Problem Statement

Cut and fill mining method is the preferred choice for steeply dipping, irregular, thin and relatively high grade orebodies. Underhand cuts (using backfill) provide several advantages from a production and safety perspectives. In fact, underhand cuts provide an additional mining horizon while the overcuts are progressing upwards. They also force stresses to be in the floor, and since backfill (usually paste) is an engineered material, it is preferred to be the roof of the cut rather than a potentially unstable rock mass (as it is the case of overcuts).

To increase the production rate, different levels may be excavated in parallel. As the underhand and overhand cuts approach each other, the diminishing pillar, referred to as ‘sill pillar’ is subject to increasing stresses. Therefore, the last planned overcut is usually subject to high stresses in the back, making it a difficult cut to mine from a production rate and workers safety standpoints.

This example propose a method to decide the ground support in such overcuts. It is important to note that there are other methods for achieving the same results. This example has been published for educational purposes and not to replicate the same recommendation for a given mine without proper tailored engineering.

Figure 1 shows the geometry of the studied cut and fill horizon. The remaining sill pillar is 40 m in vertical height. The cuts are 5m in width and height each. The particularity of the geometry is the existence of a main ore vein with an East-West strike and a series of smaller veins sub-perpendicular to the main vein, with a North-south strike. The ore body is located approx. 1,500m below surface.

![Figure 1: (a) Isometric view showing the geometry of mining and the last overcut (in light blue) before the sill pillar recovery. (b) East-West cross-section showing the underhand cuts (UC), the overhand cuts (OC) and the sill pillar in between.](image)

The host rock is modeled as linear elastic and its properties are listed in Table 1:

<table>
<thead>
<tr>
<th>Material Type</th>
<th>Linear-Elastic</th>
</tr>
</thead>
<tbody>
<tr>
<td>Young’s Modulus (E)</td>
<td>25 GPa</td>
</tr>
<tr>
<td>Poisson’s Ratio (v)</td>
<td>0.25</td>
</tr>
<tr>
<td>UCS (rock mass)</td>
<td>130 MPa</td>
</tr>
</tbody>
</table>
The pre-mining stress state is characterized by a major principal stress direction of 45° (Northeast-Southwest) and a plunge of 0°. Stresses of the major principal stress ($\sigma_{H_{\text{max}}}$) increase linearly with depth by 50 kPa/m with a constant stress of 8 MPa. The vertical and the minor principal stress components also increase linearly with depth by 30 kPa/m without an initial constant stress.

In the following section, we will contemplate how to use modeling results of part 1 into the design of ground support of the overcut. Dynamic calculations (energy release and bolt energy absorption) are addressed below along with selection of surface and bolt support.

**Recommendations for Ground Support**

Undoubtedly, support tendons should be anchored into competent rock as much as possible to provide good support performance. In the case where the excavation damage zone (EDZ) is limited to a relatively small depth, the bolt length should be at least 1 m longer than the EDZ depth. However, in case of large EDZ such as shown in figure 5, the bolt length should be determined following the equation proposed by Statens Vegvesen (2000) for moderately jointed hard rock masses:

$$L_b = 1.4 + 0.184 \times B \text{ (excavation width)}$$  \hspace{1cm} \text{Eq. 1}

From an operational perspective, the bolt length should be less than half of the opening height for roof bolts and half of the span for wall bolts in order to avoid installation difficulties. Equation 2 is also a used rule for minimum bolt length requirements based on Lang (1961):

$$\begin{align*}
\text{if Span} & \leq 6 \text{m Then } L_b \leq \frac{\text{Span}}{2} \\
\text{if } 6 \text{m} \leq \text{Span} & \leq 18 \text{m Then } L_b \leq \frac{\text{Span}}{3} \\
\text{if } \text{Span} & \geq 18 \text{m Then } L_b \leq \frac{\text{Span}}{4}
\end{align*}$$  \hspace{1cm} \text{Eq. 2}

The longer dimension from these equations should be selected. Certain bolts such as yielding connectables and cable bolts are not subject to these rules.

**Figure 5:** Schematic of an excavation in a blocky rock mass and high stress conditions showing the yield zone, the natural and the artificial arches formed (adapted from Li, 2017).
Using the above equations, the minimum length of the support is recommended to be 2.4m (8') long in the overcut.

For practicality, the spacing could be either squared or staggered (dice 5). A squared pattern is chosen in this example for illustration purposes. The bolt spacing is recommended to be the smaller of the two following rules:

\[ s \leq \frac{L_b}{2} \quad \text{Eq. 3} \]

And, \( s \leq 3 \times J_s \quad \text{Eq. 4} \)

Where \( s \) is the bolt spacing, \( L_b \) is the bolt length, and \( J_s \) is the mean joint spacing.

Equation 3 is based on a 0.5 \( L_b \) thickness of the interaction zone between adjacent bolts (Figure 5) in order to establish a strong enough artificial arch in the fractured zone (assuming 90° reinforcement angle). We arrive to a minimum bolt length and spacing of 2.4m (8') and 1.2m (4') respectively.

**Dynamic Capacity**

The following represents an example of dynamic calculation that could be carried to determine what type of dynamic capacity design should be implemented.

From the back analysis performed on the same sill pillar prior to the last overcut (i.e. mining step 1), it is shown in figure 3(a) from part 1 that macro events up to a magnitude of \( M=1.7 \) occurred in the core pillar. Since the sill pillar is diminishing with mining the last overcut (i.e. mining step 2), one could expected that a seismic event higher than \( M=1.7 \) (say \( M=2.0 \) or 2.5) to occur in the pillar core. Equation 5 indicates the relationship between the energy and the Nutti magnitude:

\[ \log(w) = 1.3 \times Mn - 1.75 \quad \text{Eq. 5} \]

If we consider the energy is traveling in the form of a sphere, then the energy per area at the boundary of the excavation, assuming a distance of 20 m from the event source (pillar core) to the boundary excavation (overcut 5) and a seismic event of magnitude \( M=2.3 \) is calculated as:

\[ \text{Energy Release} = \frac{W}{\text{Area of the sphere at the excav. boundary}} = 15 \text{ to } 20 \frac{kJ}{m^2} \quad \text{Eq. 6} \]

Tables 2 and 3 show the energy absorption for the most commonly used of bolts and surface support. It is shown that if we only use rigid support system (rebar or cables), we wouldn't be able to reach the energy absorption for such seismic event. However, if we consider bolts such as D-bolts or Cone bolts with dynamic and yielding capacity, we quickly increase the energy absorption of the support system.
Table 2: Rockbolt Energy absorption capacities of various commercially available rockbolts determined by dynamic testing (weight drop). Compiled from various open sources (after Li, 2017)

<table>
<thead>
<tr>
<th>Rockbolt</th>
<th>Maximum Deformation capacity (mm)</th>
<th>Energy absorption at 100 mm slip (kJ)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Frictional stabiliser/split set (Ø47 mm)</td>
<td>300</td>
<td>4.5</td>
</tr>
<tr>
<td>Rebar (Ø20 mm), full column grouted</td>
<td>35</td>
<td>3 @ max displ.</td>
</tr>
<tr>
<td>Yield-Lok 19 bolt (Ø19 mm)</td>
<td>300</td>
<td>10</td>
</tr>
<tr>
<td>MD-Bolt (Ø47 mm)</td>
<td>300</td>
<td>10</td>
</tr>
<tr>
<td>D-Bolt (Ø22 mm)</td>
<td>220</td>
<td>25</td>
</tr>
<tr>
<td>Cone bolt (Ø17 mm)</td>
<td>300</td>
<td>12</td>
</tr>
<tr>
<td>Cables</td>
<td>15-20</td>
<td>1.5 @ max displ.</td>
</tr>
</tbody>
</table>

Table 3: Energy absorption capacities of various types of surface support (from WASM static tests)

<table>
<thead>
<tr>
<th>Surface Support</th>
<th>Energy absorption per unit area (kJ/m²)</th>
<th>Maximum displacement at failure (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>FRS 60 mm, synthetic fibre</td>
<td>0.8</td>
<td>60</td>
</tr>
<tr>
<td>FRS 80 mm, synthetic fibre</td>
<td>2.2</td>
<td>80</td>
</tr>
<tr>
<td>Weld mesh 100 × 100 mm (5.6 mm wire)</td>
<td>1.3</td>
<td>210</td>
</tr>
<tr>
<td>FRS 60 mm + weld mesh over</td>
<td>2.1</td>
<td>210</td>
</tr>
<tr>
<td>FRS 80 mm + weld mesh over</td>
<td>3.5</td>
<td>210</td>
</tr>
<tr>
<td>Woven mesh (6 mm wire) with welded double-wire on perimeter</td>
<td>2.0</td>
<td>300</td>
</tr>
</tbody>
</table>

It’s noteworthy to mention that in fractured ground, the installation of dynamic support (yielding support) is not always an easy task as the bond strength could be compromised due to resin spinning and installation difficulties. This is why some deep mining operations adopt a mix of both rigid and dynamic support (hybrid support), to work together for different loading conditions.

This calculation is for illustration purposes and should be used in conjunction with other types of calculation (PPV, static stability, etc.) and an adequate safety factor should be applied for the final design of such static and dynamic support system.

From the above calculations, a suggested support for the last overcut (OC5) roof and shoulder could be:

- **Surface Support**: Fibre Reinforced Shotcrete (FRS) 80 mm + weld wire mesh #6ga over the FRS.
- **Rockbolt Support**: D-Bolt (Ø22 mm) OR combination of Rebar and Conebolt (depending on site conditions) on a 1.2m x 1.2m (4’x4’) pattern in a single width excavation and using longer support tendons (cable bolts for ex.) in large spans and intersections of the overcut.
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References


