the condition of the walls in which the measurements are performed, which can limit the significance of the measurements. For this reason, a back analysis was performed in order to make the results of the measurements congruent due to external factors, with the optimal configuration of the model, and to evaluate the geomechanical parameters that best represent me problem under examination.

In order to obtain a complete characterisation of the rock mass, it was necessary to evaluate the mass parameters, after having determined me parameters referring to the test scale, to insert into the modelling.

At this point, it was possible to approach the design stage of the work, imposing the stability analysis for different mining levels.

Table 1. Main mechanical properties of rock materials: in columns 6 and 7 C and cp are the shear strength parameters; in columns 8 and 9 c and ip are the discontinuity parameters.

Rock type	σ_c [MPa]	E [Gpa]	ν [-]	σ_t [MPa]	C [MPa]	φ [°]	c [MPa]	φ [°]
D	46	43	0.31	3.6	15	37	0.77	38
MA	64	59	0.38	4.0	10.0	47	0.54	40
MSV	94	66	0.22	5.2	14.5	47	2.15	32
MCV	61	57	0.40	4.5	8.7	48	0.08	40

3 DESIGN GUIDELINES

The excavation of ornamental carbonate rock is today carried out almost exclusively with mechanical cutting techniques. Diamond wire saws and chain saws are used above all; they have reached such high levels of performance that it is actually disadvantageous not to use them in an efficient production process.

The importance of using such equipment during mining is so great that it conditions the geometry of the underground chambers. In other words, efforts are made to "adapt" the geometry of the chamber for optimal performance of the chain cutters in order to optimise the yield of the cutters, which, during the production stage, can affect the costs of extraction remarkably. In the case under examination, it was considered appropriate to visualise chamber and diaphragm mining on a single level, which could then be converted into a classical room and pillar scheme, with the width of the room being equal to that of the diaphragm (9 m), and a height of 13.5 m. This precautionary choice was adopted to compensate for a certain margin of uncertainty in the results obtained from the design, which is inevitable in work of this kind. Once the mining has been started, the objective is to integrate the classification of the rock mass with new data so as to be able to take best advantage of the whole of the useful deposit.

The geometry of the underground rooms is also naturally influenced by the static state of the work one wishes to perform. In rock masses similar to that under study, the geomechanical behaviour would allow the continuation of mining in chambers similar to those just described on two superimposed levels. In this case, however, it would be necessary to precisely respect the geometry imposed at the upper level in order to avoid dangerous stresses outside the axis. This could influence the yield of the blocks, this being of great importance as this is a commercial activity which tries to obtain the greatest possible profit.

The planning of a work of mis kind can be approached in different ways: analytical methods based on theoretical simplifications are easy to apply, but supply results that, in this case, do not reach a good level of reliability; graphical methods are still quite simple to apply, but, like the previous methods, do not satisfy some indispensable planning criteria for a work similar to that dealt with here; numerical methods, though articulated and complex in their calculation procedures, can supply more precise values with a good margin of reliability. Numerical methods can be distinguished according to how one considers the rock mass. Finite element calculation codes (FEM) and finite difference codes (FDM) consider the rock body as a continuous body. Another procedure, expressed by distinct elements (DEM), considers the rock mass as a discontinuous body.

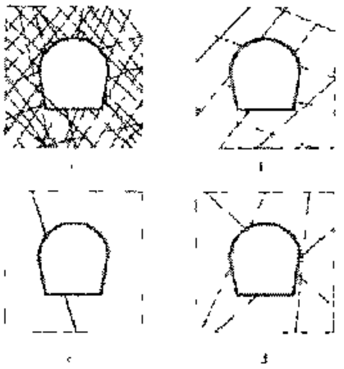

Figure 1 Geometrical schemes for modelling a) Pseudo-continuum using continuum approach (FEM, FDM); b) discontinuum approach (DEM, wedge analysis), c) continuum approach (FEM, BEM), d) intermediate case, continuum with few joints (DEM, stereograms)

From Figure 1 it is evident how difficult it is to identify the model that faithfully reproduces the behaviour of the rock mass in question. For this reason, the analysis was carried out using two different models.

The first one was that of finite differences FDM (FLAC code - Itasca Consulting Group). This model consists of 35070 quadrilateral elements with dimensions of 0.5 x 0.5 m^2 In its central part, where the mining chambers were simulated. This model allows one to analyse the rock mass around the chamber for a total width of 225 m and a height of 140 m, in such a way as to position the artificial borders of the model so that the stress and strain disturbance produced by the presence of the

Chambers is not felt (that is, at a distance in which the equitensional lines in the rock appear horizontal in correspondence to the artificial borders of the model). The upper border of the model is real and corresponds to the surface of the soil.

In order to optimise the process, the dimensions of the elements were made to increase as they became distant from the chamber, in relation to the lowering of the stress gradient, so as to limit the total number of elements and to reduce the computational burden.

Three contiguous 9-m-wide and 13.5-m-high rooms are visualised, and excavated on one single level.

At the same time, a second model of the distinct elements DEM (UDEC code - Itasca Consulting Group) was set up.

The second model takes an area 130 m wide with a depth of 90 m from ground level into consideration. This is necessary to get around tbe border effects, which can reflect on the conditions of the chamber under examination. The rock mass is divided into three areas, with different jointing levels, in order to better represent reality and avoid excessive calculation volumes that would need to be processed, which could lead to overflow errors. The first area, the most jointed, starts from the ground level and descends to a depth of 12 m; the second, subject to the excavation of the chamber, starts from a depth of 12 m and reaches a depth of 65 m; the last starts from a depth of 65 m and continues to a depth of 90 m.

The discontinuities inserted into the model are: Ki = 292/90 and K_2 « 255/64, while the attitude remains constant, as anticipated, and the values of the length of the discontinuity and spacing vary; in particular, both increase when passing from the surface area to a depth.

The attitude data refer to the central values of the frequency distribution; both the length and the spacing refer to the in-situ geostructural analysis. The mechanical characteristics of the rock, also identified from İn-situ and laboratory geomechanical analysis, are applied to all the model. However, the geomechanical parameters of the discontinuity change according to the area. The boundary conditions foresee a horizontal pressure that varies according to the depth, from the left side of the model, the other is blocked by a series of horizontal constraints (simple supports). Another series of simple vertical supports is placed on the lower border of the model.

When the results of the models are compared, the optimal correspondence between the two models of the stress analysis is first shown; however, as far as the deformations are concerned, the distinct element model has resulted in being more sensitive to the presence of the blocks, which are relatively small in comparison to the size of the chamber, that is, it influenced the result in correspondence to the walls of the chamber.

At this point, having positively evaluated the compatibility of the stress state induced by the excavation with the resistance of the rock, it was necessary to evaluate the stability of the portions of rock isolated by the mining operations on the roof and walls of future chambers. In practice, an analysis was performed according to the K-Block theory developed by Goodman and Shi (1985).

This analysis, performed with the attitude data of the discontinuities found on the external walls, only provides results indicative of the reality. This analysis showed in particular the high probability of the formation of blocks, both on the walls and on the roof, that can in some cases be considered unstable, both due to the geomechanical characteristics and the attitude of the discontinuities. In this case, it is necessary to intervene with local stabilising, that is, bolting of sufficient length to pass the destressing area with the anchorage.

4 OPERATIVE PROCEDURES

Underground mining occurs in two distinct stages: the opening, with the excavation and enlargement of the tunnel, and lowering, with the mining of the chambers through gradual lowering.

The extraction techniques used in the second stage, that İs, from the development at the top to the squaring of the blocks, are the same as those used for open pit mining; on the other hand, the excavation of top tunnels anticipates specific techniques.

The advancement operations can be performed with two different cutting configurations; one with only the use of a chain saw, as in the classical type of tunnel, while in the other this type of machine is used together with a diamond wire cutter.

Vertical and horizontal cuts are performed with the first type of technique in order to isolate a portion of the rock that corresponds to the chamber face. The cut at the back is performed using hydraulic cushions which, through vertical direction pressure, break the rock by bending.

With the second technique, a smaller portion of the rock is detached using a hydraulic jack and then the cut is performed behind using a four-pulley return frame. A noteworthy advantage of this method is the ability to control the direction of the cut.

At present, specific frames have appeared in some sites, with chain cutting machines mounted on self-propelled tracked vehicles.

This could simplify the overall cutting operations, leading to a time reduction of 20% with a saving of manpower of at least 30%. This innovation, which is potentially important in underground marble excavations, could thus improve feasibility in cases like the one under examination.

In both cases, the excavation anticipates the removal of a wedge of variable dimensions that are dictated both by the characteristics of the machine and by the space required to carry out subsequent work correctly.

5 NUMERICAL MODELLING

The numerical modelling led to the detailed evaluation of the stress state induced in the rock following excavation operations. The creation of the anticipated mining rooms was simulated in stages. Two different hypotheses were formulated concerning the geometric shape of the rooms: first rooms with dimensions of 10.5 m in height and 9 m in width on two levels (case 1), then rooms with dimensions of 13.5 m in height and 9 m in width on one single level (case 2).

The stresses calculated in the pillars and in the horizontal beam are of particular interest. These are necessary for evaluating the project thicknesses of these natural support elements and for suggesting the level and modality of any possible partial recuperation of the diaphragms in a second successive stage.

The vertical stresses in the rock at the end of construction of the six chambers for case 1 are shown in Figure 4. It should be noted how the roof of the lower chambers is subjected to traction stresses in the vertical direction and that these involve almost the entire thickness of the beam (6 m). From Figure 4 it is also possible to see a stress state inside the lower pillars that is slightly higher than that which develops in the higher pillars. The maximum values, however, do not exceed 2.5 MPa. The calculation allows one to show how the horizontal stresses in the pillars do not exceed 0.25 MPa and a localised presence of horizontal traction stresses is registered in the higher pillars at half of the borders. Horizontal stresses are noted in the beam that are always higher than 1.5 MPa, while in the central part they are always higher than 1.75 MPa. They never, however, exceed the limit of 1.85 MPa.

A more detailed examination of the stress state in the pillars can be made from Figure 5, which refers to the pillars of the column on the left.

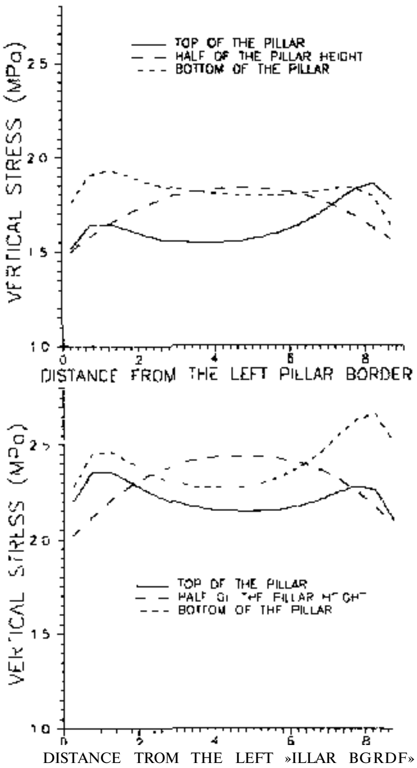

Figure 3. Vertical stresses in the upper left pillar (upper) and lower left pillar (lower) (case I).

In Figure 3 it is possible to see how in the upper part and in the lower part the vertical stress state of the pillar suffers the effect of the corner, and therefore the maximum values are concentrated towards the edges. In the central part, the maximum values are instead in the centre of the pillar, where the lateral confinement is greater. It is possible to determine an increase in the mean vertical stresses of about 0.6 MPa passing from the higher to the lower pillars. From the calculation, a certain symmetry is furthermore evident between the stress conditions in the pillars on the left and those of the right of the hypothesised mining scheme.

The studied mining scheme is adequate for the stress state that is present and for the geomechanical characteristics of the rock mass. The stresses induced in the pillars and in the horizontal pillars are, in the configuration of case 1, very much lower than the maximum permitted values. The chambers turn out to be stable on a large scale, except for the necessity of intervening with precise reinforcement elements to eliminate any possible danger of movement of unstable rock blocks. Given the low values of the vertical stresses induced in the diaphragms, it has been possible to examine the possibility of dividing them, attacking tiansversally, so as to create chamber and pillar mining on various levels.

The results for case 2, referring to the situation that occurs at the end of the excavation of the three chambers, are shown in Figure 3.

The vertical stress state in the pillars does not change substantially in comparison to the previous scheme. The width of the diaphragm increases slightly (from 3.23 to 4.15), remaining, however, at very low values.

As far as the horizontal stresses are concerned, a horizontal strip of limited height can be seen at the middle of the pillar (3.7 m at the borders and 50 cm at the centre) that is subject to traction stresses. This strip was not present in the scheme that had chambers 10.5 m in height (case 1).

The mean vertical stress inside the pillar is, in this case, a little lower than 2 MPa.

From the results of the numerical calculation obtained for the two different hypothesised geometrical configurations, the following can be stated:

- the stresses induced at the borders of the mining chambers and inside the pillars and the beam are compatible with the resistance characteristics of the rock mass, so much so that no areas were revealed in which the elastic limit was exceeded (plastic areas);
- even heights of the chambers of 13.5 m can be considered permissible;
- there are limited portions of the rock that show traction stresses in die vertical direction, horizontal direction or in both directions, which require careful observation of the conditions of the rock mass in order to verify the possibility of unstable blocks breaking off;
- the lateral thrust coefficient KB results in influencing the distribution of the stresses inside the mining voids;
- it is possible to consider a second mining stage that can attack the diaphragms, thus increasing the recuperation coefficient of the deposit; an initial quick analysis of the stress state of the diaphragms led to the belief that a further 50% recuperation of the diaphragm would be possible, which would mean leaving 9 m x 9 m pillars in situ;
- the displacements relative to the borders of the mining voids are very limited, being less than millimetric.

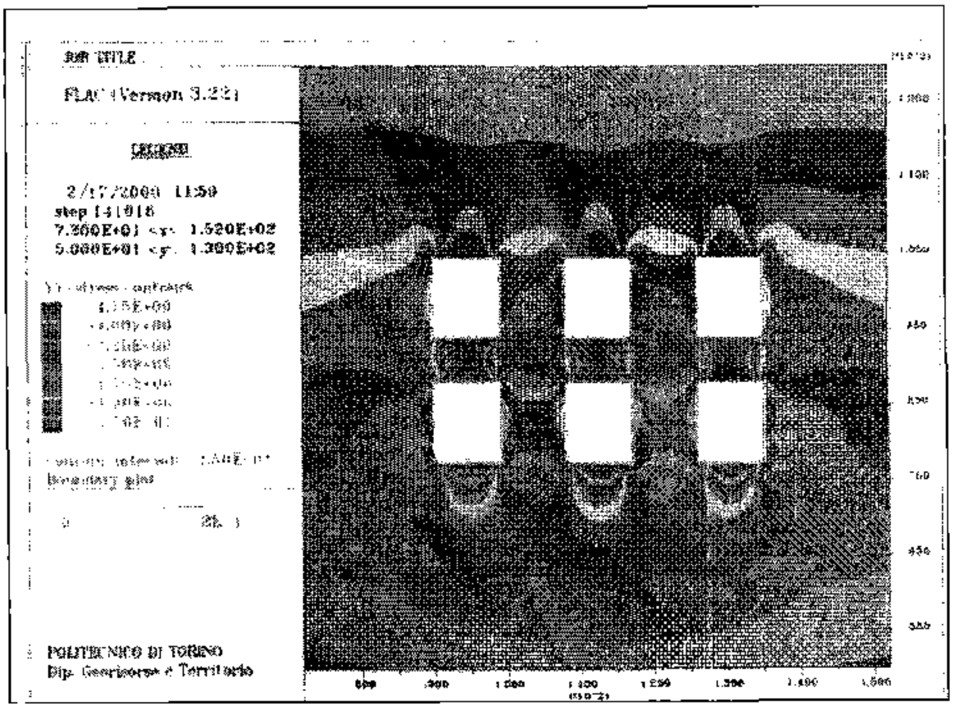

Figure 4. Vertical stresses in **the** numerical **model at the end** of **the** excavation of the six mining chambers (case I).

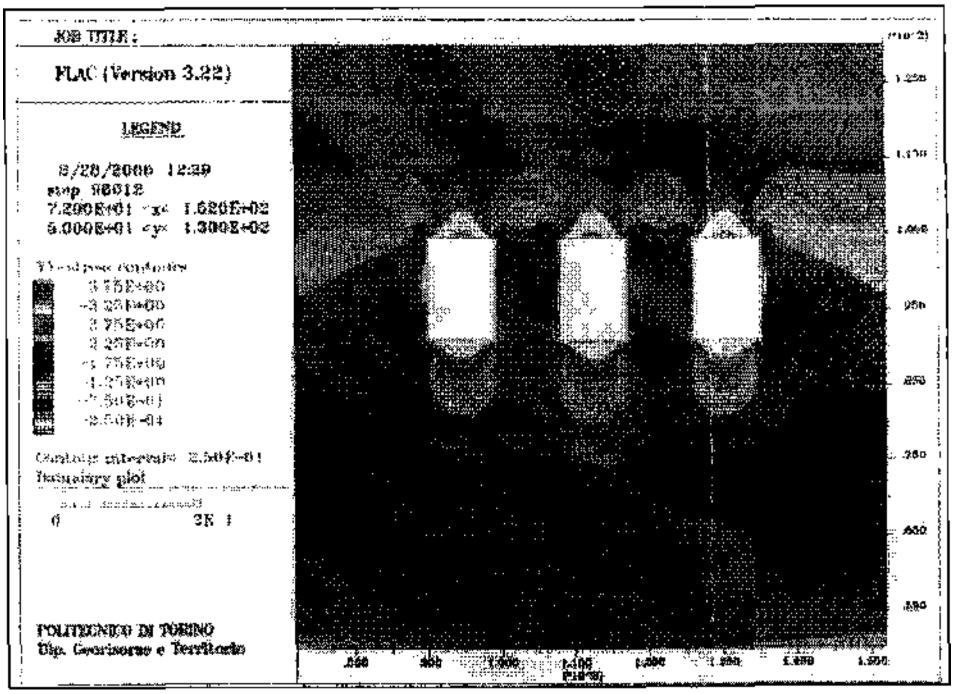

Figure 5. **Vertical stresses** in **the numerical model** at **the end** of excavation of **the** three **mining chambers** (case 2).

6 MONITORING

There are basically three aims of monitoring underground excavations for ornamental stone quarries: a) validation of die hypothesis of die numerical modelling by means of in-situ measurements (back analysis); b) control of the behaviour of natural and artificial structures; and c) updating of the stress state and geostructural conditions.

The directly measured parameters are usually displacements, stresses and loads, whilst the use of indirect methods allows one to define the fracturing state of the rock or the related physical properties (mechanical wave velocity and other geophysical parameters).

The convergence measurements can be performed next to the portals and across transverse sections of the rooms, using long-base extensometers equipped with undeformable wire and Potentiometrie transducers. In the same way, crack meters with Potentiometrie transducers can measure the movements across die walls of the discontinuities. One or two monitoring sections can be quipped with rod extensometers, also passing dirough the entire pillar, eventually equipped with vibrating wire transducers.

Induced stress measurements in the pillars can be performed using flat jack equipment, but the measurement points must be repeated adequately because of die gradient of the stress near the corners.

Alternatively, some measures can adopt the surface overcoring technique with a large diameter of the overcoring, or the overcoring in the borehole with the doorstopper method.

7 CONCLUSIONS

An initial study concerning the technical and economic feasibility of underground mining of a marble rock mass has been developed in this work. The results of significant geomechanical and structural investigations and geognostic measurements, which can be further investigated, were already available for the studied case thanks to the presence of extensive functioning sites on the surface. This led to the field of hypotheses being cut down, so that operative proposals could be developed, both as far as any further investigations and the experimental starting-up of the tunnels are concerned. This stage is fundamental in compensating for the inherent difficulty of extrapolating reliable data relative to the parameters of the rock mass directly from the geomechanical classification, whose applicability seems to be problematic for rock masses of good to optimal quality.

Two different numerical models of the stress and strain analysis of the rock mass were set up for prediction of the development of the underground production sites. The first, based on the concept of "continuous equivalent", led to an evaluation of the static situation on a large scale, with particular attention being paid to the work conditions of the pillars in abandoned rock. The second, which was instead based on *the* distinct element method, was able to analyse the stability of the underground chambers in more detail, evaluating the behaviour of the potentially unstable rock blocks at the borders of the voids. These latter results were compared with those obtained from more traditional calculation approaches, which refer to the well consolidated "block theory" and "limit equilibrium theory".

The results that have been obtained, apart from guiding technical and operative choices in the development of the underground caverns, also supply an overall view of the static situation, in terms of both production and safety, and constitute operative support for planning and updating, even during the work process. Adequate monitoring and control programmes should allow timely decisions of an operative nature. In particular, given the small nature of the expected displacements, it was decided to cany out measurements of the stress state, even though they were limited to the edges of the mining voids. The continuous comparison, through back-analysis procedures, of the measured data and those obtained from the numerical calculation with the models that were set up, will allow an improvement in me reliability of the results and make it possible to optimise the project through the increase of the recovery percentage of the rock mass.

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The authors have contributed equally to this work.

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