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Tunnel design in jointed rock

Tunnelling in jointed rock is a very common activity. In most cases, some degree of support of the opening is required. Only occasionally is the quality of the rock mass sufficiently high to allow unsupported openings. Supports for tunnels in fissured rock are generally designed using empirical procedures based on geomechanics quality indices, the more popular of which are probably the Q and RMR parameters. A good geotechnical knowledge of the rock mass is always a prerequisite; this is acquired from surface geological surveys and from information generated by boring.

Such methods present the great advantage of incorporating the benefit of past experience. Hence, it is our opinion that empirical methods should never be totally abandoned. However, in certain cases which are progressively more frequent due to the increasing complexity of the works undertaken, the use of numerical methods becomes necessary for achieving a sound design of the tunnel.

Occasionally, but unfortunately not very often, the ground can be idealized as a continuous medium. This allows the use of traditional finite element and finite difference approaches, the latter being specially appropriate for large deformation problems. However, in the majority of cases, pervasive fracturing of the rock mass makes such methods inadequate. It then becomes necessary to model the rock mass as a series of deformable blocks separated by discontinuities on which water may be exerting pressures. A preliminary theoretical analysis of such jointed rock behaviour is outlined here.

Movement of an isolated block

The main problem which arises in the design of relatively shallow tunnels in a fractured mass of competent rock is the fall of rock blocks from the walls and back of the tunnel. Hoek and Brown proposed a method for assessing the stability of such blocks based on the use of the stereographic projection. The method requires representing the planes of discontinuity which bound the rock blocks using Schmidt's chart.

The weight of the block is obviously needed for determining the anchoring force. The most probable blocks are formed by three planes of discontinuity and the surface of the back or wall of the tunnel. Their volume is usually calculated on the assumption of a rectangular tunnel cross-section, with the corresponding corrections in case of circular or other sections.

The calculation of the volume starts by projecting the plane of the back or rotating the plane of the wall on to a horizontal plane. Once the corresponding traces of the planes of discontinuity have been drawn, it is then possible to compute the base area of the block. Assuming, for example, that the block is pyramidal in shape, its volume is one-third of its base area times its height. This height is determined by rotating on to the horizontal plane the section which contains the projection of the apex of the pyramid on to the wall or back face of the pyramid.

The weight of the block can then be computed by multiplying its volume times the specific weight of the rock and times a scaling factor; the latter is a function of the scale at which the stereographic projection has been carried out.

When the block is located at the tunnel back, its fall may take place both with or without friction. In the latter case, the anchoring force required can be computed simply by multiplying the block weight by the desired safety factor. The frictional fall is identified by inspecting the projection of the apex on to the pyramid base: if the projection lies outside the base, friction will be developed and the anchoring force is computed using the stereographic projection.

If the block is falling from the wall, friction is necessarily developed in one or two planes. The problem is solved using the stereographic projection once the block weight has been calculated. The calculation of the anchoring force begins by selecting the intersections of pairs of discontinuity planes which dip towards the tunnel. In the case of a pyramid, the block is defined by three planes of discontinuity and the plane of the wall so there will be three intersections. Those dipping towards the tunnel, and thus kinematically unstable at one of the walls, will dip towards the rock mass, and thus be stable, at the opposite wall.

The study of an intersection which dips towards the tunnel is based on several stereographic projection criteria designed to define the type of fall produced: sliding on one of the three planes of discontinuity or sliding as a wedge on two planes and their edge of intersection. In the former case, the calculation must use as friction angle that of the plane of discontinuity. In the latter, the friction angle must be determined by means of the stereographic projection as a combination of the friction angles of the two planes of the wedge on which sliding is to take place. In this case, the block will fall along the edge of intersection of the two planes while, in the case of plane sliding, the block will fall along the dip line of the plane.

In the case of the wedge, the safety factor is defined as the ratio between the tangent of the friction angle $\phi$ and that of the angle $\alpha$, formed by the weight of the block and the normal to the falling trajectory:

$$ F = \tan \phi / \tan \alpha, $$

To increase the safety factor, it is necessary to compute the required angular deviation of the block weight towards the normal to the trajectory. That deviation is achieved by combining the weight with the anchoring force; its value must be such that the ratio of the tangents equals the safety factor sought.
The angle \( \alpha \) (see Figure 1) can be calculated using Equation 1. Once the anchor direction and the weight of the block are known, the anchoring force is calculated.

It is evident that the described process can be programmed in a computer and carried out automatically rather than developed using the stereographic projection. This is obviously faster, but the process is less easy to visualise and rely on equipment which is not always available in the field.

Up to this point, no special consideration has been given to existing horizontal stresses in the rock. Such a case is studied below for the tunnel back; the same calculations are valid for the walls with some minor modifications.

The forces acting on a block at the tunnel back are the following: its weight \( W \), calculated as mentioned above; the normal \( N \) and tangential \( T \) components of the force applied at the block faces, which depend on the stress field in the rock mass; the resultant of the water pressures which may be acting on the block faces; and, lastly, the anchoring force \( A \). In the procedure described below, the resultant of water pressures is not taken into account.

The section presented in Figure 2 corresponds to a roof block, shaped as a mathematical triangular prism and with its axis aligned with that of the tunnel. The calculation of the anchoring force is carried out by establishing the force:

\[
F = A + P = F = 2(T \cos \phi - N \sin \phi)
\]

(2)

The value of \( T \) is:

\[
T = N \tan \phi
\]

(3)

where \( \phi \) is the friction angle of the faces of the block.

Combining Equations 2 and 3:

\[
F = 2N \sin(\phi - \alpha) \cos \phi
\]

(4)

This means that if there are horizontal stresses which produce \( N \) on the block faces, \( F \) will point upwards only if \( \phi < \alpha \). If \( \phi > \alpha \), the block will fall even if it had no weight.

The calculation of \( F \) from the horizontal stresses (Brady and Brown, 1985) is carried out by the stress relaxation method. First, an elastic analysis is carried out on the assumption that the joints have a very high stiffness. This allows analysing the rock mass as a continuum and determining the stress state at the locations of the joints. With the areas of the block faces and their stress state, the forces acting on the block faces are immediately calculated. Then, the force \( F \) defined previously can be evaluated:

\[
F = 2N \tan(\phi - \alpha)
\]

(5)

where \( H \) is the horizontal force compressing the block.

Finally, the safety factor is obtained as a ratio between the force \( F \) and that resulting from the combined action of the block weight \( P \) and the anchoring force \( A \).

When drilling a tunnel in a rock mass with several joint systems, blocks of various shapes and sizes will appear at the walls and back of the tunnel. Some of those blocks may fall into the tunnel, while others will remain stable either due to their attitude and geometry, or because their movement is impeded by other blocks. When the unstable blocks fall, some of the previously stable ones will start moving as they lose their confinement. This process continues until final collapse of the excavation, unless one or more stable blocks act as a barrier for the others. Such blocks are termed “key” blocks and their stability is most important for the global stability of the excavation.

It is therefore advisable to identify the new key blocks as the drilling process proceeds. Once identified, special attention must be dedicated to their stability and adequate supports will have to be provided if necessary. The identification of key blocks is generally complex due to the three-dimensional character of the problem.

**Behaviour analysis**

The methods discussed in the previous section provide guidance for analysis of isolated blocks located in the walls and back of the tunnel. However, they do not allow solving for the displacements undergone by the array of blocks surrounding an underground opening. Cundall was the first to develop a procedure for modelling the mechanical behaviour of rock masses as arrays of individual blocks or particles. Following some developments, his method is now well known as the distinct element method.

Many investigators have followed this line of work and the initial code, designed for two-dimensional rigid blocks separated by dry, plane, elastic joints, has now become a tool able to deal with three-dimensional deformable blocks, non-linear discontinuities and water flow. The codes, currently known as UDEC (Universal Distinct Element Code) and 3DEC (3-D Distinct Element Code), constitute very useful tools for analysis of jointed rock masses.

The first step in the numerical analysis of a rock mass consists of the mathematical idealisation of the three-dimensional array of blocks around the excavation. Lain (1985) carried out an attempt to provide some automatic generation of blocks. Similar initiatives have been conducted in other places and a Block Generation Language (BGL) has been proposed by Halot.

The compressive and shear behaviour of the joints, as well as their dilatancy, are all non-linear processes which programmes like UDEC must represent in a realistic fashion. Barton has dedicated a major part of his scientific endeavours to studying joint behaviour and has proposed for its representation models of great practical use. Typical stress-strain curves for jointed specimens are shown in Figures 3 and 4; they are concave for the behaviour in the normal direction and convex in the shear direction. The influence that the joint behaviour exerts on that of the rock mass can be observed in Figure 5. The data required for the models is relatively easy to acquire from shear tests under small normal loads and using Schmidt’s hammer.

**Hydraulic conductivity**

The hydraulic conductivity of the discontinuities is related to their geometry and their stress state. Two types of aperture can be distinguished in a joint: the actual physical aperture and a theoretical equivalent aperture. The latter is that of a smooth joint with identical conductivity. In models, these two apertures are represented by \( E \) and \( e \), respectively. The conductivity \( K \) of a joint
can be obtained from:

\[ K = \frac{\delta}{12} \]  

(6)

Codes such as UDEC, which are utilised for modelling the rock mass, must be able to simulate the mechanical and thermal processes which influence the real aperture \( E \) and to express the results in terms of the theoretical aperture \( e \).

Not only do the discontinuities display non-linear behaviour, but the blocks themselves may also behave in a non-linear fashion, be it of a plastic, viscous or viscoelastic nature. These types of behaviour have been extensively investigated using both laboratory tests and on-site measurements. The corresponding constitutive laws are usually known with considerably better approximation than those of the joints. The blocks are modelled as continuous structures, using finite element or finite difference approaches. The implementation of the constitutive laws in explicit integration codes such as UDEC is straightforward even for complex laws, although difficulties increase when dealing with static or implicit integration solvers.

Following the determination of the geometry of the rock mass discontinuities, their thermal, hydraulic and mechanical behaviour, and the thermo-mechanical behaviour of the rock blocks, it is then possible to simulate the global response of the rock mass.

With codes such as UDEC, the simulation process starts by solving for the initial conditions, that is, the rock pre-existing mass with gravity and other applied forces. Once equilibrium is achieved under those loads, the excavation process is simulated by progressive removal of blocks to create the opening. Deformations around the tunnel typically grow with time until a stable condition is reached or global collapse takes place.

The lining is introduced as a series of curved structural elements and their mechanical properties are assigned so that they represent the lining support characteristics.

Analytical approaches have been presented for studying the behaviour of jointed rock masses. Both limit equilibrium and numerical methods have been proposed as tools for the design of tunnels and other underground openings. Traditional approaches based on geomechanical indices are also proposed as a first approximation in any case.

Limit equilibrium procedures, such as the key block method, are useful for studying the stability of isolated roof and wall blocks and for determining the forces required in order to support them. When the movements expected around the excavation affect a considerable region of the rock mass, numerical mod-

![Figure 3](image3.png)

![Figure 4](image4.png)

![Figure 5](image5.png)
...such as UDEC are recommended. Application of the latter requires the previous determination of the constitutive behaviour of joints and blocks by means of the appropriate experimental campaigns.

Finally, it must be stressed that any of the theoretical solutions described above for solving tunnel stability problems requires a very detailed knowledge of the site geology. This constitutes an unavoidable prerequisite for any serious study of an underground project.

Joacim Martí became a civil engineer in Madrid and proceeded to earn a PhD from Texas A & M University. He is currently associate professor of mechanics and an engineering consultant. His work has developed primarily in relation with the numerical simulation of dynamic non-linear problems for geotechnical and structural applications.

Pedro Ramírez is a mining engineer. His BSc and PhD degrees are from the Madrid School of Mines and his MSc from Newcastle University. A former partner at Dames and Moore he is at present the professor of Rock Mechanics at the Madrid School of Mines. He has been an active consultant for 20 years in the rock mechanics and geotechnical engineering disciplines for mining and civil projects.

Ricardo Lain obtained his BSc and PhD degrees in mining engineering in Madrid. He is an associate professor of Rock Mechanics. His major interests are rock cutting, the mechanical behaviour of jointed rock masses and their numerical simulation, particularly as applied to mining and underground space activities.