Modelling and Design of an Underground Slate Room & Pillar Mine

L.R. Alejano¹, J. Taboada¹, E. Alonso¹ & F. Varas²

¹ Natural Resources and Environmental Engineering Department
² Applied Mathematics Department
¹² E.T.S.I.I. y Minas, Universidad de Vigo, E-36.201 Vigo, Spain

Abstract: In this paper it is studied the geomechanical and economical viability of the underground mining of a slate deposit, which has been quarried up to the moment. Firstly, the geology is described. Then, the characterization of the rock mass is carried out, starting from field data and laboratory tests. According to these parameters, the feasibility of the room & pillar mining method is assessed by means of rock mechanics standard criteria, which allowed to propose a preliminary mine design. Then, a numerical simulation is carried out, in order to control mine behaviour. Finally, the project is analysed from a mining-economical point of view.

Key Words: underground room & pillar mining, slate, FDM modelling.

Introduction

Vilarachao Slate Quarry is located in the region of Galicia (NW-Spain). Slate has been mined there for the last fifteen years. Economic data have been recently taken showing that the ratio -m² of waste/ tons of mineral- was dangerously increasing, and that it could attain not-economic values soon. So it was decided to carry out a feasibility analysis in order to evaluate the possibility of the underground room & pillar method mining.

Geology

The quarry is located in a very folded Ordovician sedimentary area which suffered low level metamorphism in its geological history. Tectonically, the zone presents very tight NS folds affected by thrusting. In the quarry zone the following strata could be identified from top to wall, as Fig.1 shows:

1. Aguilera formation slates.
2. Slate with quartz cm. lenses (100 m).
3. Transition Top bed. Slate with quartz and sandstone mm. interbeded (8-10 m).
5. Transition wall bed. Geologically similar to 3 (10-15 m).
6. Wall quartzite.

Fig.1. Geological scheme of the quarry
The mineral bed dips between 20 and 25°. This geological scheme has been obtained from borehole and outcrop data.

**Rock mass field characterization**

The field work involved taking enough data related to the presence of discontinuities in the outcrops and rock cores in order to classify the rock mass according to standard classification systems -Bieniawski’s RMR and Barton’s Q-. As a result, average values of RMR = 63 and Q=6.8 were obtained.

A good number of samples were taken to perform laboratory standard rock mechanics tests including: *U.C.S.* tests with Young’s modulus and Poisson’s ratio measurements, triaxial, Brazilian, direct shear box (on cleavage weakness planes) and density tests. Most part of these tests were performed on samples coming from the mineral bed, whose mechanical behaviour is a key factor for the general stability of the mine. However, some tests were carried out on representative samples of the rest of the materials.

**Rock mechanics standard preliminary design**

This empirical point of view is based on the rock mass classification systems and on the tributary area analysis of pillar support. The former helps to estimate maximum (drift-shape) room width whilst by means of the the latter the safe pillar width can be obtained.

A conservative design is initially preferred due to the features of this type of exploitation which are as follows: high amount of reserves, lack of mining experience, need of simplicity and safety and low costs of rock support. According to future developments and in-situ tests this initial design could be optimized.

The obtained RMR and Q values together with operational requirements made the authors to propose the following size and shape for the rooms: 15 m high and 20 m wide rooms with the upper part semicircle-shaped, being the radius of this circle 10 metres. The required support is presented in Fig. 2. The structurally controlled failures -rock wedges and falls- are analyzed checking that this support can cope with this type of failure.

It has also been calculated that for the length of rooms -200 m long- three of them will allow the work for at least five years.

The tributary area analysis provides a safety factor for every room and pillar width input, starting from the estimated *U.C.S.* of the pillar -taking into account shape and size corrections- and from the depth of the mined bed, as shown in table 1.

The tributary area analysis is applied according to the above presented data -geology, geometry, mechanics- for a conservative value of the safety factor =2.5. As a result, the width of the 1st and 2nd pillars -separating the 1st and 2nd and the 2nd and 3rd long rooms- must be 15 and 20 m respectively.
Table 1. Tributary area analysis

<table>
<thead>
<tr>
<th>Factor of safety</th>
<th>$F_{o.S.} = \frac{\sigma_v}{\sigma_p}$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Vertical stress in the pillar</td>
<td>$\sigma_v = \rho \cdot g \cdot h \cdot \frac{W_p \cdot W}{W_p}$</td>
</tr>
<tr>
<td>U.C.S.</td>
<td><strong>Size correction</strong></td>
</tr>
<tr>
<td>of the pillar</td>
<td>(Hoek-Brown)</td>
</tr>
<tr>
<td></td>
<td>$\sigma_p = \sigma_{c0} \cdot \left( \frac{0.05}{d} \right)^{0.18}$</td>
</tr>
<tr>
<td></td>
<td><strong>Shape correction</strong></td>
</tr>
<tr>
<td></td>
<td>(Obert &amp; Duval)</td>
</tr>
<tr>
<td></td>
<td>$\sigma_p' = \sigma_p \left( 0.778 - 0.222 \cdot \frac{W_p}{H} \right)$</td>
</tr>
</tbody>
</table>

* $\sigma_{c0}$ is the minimum observed U.C.S. in the laboratory tests with standard rock core samples.
** $W_p$ is the average diameter of the pillar, taking into account its width and length.

Material behaviour parameters estimate

A painstaking characterization of the slate bed is carried out, since the exploitation rooms are mined in it. As far as the material behaviour model is concerned, and according to the authors’ experience, it is considered clasto-plastic. Its strength criterion is taken as anisotropic, due to the fact that slate presents very marked weakness planes, known as cleavage. This means that this rock may fail either by these cleavage planes or across the material itself. In order to analyze what happens once the material has been broken, post-failure or residual strength parameters are needed too.

This behavior model is already implemented in the code used, and it requires the following parameters: elastic Young’s modulus, Poisson’s ratio, density, and peak and residual cohesion, friction angle and tensile strength of the material and cleavage planes.

Deformability parameters of the slate

In this type of rock masses, the authors have observed that to estimate Young’s moduli starting from RMR rating, the better approaches are Serafim & Pereira’s (1986) and Afrouz’s (1992):

\[
E_{RM} = 10^{(0.60R - 10) / 40} \quad (1)
\]

\[
E_R = E_{RM} \cdot e^{-0.0217 \cdot RMR - 2.17} \quad (2)
\]

Where $E_{RM}$ and $E_R$ are the Young’s moduli of the rock mass and the rock respectively. Expression (1) applies to hard rock masses, whereas formula (2) applies to soft rock masses presenting more or less horizontal stratification or joints. An average value of $E_{RM} = 20$ GPa is obtained, starting from the laboratory intact rock Young’s modulus -38.77 GPa- and from RMR, and considering both formulas.

Poisson’s ratio is supposed to be equal to the one obtained in laboratory for intact rock and then, $\nu = 0.19$. Measured density is 2680 kg/m$^3$. 

351
Strength Parameters of the slate

Values of $m=14.1$ and $\sigma_c=192$ MPa have been obtained for the intact rock. This is done starting from uniaxial and triaxial compressive strength tests on rock samples and according to Hoek-Brown criterion. To estimate the intact and broken rock mass parameters $m$ and $s$, Hoek-Brown formulae depending on $RMR$ are used. From these data; peak and residual cohesion, friction and tensile strength are obtained as Table 2 shows.

In order to estimate the shear strength of the cleavage planes, two types of tests have been analysed: standard compressive tests where samples broke along cleavage planes, and direct shear tests on this type of planes.

In the first type of test, when the sample broke through cleavage, then in that plane:

$$\tau = \frac{1}{2} \cdot (\sigma_1 - \sigma_3) \cdot \sin \beta$$

$$\sigma = \frac{1}{2} \cdot (\sigma_1 + \sigma_3) - \frac{1}{2} \cdot (\sigma_1 - \sigma_3) \cdot \cos \beta$$

Where $\tau$, $\sigma$ are the shear and normal stresses in the cleavage plane, $\sigma_1$ and $\sigma_3$ the principal major and minor stresses applied to the sample and $\beta$ the angle subtended between the vertical axis of the sample and the cleavage plane. With two or more of these tests, a linear regression approach could be performed, in order to calculate the peak cohesion and friction of the cleavage planes. In this case the produced results are $C_{\text{cleav}} = 0.71$ MPa and $\phi_{\text{cleav}} = 43^\circ$.

The second type of approach based on direct shear test results helps to calculate the cleavage plane’s residual strength parameters. When the shear tests are performed, the two parts of the sample separated by the cleavage plane were already pulled apart. The average values obtained in this way are $C_{\text{cleav}} = 0.29$ MPa y $\phi_{\text{cleav}} = 23^\circ$. According to a back analysis based on the $RMR$ and on the peak and residual parameters presented, mean values of these strength features are estimated and presented in Table 2.

Other materials parameters

The rest of materials -the top and wall beds and the quartzite- are not as significant as the slate for the performance of the mine. So behaviour models which are used to simulate them are not as complex. The top and wall strata are considered isotropic elasto-plastic, whereas the quartzite is modelled as elastic, since it is not likely to fail. These rock data are shown in Table 2.

Table 2: Slate and other material mechanical parameters

<table>
<thead>
<tr>
<th></th>
<th>SLATE</th>
<th>OTHER MATERIALS</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Intact/peak</td>
<td>Broken/residual</td>
</tr>
<tr>
<td>$E$ (GPa)</td>
<td>20</td>
<td>14</td>
</tr>
<tr>
<td>$\nu$</td>
<td>0.19</td>
<td>0.16</td>
</tr>
<tr>
<td>$\rho$ (kg/m$^3$)</td>
<td>2,680</td>
<td>2,670</td>
</tr>
<tr>
<td>$C$ (MPa)</td>
<td>5</td>
<td>2</td>
</tr>
<tr>
<td>$\phi$ ($^\circ$)</td>
<td>59</td>
<td>51</td>
</tr>
<tr>
<td>$\sigma_1$ (MPa)</td>
<td>1.15</td>
<td>0.5</td>
</tr>
<tr>
<td>$C_{\text{cleav}}$ (MPa)</td>
<td>0.78</td>
<td>0.26</td>
</tr>
<tr>
<td>$\phi_{\text{cleav}}$ ($^\circ$)</td>
<td>31</td>
<td>0.12</td>
</tr>
</tbody>
</table>
Numerical Modelling

The FDM based 'FLAC-2D, vs.3.30' code (Itasca, 93) is selected to perform simulations. Since the code is 2D, the following assumption is made: 'there will not be deformations normal to the plane of the model'; that is to say plane-strain conditions are supposed.

Geometry and Area of Discretization

Starting from the geological sketch already presented, the three upper layers are put together into one equal-behaving material. The other materials are considered separately. The rooms and pillars are modelled as defined in the first empirical approach.

The area of discretization is chosen, starting from different size temptative simulations, where displacements were recorded. When it is observed that these results do not change significantly with an increase in the size of the model, then this area is selected. This discretization area, together with geology, room geometry and boundary conditions are presented in Fig. 3.

Boundary and Initial Conditions

Boundary conditions are defined as illustrated in fig. 3. Horizontal displacement is restrained in both left and right borders. These borders are located far enough from the mined seam so as not to influence stress-strain patterns around holes. Vertical displacement is restrained along the bottom axis of the model. Mesh average dimension decreases from the boundaries to the surroundings of the excavation, where significant stress and displacement gradients take place.

The initial stress field has shown to be a very significant input feature in order to analyse the stability of an underground excavation. In this particular case two different assumptions could be made: isotropic or elastic stress natural field. Since it has been checked that the excavations are more likely to be stable under an isotropic stress field, this type of stress field is selected to conservatively perform the model. During the mining of the first room, flat jacks stress measurements will yield the actual stress field and a more realistic simulation approach will be carried out with these new data.

![Fig.3 Simulation: geometry, area of discretization and boundary conditions.](image-url)
Two kinds of strategy are followed to perform the model. In the first one all the materials behave elastically. So, once the model is run, the stress state of every single zone of the model is compared to its strength criterion, and factors of safety are calculated. In this way, the zones where yield appears can be tracked and analysed.

In the second one, the elasto-plastic simulation is carried out. In this case all the yield possibilities are taken into account. So the outputs will show the broken zones according to the predefined failure criteria -across the rock or along cleavage planes-.

**Elastic simulation results**

Once mined the rooms, stress concentration is analysed. Fig. 4 presents the distribution of the stress difference ($\sigma_I - \sigma_J$) in the surroundings of the mined rooms. The higher this parameter, the more ability of the material to suffer compressive failure. It can be observed that compressive stress concentration takes place in the rooms walls and pillars, whilst on the floor and roof tensile stress concentration occurs.

![Fig. 4. Principal stress difference contours after room mining.](image)

In order to study if these stress levels are large enough to produce failures, Fig. 5 is presented. It illustrates the plot of the stress state of the zones together with the Mohr-Coulomb failure envelope of failure through material -Fig. 5a- and along cleavage planes -Fig. 5b-.

In Fig. 5a it is observed that only a couple of zones are failed, in this case because of tensile stress. These zones are on the room’s floor. This is not a problem for the mining development.

Fig. 5b illustrates various failed (shear failure) zones along cleavage planes. They are located in the left-hand roof of the two deepest rooms. Initially, these instability problems should be controlled by the already presented empirically designed support.
Elasto-plastic simulation results

In this model, the type of yield of every single zone can be controlled. Fig. 6. illustrates a detailed view of the mining area, where the different material, the areas submitted to tension or compression and the type of yield of the zones can be observed.

Tension zones are located on the floor and roof of the rooms. Local yield is observed on the floor, but this is not a real problem for mining. Yield over the roof, partly due to the shear and tension failure along cleavage planes, could be more dangerous. It could locally involve bed separation, which should have to be controlled by means of the support. In the deeper rooms these phenomena are more extended. An adequate support installation—in due time with full working rock-bolting—should avoid the appearance and generalization of this type of instability.

Nevertheless, in order to avoid any failure a stronger support is proposed in the mine design; so that safe working conditions can be ensured. This support is based on the empirically designed one, adding three long fully grouted bolts (6 m. long, with transversal and longitudinal spacing equal to 3 m.) in the left roof of the rooms, as fig. 7. illustrates.

Fig. 6 Elasto-plastic simulation. Tension and compression zones. Yielded zones.
In the first stages of mining, in site observations will help to “fine tune” this support.

**Economic Scope**

In the present paragraph the underground mine project is focused from an economical scope. Firstly, the concept of slate ‘grade’ or recovery index is defined. Then, the authors present an approach to the underground mining and quarrying costs in order to know when it would be adequate to carry out the mine.

**Slate ‘grade’**

Slate quarrying has been thriving in NW-Spain in the recent years, with the result that this country has become the first producer and exporter all over the world. This ornamental rock is mainly used in the construction industry to roof. In the increasingly important market of ornamental rocks the concept of ‘grade’ or percentage of mineral that can be finally sold is not easy to appraise and predict. In the early nineties, efforts have been made to define methodologies to estimate correctly this recovery index for ornamental rocks. Unlike the metal mining world, this ornamental rock ‘grade’ should comprehend various aspects of materials.

So it was proposed a recovery index for slate (Taboada, 93) based on three efficiencies:

Mining efficiency \( E_{M} \) is the ratio of mined material occurring in adequate size and shape blocks after mining. \( E_{M} \) is estimated starting from the type and direction of cuts in the banks and from the rock mass original blocks size and shape distribution. This last issue can be obtained according to discontinuity presence, persistence and spacing.

Quality efficiency \( E_{Q} \) is the ratio of slate already mined and cut into adequate blocks, that do not present defects or flaws. These flaws include kink-bands, weathering, oxide stains, carbonate ‘moons’, sulphide presence, quartz threads... and they turn the mineral unmarketable. So it is not worthy to transport the very low quality blocks to the plant to produce a few tiles. Further analysis divides this \( E_{Q} \) into two ratios, according to the use given to tiles and their price (1st & 2nd quality).

Plant efficiency \( E_{P} \) is the ratio of the material received in the plant that can be sawn into marketable tiles. \( E_{P} \) can be estimated according to the shape of the tiles and to the joint analysis of blocks. Some tiles have to be retired at this stage due to presence of defects; this loss of material is logically included into \( E_{Q} \).

The product of these three levels of efficiency, called recovery index, let the authors know the ratio of mineral in the deposit to be finally sold. That is why it is called ‘grade’:

\[
\text{slate ‘grade’} = r, i, \left( \frac{\text{t of marketable tiles}}{\text{t of slate in deposit}} \right) = E_{M} \cdot E_{Q} \cdot E_{P} \tag{5}
\]
This grade varies significantly for the different mining areas and mining methods. However, the results of different deposits located in the same belt are usually consistent. For instance in the quarries of Galicia this r.i. varies from 6 to 15%. Methodologies are still being developed and improved in order to accurately estimate these efficiencies starting from observations of rock cores and outcrops and, from mining planning and design issues. In Vilarchao Quarry the up-to-date r.i. has shown to be actually 9.2% - $E_{fr}$ = 43.3%, $E_{fQ}$ = 73.1% and $E_{fn}$ = 27.8%.

Underground Mining

Underground mining sequence is proposed to be as shown in Fig. 8. A more detailed scheme will be given as long as the mine works. In the banks cuts will be performed by sawing and diamond thread systems to improve mining efficiency. This $E_{fr}$ would attain 58.2%, which is considered a significant improvement.

![Diagram of mining sequence](image)

Fig. 8 General idea of the mining sequence. Drifts: blasting; Banks: mechanical cut with saws and diamond thread. Estimated $E_{fr}$ = 58.2% and $E_{fQ}$ = 74.4%. Estimated final grade 12.8%.

The efficiencies for the Vilarchao deposit according to the proposed underground mining method are calculated starting from data taken in the outcrops and rock cores and using a geostatistical approach. In this way, values of $E_{fr}$ = 58.2%, $E_{fQ}$ = 74.4% and $E_{fn}$ = 29.6% are obtained, resulting a r.i. of grade of 12.81%.

This grade, larger than the one of the quarry is due to the substitution of blasting for mechanical cutting and because the deeper part of the deposit is less weathered and jointed.

Underground Mining vs. Quarrying (costs assessment)

In NW-Spain there are today approximately one hundred slate quarries. Nevertheless, only one underground slate mine has proved to be profitable and it is still working today.

In table 4 it is presented an economical comparison between the so far observed quarrying costs and the estimated future underground mining costs when producing a ton of the final elaborated product in Vilarchao deposit.

According to this estimate, underground mining would be preferred for waste/slate ratios larger than 2 m³/t. This average ratio will be achieved after excavating 25 metres in the whole main slope of the quarry. It would take about two years to get to this situation. The decision to run this project will be then made, taking into account this study together with other aspects not deep analysed in this paper such as environmental, social, legal and updated financial issues.
Table 3: Underground mining vs. quarrying. Costs estimate.

<table>
<thead>
<tr>
<th>QUARRYING COSTS</th>
<th>UNDERGROUND COSTS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Let R be the ratio: m$^3$ of waste/ t of extracted slate.</td>
<td>Und. workers (9 p.)</td>
</tr>
<tr>
<td>Let $\rho$ be the slate density: 2.8 t/m$^3$</td>
<td>Support &amp; Materials</td>
</tr>
<tr>
<td>Let r.i. be ratio: recovery index (t of elab. mineral to be sold/ t of extracted slate). For the last 10 years the average ratio has been 9.2 %.</td>
<td>Energy Supply</td>
</tr>
<tr>
<td>Unit Cost of waste removal $\frac{S/m^3}{3.40}$</td>
<td>Maintenance</td>
</tr>
<tr>
<td>Unit Cost of slate extraction $\frac{S/m^3}{14.50}$</td>
<td>Financing</td>
</tr>
<tr>
<td>Total Cost of waste removal: $\frac{3.40 (S/m^3) \cdot R (m^3/t) \cdot r.i. (t)}{36.96 \cdot R S/t elab. mineral}$</td>
<td>Unexpected costs</td>
</tr>
<tr>
<td>Total cost of slate extraction: $\frac{14.50 (S/t) \cdot \rho (t/m^3) \cdot r.i. (t)}{56.29 S/t elab. mineral}$</td>
<td>Underground Mining Costs:</td>
</tr>
<tr>
<td>Up-to-date average Quarrying Unit Cost: $36.96 \cdot R + 56.29 S/t elab. min.$</td>
<td>Estimate average month production:</td>
</tr>
<tr>
<td>Estimated Underground Unit Cost: $130.96 S / t elab. mineral</td>
<td></td>
</tr>
</tbody>
</table>

Conclusions

The following conclusions related to the mine design, required support and economics of the underground slate exploitation deserved to be outlined:

The geomechanical feasability and economic viability of the exploitation is ensured.

The required support for the first room is shown in fig 2. In the deeper rooms a stronger support is initially preferred, including three more bolts as mentioned above.

The conservative design of the pillars ensure the general stability of the mine.

In situ validation of the support and pillar performance must be ineluctably carried out in the first stages of mining. This will lead to more economic designs, increasing mining ratios and keeping safe work conditions.

Acknowledgements

The authors thank the autonomic government of Galicia (Xunta de Galicia) for the financing of this applied research project, under contract code XUGA 32102A95.

References


