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An Alternative Cemented Hydraulic Sand Backfill Method for Large Spans in an Underhand Cut-and-Fill Deep Metal Mine

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ABSTRACT: This paper documents the development and implementation of an alternative backfill design method using cemented hydraulic sand backfill (CHSB) for mechanized underhand cut-and-fill (UCF) mining along a wide vein structure by NIOSH researchers and mine staff for U.S. Silver Corporations' Galena Mine. The mine historically mined discrete silver-copper veins with widths ranging from 0.3 m up to 3 m using conventional overhand cut-and-fill mining and hydraulic sand backfill. A cement-mixing circuit added to the backfill plant in 2001 enabled backfilling with CHSB for mechanized UCF mining of the 72-Vein and extraction of remnant pillars. The CHSB design method, initially developed in 2001, allowed for a stable exposed backfill span up to 3.6 m wide for distances less than 6 m in length. In the mid-2000s, a high-grade lead-silver vein was discovered approximately 1,580 m below surface with ore zones up to 10 m wide and 21 m long. To facilitate UCF mining along the wide vein structure, an alternative CHSB design and support method was required to allow for stable backfill spans in excess of 3.6 m for extended lengths. The alternative CHSB design and support method developed, using available materials and equipment, allowed for safe extraction of the wide, high-grade ore with minimal delay to mining and only marginal increases in overall costs.

1. INTRODUCTION

The Galena Mine historically mined along narrow silver-copper veins that rarely exceeded 4 m in width. Lead veins and areas of disseminated lead were not explored or mined due to their relatively low economic value compared to the silver-copper veins. In the mid 2000s, a number of wide, high-grade lead-silver veins were discovered approximately 1,580 m below surface, with widths up to 10 m and strike lengths up to 21 m. Underhand mining would be required to extract the ore due to limitations in accessing deeper levels. The current backfill infrastructure and support methods allowed underhand cut-and-fill (UCF) mining along the narrow silver-copper veins. Modification to the backfill infrastructure was not an option, so engineering design and support parameters were modified to utilize available resources and underground operator skills to provide a stable cemented hydraulic sand backfill (CHSB) beam that could be undermined safely in the wide lead-silver vein stopes. NIOSH researches worked with mine staff to conduct mechanical property tests on the CHSB to determine design properties and help develop the support modifications. This paper summarizes the methodology and effectiveness of the modification.

1.1. Location and Geotechnical Conditions

U.S. Silver's Galena mine is located in northern Idaho, U.S.A., in the Coeur d'Alene Mining district along the Silver Belt, producing primarily silver and lead concentrates with some copper byproducts. Figure 1 shows the location of the mine within the district.

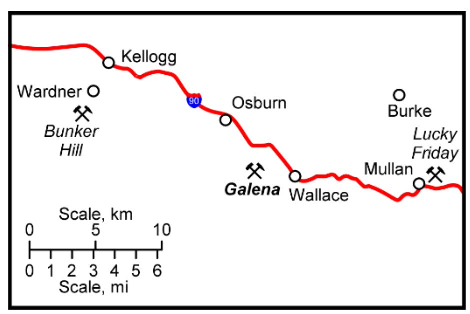

Fig. 1. Location of the Galena mine in the Coeur d'Alene Mining District.

The region is structurally complex and dominated by steeply dipping shear structures associated with the Lewis and Clark Line (White, 1998). The shape of the mine aligns with the major WNW faults in the area and the associated shear structures between them. Mineralized zones commonly occur as structurally controlled narrow veins along discrete sections of the faults and shear structures, primarily where they intersect with the Upper

Revett Formation (Mauk and White, 2004). Vein widths are usually less than 4 m. Recent discoveries have identified zones of economically viable mineralization with widths up to 10 m.

Mineralized zones typically have hydrothermal bleaching and silica flooding. Rock Mass Rating (RMR) values are normally between 25–70, driven primarily by moderate to strong fracturing and bedding plane detachments. Strengths vary from 77–225 MPa and are strongly anisotropic due to bedding (Chan, 1971).

Current mining occurs from 750 m to 1,585 m below surface. The principal stress is oriented parallel to the WNW striking regional lineaments with magnitudes between 1.2–1.5 times the vertical overburden (Ageton, 1967). The combination of depth, high horizontal in-situ stress, and relatively strong wall rocks is conducive to mining-induced seismicity. Figure 2 shows a simplified plan view of the mine at depth.

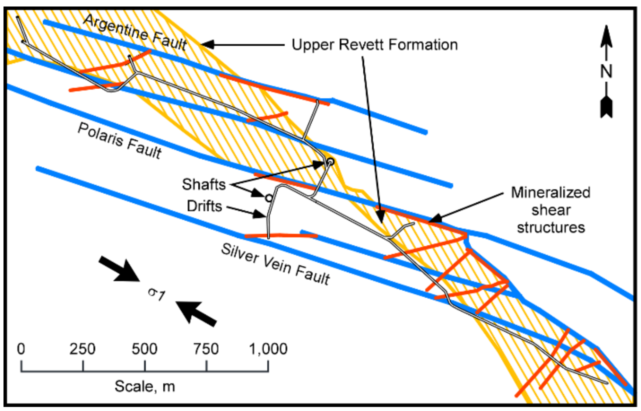

Fig. 2. Plan view of mine layout and alignment to sigma 1 and Polaris Fault.

2. BACKFILL OPERATIONS

2.1. *Hydraulic Sand Plant and Backfill*

Mining at the mine from the late 19th century to the early 1950s primarily utilized timbered cut and fill to depths of up to 600 m with minimal backfill. Discovery of the Silver Vein on the 3000-Level (914 m below surface) in 1953 and subsequent mining to greater depths encountered increased stress loads, resulting in increased dilution from overbreak and damaging mining-induced seismicity. To increase the ore grade and mitigate the seismic activity, the mine built a hydraulic sand backfill plant in 1956 (Visnes, 1957). The plant has been in continual use since its completion with few modifications.

Whole tails are pumped directly from the mill to the backfill plant. Cyclones remove the ultra-fine particles, which account for approximately 20% by weight of the whole tails and increase the percent solids from 28% up to 60-72%. Figure 3 shows the particle size distribution for whole and cyclone tails. This data represents tailings since August 2014 when the mine began processing primarily lead-silver ore, which has a higher percentage of fine particles compared to the silver-copper ore historically processed. Three 65,300-liter-capacity storage tanks provide 100 tons/hour of hydraulic sand to the stopes via a gravity-fed distribution network, consisting of a 63.5-mm inner diameter, rubber-lined, steel pipe down the Galena shaft with 78-mm, unlined steel pipe laterals on all the levels. Decanting requires 5–12 hours before production resumes.

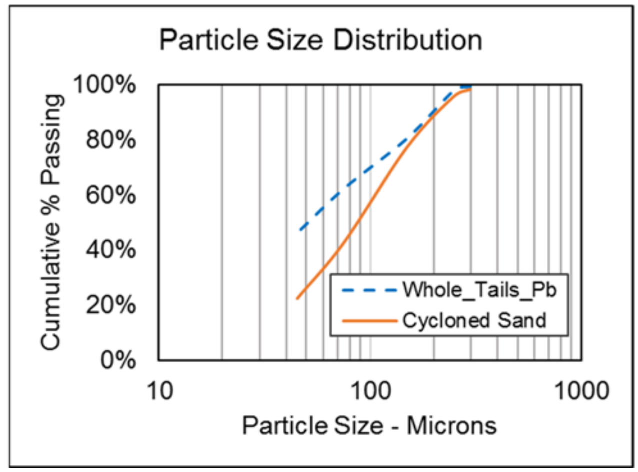

Fig. 3. Particle Size Distribution curves for whole tails versus cyclone sand. Both are from lead-silver ore.

2.2. *Cemented Hydraulic Sand*

A project in August 2000 with Itasca Consulting added a cement-mixing circuit to the existing backfill plant to allow production of CHSB for UCF mining in timbered and mechanized stopes (Pierce, 2000). The original design produced a CHSB product at 75% solids with 10% binder using the silver-copper tailings. The increased fines in the lead-silver tailings required an increase in the binder content to produce an equivalent CHSB product. Currently, the CHSB produced has 68–72% solids with an approximate binder content of 14%–19%, delivered underground at a rate of 63 tons per hour via the gravity-fed distribution network.

Removing a larger portion of fines from the tails via the cyclones is possible, but the reduction in solids dramatically increased the time required to produce a sufficient amount of sand for backfill, thereby affecting the production schedule. The economic conditions are not currently favorable for replacing the backfill plant with a paste or other high-density backfill plant. Conversion to a high-density backfill product would also require replacement of the current distribution network, estimated at over 15 kilometers in length.

The backfill plant can only provide approximately 200 tons (109.4K Liters) of CHSB in any one pour due to limitations in the cement silo capacity. Cells larger than 200 tons require special planning and coordination between the surface backfill plant and backfill crews underground. This volumetric restriction requires that stope cuts are poured in a series of cells from face to entry, separated by wooden barricades. Normally, one cell of CHSB is poured per shift taking an average of 2.75 hours. Un-cemented hydraulic sand is poured during the rest of the shift until the cement silo can be refilled. Backfill cells are laid out by either the engineering department or supervisors based on the average width of the drift to ensure they are correctly sized. The narrow silver-copper veins often had very linear ribs, and cells could be measured out by taking a few width measurements with a tape measure and then pacing out the calculated lengths. Table 1 gives examples of cell sizes based on the average width of the mined drift.

Table 1. Cell Lengths for CHSB Pours

Average Drift Width (m)	Strike Length Between Barricades (m)
1.8	29.3
3.0	17.5
4.5	11.7

While the requirement of building multiple backfill barricades is time consuming and increases the overall costs, it provides a few advantages. Backfill depths can easily be maintained throughout the length of the cut, which usually have a 2% incline from start to finish. This minimizes the potential for a too shallow final fill depth at the far end of the cut. It also allows for full decanting between pours, reducing the dewatering load on the pumps in the area and reducing the potential for decant water to flow through freshly poured CHSB, which is detrimental to final strengths.

3. UCF IN NARROW VEINS

At the time of the CHSB project, mining was primarily occurring along the 72-Vein, located in a section of the Polaris fault below the 5500-Level (1,675 m) where vein widths were between 2 and 3 m. As part of the project to add a CHSB mixing system, Itasca Consulting developed UCF design and support parameters using numerical modeling and empirical methods for mining along the 72-Vein. Cut heights were restricted to 3 m to accommodate installation of ground support with pneumatic, hand-held jack-legs and to control squeezing of the ribs by minimizing the effective hanging wall span.

The design parameters required final fill depths to be no less than half the span of the expected cut below, with a minimum CHSB unconfined compressive strength (UCS) of 1 MPa. Because of the restricted cut height, fill barricades were built to a height of 2.4 m, leaving enough room for a 0.45-m-thick layer of prep rock along the sill and backfill distribution pipes to pass over the barricade. Test work showed that the CHSB had 11% settlement after decanting, which resulted in a final beam depth of 2.1 m. This fill depth could support a stable exposed backfill span of 4.2 m on the next cut. Vein widths were rarely more than 3 m, and fill failure from overexposed spans was not an issue.

Part of the support plan for using CHSB also required reinforcement of the backfill beam. Upon completion of a cut, stopes were prepped for backfilling. Preparation entailed first cleaning out as much remaining ore as possible by scrapping the sill with an LHD. After the clean out, a 0.45-m-thick layer of coarse prep rock, consisting of low-grade ore retrained underground from the stope, would be laid down and leveled. A layer of prep rock is standard for most types of cemented backfill. It is critically important for CHSB because it provides adequate drainage for the increased percentage of decant water as compared to higher-density backfill material such as paste.

For internal reinforcement, rows of 1.8-m-long, #7 all-thread bolts were installed vertically on 1-m × 1-m spacing in the prep rock. A set of bearing nuts and plates that have been spot-welded together were threaded onto each bolt at both ends to create a bearing surface at the base and top of the backfill beam. The supports were referred to as "helicopters." Figure 4 is a typical example of a backfill barricade with "helicopters" in the foreground. Figure 5 shows a schematic cross section of the internal support plan. More details on the CHSB support system can be found in Williams et al. (2007).

Fig. 4. Picture of a backfill barricade prior to completion in the background. Rebar helicopters and a stull for the next cell are in the foreground.

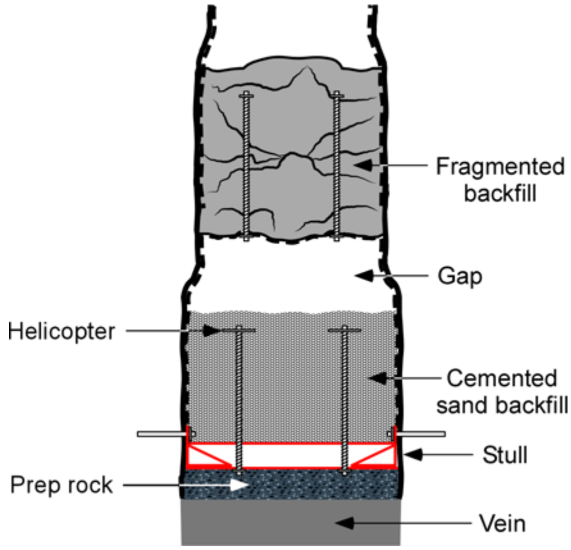

Fig. 5. Cross section view of typical UCF CHSB support prep. Stulls are used in wide span areas only and are discussed in later sections.

To ensure adequate strength requirements, two quality control samples are taken during each pour for 7-day and 14-day UCS testing, using 75-mm × 150-mm cylinders. The cylinders are tested in a small onsite lab. Figure 6 shows the QA/QC data from 2014–2016. Although the initial Itasca Consulting study specified a 1-MPa minimum strength requirement, in the years following commissioning of the CHSB circuit a 2-MPa minimum UCS from the QA/QC cylinders was found to provide adequate strength. Pours with UCS values below 2 MPa were more prone to failure due to overpressure from blasting especially in wider areas such as intersections. The 2-MPa requirement is referred to as the Functional Minimum UCS.

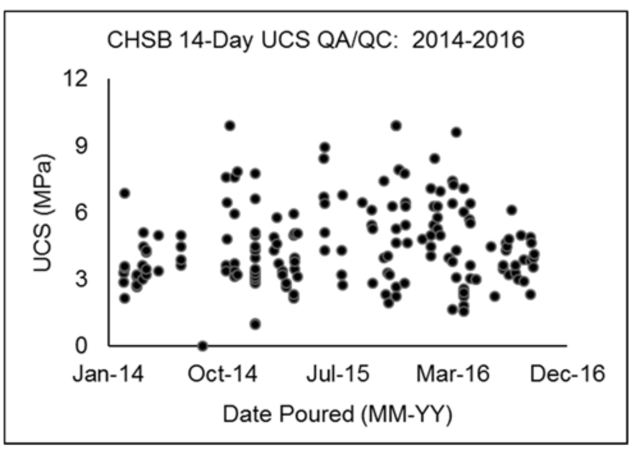

Fig. 6. 14-day UCS QA/QC data from in-house testing for 2014–2016.

Following the change to processing primarily lead-silver ore, samples of the lead-silver CHSB were made and tested at the National Institute for Occupational Safety and Health (NIOSH) Spokane Research Laboratory for UCS and tensile strength to ensure the new tailings created an adequate CHSB product. Table 2 lists a summary of the results. All results exceeded the 2-MPa minimum functional UCS.

Table 2. UCS and tensile strength of lead-silver CHSB after 7 and 14 days of curing (Binder content, 18% by dry weight).

Property	7-Days (MPa)	14-Day (MPa)
Average UCS	3.51	4.17
Minimum UCS	2.52	3.82
Maximum UCS	4.03	4.79
Tensile Strength	0.40	0.55

4. UCF IN WIDE VEINS

Historically, lead veins and areas of disseminated lead were not explored or mined due to their relatively low economic value compared to the silver-copper veins. In the mid 2000s, a number of wide, high-grade lead-silver veins were discovered approximately 1,580 m below surface at the 5200-Level. The two main lead-silver ore structures were identified as the 175-Vein and 207-Vein. These veins contained high-grade ore in zones up to 10 m wide.

Rock mass characterization of the veins had RMR values of 40–70 with well-developed shear fracturing in the ore between vein parallel slips. Both of the veins were oriented subparallel to the maximum stress, reducing the potential for rockbursts and squeezing ground.

Improvements to level access and ventilation in 2013 allowed for mechanized development and production along the 175-Vein and 207-Vein. Mining of the veins above the 5200-Level using timbered horizontal overhand cut-and-fill (OCF) was completed in early 2014. With no access to the veins from the next level below, mine plans were developed that would utilize mechanized UCF below the 5200-Level.

4.1. Wide Vein Design Criteria

OCF mining above the 5200-Level had encountered ore zones along the 175-Vein and 207-Vein with widths between 5 to 10 m and with ballroom areas greater than 4.2 m wide for strike lengths up to 21 m. Ballrooms in the OCF mining were stabilized effectively with longer ground support. Underhand mining would require modification to the current CHSB design parameters and support standards to allow stable spans over 4.2 m.

Analysis of reported CHSB failures or problems in the narrow-vein UCF areas since 2001 showed that issues with CHSB stability began occurring when stope widths exceeded 4.2 m for strike lengths of 6 m or more. In zones where cut widths exceeded 4.9 m for more than 6 m in strike length, failure of the backfill was almost certain. A large portion of the 175-Vein and 207-Vein exceeded this

4.9-m limit. Using this empirical analysis, an exposed span limit of 4.2 m was applied to the CHSB support design

Increased cut heights of 3.6 m were evaluated and implemented in certain sections. The increased height would accommodate a deeper backfill pour of 2.7 m, as compared to the normal 2.1 m, allowing a wider backfill span to be exposed. The deeper fill mat could potentially support spans up to 5.4 m, but at the cost of additional timber for taller backfill walls and increased pour times. It would still not provide adequate support for the very wide and long ore zones. Cut heights greater than 3.6 m were not feasible due to equipment constraints and ground control issues.

The solution for the wide span areas was to install 25 × 25-cm timber stulls (Staley, 1962) between the hanging wall and footwall at calculated intervals within each backfill cell in order to constrain the unsupported backfill span to 4.2 m or less. Stulls were placed over the prep-rock at locations predetermined by engineering and were anchored to the ribs with wooden wedges or custom-fabricated steel "chairs," which could be bolted to the rib using grouted all-thread rebar. This augmented CHSB design became known as a Super-Prep. Figure 5 shows the location of a stull in a cross-section view of the Super-Prep. Figure 7 shows a timber stull set in a chair during preparation of a Super-Prep.

Fig. 7. Span mitigation stull positioned in custom "chair," anchored to the rib with all-thread rebar. Note the vertical piece of wood wedged between the stull and the rebar bolt to keep the stull from floating during pouring of the cell.

4.2. Engineered Design of Wide-Span Backfill Cells

As discussed earlier, special preparations were necessary for the backfill plant to supply more than 220 tons of CHSB in one pour. The wide nature of the lead-silver veins dictated that backfill cells were either extremely short in length between fill walls, which was highly uneconomic, or each cell would need an accurate volumetric design that would allow effective planning between the underground operations and backfill plant to allow for larger than normal pours.

After each finished cut was surveyed, an engineer would develop a cell design map based on the locations of the backfill barricades and the pour size between them. Barricade location was critical, not only for the final pour size, but for the stability and safety of the barricade during pouring. The wide veins and taller cuts meant that each barricade would be required to maintain stability under increased hydraulic loading compared to normal narrow vein backfill pours. Barricade failures during pouring can be dangerous and potentially fatal to miners working nearby (Potvin et al., 2005). Barricades were placed in narrow points, no greater than 4.8 m wide, along the stope to create cells reasonably close to the 200-ton pour limit. Proper placement of barricades takes priority over the final pour volume.

Two incidents occurred when barricades were placed in wide areas without engineering control. One barricade had a minor blowout that was quickly controlled. The other had a major blowout of a rib anchor resulting in a near-miss, highlighting the need for proper barricade placement.

Once the barricade locations are identified, the location of the stulls are determined by identifying zones within each cell where a 4.2-m-diameter exposed span limit circle does not intersect with the hanging wall, the footwall, and a fill barricade or an adjacent stull. A stull is placed where an intersect of the 4.2-m-diameter span limit circle is needed, then the distance to the far fill barricade is annotated to assist operations in correct final placement of the stull. Figure 8 is an example of an engineering cell design.

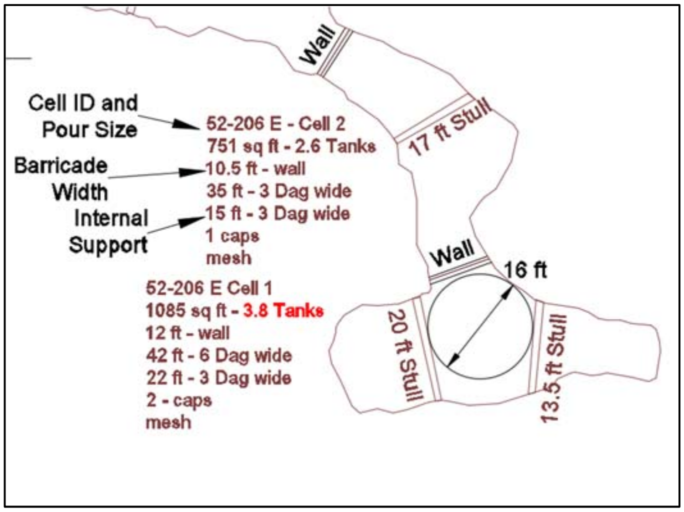

Fig. 8. Map from the mine showing a wide-span CHSB (Super-Prep) cell design with the 4.2 m span limit circle identifying the potential exposed span in Cell 1. Dimensions are in imperial units as that is the convention used by the mine.

4.3. *Extremely Wide Span Areas*

Due to geometric constrains at the shaft stations, stulls longer than 5.1 m could not be unloaded. This constraint meant that stulls and fill barricades could not be placed in areas with widths exceeding 5.0 m. While there were few sections of the veins where widths exceeded this value for any distance, some of the entry ramp intersections did exceed 5.1 m in effective span. In these areas, special designs utilized crossed cable slings anchored to adjacent nose pillars and overlain with chain-link mesh. The cable slings and mesh were laid over the prep rock. For internal support, welded wire mesh panels were stood up on edge and were tied together in a fashion similar to that used at Vale's Coleman Mine near Sudbury, Ontario (Townend and Sampson-Forsythe, 2014).

5. PERFORMANCE AND ANALYSIS

5.1. *Performance*

Careful survey and cell planning resulted in detailed maps for the Super-Prep cells, allowing for efficient preparation of a cut for backfill, even with all the additional materials required. A few issues occurred where crews had to relocate fill barricades or stulls due to inaccurate measurements from the engineering staff. Where relocation was necessary, engineering staff worked with the crews to find a suitable new location and adjust the cell design for the remainder of the cut.

Correct placement of the fill barricades also insured that cells were properly sized, which meant that backfill plant operators on surface knew to within 5%–10% how much CHSB would be required and were ready to pour each cell without interruption, even in very large intersections. Significant cost savings were realized by eliminating the practice of adding additional cement that was common in some of the narrow vein stopes where cells were laid out by stride count. More importantly, from a safety perspective, the accurate cell sizing ensured that pours did not contain cold joints from interruptions during a pour or that the cement content was too low because the backfill plant did not have the required cement available.

Crews mining below the Super-Prep commented that mining proceeded efficiently, even with the increased widths, and they usually cycled one round per day, which was normal for a mechanized UCF stope. Occasionally prep-rock would stick to the CHSB, a situation that can create a loose-ground hazard. There were very few instances where overpressure from blasting caused any damage to the Super-Prep requiring repair. Where blast damage did occur, or where a proximal seismic activity caused fracturing to the CHSB, the stulls contained the damage between them and prevented a progressive failure, thereby reducing the repair time involved. In the extremely wide span areas, the crossed cable slings and mesh in the intersections performed extremely well with no issues when crews excavated below them.

The robust nature and appearance of the Super-Preps also increased the miners' confidence about working in the wide areas. Most of the crews were accustomed to the narrow vein headings.

5.2. *Economic Analysis*

The Super-Prep, while effective, did require a notable increase in the materials and labor required for production, so an economic analysis was completed to compare the increased costs. The labor costs, material costs, and time from a recently mined, mechanized, narrow, UCF silver-copper vein were analyzed to create a base cost per ton. Two of the new Super-Prep cuts were analyzed in the same manner. The increase in cost per mined ton of the wide-vein Super-Prep compared to traditional narrow-vein UCF design varied considerably between locations. The relative increase in cost per mined ton for narrow-vein UCF and wide-vein Super-Preps over the normal OCF mining is shown in Table 3.

Table 3. Increased cost per ton of Narrow UCF and Super-Preps compared to normal overhand cut-and-fill (OCF).

Location	Increased Cost per ton of UCF over OCF
Narrow UCF Prep	10.0%
Super-Prep 1	31.5%
Super-Prep 4	13.3%

Super-Prep 1 in the table was the first wide-vein UCF cut mined and backfilled using a Super-Prep, while Super-Prep 4 was the fourth cut mined and backfilled using a Super-Prep. There was a significant reduction in overall costs for Super-Prep 4 demonstrating the learning curve for engineering and operations to effectively apply the new design method. While the increased costs are not optimal, the safety of the crews working below the CHSB combined with consistent production and the cement cost savings at the backfill plant from the detailed cell design, compensated for the cost per ton increases.

6. CONCLUSIONS

Utilizing empirical design methods, engineering control of backfill cell design, and excellent coordination between underground and surface backfill plant crews allowed for the successful implementation of a modified design and support method for widespan areas in mechanized UCF stopes.

Empirical design using data from wide areas in the historically narrow silver-copper veins was used to identify the maximum stable span of exposed CHSB at the Galena Mine. The engineering staff then developed modified CHSB support plans and cell layouts that included timber stulls to constrain the exposed unsupported CHSB span to 4.2 m on subsequent cuts. Support plans also used novel internal support methods such as welded wire mesh and cable slings in extremely wide intersections to successfully insure their stability.

To ensure proper barricade and stull placement all cell designs were completed by the engineering department. The proper location of barricades in more narrow locations also insured their stability under increased hydraulic loading due to wider than normal widths and increased wall heights. The resulting highly accurate volumetric cell designs were utilized by the backfill plant operators to allow for larger than normal pours, which helped insure stable CHSB beams free from cold joints or low-strength material while also keeping costs down from accidental overdosing of cement.

The overall design proved highly effective with few stability issues and allowed consistent production from veins with widths far exceeding the historical norm.

7. ACKNOWLEDGEMENTS

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8. DISCLAIMER

The findings and conclusions in this paper are those of the author(s) and do not necessarily represent the views of the National Institute for Occupational Safety and Health (NIOSH). Mention of any company or product does not constitute endorsement by NIOSH.

REFERENCES

1. Ageton, R.W. 1967. Stress ellipsoid determination in a Rock-burst-prone area at a 4,000-foot depth, Galena Mine, Wallace, Idaho. U.S. Dept. of the Interior, Bureau of Mines, RI 6997, 23 pp.
2. Chan, S.S.M. 1971. Deformation behavior of Revett quartzite under uniaxial and triaxial loading. In *Proceedings of the 6th Canadian Rock Mechanics Symposium, Montreal, May 1970,* Pub. Mines Branch – Dept. of Energy, Mines and Resources, Ottawa, pp. 9-31.
3. Mauk, J.L. and B.G. White. 2004. Stratigraphy of the Proterozoic Revett Formation and its control on Ag-Pb-Zn vein mineralization in the Coeur d'Alene District, Idaho. *Economic Geology*, Vol. 99, pp. 295–312.
4. Pierce, M. 2000. Cemented backfill system design for the Galena Mine. Internal Report, Itasca Consulting Group, Inc., Minneapolis, MN, August.
5. Potvin, Y., E.G. Thomas, and A.B. Fourie. 2005. Introduction to hydraulic fill. In *Handbook on Mine Fill*, Australian Centre for Geomechanics, pp. 67–78.
6. Staley, W.W. 1962. Timbering and support of underground workings for small mines. *Idaho Bureau of Mines and Geology, Bulletin No. 21,* Moscow, Idaho, p. 18.
7. Townend, S. and A. Sampson-Forsythe. 2014. Mitigation strategies for mining in high stress sill pillars at Coleman Mine – A case study. In *Proceedings of the Seventh International Conference on Deep and High Stress Mining, Sudbury, 16–18 September 2014,* eds. M. Hudyma and Y. Potvin, pp. 65–78. Western Australia: Australian Centre for Geomechanics.
8. Visnes, N. 1957. Sand filling at the Galena Mine. *Mining Engineering, January,1957.* The American Institute of Mining, Metallurgical, and Petroleum Engineers, pp. 49–52.
9. White, B.G. 1998. New tricks for an old elephant: Revising concepts of Coeur d'Alene geology. *Mining Engineering,* August, pp. 27–35.
10. Williams, T., T. Brady, D. Bayer, M. Bren, R. Pakalnis, J. Marjerison and R. Langston. 2007. Underhand cut and fill mining as practiced in the three deep hard rock mines in the United States. In *Proceedings of the 109th CIM Annual General Meeting, Montreal, Quebec, Canada, April 29–May 2, 2007,* 11 pp.