

Design Spans – Underhand Cut and Fill Mining

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Abstract

This paper describes a focus of work presently being conducted at the Rock Mechanics Research Group at the University of British Columbia. The underhand method under consolidated fill ensures a high recovery under an engineered back that is comprised of cemented rock fill and/or cemented paste fill. This method of mining is generally necessary either due to a weak rock mass comprising the immediate back and/or high induced back stresses. A major concern in the design of sill mats is the loading and strengths associated with the overlying sill mat. This paper reviews past practice coupled with present observations and measurements from over ten(10) mines throughout North America. It outlines areas of concern in terms of design requirements.

1. Introduction

Backfilling in North America has been practised since the turn of the century. Souza et al. (2003) has summarized the advancements in backfill with the introduction of hydraulic fills in the 1950's and the addition of cement in the 1960's. This coupled with cemented rock and paste fills being introduced in the 1980' and 1990's respectively resulted in the implementation of mining methods that require extraction under a consolidated back largely comprised of fill rather than timbered mats/cables (Marcinyshyn, 1996). The increased use of consolidated fills in the late 1990's to present under engineered conditions with a high degree of reproducibility in terms of strengths and predicted behaviours has enabled man-entry methods such as underhand cut and fill to be implemented under greater controlled spans resulting in a safe

and economic alternative to conventional cut and fill mining, (Mah, 2003). A database of twelve (12) underhand cut and fill operations was compiled as part of this project through mine visits. The operations were located throughout North America and summarized in Table 1.

The placement of consolidated fill either cemented rock fill or paste requires one to understand the overall factors affecting design. Figure 1 graphically summarizes some of the parameters that are being investigated in terms of their implication on developing a design span enabling man

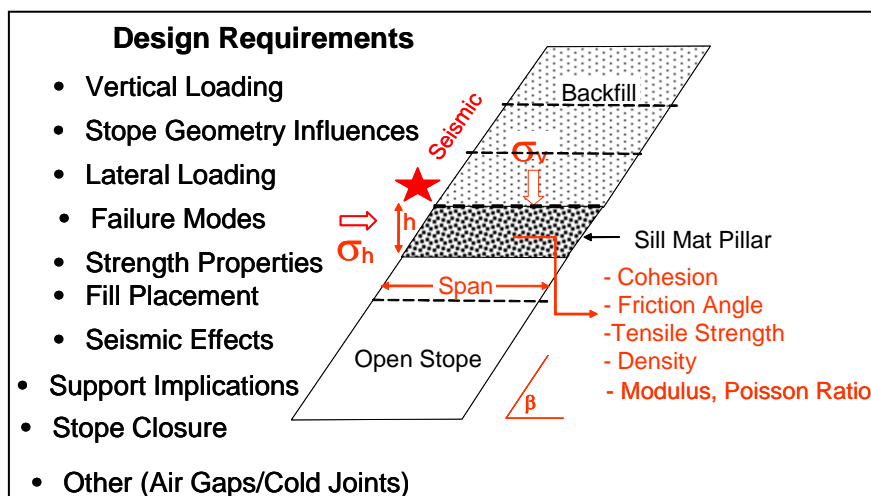


Figure 1: Mining underneath consolidated backfill.

entry access. A sill for this study is defined as a consolidated layer of previously placed fill immediately above the mine opening that is being excavated. This sill may be comprised of one large vertical height placed by bulk mining as shown in Figure 2a or by single lifts as placed by conventional drift and fill and/or underhand cut and fill as shown in Figure 2b. The major difference is that in Figure 2a one is largely operating remotely from the immediate filled back, whereas in Figure 2b one is mining by man-entry methods. This necessitates that the factor of safety for the man entry be substantially greater than the non-entry approach.

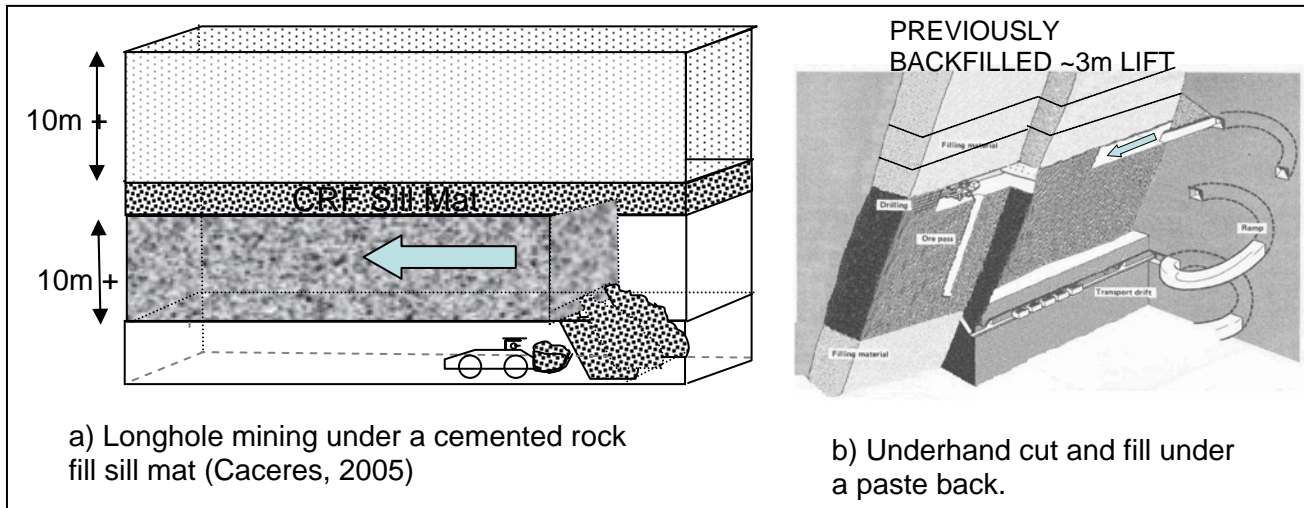


Figure 2: Mining under a consolidated back.

2. Design Constraints

Figure 1 shows the factors that have to be accounted for in terms of mining under an engineered back. These will be outlined in this paper from a general perspective with focus on the analytical, numerical assessment of span and applied loading conditions.

2.1 Design Load

A critical factor is estimating the design loads onto the sill mat. A recently completed MASC thesis by Caceres (2005) employing the Musselwhite mine of Placer Dome as a case study had looked at the loading conditions that exist on a cemented rock fill mat as shown schematically in Figure 2a. Knowing the loads is critical to determining the strength required of the sill mat for the given stope geometry. Under-estimating can cause a premature failure of the sill mat once mining exposes the mat whereas overestimating can result in unnecessary expense due to the cost of the cement in place. Knowing the vertical loading is not a trivial solution as many factors affect the overall loading conditions as evident from the many theoretical derivations that are available as per Janssen (1895), Terzaghi et al.(1996), Reimbert (1976) and Blight (1984) all of which have significant assumptions in terms of coefficient of lateral earth pressure “K” as detailed by Marcinyshyn (1996). The value for “k” describes the ratio between the horizontal and vertical stresses in the fill and indirectly the ability of the degree of load transfer by arching. When fill is placed initially, very little shear resistance is mobilized through grain interaction, the coefficient of earth pressure at rest (K_0). Subsequent placement of fill results in the fill mass to settle and compact, increasing the shear resistance and transfer load to the abutments through arching. As mining underneath the sill pillar progresses, a void is created toward which the fill mass will tend to move if unconsolidated with transfer of vertical stresses laterally through arching. This condition is described by the passive earth pressure coefficient (K_p) where full shear resistance is mobilized. This is analogous to classical embankment theory where the walls are moving into the fill. The effect of K employing individual fill load formulae is shown in Figure 3a. The analytical methods shown in Figure 3a assume vertical stope walls which is generally a conservative estimate of vertical loading as shown by Caceres (2005). The typical geometry was modelled employing FLAC^{2D} (Itasca, 2005) which did not have the constraints the analytical methods had in terms of ‘K’ and stope inclination. An analytical approximation as shown by eqn. 1 was derived by Caceres (2005) relating the numerical simulation to an equivalent relationship as shown in Figure 3b.

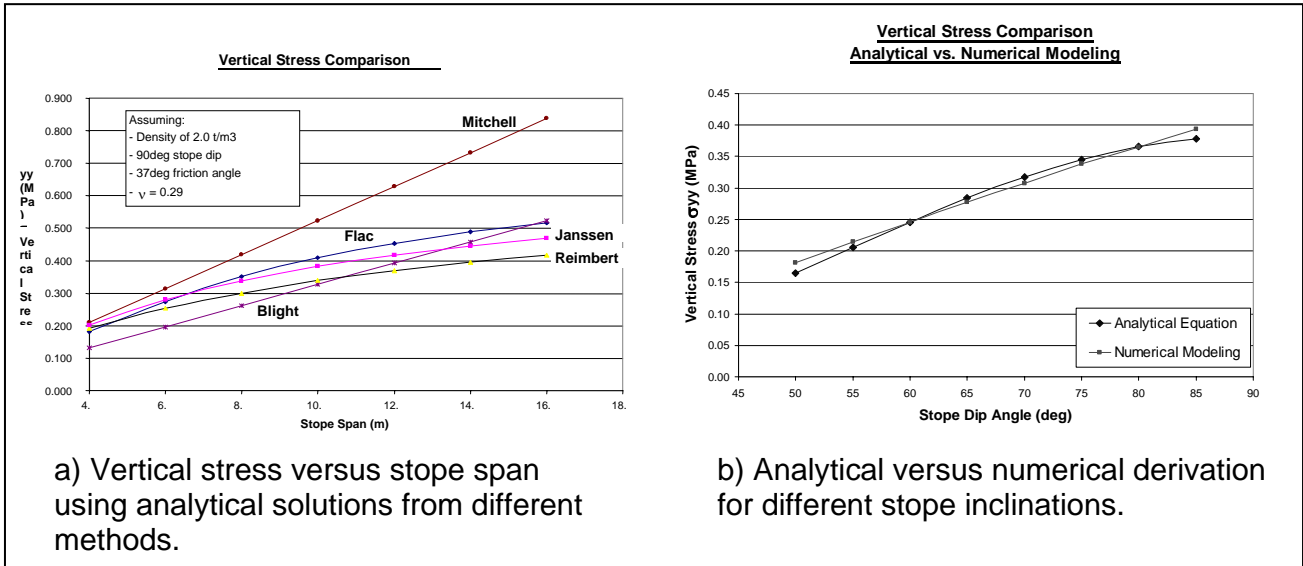


Figure 3: Estimation of vertical loading onto sill mat.

Eqn. 1:

$$\sigma_y(z) = \left(\frac{\gamma \cdot L}{2 \cdot K \cdot \tan(\phi)} \right) \cdot \sin^2(\beta) \cdot \left[1 - \exp\left(-\frac{2 \cdot K \cdot \tan(\phi) \cdot z}{L \cdot \sin^2(\beta)} \right) \right]$$

Where:

- L = Span of the stope
- z = Height of rockfill
- K = Coefficient of lateral earth pressure $\sim 1.4\sin^2(\phi) - 2\sin(\phi) + 1$
- γ = Rockfill's unit weight
- ϕ = Rockfill's friction angle
- β = Stope dip angle

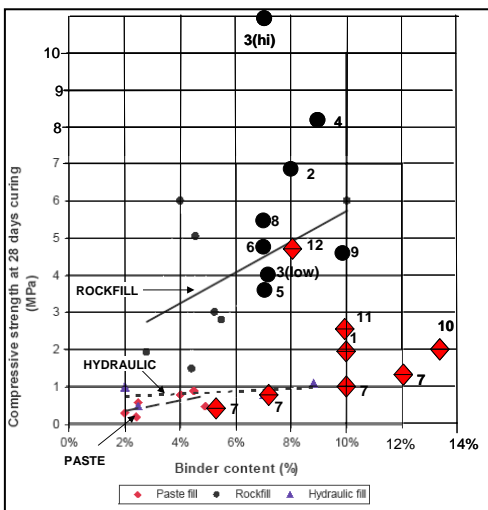


Figure 4: Backfill strength vs binder content (Table 1).

The above was derived for cemented rock fill, however, the analytical solution would be similar to that of paste as the input parameters would define the loading conditions.

2.3 Failure Mechanism

The methodology of span design under consolidated fill is complex as many factors control the overall stability as shown in Figure 1. The failure modes and combination thereof must be analysed with respect to the placed fill, stope geometry, loading conditions, seismic affects, stope closure, and support placement as well as other factors that are due to filling practises such as cold joints and gaps between successive lifts among others. This paper employs analytical, numerical and empirical tools to attempt to provide an initial tool for the operator for design. The database of underhand stopes observed by the author is shown in Table 1 which is comprised of twelve(12) operations which include seven cemented rock fill and five having paste within the immediate back.

The unconfined compressive strength is typically the parameter employed to benchmark the overall stability of the immediate back. Figure 4 shows the compiled database (Table 1) of backfill unconfined compressive strengths adapted from Souza et al.(2003).

The design methods (Table 1) all employed a form of limit equilibrium analysis coupled with modelling. The failure modes are summarized by Mitchell (1991) and shown in Figure 5.

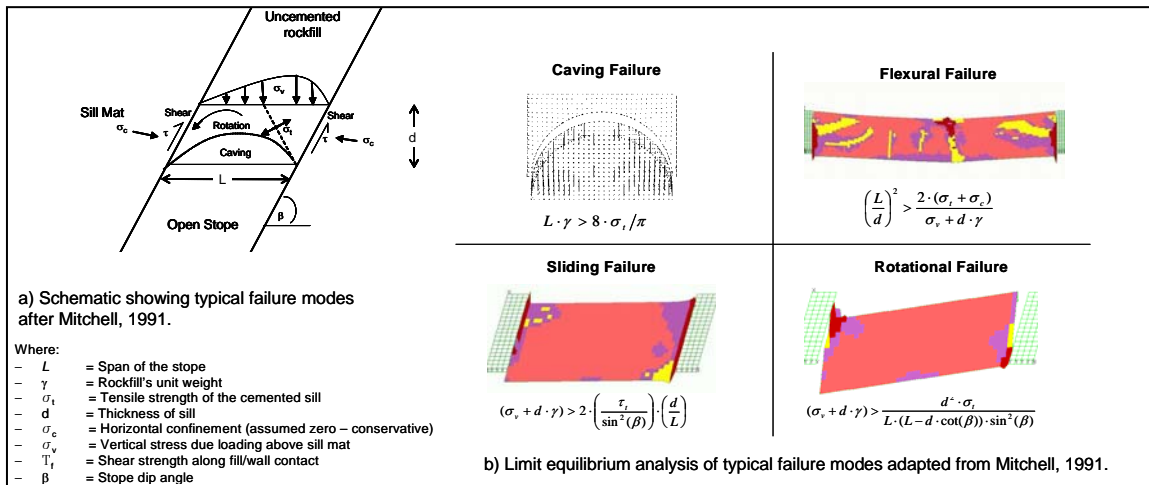


Figure 5: Limit equilibrium criteria adapted from Mitchell, 1991.

Flexural instability was found to be most critical in the absence of rotational instability and closure stresses (σ_c) which

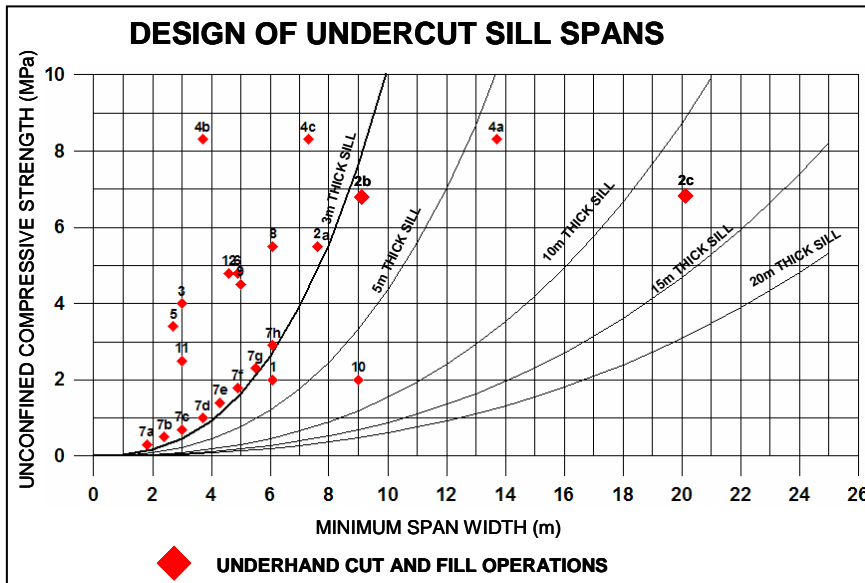


Figure 6: Stability chart for the design of undercut sills with vertical sidewalls with a FS of 2. Chart is based upon fixed beam bending failure

related to actual field observations. Generally the mine data was found to be more conservative than the required for a Factor of Safety of 2.0. This may reflect the quality control requirements at individual operations, along with other factors such as seismicity and stope geometry among others as shown in Figure 1.

The limit equilibrium approach shown in Figure 5 was simulated by Caceres(2005) employing FLAC^{2D} models (finite difference code) for a given value of cohesion, span and stope dip. The cemented rock fill properties assigned are for a Mohr-Coulomb type of material with strain-softening behaviour where integrity is lost after 1.5% strain (Swan and Brummer, 2001). The resultant mode of failure was analysed for 90° to 75° dip stopes with cohesion on the hanging wall

have to be evaluated separately. Stone (1993) had concluded that for cemented rock fills that crushing, caving, sliding are generally negated when the sill thickness exceeds 0.5 x span , absence of closure stresses and the unconfined compressive strength of the cemented rock fill is greater than 1.5MPa and that rotational instability where kinematically possible has to be analysed separately. Figure 6 shows the database that has been compiled in Table 1 and plotted onto a stability chart adapted from Stone(1993) and developed for the design of sills with vertical sidewalls with a Factor of Safety of two. The chart is based upon flexural instability employing fixed beam analysis with surcharge loading after Eqn. 1. It shows the unconfined compressive strength required (FS=2) for a given sill thickness and span exposed and

contact varying from zero to maximum (cemented rock fill cohesion) as shown in Figure 7 for the 75° stope dip. The analytical approach after Mitchell assumes no hanging wall cohesion for the rotational instability and this was found to result in a high degree of conservatism. The FLAC^{2D} simulation shows the failure mode that results for a given stope span, sill height and cohesive strength for various assumptions on wall friction. The design curves have been developed for cemented rock fill operations, however, they are largely dependent upon material strength characteristics which are

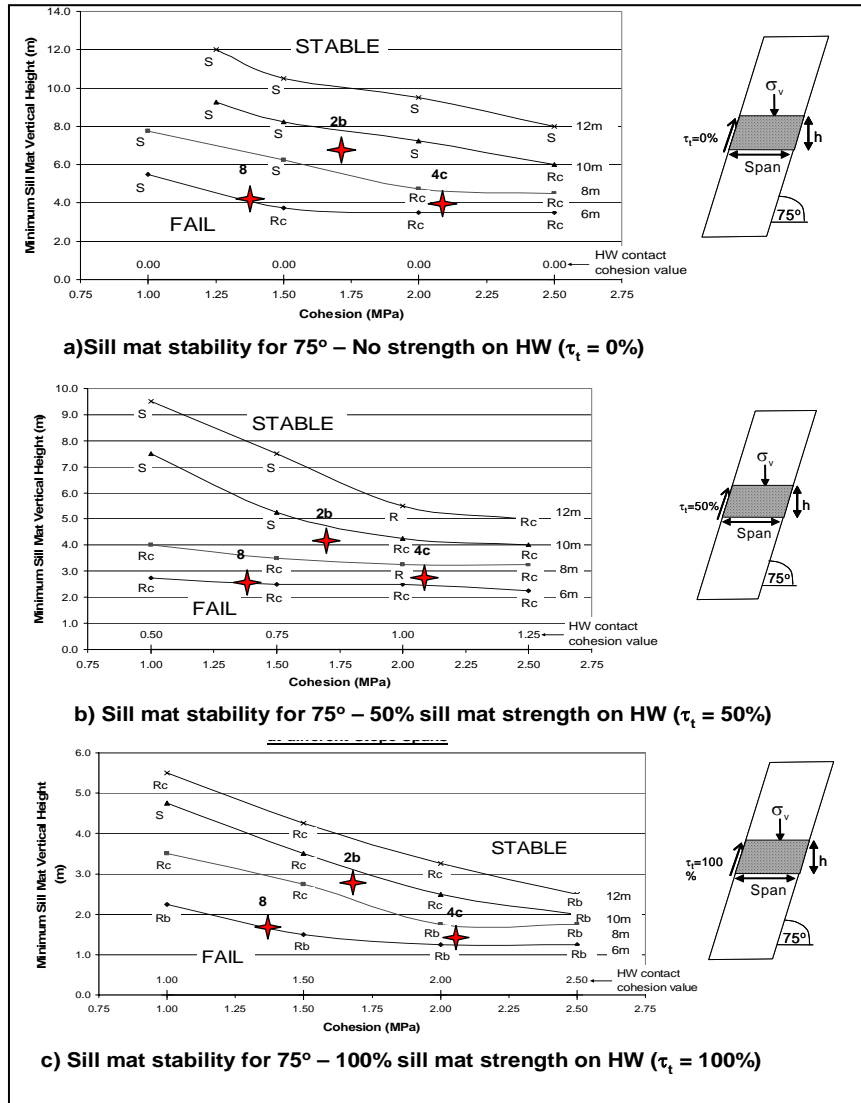


Figure 7: Sill mat stability as derived from numerical modeling. Factor of Safety ≥ 1.0

play a significant role in ensuring a safe exposed operating span. A major benefit of mining under paste is the mitigation of the hazards posed by bursting (Blake et al, 2004).

3.0 Seismic Case History – Managing Rockbursts at Hecla’s Lucky Friday Mine, Mullan Idaho

The following have been compiled by Blake and Hedley, 2003. Its importance is that the underhand mining as practised at Lucky Friday (Mine #12 in Table 1) is the first to incorporate paste to mitigate burst damage and the method has been adopted at mines throughout North America such as the Red Lake Mine in Ontario (Mah et al., 2003) and the Stillwater Mine in Montana (Jordan et al., 2003).

Hecla initiated overhand cut-and-fill mining on the Silver Vein at the Lucky Friday Mine in the late 1950’s. By the mid 1960’s mining had progressed down to the 3050 level (~930m below surface), and the mining geometry consisted of long, flat-backed stopes, all at the same elevation, being carried up from two or more levels simultaneously. A burst

input into the model. This method does not have the constraints associated with the limit equilibrium techniques shown in Figure 5 as the relationships have been derived through numerical simulation of modeled conditions. The critical parameters identified in Figure 7 do not incorporate a factor of safety as they are identified in the numerical model by being either stable or unstable. Therefore one must apply a safety factor on the input parameters to ensure an adequate design factor of safety is incorporated. The cemented rock fill database shown in Table 1 was overlain onto Figure 7. The type of failure occurring is indicated for each curve by either “S” for sliding, “Rc” for rotational-crushing, “Rb” for rotational-breaking, and “F” for flexural failure modes. The degree of cohesion at the wall contact allows flexibility in design depending upon the quality of fill placement. It must be recognized that based upon the cases identified in Table 1 and plotted onto Figure 7 that the depth of mat largely corresponds to 50%-100% cohesion on the hangingwall contact.

2.3 Other Factors

The above attempts to outline a methodology for span design. It is critical that the method be calibrated for individual sites, incorporating critical factors such as seismic conditions, installed support, and methods of fill placement as these all

prone sill pillar was formed when mining from below would approach the overlying mined out level. As a result of a double rockburst fatality in 1969, the mining front was changed to a “centre lead stope” geometry. In 1973 the first computer controlled seismic monitoring system was installed, and pillar distressing was routinely carried out when a sill pillar was mined to approximately 12m (thickness).

This rockburst strategy allowed mining to proceed safely down to below the 4660 level (~1420m below surface). In 1982 the mining front entered a highly burst prone formation, and serious rockburst problems were encountered. As a result of rockburst fatalities in 1984 and 1985, Hecla initiated an experimental underhand cut-and-fill stope along the east abutment of the mine. After another rockburst fatality in March 1986 Hecla realized that it was not possible to manage their rockburst problem with overhand cut-and-fill mining. Production mining at the Lucky Friday was stopped in April

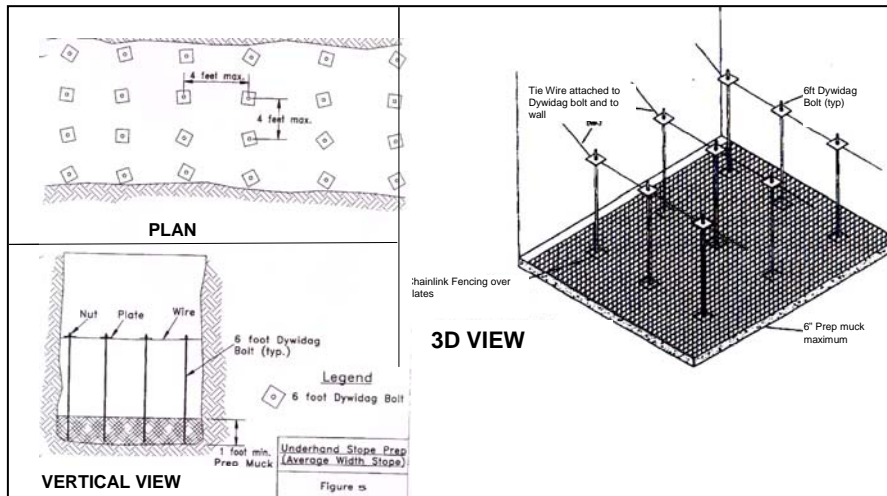


Figure 8: Lucky Friday Mat

1986, and plans were made to convert the entire mine to mechanized underhand cut-and-fill mining geometry, which they named LFUL – Lucky Friday underhand longwall. The key features of this mining method were that pillars would never be formed, and the mining would be carried out under a stable, engineered, paste type fill back.

Production mining at Lucky Friday resumed in October 1987 incorporating the above changes. Despite increased rates of rockbursting, as well as larger magnitude bursts (M1 4.1), underhand cut-and-fill mining at

the Lucky Friday has been carried out without any serious rockburst injuries or fatalities, and with greatly increased productivity at significantly reduced costs. Underhand mining has allowed Hecla to very effectively manage their rockburst problem. The miners have a higher sense of security working below an engineered back. *Management has said that the mine would likely have never reopened after 1986 had it not been for the all the benefits of LFUL mining.*

Finally, the paste backfill is only very rarely damaged by the effects of nearby rockbursts. The only burst induced fill failure at the mine occurred in 1991 during mining of a remnant pillar where a 3.5M1 burst caused the wall to fail and in turn undercutting the past back which collapsed. The peak particle velocity at the hangingwall/fill mat was approximately 1m/s. Despite closure from ongoing mining, as well as closure and shock loading from the burst, the fill was not rumbled as might have been expected.

4.0 Other Observations

The focus of the present research is towards mining under paste, however, a much larger database of information exists for mining under cemented rock fill as derived from the Nevada database of mines operating within a weak rock mass (Brady et al., 2005) and the database shown in Table 1 reflects this. The limit equilibrium methods are similar whether one is working with a cemented rock beam or a consolidated paste. The differences lie in the resultant strengths associated with each as shown in Figure 4 where the cemented rock fills exhibit strengths (UCS) generally in excess of two to three times that of the cemented pastes. This is largely due to cement being able to be more evenly mixed with the larger aggregate as compared to the paste which is generally between (60% passing) 20 microns (fine) to 100 microns (coarse) for Mine #1 and Mine #12 in Table 1. The typical cemented rock fill optimum aggregate size is 50 000 microns or 5cm (2inch) for the database analysed.

The cohesion for paste fills was estimated as being 0.25 times the unconfined compressive strength based upon an internal angle of friction (φ) of 30° and derived from the Mohr-Coulomb relationship where the “Unconfined compressive strength = $2 * \text{Cohesion} * (\cos \varphi) / (1 - \sin \varphi)$ ”. A value of internal angle of friction of $35-40^\circ$ was employed for the rock fills.

The tensile strength was generally derived from the unconfined compressive strength for consolidated fills and a value of “ $0.1 \times \text{UCS} = \text{Tensile Strength}$ ” was employed (Jaeger et al., 1976).

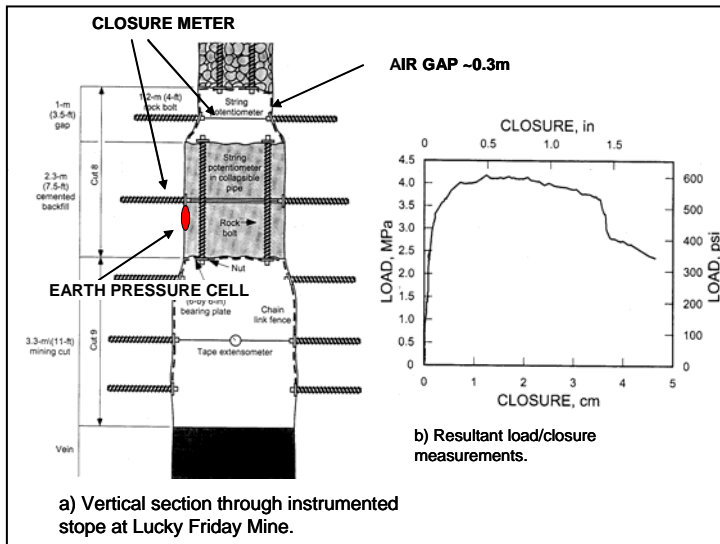


Figure 9: Instrumented underhand stope at Lucky Friday (Williams et al., 2001)

reach 40°C with stresses measured in excess of 69kPa due to the temperature difference. The vertical pressures that arise for a 3m pour height of paste fill would approach fill pressures of 0.02MPa/m of pour height ($\text{SG}=2.0$) or 60kPa of vertical pressure which is largely equivalent to the temperature correction on the earth pressure cell. The horizontal pressure would be a fraction of this. It is important to recognize the correction factor can be equivalent to the absolute value measured.

5.0 Conclusions

Mining under consolidated fills is becoming competitive to conventional cut and fill mining as increased spans and productivities are realized through reduced placement of ground support and more control on the mine cycle due to working under an engineered back. This requires a thorough understanding of the mechanism of support that one is relying upon which is the consolidated fill immediately above. The fill may be supported in terms of conventional bolts and screen in order to counter “cold joints” that may develop in the fill, account for variability in fill quality control and/or increase the overall factor of safety required due to seismic events in the close proximity. This requires an understanding of the stabilization affect of the consolidated fill and the mine environment that it is placed within. Through the gathering of site data, modelling of behaviour either analytically and/or numerically coupled with observation and measurement one will be able to advance the overall design criteria to provide a safe and cost effective workplace.

6.0 Acknowledgements

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A Young’s modulus for paste ranged from 0.6GPa to 3MPa (laboratory) for 10% binder and these values were field calibrated (Williams et al., 2004) to reflect field data through a combination of earth pressure cells embedded within a paste stope at the Lucky Friday mine (Mine #12) and the closure recorded and related by the relationship “ $\text{Stress} = \text{Modulus} \times \text{Strain}$ ” where the strain was measured by means of closure meters divided by the stope width. This is shown in Figure 9 with the resultant closure/load history. It is interesting to note that the loads upon reaching 4MPa (UCS) showed yielding of the paste. The closure at the Lucky Friday was in excess of 25mm. The rock mass of the wall contact was under 50% (RMR_{76}). Mine #1 showed closure values of under 10mm at similar depths, with the wall contact having an $\text{RMR}_{76} = 75\%$. Both sites are considered to be burst prone.

A further observation by Tesarik et al., 2005 was that the earth pressure cells are significantly affected by the paste cure temperature which can

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Table 1: Database of Underhand Cut and Fill Operations

UNDERHAND CUT AND FILL MINING UNDER CEMENTED FILL

MINE	%BINDER	SPAN (m)	SILL THICKNESS (m)	UCS (MPa)	COMMENTS
1	10	6.1	3	2	PASTE
2a 2b 2c	6.5 8% 8%	7.6 9.1 21	4.6 4.6 4.6	5.5 6.9 6.9	CRF CRF Design Mined Remote - No Cave 2" Minus Aggregate Go Under a Minimum of 14days Wall CRF 5-6% Binder Jam Tight to Back/Steep
3	7	3	3	4-11	Go Under in 14Days(9MPa UCS) CRF (4MPa Design) UCS is 11MPa (28 DAY)
4a 4b 4c	9 9 9	13.7 3.7 7.3	4 3 3	8.3 8.3 8.3	CRF Test Panel CRF Drift & Fill CRF Panel
5	7	2.7	3	3.4	CRF
6	6.75	4.9	4.3	4.8	CRF Go Under in 28 Days
7a 7b 7c 7d 7e 7f 7g 7h	10	1.8 2.4 3 3.7 4.3 4.9 5.5 6.1	2.7 2.7 2.7 2.7 2.7 2.7 2.7 2.7	0.7 0.3 0.5 0.7 1 1.4 1.8 2.3 2.9	PASTE (FS=1.5) Go Under in 7 Days - 28 Days (5% binder - 0.5MPa UCS 28D) (7% binder - 0.7Mpa UCS 28D) (10% binder - 1MPa UCS 28D) (12% binder - 1.2MPa UCS 28D)
8	7	4.6-6.1	4.6	5.5	CRF
9	10	5	5	4.45	CRF
10	12.8	6-9	6	2	High Density Slurry (78% WT SOLIDS)
11	10	3	3 (includes 0.9m air gap)	2.5	10% Cemented Hydraulic Fill Go Under in 7days (2.5MPa UCS) (73-75% Wt Solids) (UCS after 7 days)
12	8	2.4-4.6	3 (includes 0.6m air gap)	4.8	Go Under in 3days (2.4MPa UCS) 8% Paste (Coarse Tails) (no free water)
13	2-10%	NO UNDERHAND AT THIS TIME		0.2-2	(2% binder - 0.2MPa UCS 28D) (5% binder - 0.8Mpa UCS 28D) (7% binder - 1.2MPa UCS 28D) (10% binder - 2MPa UCS 28D)