Practical Geotechnical Design in Backfill at the Turquoise Ridge Mine, Nevada

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ABSTRACT

Cemented rock fill (CRF) commonly referred to as backfill has been utilized as the main support in the underhand cut and fill (UCAF) mining method at the Turquoise Ridge Mine (TRJV). The design is based on flexural instability utilizing fixed beam analysis, originally developed by Mitchell and Roettner (1989) and later Stone (1993). The methodology has been utilized to increase our average undercut width from 20 to 30 feet. The opportunity also exists to lower mine costs by adjusting the cement content without jeopardizing necessary mine design safety factors. Numerical modeling and instrumentation has also substantiated minimal pressures and movement from current designs, and could facilitate even wider undercut widths or expanded mining methods in the future.

INTRODUCTION

Cemented rock fill (CRF) commonly referred to as backfill has been utilized as the main support in the underhand cut and fill mining method at the Turquoise Ridge Mine. The TRJV mine site is located in northern Nevada, and is situated within the Basin and Range province, near the northeast end of the Osgood Mountain range (see Figure 1).

Figure 1. Location map of Turquoise Ridge Joint Venture

GEOLOGIC SETTING

According to Sandbak, Rai, Howell, and Bain (2012) the regional geology of the Getchell-Turquoise Ridge area is structurally and stratigraphically complex. Lower Paleozoic chert, pillow basalt, and deep-water sediments, and Paleozoic shallow-water to deep-water marine sediments were later intruded by Cretaceous granodiorite dikes and sills. This intrusive and surrounding metamorphic halo forms the core of the Osgood Mountains. Gold mineralization in the area is typical of other Carlin-type gold deposits. Gold occurs atomically in the lattice of arsenic-rich iron sulfides that formed chiefly within carbonate-bearing rocks that have undergone decalcification and argillization. Mineralization was apparently passive and exploited existing fracture porosity rather than whole-rock permeability.

MINING METHOD

The underhand cut and fill (UCAF) mining method is designed to accommodate the very low RMR, and low dipping (20-45 degree) ore deposits present at the TRJV property. In underhand cut and fill, the ore is initially mined out in 14'Wx14'H panels referred to as topcuts. Drifting is accomplished using conventional drill and blast techniques or by underground LHD, commonly referred to as muck advance. The typical drill pattern is smaller than the excavation size, and the amount of explosive is minimized depending on rock quality. Mucking is performed with either a 3 or 6 yd^3 LHD. After the completion of a panel, that panel is completely backfilled, tight to the back, with CRF. The next panel is advanced adjacent to the newly backfilled drift. This cycle is repeated in adjacent drifts, with one rib following the backfill of the adjacent drift until the desired width of the stoping area is reached. The backfill creates a pillar of strength that reduces the possibility of back failure by replacing the weaker host rock with a stronger, more homogenous material. The ideal sequencing of a top cut panel will involve driving an initial access panel down the center of the ore zone and mining the level on retreat towards the access. This procedure is used throughout the entire ore zone of the topcut level. Topcuts are occasionally mined and backfilled on advance due to the instability of the ground, however, this is typically avoided as it reduces the efficiency of the level. Figure 2A and Figure 2B show a schematic of an idealized UCAF design configuration and an example of an actual mining block from TRJV, respectively.

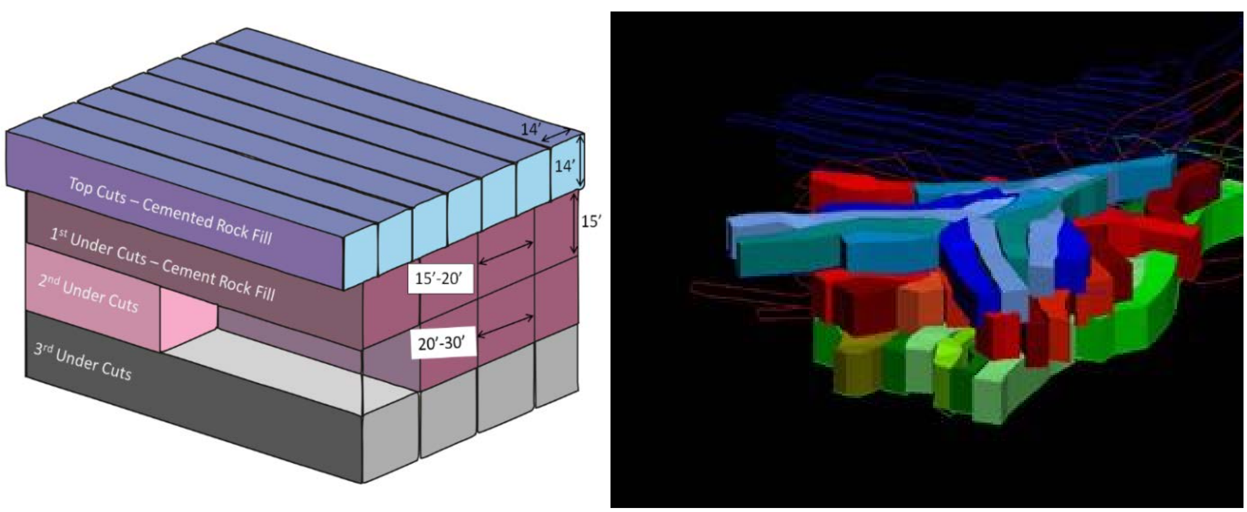


Figure 2A and 2B. Idealized UCAF schematic and actual panel layout of UCAF at TRJV

Upon the completion of the topcut level, the next level is advanced below the previously backfilled level. This level is referred to as an undercut. The undercut level is typically mined at a height of 15 feet, using the backfill from the topcut as the back. The height may vary from about 12 feet to over 17 feet, based on actual mining profiles. The width of an undercut can vary from 15-30 feet. At TRJV, it has been demonstrated that a span of at least 30 feet in width can be safely excavated beneath the consolidated backfill for the new undercuts. Access to an undercut is typically similar in design to topcuts, and often times utilizes the same tertiary access by removing the sill to access the undercut level. When mining beneath a backfilled topcut, a minimum of 8 feet of backfill is required to ensure sufficient support along the back of the drift. Similar to topcuts, an access panel is typically advanced to the end of the ore body with the remainder of the ore body mined on retreat to maximize efficiency. In the wider and more massive zones, two headings in a production panel may be worked at any one time to maximize efficiency.

ANALYTICAL MINING DESIGN PARAMETERS

It is long recognized that it is feasible to mine very wide undercuts under fill. Tesarik (2007) completed a study where a 45 foot undercut was opened up in the South Zone of the mine (TR 4050 Level, ~1100 feet below surface). The area was mined in stages, rather than full face rounds, to minimize the amount of open ground at any given time. Previous backfill tests and instrumentation at TRJV involved the use of borehole extensometers, biaxial stress meters, earth pressure cells, and deformation meters. The entire 45 foot excavation remained stable after approximately 250 days until the access drift was closed off. The maximum deflection measured in this area was only 0.2 inches (5 mm); and the calculated pressure on individual pillars was only about 60 to 90 psi (0.4 to 0.6 MPa), much smaller than the high pillar strengths of 600 to 1100 psi (4.1 to 7.6 MPa).

Backfill achieves 60 to 70% of the ultimate strength at 7 days, and 90% of its total strength at 14 days. The average strength of CRF at TRJV is 1000 to 1200 psi (6.9 to 8.3 MPa) after 28 days, nearly double the 600 psi (4.1 MPa) design standard. Because of the high strength achieved by the CRF at TRJV, a 14 day minimum cure time is required to ensure sufficient strength is achieved prior to working beneath recently placed fill.

Pakalnis (2009) did a study based on the historical CRF strengths achieved at TRJV using basalt and limestone aggregates. The study looked at the various failure mechanics associated with sill matt failure, and flexural failure specifically, based on work done by Mitchell and Roettner (1989) and Stone (1993). The conclusion of the study showed that undercut widths of 30 feet or greater could conceivably be mined under multiple layers of backfill. Even using the most conservative and lowest strength of basalt aggregate CRF, and reducing the strength further by 65% to 435 psi (3.1 MPa) due to aggregate size factors, a safety factor of 2 was established for a 25 foot undercut width. It also shows stable conditions for a sill height greater than 12 feet, and that the greater the sill height, the more stable the back. These results are outlined below in Figure 3, taken from Stone (1993).

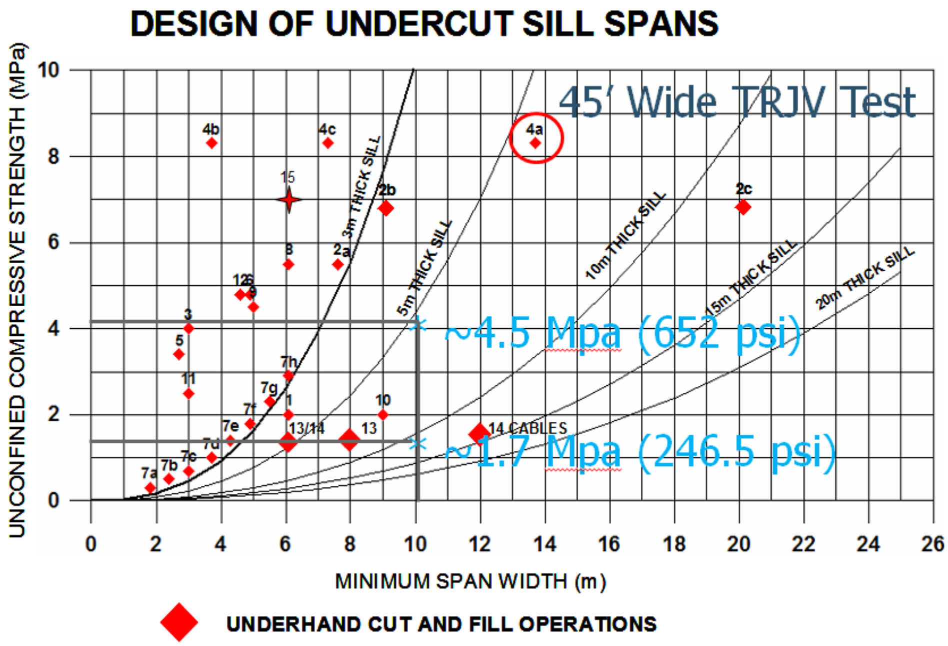
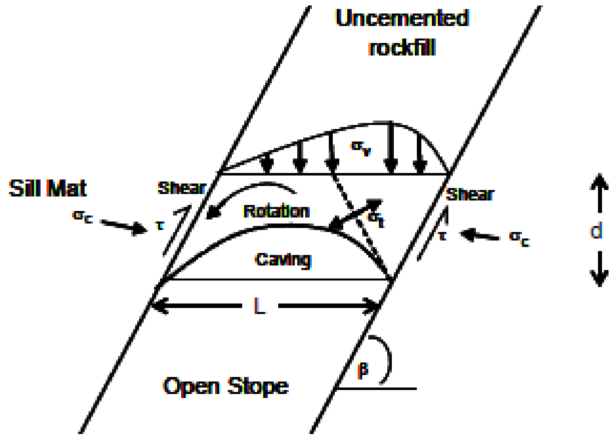


Figure 3. TRJV backfill strength stability chart versus undercut width from Pakalnis (2009)

ANALYTICAL ANALYSIS DISCUSSION

The analytical analysis utilized in UCAF mining is derived from work by Stone (1993), Mitchell and Roettger (1989) and Pakalnis (2009). The analytical methods look at methods of failure for various sill mat widths, heights, and strengths based on caving, flexural, and sliding failure. Figure 4 shows the various types of failures, and the equations associated with each. It should be noted that rotational failure is omitted from the analysis since a rotational failure is not kinematically possible with vertical side walls, as both sides must be detached. A rotational movement with vertical walls will induce a flexural type failure and is poorly accounted for with the rotational equations presented by Mitchell (1989).

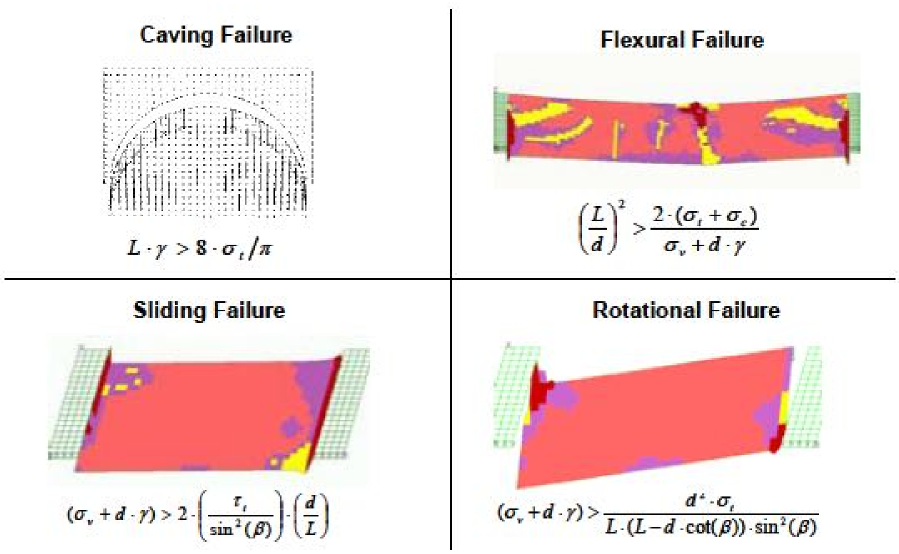
A few observations from the analytical analysis include that flexural failure dominates failure for wider sills, while narrower sills fail due to caving (see Table 1). Note that the critical strength for the CRF is less than 300 psi for nearly any size opening for both caving and sliding mechanisms. Flexural failure occurs when the induced tensile stresses along the bottom of the beam exceeds the flexural strength of the beam material. When a beam begins to sag, tensile stresses are induced at mid-span along the bottom of the beam and along the corners at the top of the beam. Flexural failure is only likely to occur if there are extremely low horizontal stresses acting on the CRF (via horizontal stope closure ground movement), or if the ground is in tension. Compressive failure can begin to occur when the horizontal stress reaches 50% of the unconfined compressive strength (UCS). Each failure type and the parameters associated with it are highlighted in Figure 4A and 4B.



a) Schematic showing typical failure modes after Mitchell, 1991.

Where:

- L = Span of the stope
- γ = Rockfill's unit weight
- σ_t = Tensile strength of the cemented sill
- d = Thickness of sill
- σ_c = Horizontal confinement (assumed zero – conservative)
- σ_v = Vertical stress due loading above sill mat
- T_f = Shear strength along fill/wall contact
- β = Stope dip angle



b) Limit equilibrium analysis of typical failure modes adapted from Mitchell, 1991.

Figure 4A and 4B. Sill mat failure schematics based on work done by Mitchell (1991) and Mitchell and Roettger (1989)

Shallow sill mats require greater CRF strength, as the width/height ratio is increased. This analysis has numerous assumptions built into the calculations, including that there is no additional support effects in the CRF from bolting in the sill mat, and high roughness on the side walls is ignored. No extra support besides the CRF itself is needed. A safety factor of 2 is employed for the conceptual design stage.

Table 1. TRJV Results of critical strengths, in psi, needed for support of caving, sliding, and flexural failure for sill thicknesses of 10 feet, 12 feet, and 15 feet, using a safety factor of 2.0. These results are based on work by Pakalnis (2009) and Hughes (2009).

Sill Thickness	Sill Width feet	Sill Width m	Caving psi	Sliding psi	Flexural psi
3m (10')	15	4.6	120	109	149
	20	6.1	160	165	352
	25	7.6	200	233	689
	30	9.1	239	312	1190
3.7m (12')	15	4.6	120	100	103
	20	6.1	160	151	245
	25	7.6	200	212	479
	30	9.1	239	280	827
4.7m (15')	15	4.6	120	93	67
	20	6.1	160	138	157
	25	7.6	200	190	306
	30	9.1	239	248	529

30 FOOT UNDERCUT WIDTH RATIONALE

Based on the above rationale, the decision was made to increase the maximum undercut width from 20 to 30 feet. Until 2 years ago, 20 feet wide undercuts were based on practical limits, not necessarily on optimum design. The results for a wider undercut study used a conservative CRF strength of 435 psi, based on a reduction correction factor for aggregate size strength. The study shows stability and sufficient safety factor for a 25 feet wide span for a 12 foot thick sill mat (see Figure 5). The analysis shows that for flexural stability 30 feet wide spans under backfill could be used based on CRF strengths of only 500 psi, and a 15 foot sill thickness. In practical terms, we could go wider, as there has been no predicted movement or slump, and our CRF design is a minimum of 600 psi with typical averages ranging from 1000-1200 psi for 28 day cure times.

Numerical Modeling shows that a homogeneous 450 psi CRF strength is stable under the proposed span of 15 feet (4.6 m) , 20 feet (6.1 m), 25 feet (7.7 m), and 30 feet (9.1 m) with a sill thickness of 16 feet (4.9 m). This is displayed below in Figure 5. The 30 foot wide undercut is not unprecedented and has been utilized with success for several years at Barrick's Cortez Hills property. Figure 5 and Table 2 summarize the results regarding the types of failure and calculation of critical CRF strengths necessary at TRJV.

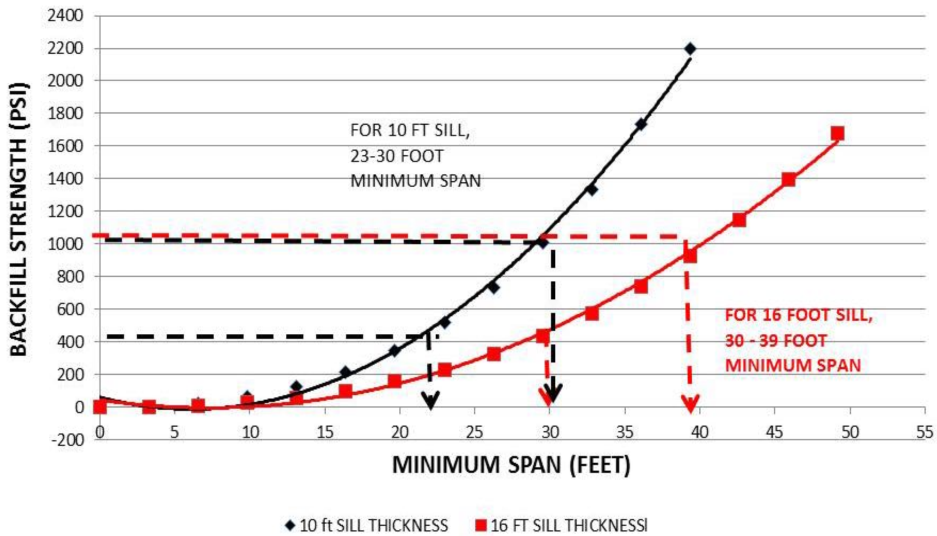


Figure 5. Design criteria for undercut widths and critical CRF strengths with sill thicknesses of 10 and 16 feet. Based on a safety factor of 2 and an assumed flexural bending failure, taken from Stone (1993).

Table 2. Critical CRF strength versus sill mat thickness (using flexural beam analysis). Typical CRF strengths achieved at TRJV are highlighted in grey.

Undercut Span	Undercut Span	Sill Mat Thickness					
		feet					
		10	12	14	15	16	17
m	feet	Critical Strength (psi) for CRF Backfill					
0	0	0	0	0	0	0	0
1	3	5	4	3	3	3	3
2	7	23	18	15	14	13	12
3	10	61	46	38	34	31	29
4	13	124	92	74	67	62	57
5	16	217	160	128	116	105	97
6	20	346	253	200	181	164	151
7	23	517	376	296	266	241	221
8	26	735	531	416	374	338	308
9	30	1007	724	564	506	456	416
10	33	1337	957	742	665	599	545
11	36	1731	1234	955	854	768	697
12	39	2196	1560	1203	1075	965	875
13	43	2736	1937	1490	1330	1192	1081
14	46	3358	2371	1819	1622	1453	1315
15	49	4067	2864	2193	1953	1748	1581

HGB 3159 PANEL RESULTS

The first 30 foot wide undercut production panel at TRJV was reviewed in a study by Rodriguez, Powell, and Sandbak (2015). The mining of the panel began in March of 2015, and was completed in less than 3 months. Rather than mining in broken up sections, as done in the past, the entire 30 foot wide drift was mined with each round of advance. As mentioned above, the design criteria for minimum access under a CRF back is 14 days, a minimum strength of 600psi (4.1MPa). The average CRF strength was found to be 1312 psi (9.1 MPa) with 95% of samples producing strengths greater than 600 psi (4.1 MPa) and 80% greater than 1000 psi (6.9 MPa).

An audit of the HGB 3159 was conducted using a MAP-3D finite element model by Pakalnis (2015). This model is shown below in Figure 6. The analytical design shows that a 30 foot (9.1 m)or two lift height CRF slab exists for a 37.8 foot (11.5 m) wide undercut, and requires only 290 psi (2 MPa) as the critical CRF compressive strength. It must be noted that these values must be in situ, and QA/QC is essential. Presently, these spans have been shown to be stable with strengths in excess of 600 psi (4.1 MPa) and approaching 1320 psi (9.1 MPa) for the 3” minus aggregate CRF.

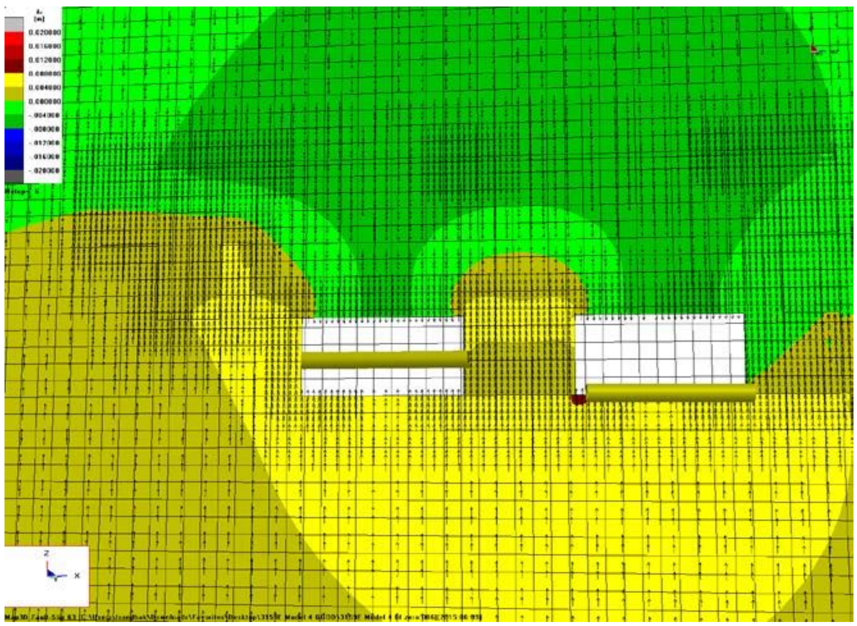


Figure 6. Map 3-D modelling showing predicted horizontal drift movement concentrations. Model shows 0.016 feet (0.005 m) (yellow) left rib x displacement and 0.007’ (0.002m) (brown) right rib x displacement.

HGB 3159 INSTRUMENTATION

For this trial of a 30 foot wide undercut, the following instrumentation was utilized. The instrument locations and a photo of the drift are shown below in Figure 7A and 7B respectively.

1. Placing 2 MPBX (Multi-Positional Borehole Extensometers) 20’ into the back anchored into previous sill cuts to access roof convergence (flexural) and to determine potential for slip at layered or cold joint contacts
2. CDX (Cross Drift Extensometer) points to measure horizontal convergence or closure between the walls.
3. Vibrating Wire Stress Meters at two locations to try to monitor stress changes
4. Observations/quality control design standard minimum as practiced (minimum design strength of 600 psi at 28 days).

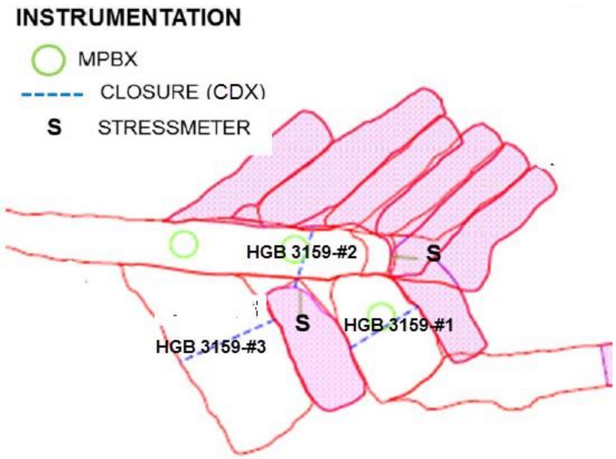


Figure 7A and 7B. Plan map of HGB 3159 level, showing MPBX, CDX, and stressmeter locations. Photo of wider undercut area using tape extensometer for horizontal convergence measurement (HGB 3159-#3).

MPBX results: Thus far we have not seen any movement in the back from the MPBX's. There has also not been any movement along the cold joints between the two adjacent fills pictured in Figure 7. Numerical modeling suggested a flexural vertical movement of at least 0.25 inch for a 30 foot span. Earlier studies by Hughes (2009) using a finite element FLAC 2_D Model showed potential failure with a 360 psi (2.5 MPa) CRF strength.

CDX results, and stress estimates. The vibrating wire stress meter data results were inconclusive, as both vibrating wire stress meters quickly registered near zero changes, and have remained at near zero since installation. Minimal stress concentrations on the pillar are expected due to the de-stressing effects of the overlying CRF.

The cross drift or horizontal closure movement in the ribs has been minimal, but more than anticipated from the stress modelling, showing approximately 0.15'' to 0.4" total convergence over 16 months. With a current horizontal closure rate of less than 0.001"/day, and based on a threshold for rapid convergence concern at TRJV of 0.04"/day, there is minimal concern that the drift is at risk of failure. Despite this minimal concern, the level is still being monitored regularly. Stress analysis based on the actual convergence should be calibrated as is presently undertaken for the HGB 3159, and is expressed as Stress = Strain multiplied by the Modulus (see Table 3).

Table 3. CDX instruments and calculated stress based on CRF modulus of 1320 MPA.

CDX Point Location	Span Width	Span Width	Total Convergence	Calculated Vertical Stress
	feet	in	in	psi
HGB 3159 #1	27.67	332	0.156	90
HGB 3159 #2	24.17	290	0.442	292
HGB 3159 #3	37.83	454	0.156	66

The lateral closure through modeling, using a modulus of 191.4 Kpsi (1320 MPa), indicates that the 0.15 inches of closure recorded over a 37.8 foot (454 inch) span displays an induced back stress of only 66 psi (0.46 MPa). This is determined using the following equation:

$$\text{stress } (\sigma) = \text{strain } (\varepsilon) \times \text{Modulus } (E), \text{ or } (0.156/(454)) \times 191400\text{psi} = 66 \text{ psi}$$

This shows that the lateral stresses in the ribs and backs are much less than the strength of the CRF and are of minimal concern for crushing. These values should be established in terms of modulus (E), relating actual measured horizontal and vertical back deformation, and induced stresses upon mining of the individual panels, replacing with CRF and relating to the numerical model as being developed by TRJV.

CRF RECIPES

The backfill system utilized at TRJV is a typical cemented rock fill system. The CRF consists of a cement/fly ash mixture or just cement with crushed and sized aggregate. The material is batched in a mixing plant, hauled to the particular excavation to be backfilled and jammed into place using a modified LHD with a push plate as the jamming tool. In the top cut ore headings, the backfill may have to be trammed to the backfill face using an LHD. A backfill replacement factor of 0.85 tons for each ore ton removed has been found to be accurate for estimating quantities required. Backfill is an important part of the total ground support in active mining panels. The major benefits of backfill are as follows:

- Provides lateral support to pillars to stop spalling and prevent collapse
- Reduces stope wall closure
- Reduces subsidence of mine workings
- Creates stability to mining block
- Solid, uniform backfill is usually better than jointed rock it replaces

The samples are designed to be tested at a 7 day, 14 day, and a 28 day interval. The final strength of the backfill is designed to average or exceed 600 psi compressive strength. The formulas are based on the percent of cementatious material used and flyash (see Table 4).

Table 4. Backfill batch plant recipes for the North Zone Batch Plant (NZBP) and the 1250 Batch Plant (1250 BP).

Formulas		Water	Cement	Flyash	Aggregate	Admix (Plasticiser)	Cement Content	Flyash Content	Total Cementatious
NZBP	1250 BP	lbs	lbs	lbs	lbs	oz	%	%	%
01 (50:50 cement:flyash)	24	1000	888	888	22,000	88	3.6	3.6	7.2
02 (All cement 5%)	6	800	1150	0	20,000	88	5.2	0	5.2

The four most common formula recipes used here are based on a 7.2% total cementitious with a 50:50% flyash-cement ratio, or a straight 5.2% cement content. The opportunity also exists to lower backfill costs with lower cement content without jeopardizing necessary mine design safety factors, as the average strength of CRF is well above the minimum design standard. The most practical solution would be to test the strength reduction undergone by the backfill by reducing the cement content in the mix. The optimal cement content will minimize the cost of the backfill while still maintaining the minimum design strength.

CONTINUED USE OF 30 WIDE UNDERCUTS

As plans have progressed at TRJV, additional 30 foot wide undercuts have been successfully mined. With the completion of these areas, various advantages and disadvantages have been realized. The most important advantage is the increased efficiency and production achieved from the level. This increase in efficiency was achieved for several reasons. The first being an increase in the tons mined with each round of advance. Also, by mining a larger area, additional backfill cycles were removed from the mine plan, allowing for a reduction in the total down time of the level between the completion of a panel and the last load of backfill being placed. This also helped to reduce the travel time to and from the level, as it allowed for, what would have been multiple cycles worth of steps, to be compiled into one cycle. Lastly, by utilizing similar mining techniques (i.e. drill pattern, bolting pattern, round length, etc.) the operations group was able to begin mining in these headings with little to no additional training.

Along with these advantages, several disadvantages were discovered as planning progressed. The first, and most important, was the difficulty to get total buy-in from operations. One of the biggest obstacles from achieving complete operational buy-in was due to the history of ground control issues and ground falls the mine has experienced. Due to the poor ground conditions present at TRJV, the mine has many historical issues related to large drift sizes resulting in ground failure. Because the mine recently increased top cut drift sizes from 10’x10’ to 12’x12’ and then shortly after to 14’x14’, the majority of personnel were more willing to embrace changes to standard design parameters. Another major push-back that was received was based on the sheer amount of material produced as the drift advanced. A single round taken with 12 feet of advance required 74 buckets and nearly an entire shift to muck out. This was addressed by creating additional areas for muck to be stored and coordinating with the support group to maximize the efficiency of haulage. With the increased drift size, the opportunity to load trucks directly from the face of the drift was taken advantage of. This helped alleviate some of the burden of the large amount of muck generated. This also helped move muck from underground to the skips to be hoisted to surface faster.

Due to the relatively new inclusion into the mine plan, it was important to find ways to effectively monitor and maintain ground support in these areas. This brought about additional challenges and disadvantages. Because of the nature of the native rock, it is typical for the drift to undergo squeezing over the design life, leading to the ground support visually taking weight, deflecting, and yielding. Due to the large drift size operators typically spend less time in close proximity to ground support, making these visual clues less noticeable. To combat this issue, special importance was placed on visual inspections of ground support when working in the larger drift size. If any issues were encountered, they were to be reported to the geotechnical group who would then evaluate the stability of the drift. Another major issue is the amount of time that the drifts are left unsupported. Because of the increased amount of time required to muck, drill, and bolt for each round, the heading is left unsupported for longer periods of time. This could become an issue if the heading is planned in poor ground conditions. To alleviate this concern, the geotech group was consulted with each drift to ensure that the drift dimensions were adequate for the ground conditions present.

GOING FORWARD

By continuing to apply improving technologies and new ideas to the mine design, several different opportunities exist for future improvements at TRJV. One potential improvement is increasing the size of undercuts beyond the current maximum of 30 feet. Current backfill strengths suggest that undercut spans of 45 feet or more could be effectively employed under two layers of fill. This would allow for a further increase in production from undercut levels with minimal change to the mining plan.

Another potential benefit to these increased dimensions would be the potential utilization of mechanized mining in the area. The use of roadheaders is being investigated for use in the very weak ore body, and with continued shotcrete support for rib rock. Current research is being conducted into the use of early strength shotcrete that could yield a 145 psi (1 MPa) compressive strength within one hour, enough time to utilize bolting if needed along the ribs of the undercut drifts.

As the undercut size increases, the effectiveness of a mechanized miner would also increase due to the large amount of material it could mine with minimal time spent moving from one heading to another. One of the most common issues at TRJV is the overall poor rock quality encountered in and around ore zones. By mining and backfilling sections of this poor quality host rock, drifts can be designed to be surrounded on all sides with backfill, essentially eliminating the issues associated with the native rock. This could potentially allow for larger bulk extraction methods, such as benching or stoping. The design criteria for wider undercut widths could expand our current thinking on alternate mining methods; including taller undercuts based on mining experience and monitoring of drifts in and around the undercut openings. With proper ground support of the rock pillars, it also facilitates larger stopes that could be contained on the top by caving that would end at the backfill cap. These bulk extraction methods could greatly increase production from ore headings. In total, effective utilization of backfill at TRJV has allowed for increases in mining efficiency and continues to be a driving factor in the evolution of the life of mine plan.

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