Problem Statement

The main objectives of backfilling are to improve ground stability and reduce ore dilution (Li and Aubertin, 2014). Consequently, the placed backfill in a stope must possess a minimum strength to remain self-standing during mining of an adjacent stope.

3D Numerical modelling represents a valuable tool to understand the behaviour of the rock mass, the placed backfill and the interaction of these two elements. In fact, numerical modelling can be used in conjunction with empirical analysis (Mitchell et al 1982; Li Li 2013), field observations, model calibration and experience to evaluate the dilution potential and further optimize backfill strength requirements in an underground mine.

The purpose of this example is to evaluate the behavior of the backfill in an underground stope following mining adjacent block and exposing the end wall. Another simulation is performed to evaluate the factor of safety associated with mining underneath the backfill. 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 area. The shape of the opening is based on stope CMS that has been performed before backfilling the stope. Block 2 (Figure 1-b) is the planned stope that will be mined adjacent to the backfilled stope 1.

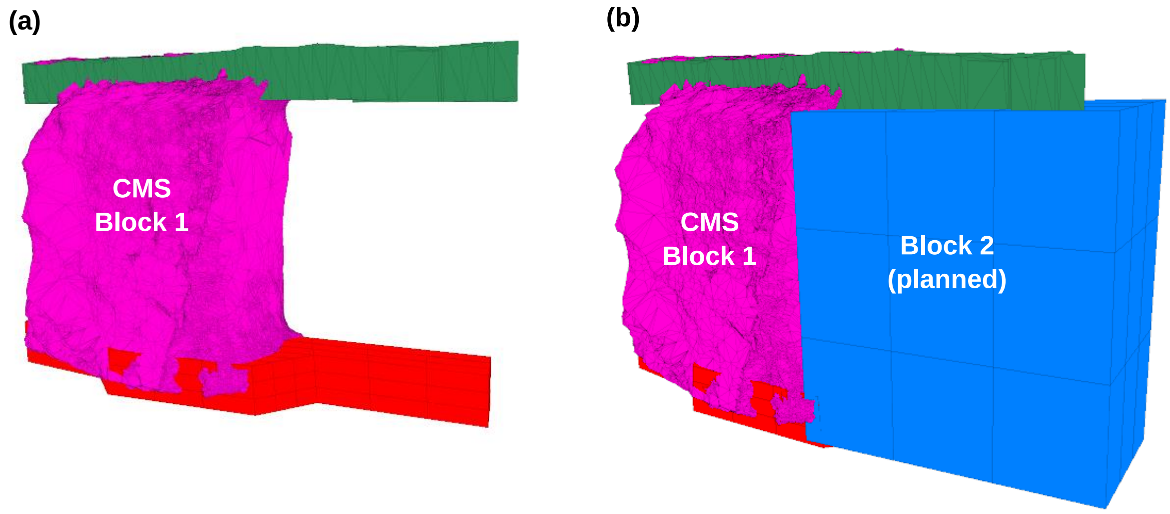

Figure 1: (a) Isometric view showing the geometry of the mining area consisting of top and bottom drifts and a mined-out stope. (b) Planned Block and Exposure of the End wall of the CMS

Three (3) backfill strengths have been modeled to analyze the stability of backfill when exposing the end wall from the extraction of block 2. The scenarios are listed in Table 1:

Table 1: Host Rock and Backfill Properties

Materials		Host Rock	Backfill 1	Backfill 2	Backfill 3
Constitutive Model		Elastic	Mohr-Coulomb	Mohr-Coulomb	Mohr-Coulomb
Material Properties	Young's Modulus (E)	50 GPa	100 MPa	250 MPa	500 MPa
	Poisson's Ratio (v)	0.2	0.2	0.2	0.2
	Density (t/m^3)	2.7	1.9	1.9	1.9
	Friction Angle (ϕ)	NA	25	27	30
	Cohesion (c)	NA	20	50	100

The pre-mining stress state is characterized by a major principal stress direction of 90° (East-West) and a plunge of 0°. The stress magnitude in the study area is 10, 5 and 3.5 MPa for the major, intermediate and minor principal stress components respectively. It is assumed that the stress redistribution due to mining do not affect the stresses within the backfill. Other analyses, using the same methodology as presented here, can be conducted to understand the impact of convergence of the rock walls on the stresses within the backfill.

The modelled stope is 30 m long (strike-length), 30 m in height, 15m in width and has a 80° HW dip angle. The adjacent face that will be exposed is 15m width and 30m height.

Modeling Procedure

The software used in this example is FLAC3D (Itasca, 2020). FLAC3D is an explicit finite difference program to study, numerically, the mechanical behaviour of a continuous three-dimensional medium as it reaches equilibrium or steady plastic flow.

Three types of backfill have been studied in this example: 1)- low strength (e.g. soft paste for example); 2)- medium strength (e.g. consolidated paste); 3)- high strength backfill (e.g. cemented rock fill). Note that these descriptions are used for illustration purposes and do not represent the general characteristics of backfill.

Results and Discussion

Figures 2 (a) and (b) show the contours of vertical stresses of the backfill after placement in the stope. It can be noted that vertical stresses at the bottom of the stope are significantly higher for backfill 1 (250 kPa) as compared to backfill 2 (130 kPa). This is explained by the significant arching effect that takes place in backfill 2 due to the higher friction angle and cohesion, translated to higher shear stress in the backfill-rock wall interface and the transfer of load to the adjacent rock.

Figure 2 (c) shows the contours of displacement in backfill following mining block 2 and the end wall exposure of the backfill. A factor of safety can be calculated based on the shear strength reduction method. The variation of the safety factor in function of the backfill strength is shown in Figure 3.

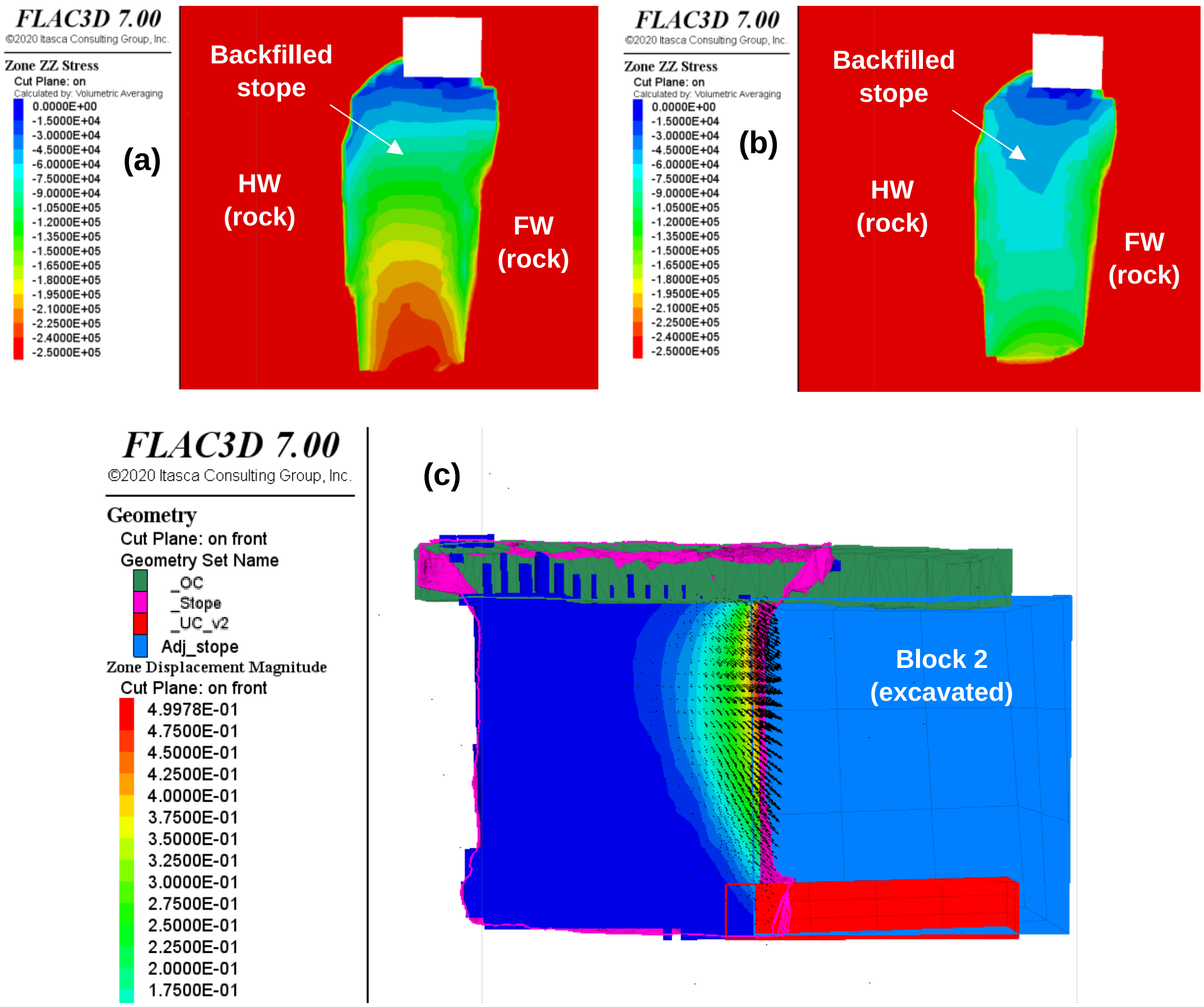

Figure 2: Isocontours of the vertical *stresses* (Szz) on cross section view at mid-strike length of *(a) backfill 1 (low strength)*; (b) backfill 2 (medium strength) and (c) the Unstable section of backfill with displacement contours and vectors (long section at mid-width)

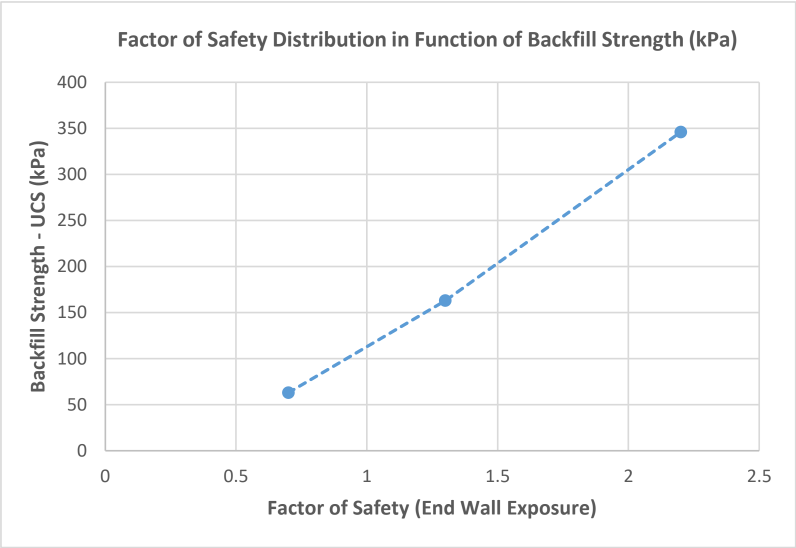

Figure 3: Evolution of the safety factor associated with the end wall backfill exposure in function of backfill strength (kPa)

Similarly, we can evaluate the stability of the backfill with a certain strength by mining underneath and design a ground support system to obtain satisfactory factors of safety. Figure shows the displacement induced by mining underneath the backfill and the corresponding displacement

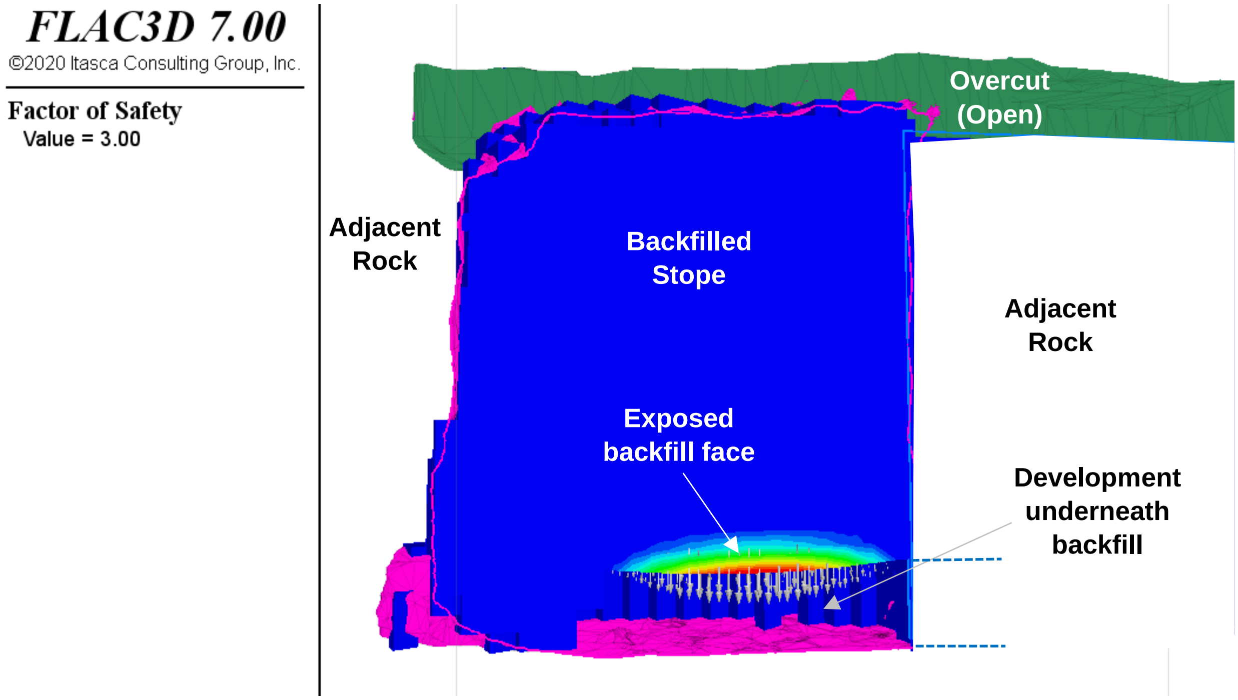

Figure 4: Displacement Magnitude Isocontours and Velocity Vectors showing the area of potential Instability of the Backfill and the corresponding deformed mesh (Exaggerated x 1.3)

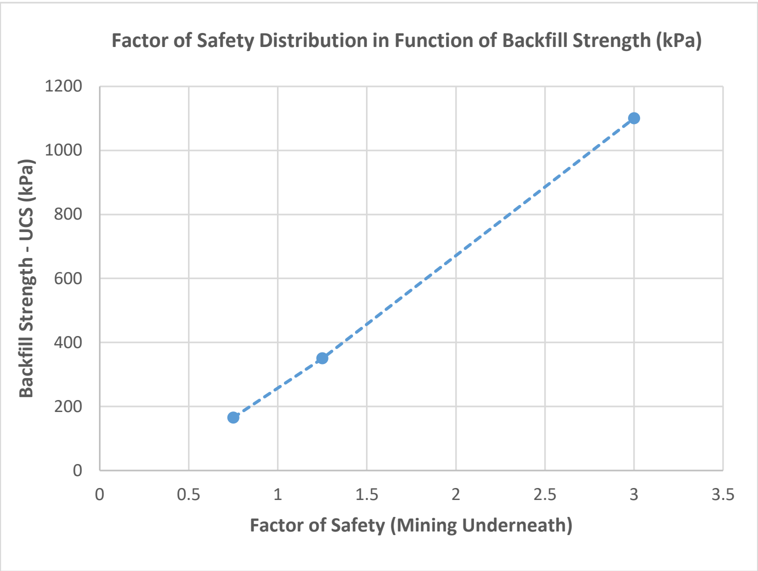

Figure 5: Evolution of the Safety Factor Associated with Backfill Sill Exposure in Function of Backfill Strength (kPa)

In this example, the backfill strength required for safe extraction of adjacent blocks and to minimize backfill dilution can be determined from numerical modelling with FLAC3D. The same type of exercise can be used to determine what is the required ground support for remnant mining strategies or to mine underneath or around a placed backfill with a given strength.

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