GEOTECHNOLOGY INC
FROM THE GROUND UP

December 2, 2020

Mr. David Olsson
Streets of West Pryor, LLC
7200 W 132nd Street, Suite 150
Overland Park, Kansas 66213

Re: Proposal for Underground and Construction Monitoring Services
Filling of Portions of Former Union Quarry Mine
Lee's Summit, Missouri
Geotechnology Proposal No. P035637.02

Dear Mr. Olsson:

This letter provides you with a proposed scope of work and estimated fees for consultation services regarding remediation measures for the undermined portions of the Streets West of Pryor development.

1.0 BACKGROUND

Based on the previous work performed by Geotechnology for the subject mine space, a majority of the mine area is underwater with depths of up to 10 feet and a number of dome outs are located along the major joint directions. These dome outs extend multiple rooms and evidence of propagation can be seen in the surrounding rooms.

Filling of the mine space is proposed in order to eliminate the need for maintenance and inspection of the mine space and limit the propagation of the existing dome-outs. We propose mine filling be accomplished by drilling a series of large diameter (9-inch) holes from the surface into the mine space. Backfill would be placed using the rock slinging device being developed by Drill Tech & Shoring, Inc. In order to finalize a design methodology, we recommend testing of the rock slinger.

2.0 SCOPE OF SERVICES

Phase 1

Preparation of Submittals for the City of Lee's Summit: Geotechnology will cooperate with consultants hired by the City of Lee's Summit for the preparation of documents to be completed for the City prior to the implementation of a mine filling program. Our scope will include correspondence and meetings with other consultants and preparation of technical specifications and other documents as required by the City.

Preliminary Quality Assurance/ Quality Control Plan: Geotechnology will develop a quality assurance/ quality control plan based on the available materials and calculations (attached) regarding the expected bulking factor of the materials. Documents regarding standard industry practices for blind backfilling methods are attached. The QA/QC plan will be dependent on the performance of the rock slinger as demonstrated in the test hole(s) to be performed.

Finalizing Project Survey: Surface and subsurface surveys should be referenced to the same control point, combined, and overlaid with the proposed development. Once the surveys are combined, the surface locations to be drilled should be marked with a stake.

Test Hole(s): We recommend a test hole(s) be drilled by Drill Tech & Shoring, Inc. Fill should be placed to test the efficacy of the slinger and refine the filling methodology. The underground should be observed to determine the maximum spread achieved by the rock slinger.

A letter documenting the performance of the rock slinger and, if needed, changes to the quality control plan will be provided to the City for review prior to finalization of the design methodology.

Finalization of Design Methodology: Geotechnology will work closely with Drill Tech & Shoring, Inc. to finalize the design plan for filling of the mine space. Based on the results pf the test hole(s) and the required fill height, criteria should be developed for the minimum volume of material to be use per hole and selection of a suitable backfill material. Geotechnology will not be held responsible for the mobilization or maintenance of equipment on the site.

Phase 2

Construction Monitoring: Geotechnology will provide a representative to monitor the placement of fill and collect samples to be tested in the laboratory. SCHEDULE AND FEE

Preparation of a boring plan as well as some preliminary engineering analyses can commence as soon as a survey of the surface and subsurface with the same control point can be generated.

Our services will be performed on a time and material basis. For Phase 1 and 2 we estimate a total fee of One Hundred and Ninety-Eight Thousand and Seven Hundred and Fifty-Five Dollars ($198,755). A breakdown of our rates and fees for the different tasks is provided on the attached cost estimate.

4.0 AUTHORIZATION

If this proposal, including the contractual terms, is acceptable, please sign in the space provided on the following Terms and return one executed copy of the Terms and this proposal to our office as your authorization for us to proceed.

* * * * * *

Please contact the undersigned if you have a question or comment about this proposal.

Very truly yours,

GEOTECHNOLOGY, INC.

Andrea Prince, P.G.
Senior Project Manager

ALY/ALP:aly

Attachments: Appendix 1 – Mine Filling: Quality Control Guidance
Appendix 2 - Calculations
Appendix 3 – Contract Documents
Table 1 – Estimated Mine Remediation Consultation Fees
Terms for Geotechnology's Services

Mine Filling: Quality Control Guidance

Monitoring Blind Backfilling in Abandoned Mines

Richard E. Thill, Peter J. Huck, and Bruce G. Stegman

Introduction

Backfilling of mine voids is used to prevent or control the effects of subsidence on surface structures. The pumped-slurry, blind backfilling process has been successful for stabilizing ground over abandoned and inaccessible underground coal mines.

This article describes the systematic evaluation of potential monitoring systems that would improve backfill monitoring technology. It also details field proof-of-concept tests that were conducted on two systems and the difficulties encountered in implementing them.

Ground subsidence over abandoned mines can have devastating effects on overlying urban areas decades after the closing of room-and-pillar mines. The result may be cracked foundations, ruptured gas and water lines, broken sewers, distortion or cracking in superstructures, and sinks and potholes in the ground surface.

About 32,000 km^2 (8 million acres) of land in the US has been undermined for coal. According to the US General Accounting Office, 8,100 km^2 (2 million acres) have undergone subsidence and another 8,100 km^2 (2 million acres) are expected to subside by the year 2000. Actual figures on land affected or threatened by subsidence are probably much higher, since these estimates were based on late 1960s studies.

For more than two decades, the US Bureau of Mines has been engaged in stabilizing ground in urban areas having high risk potential for subsidence costing millions of dollars. Thousands of acres have been stabilized, protecting property worth hundreds of millions of dollars. In earlier years, backfilling was by in-mine stowing in accessible mines or by gravity-feed sluicing methods in inaccessible mines.

Pumped-slurry methods have been used extensively in the past 10 years. More slurry can be injected for fill from fewer injection holes over extensive areas in abandoned mines. Since this method requires fewer boreholes, there is less disruption of surface facilities.

An investigation was conducted to determine the feasibility of using advanced, remote sensing or monitoring technology for application in monitoring fill placement. Specifically, the objective was to develop conceptual systems for blind backfill monitoring that require fewer boreholes and give better definition than is now possible.

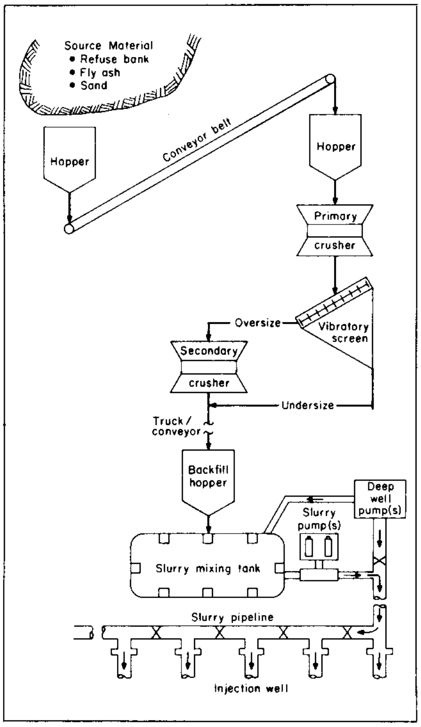


Fig. 1—Pumped-slurry backfilling operation.

Blind Backfilling Methods

Blind backfilling operations are conducted from the surface and do not require personnel and equipment underground. These methods are applicable to abandoned, inaccessible workings. Two categories of blind backfilling are point support, practiced by civil engineers to protect individual structures or surface facilities, and areal backfilling methods, used to protect large areas against subsidence.

Point Support Methods

Point support methods are usually gravity-feed systems. In general, point support methods use small volumes of expensive fill material and require a large number of boreholes within a constructed site. Because of the processed material used for backfilling and the close spacing of the injection boreholes, control and monitoring often may be accomplished by borehole cameras.

Areal Backfilling

Areal backfilling is conducted mainly by the pumped-slurry injection process (Fig. 1). In eastern and interior coalfields, material used is often mine refuse or flyash, both being undesirable on the surface and costly to dispose of in an environmentally acceptable manner. Hence, the use of

Richard E. Thill, member SME, is a group supervisor for the US Bureau of Mines, Twin Cities Research Center, 5629 Minnehaha, Minneapolis, MN 55417; **Peter J. Huck** is president of Huck Research Corp., Columbia, MD; and **Bruce G. Stegman** is a research engineer for EarthTech Research Corp., Baltimore, MD.

waste bank or preparation plant refuse materials in backfilling is attractive from an economical and environmental standpoint.

However, when these materials are unavailable near the site or may have higher value for coal content, screened sand is used.

Backfilling materials are loaded at the source and trucked to a central slurry mixing plant. If mine refuse is used, it may be scalped or crushed to about 10 mm (0.4 in.) maximum grain size, and the carbonaceous material recovered for sale. Flyash is taken from fossil powerplants or disposal sites without processing.

At the mixing plant, solids are dumped into a surge hopper and loaded by conveyor belt into a slurry mixing tank. A belt scale records the weight of solids fed across the conveyor so control of the slurry solids content may be maintained.

A mine refuse slurry contains 11-21% by weight solids, whereas a flyash slurry contains 70-75% solids and still retains excellent pumpability. Water for the slurry is provided by a submersible pump lowered into the abandoned mine at a location removed from injection holes so no injected solids will damage the pump. From the mixing plant, the slurry is pumped through surface pipelines as far as 1 km (0.5 mile) to the active injection borehole.

Injection pipes range from 150-355 mm diam (6-14 in. diam), carrying slurry at a velocity of 3-14 m/s (10-46 ft per sec). By gravity methods, only about 60 m^3 (2,120 cu ft) are sluiced through a typical borehole, but up to a few hundred thousand cubic meters of fill may be injected in a single hole by the pumped-slurry method.

The injection borehole is cased and cemented to within several feet above the mine roof so positive pressure can be exerted on the slurry.

At injection, solids entering the mine swirl beneath the injection borehole then flow radially outward. As the slurry moves into the mine spaces, the flow velocity decreases and solids settle out of the slurry to build an annular deposit surrounding the injection borehole.

The mine space immediately below the injection borehole is kept open by turbulence from the entering slurry. With continued injection, the annular deposit gradually builds up until it contacts the mine roof. When contact is established, the slurry will channel through the last opening. Solids drop from the slurry onto the outer face of the deposit.

As injection continues, the deposition front extends away from the borehole, leaving a single slurry flow channel through the deposited backfill material along the mine roof. As the flow channel lengthens, pressures within the deposit gradually increase due to head losses in the longer flow channel.

Eventually, pressures within the deposit will increase sufficiently to cause breakthrough, blowing a new passage through the top of the annular deposit in a new direction. A new flow channel begins to grow while the older channel plugs with solids.

The sequence of deposition and breakthrough gradually builds up a scalloped-shaped backfill deposit extending throughout the mine in all directions from the injection boreholes. Each breakthrough should occur at a higher pressure than the previous one, and, at some point, the breakthrough pressure will exceed a safe injection pressure limit based on pump capacity. Borehole injection is complete when the safe upper limit of pressure is reached.

The deposition-breakthrough cycle may be greatly subdued or absent with flyash or fine-grained slurry. A fine-grained, highly pumpable slurry may establish a uniform radial sheet flow across a wider annular deposit and approach the mine roof slowly, without making actual contact.

State of the Art in Backfill Monitoring

Currently, monitoring the pumped-slurry process consists of preinjection surveys of void conditions in the abondoned workings, monitoring tonnage of injected material, and using sounding lines to detect the height of fill in monitoring or injection boreholes. Intermittent monitoring of the pressure head is made at the injection hole to establish when rejection occurs.

The preinjection assessment of mine voids requires accurate mine maps and may include a borehole television camera or sonic caliper in exploration holes. A network of boreholes for line sounding indicates the extent and height of backfill directly beneath the boreholes.

Monitoring technology has several shortcomings, including obstructions in the mine, making it difficult to estimate cavity size from the boreholes. Television tools are limited or negated by murky water, and sonic caliper (echo-location) devices are limited in range by energy coupling conditions. The mines sometimes experience partial caving with reduction or migration upward of void.

Drilling is expensive and causes disruption at the surface. Some holes may not intersect voids because of ground caving or settlement, or intersection with pillars. Borehole spacing may also be difficult to maintain because of urban development on the surface. At times, the monitoring boreholes become pressurized from the injection process and risk blowout.

Given accurate mine maps, assessment of mine void, measurement of injection tonnage, knowledge of flow characteristics of the fill material, and a network of monitor boreholes for sounding, reasonable estimates can be made of the extent and effectiveness of backfill. Due to the difficulty in obtaining accurate assessments of these, remote monitoring technology is needed to supplement and improve the existing monitoring technology, especially to interpolate the geologic conditions and location of fill between monitoring boreholes.

Evaluation of Potential Monitoring Methods

The nature of backfilling, including conditions at the surface and the complexity of coal measure strata over the abandoned mines, present numerous constraints to monitoring systems.

Ideally, the backfill monitoring system should be capable of detecting the location and height of fill at any location surrounding the injection hole. The abandoned mines may be as shallow as 10 m (33 ft) below the surface or range to more than 200 m (656 ft).

Coal measure strata, sedimentary rocks that occur in cyclic association with coal seams, typically are stratified and contain numerous bedding planes and other discontinuities, and are often overlain by unconsolidated glacial till or alluvial deposits. The mine may be above or below the water table, and flooded or dry.

These characteristics present difficulties that have not been overcome for most surface and borehole geophysical methods. From a cursory review of the literature, many geophysical remote sensing techniques can be eliminated on the basis of impractibility of deployment and difficulty in interpreting results. The remaining systems that indicated some potential in backfill monitoring were evaluated using a multiple objective ranking matrix.

The ranking matrix (Fig. 2) comprises methods to be evaluated and weight objectives to be met. The product of the degree S_i, where a particular method satisfies the objective j, times the weight W_j of that objective is listed in the appropriate matrix cell. The summation of the S_iW_j products gives the total score of each candidate. Thus, the matrix becomes a system of organizing the judgmental processes, permitting complex problems to be handled, and exposing the thought and consideration that went into the process. This process often reveals that two or more candidates can be synthesized into a superior system.

	Technical objective 1	Technical objective 2	Technical objective 3	Technical objective 4	Technical objective 5	Cost objectives	Environmental impact objective j	Weighted ranking
Objective weights	W_1	W_2	W_3	W_4	W_5		W_j	Total score
Technology 1	$S_1 W_1$	$S_1 W_2$	$S_1 W_3$	$S_1 W_4$	$S_1 W_5$		$S_1 W_j$	$\Sigma (SW)_{1,j}$
Technology 2	$S_2 W_1$	$S_2 W_2$	$S_2 W_3$	$S_2 W_4$	$S_2 W_5$		$S_2 W_j$	$\Sigma (SW)_{2,j}$
Technology 3	$S_3 W_1$	$S_3 W_2$	$S_3 W_3$	$S_3 W_4$	$S_3 W_5$		$S_3 W_j$	$\Sigma (SW)_{3,j}$
Technology 4	$S_4 W_1$	$S_4 W_2$	$S_4 W_3$	$S_4 W_4$	$S_4 W_5$		$S_4 W_j$	$\Sigma (SW)_{4,j}$
— — —								
Technology i	$S_i W_1$	$S_i W_2$	$S_i W_3$	$S_i W_4$	$S_i W_5$		$S_i W_j$	$\Sigma (SW)_{i,j}$

Fig. 2—Example of partial multiple objective ranking matrix.

High Precision Local Measurements

Within the vicinity of a monitoring borehole, it is advantageous to determine whether solids have been deposited and to what depth and density. It should be noted, however, that in an optimumly-designed backfilling project, the material injected at one borehole would extend perhaps halfway to adjacent boreholes. Thus, if we identify backfill material at an adjacent borehole, we have probably placed the injection boreholes too close together, or the backfilling itself is out of control.

Borehole Samples

Samples of collected material in or beneath the monitoring borehole confirm when the backfill deposit has extended to the monitoring borehole. This helps determine the direction the slurry is flowing underground and gives information on the density of the fill.

Borehole Sounding

Boreholes adjacent to the injection borehole are sounded to determine mine pool elevation and depth of a backfill deposit beneath the borehole. The latter measurement may give false readings since backfill may surge into the borehole, or soft rock in the uncased portion may slough down.

Borehole Camera and Sonic Calipers

A borehole camera can provide excellent data from the vicinity of a monitoring borehole if mine spaces remain open and the water in a flooded mine is clear enough for vision. Its disadvantage is its expense and operating costs, including the high level of skill required.

Areal Measurements

Because it is economically and environmentally beneficial to reduce the number of monitoring boreholes, sensing systems are needed that obtain approximate information from regions remote from monitoring boreholes. It would even be of benefit to determine the general direction that the slurry is flowing from the injection borehole.

Tracers

It is conceivable that conventional ground water tracers may be used for locating fill placement. A quantity of tracer may be injected into the injection borehole before pumping begins for a particular shift. The tracer will pass rapidly through the slurry flow channel, exiting into the flooded mine spaces at the deposition front. From there it will slowly migrate through the large mine opening as additional slurry is injected behind it.

If the tracer can be identified in any of the monitoring boreholes adjacent to the injection borehole, the general direction of slurry flow at the time of injection will be known.

This system requires that the mine be flooded, and would be ineffective if the mine pool contains significant currents that would distort the flow of the tracer marked water moving away from the deposition front. It is also slow, and the cost of tracer would limit its use.

It does, however, provide one way of locating the approximate position of the deposition front, a difficult task under any circumstances.

Acoustic Emission

Acoustic emission (AE) monitoring is used in two distinct modes. During injection, an AE sensor is lowered into the mine spaces at monitoring boreholes surrounding the injection borehole to detect noise from the flowing slurry. Detection of AE in some holes, but not in others, could provide an indication of the direction of slurry flow.

AE monitoring is used before and after backfilling to determine whether subsidence activity is underway prior to backfilling and whether the process has eliminated it. In this mode, AE monitoring would be used in its conventional application of localizing areas of high structural distress in the rock mass.

It is limited to those cases where subsidence activity is intense enough to produce detectable acoustic emission. Such emission, associated with strata adjustments and fracturing, can occur well before subsidence reaches the ground surface.

Although the objective of the backfilling process is to provide firm contact between the backfill material and mine roof, time is required before the mine roof settles onto the backfill material, relieving structural distress and reducing acoustic emission.

Microgravity Surveys

Microgravity surveys have potential for detecting the gravity anomaly caused by backfill material in shallow mines. Such a survey requires sensitive gravimeter instruments and before and after backfilling surveys, with data averaged from many spatially distributed measurements.

With sensitivities of 0.02-0.05 $\mu m/s^2$, anomalies might be detected to a depth of about 20 m (66 ft) in flooded mines or about 30 m (98 ft) in dry mines, if errors associated with latitude, elevation, topography, and tides are eliminated.

By obtaining before and after differences in gravity potential at each survey station, lithologic "noise," which makes it difficult to detect gravity anomalies near the level of resolution of the instrument, might be reduced or eliminated.

Non-Geophysical Methods

Several methods not involving geophysical systems have potential for improved monitoring and have been applied in active backfilling projects. Any method that has been proven effective in the past should be incorporated into the overall monitoring scheme.

Process Monitoring

The current practice of backfilling injections measures weight of solids injected and controls the injection itself on a minute-to-minute basis by observation of pressures at the mixing plant and the injection borehole.

In the past, permanent records of injection data on pump pressure and tonnage of injected solids have been collected daily. Only in a few cases has pressure data been recorded at more frequent intervals. To prevent overlooking pressure signatures that may be indicative of important events in backfilling conditions underground, records of injection parameters should be recorded more frequently.

As a monitoring tool, process monitoring involves recording injection pressure, flow rate, and other parameters at close intervals to distinguish underground events, such as breakthrough, and to manipulate or display these data in a format that can be easily interpreted.

Production work will probably require automated data acquisition and data reduction systems.

Blind backfilling could be expected to produce characteristic pressure signatures. They display low injection pressure until the backfill material has filled the area immediately around the injection point. Since continued injection requires sufficient pressure to maintain flow channels between the top of the backfill material and the mine roof, this phase is characterized by long plateaus of low to moderate pressure.

The final stage of injection produces abrupt increases in pressure as the flow paths through the injected material become blocked. These abrupt increases in pressure are followed by gradual decreases in pressure due to the injected material being displaced radially from the high pressure. This phenomenon of rejection pressure buildup and breakthrough is related to the principles of hydraulic transport with the mine cavity.

Process monitoring is applicable to any backfill project if signature characteristics of underground events can be identified and interpreted. The cost is relatively small and its use enhanced as experience is gained.

Maps

An excellent technique used in former backfilling operations is mine map making. The procedure is simply to mark on existing mine maps the area that represents the amount of the open mine space that could be backfilled by each week of production.

Although it may not be known that the material injected in a particular week has backfilled a specific part of the mine, the general results agree with observations that are available. In one case, confirmation of the validity of mine marking was obtained by sending personnel underground through an access shaft.

Selection of Techniques for Field Tests

Sounding and sampling from monitoring boreholes, using borehole television cameras or sonic caliper, and intermittent process monitoring, have been used in previous backfilling projects and have proven themselves fieldworthy. Other techniques suggested by the ranking process, however, have never been tried in the field for backfill monitoring applications.

Acoustic emission monitoring, continuous process monitoring, and tracers were selected for evaluation and proof-of-concept testing under backfilling conditions at a site near Scranton, PA. It was apparent from the ranking process that no single monitoring technology would be sufficient to satisfy the needs of a successful backfill monitoring system. Therefore, a synthesis of successful candidates into an integrated monitoring system was necessary.

Field Trials

Field trials were scheduled for the three concepts at an active backfilling site in the Borough of Taylor, PA. The site covered about 20 square blocks and involved injection into five seams of abandoned, room-and-pillar anthracite mines.

Tracer Feasibility

Laboratory testing suggested that tracers would not be feasible. Radioactive and toxic dyes were excluded because of possible harmful effects.

Fluoroscopic dyes require laboratory analysis and would not conveniently allow for field determination of the presence of dye.

A water soluble, nontoxic dye showed promise, but laboratory studies indicated that a construction of 100 mg/L (379 mg/gal) was required to detect the dye in backfilling slurry.

Process Monitoring

Process monitoring is used to obtain characteristic injection signatures that are indicative of events occurring underground relating to fill placement. Observations at backfilling sites indicated that injected coal refuse tailings produced discrete pressure signatures associated with blockage and breakthrough of flow channels, when slurry is packed to the mine roof. These fill materials were fairly coarse-grained and scalped to less than 10 mm (0.4 in.).

Pressure signatures gradually increased in pressure, followed by a 257-690 kPa (40-100 psi) spike.

Unfortunately, about one month before process monitoring, backfill operators changed material size from coarse refuse to fine sand silt size material from another source. Thus, the behavior of the injected slurry was different than expected. Rather than the deposition-breakthrough behavior previously experienced

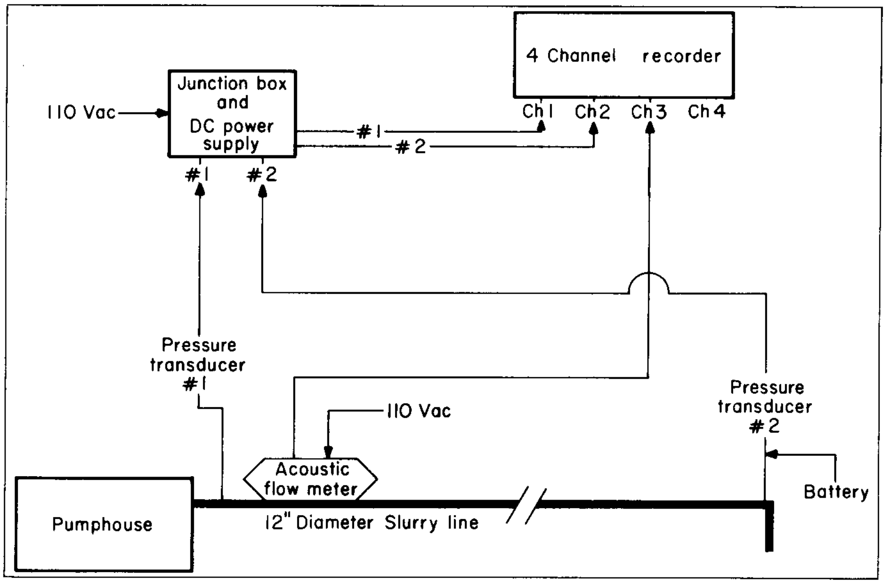


Fig. 3—Process monitoring system.

with coarse material, a uniform pressure-time history was recorded. Signatures of underground events were subtle and required intensive interpretation. This experience emphasized the importance of knowing the grain size used in the slurry.

Process monitoring instruments consisted of two pressure transducers, with a nonintrusive sonic flowmeter and a multichannel stripchart recorder (Fig. 3).

The pressure transducers continuously monitored injection pressure at the pumphouse and injection well. The flowmeter transducer was clamped into the slurry pipeline to determine velocity of the moving fluid by the Doppler effect. High ground noise and coupling problems, however, caused difficulties in obtaining reliable results from the flowmeter and it was discontinued. Since flow rates could not be determined continuously, some uncertainty exists in the analysis of pressure spikes.

Injection pressure results were interpretable only for the record of pressure-time history recorded at the pumphouse. The pressure-time history at the injection hole was complicated because of the reduction in pipe size for slurry line to casing and from turbulence as slurry exited the injection borehole into the mine opening. Most of the time negative pressures were recorded.

Although the pressure-time history at the pumphouse also was fairly complex and differed substantially from anticipated signatures, subtle signatures were detected that could be deciphered in terms of backfill conditions underground. Since nearly 3 m (10 ft) of pressure time records were collected daily over most of the 32-day injection cycle, the considerable data reduction and analysis required a computer. Examination of records revealed sequences of discrete pressure pulses ranging from 7-70 kPa (1-10 psi) for several hundred seconds.

The computer averaged the daily pressure, the sum of the area under the pressure-time curve divided by total time, the pulse rate, average pulse pressure, and the average pressure pulse duration.

Monitoring began with the injection into the New County bed, following completion of injection in the lower Clark bed (Fig. 4).

Average daily pressure decreased up to about the 23rd day, where it experienced a few major spikes and trends upward in the final days of injection. Average pulse pressure remained fairly constant through the 23rd day, experienced a sharp and sizable increase in the 24th day, returned to its former level, and began an upward trend from the 25th day to completion of the cycle.

Pulse duration was low at first, but increased steadily to a peak on about the seventh day, then followed a fluctuating trend of continually decreasing pulse duration interrupted by a few major spikes occurring several days before completion. Pulse rate showed an increasing trend in the number of pulses per hour throughout the injection period defined by the linear least-squares regression fit to the data with correlation coefficient r = 0.61.

During the final days of injection, visual inspection of the recorded pressures showed an increase in the number of pulses, agreeing with the pump operator's observation that the injection pressures "acted funny" near the end of injection.

Although the recorded pressure-time data were complex and

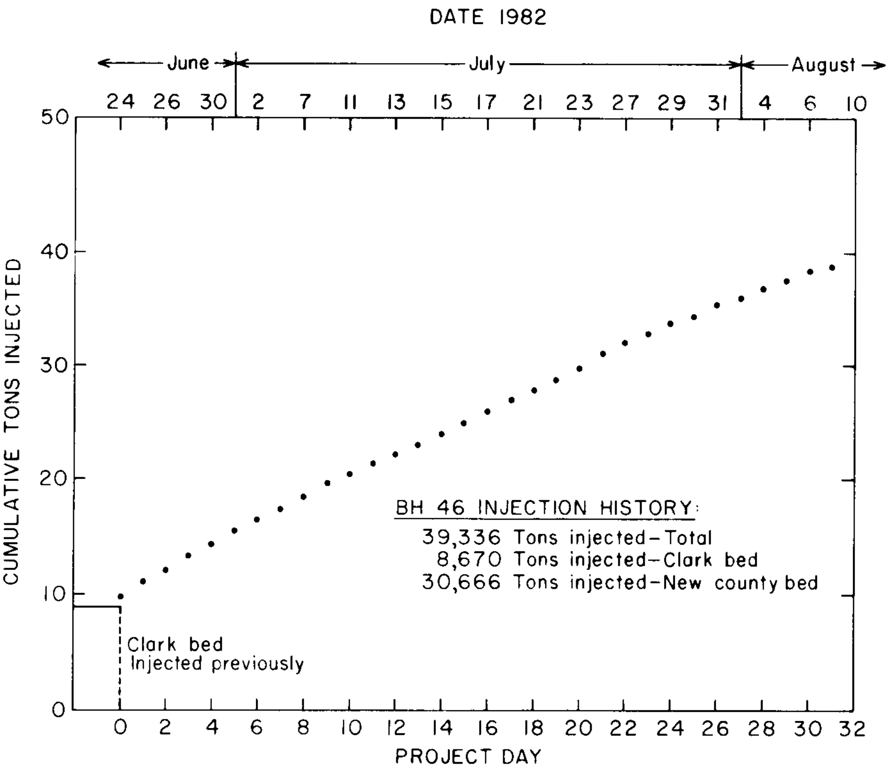


Fig. 4—Injection record for borehole No. 46.

not easily interpreted, several observations are worth noting:

- Pressure pulse rate tends to increase and pulse duration decreases throughout the injection process.
- Average daily pressure and pulse pressure exhibit a definite increase in the final stages of injection before total refusal.
- Major positive spikes in pulse duration, pulse pressure, and average daily pressure are probably associated with major blockage of flow channels in the final stages of the cycle.

It was also observed that several days before complete refusal, the daily pressures changed from a relatively uniform to a more complex signature. Encouraging was that from observations of changes occurring in the daily chart, the frequency rate of pulses and pulse pressure, blockage was predicted by field personnel a few days before the actual event.

Acoustic Emission Monitoring

Acoustic emission monitoring was done by listening at monitoring boreholes near the injection borehole for anomalously high acoustic levels from the sound of flowing slurry. Typically, the AE system would be moved several times a day.

Because of the limited field effort, it was not possible during AE monitoring to search for areas of high structural distress before backfilling and to follow-up with evaluation to see if any such areas had been quieted by the backfilling operation. Thus, one aspect of AE monitoring remains untested.

The system consisted of a hydrophone transducer for detecting acoustic noise events from a borehole location and a portable one channel, self-powered acoustic emission monitor with signal conditioning capability. Signals were amplified 200-5,000 times, and band-pass frequency set between 0.2-50 kHz. Later, the monitoring frequency band narrowed to 0.2-5 kHz.

The AE monitoring rendered mixed results. It was difficult to determine whether signals heard were the result of backfilling or were background activity. The monitoring boreholes were found to be quite noisy for the first meter below the water surface. This was probably caused by mine pool surface effects such as particles of fine sand and dripping water in the borehole.

The long-term monitoring was unable to detect an increase in acoustic activity associated with the pumping shift, as anticipated. Apparently, noise levels from the movement of the slurry, behaving more like a sludge, were not high enough to be detected over ambient noise.

In one instance, though, increased acoustic emission activity was positively correlated with the movement of slurry. The activity increased for about an hour to an intensity that saturated the amplifier, when the transducer was engulfed in the slurry. This occurred on the 27th project and is believed to be associated with a channel breakthrough.

Conclusions

Improvements are needed in backfill monitoring methods to establish where backfill is being placed and how effectively it is packed into the workings to provide surface support.

Continuous process monitoring, acoustic emission monitoring, and tracer analysis were chosen for testing. Tracers were eliminated based on cost and practicality.

Continuous process monitoring demonstrated a capability for interpreting events associated with backfill placement underground. With improvements for monitoring slurry velocity and for onsite data reduction and analysis by a small computer, this could be incorporated into future backfilling operations. It could extend capabilities in recognizing events associated with the packing of fill against the roof, channel breakthrough and plugging, and ultimate refusal.

Acoustic emission monitoring encountered high background noise in the frequency range of monitoring that did not permit detection of slurry movement, except in one case where the slurry passed directly beneath the monitoring holes a short time after the build up of emissions.

Process monitoring and acoustic emission monitoring can be expected to be more applicable and produce more distinct signatures in backfilling operations when coarser, coal refuse materials are used. In these field trials, the fine texture of fill probably caused it to behave like a sludge, perhaps with sheet flow occurring between the top of the sludge and the mine roof and sloughing at the fringes of advance.

Continued field trials of process and acoustic emissions monitoring systems at different sites under varied backfilling conditions are necessary to establish their usefulness in improving backfill monitoring.

Improvements in acoustic emission monitoring can be made by establishing a network of AE sensors in concentric rings away from the injection hole. In very shallow workings, high resolution seismic and microgravity techniques might be tested for proof- of-concept.

Continued field trials of remote backfill monitoring should point the way to major improvements for locating the movement of backfill underground and assessing the completeness and effectiveness of the backfilling operation. ■

Acknowledgments

The work reported was performed under US Bureau of Mines contract JO295061, by EarthTech Research Corp., Baltimore, MD. The cooperation of personnel at the Bureau of Mines Wilkes-Barre field office and the Bruceton Mining Research Center contributed to the success of this effort.

References

Ash, S. H., and Westfield, J., 1946. "Backfilling Problems in the Anthracite Region as it Relates to Conservation of Anthracite and Prevention of Subsidence," BuMines IC 7342, 18 pp.

Allen, A. S., 1978, "Basic Questions Concerning Coal Mine Subsidence in the United States," Bull. of the Assoc. of Engineering Geologists, Vol. 15, No. 2, pp. 147-161.

Candeub, Flessig, and Assoc., 1971, "Demonstration of a Technique for Limiting the Subsidence of Land Over Abandoned Mines," 65 pp. (and appendix). Sponsored by Urban Renewal Agency, Rock Springs, WY, and available from National Technical Information Service, Springfield, VA, PB 212708.

Carlson, E. J., 1973, "Hydraulic Model Studies for Backfilling Mine Cavities," US Bureau of Reclamation Report REC-ERC-73-19, Oct.; also Appendix, 1971, BuMines IC 8667, 32 pp.

Carlson, E. J., 1975, "Hydraulic Model Studies for Backfilling Mine Cavities," (second series of tests), US Bureau of Reclamation Report REC-ERC-75-31, March, also Appendix, 1981, BuMines IC 8846, 38 pp.

Colaizzi, G. J., Whaite, R. H., and Donner, D. L., 1981, "Pumped-Slurry Backfilling of Inaccessible Mine Workings for Subsidence Control," BuMines IC 8846, 36 pp.

Comptroller General, 1979, "Alternatives to Protect Property Owners from Damages Caused by Mine Subsidence," report by the Comptroller General of the US, US General Accounting Office Report CED-79-25, Feb., 41 pp.

Gray, R. E., Gamble, J. C., and Rogers, D. J., 1974, "State of the Art of Subsidence Control," Appalachian Regional Commission Report ARC-73-111-2550, Dec., 265 pp.

Marrolas, P., and Schechtman, M., 1981 "Coal Mine Subsidence: Proceedings from a Citizen's Conference," The Illinois South Project Inc., Herrin, IL, Sept., 45 pp.

Quan, C. K., 1979, "Overview of the Bureau of Mines Subsidence Research Program, SME-AIME Annual Meeting, New Orleans, LA, SME Preprint 79-84, 9 pp.

Ruskey, F., 1981, "Seismic and Resistivity Techniques for Locating Abandoned Mine Workings," Reprint Publication of SEG, 51st Annual Meeting, Los Angeles, CA, 21 pp.

Whaite, R. H., and Allen, A. S., 1975, "Pumped-Slurry Backfilling of Inaccessible Mine Workings for Subsidence Control," BuMines IC 8667, 83 pp.

Mine backfill design and testing

ROBERT J. MITCHELL
Professor of Civil Engineering
Queen's University, Kingston, Ontario
JOHN D. SMITH
J. D. Smith Engineering Associates Ltd.
Kingston, Ontario

ABSTRACT

A general approach to mine backfill design, including laboratory testing procedures, is outlined in this paper. Relationships between certain design parameters are developed and the effect of other factors is discussed. It is concluded that the inclusion of backfill design calculations at an early stage of mine design can often produce over-all design economies. Not only are the data needed for backfill plant and tailings disposal area design, but also the pillar recovery method and over-all mine stability are dependent on the attainable fill properties.

Introduction

Hydraulically transported and placed tailings and other granular materials have been employed by the mining industry for many years as a working platform and for ground control. A study of the available literature (and the lack of literature on some aspects) has led the authors to conclude that there is a need for an analytical approach to the general problems of backfill quantity and drainage requirements. There is also a need for standardized laboratory testing methods to determine the pertinent fill properties. The trend toward more bulk mining and the need for larger, higher bulk pours, with and without cement addition, increase the importance of early engineering design.

It is suspected that many mining companies base their backfill plant design on field experience. An adequate percolation rate appears to be the basis for design, but what is "adequate" for one design is not necessarily so for another. If it has been found from experience that a 10-cm-per-hour percolation rate in one cut-and-fill operation allows for a short cycle time, then another operation contemplating the same mining method would need the same grind, specific gravity, pouring rate, pulp density, settled porosity and plan area of pour in order for the percolation requirement to be similar. This is rarely the case. The optimum design of backfill plants, drainage systems, bulkheads and tailings disposal areas would certainly benefit from an analytic procedure that could be used for any mining method and mill waste product.

Each mine has individual backfilling problems (because of the large number of site-specific factors involved), but the general approach to backfill design is the same. Furthermore, most tailings sands are sufficiently similar in characteristics that some general relationships, on which to evaluate test programs and preliminary designs, should be considered. Tailings or backfill testing should also be standardized for the benefit of the mining industry.

The purpose of this paper is to outline the standard design calculations, propose standard testing procedures, and discuss the interrelation between backfill design and mine layout.

Backfill Quantity Calculations

The backfill plant designers need to know, from the mine planners, the quantity of fill required and the rate of filling. This establishes the weight recovery that the cyclones must achieve, their number and storage capacity, and at the same time fixes the surface disposal requirements for the discarded material.

To completely fill a unit volume of mine opening, a backfill solids volume of (1 - n) is required, where n is the average porosity of the in-place backfill; n = volume of void space in unit volume of backfill. Porosity is used (instead of density) as a basic design parameter because it is not dependent on the specific gravity of the solids.

In-place porosities are generally found to lie within the following limits: $0.42 \leq n \leq 0.48$ for hydraulic pouring and $0.35 \leq n \leq 0.42$ for densified backfill.

Ore solids volume is reduced by the extraction of concentrates (valuable minerals) and, sometimes, by the removal of waste rock (float reject) prior to milling operations. The weights of these materials and their specific gravities will be known, for a given ore zone, from assays and surveys of the ore zone. The percentage of solids volume returned as available backfill (mill tailings ≙ backfill plant feed) is given as:

$$V_T = 100 - \left(W_w \frac{G_{so}}{G_{sw}} + W_c \frac{G_{so}}{G_{sc}} \right) \quad \ldots\ldots (1)$$

where V_T = volume of tailings as a % of total ore volume
W_w = weight of float reject as a % of total ore weight
W_c = weight of concentrate as a % of total ore weight
G_{so} = specific gravity of bulk ore
G_{sc} = specific gravity of concentrate
G_{sw} = specific gravity of waste rock rejected from the ore (float reject)

Using per cent weights and volumes is recommended, as the conversions can be made directly using specific gravities without reference to units and the unit weight of water in the units system selected. It is useful to tabulate the data, as exemplified in Table 1, so that all quantities can be quickly and correctly evaluated. The specific gravity of the concentrate is evaluated as:

$$G_{sc} = W_c \ / \ \sum_{m=1}^{N} \frac{W\%\,[M]}{G_{s[M]}} \quad \ldots\ldots (2)$$

where m, from 1 to N, represents the valuable minerals
and $G_{s[M]}$ = individual specific gravities of each mineral
$W\%[M]$ = percentage of the concentrate represented by each mineral

The specific gravity of the tailings is evaluated as:

$$G_{ST} = W_T \Big/ \left(\frac{100}{G_{so}} - \frac{W_w}{G_{sw}} + \frac{W_c}{G_{sc}} \right) \quad (3)$$

The required recovery of mill tailings as classified backfill is calculated as:

$$R\% = \frac{(1\text{-}n)100}{V_T\%} \times 100\% \quad (4)$$

To avoid slimes problems and promote acceptable drainage of hydraulic backfills, it is generally necessary that $R < 70\%$. When the calculated value of R from Equation (4) is considered too high, three alternatives are available:
(1) obtain other tailings or outside borrow material that can be added to the classified hydraulic tailings;
(2) use waste rock to fill portions of the mine openings;
(3) provide the stability requirements by selective partial filling of mine openings.

In terms of weights, it would require

$$W_B = \frac{W_T\% \times R\%}{10^4} \text{ kN (or tons) of dry tailings}$$

to fill the void created by the removal of 1 kN (or ton) of ore. Makeup requirements, when necessary, are normally calculated as equivalent weights where W_M, the weight of makeup per unit weight of mined ore is:

$$W_M = W_B - \frac{W_T\% \times R_A\%}{10^4} \; \frac{G_{SM}\,(1\text{-}n_M)}{G_{ST}\,(1\text{-}n)} \quad (5)$$

where R_A = % recovery available as classified backfill
G_{SM} = specific gravity of makeup material
n_M = porosity of makeup material

Other symbols as previously defined.

In making these calculations, it must be realized that the mixing of two materials of differing grain size distribution can drastically alter the overall backfill porosity. For example, mixing waste rock with hydraulic tailings would result in an over-all porosity decrease, because the hydraulic tailings would fill the large voids in the waste rock, reducing the effective porosity of the waste rock to zero. Of special note is the observation that hydraulic tailings will not generally intrude deeply into waste rock dumped, before hydraulic filling, into the base of stopes to block off drawpoints and provide an underdrain for seepage waters.

The assumed porosity [Equation (4)] used to make preliminary calculations (0.45 is recommended for open stopes and 0.42 for cut-and-fill operations) should be checked by laboratory testing before finalizing backfill plant requirements.

The above-noted procedure assumes that the backfill plant can be completed prior to the need for backfill underground and that individual openings are backfilled immediately after completion of mining. Rapid backfilling of production openings must be considered, in the mining method and scheduling studies, to ensure the optimum use of mine waste and to provide the often-needed mine stability. Reclamation of backfill from a tailings pond is generally not economical.

Backfill Drainage Requirement

The rate of backfill drainage is of considerable importance. The need for short delays between backfilling and production cycles is obvious in mechanized cut-and-fill stopes. Bulk pours in open stopes require proper bulkhead design and a decision on the need for decant towers. The use of cement complicates the analysis considerably. Blasting safely near or adjacent to bulk pours requires a knowledge of their degree of saturation and, hence, production schedules can be affected.

Hydraulic pouring produces a saturated, settled backfill with an excess layer of free water. The amount of excess water depends on the pulp density of the slurry (PD = wt. of solids per unit weight of slurry material delivered), the settled porosity, n, and the specific gravity of the tailings, G_{ST}. This quantity is expressed as:

TABLE 1. Backfill weight-volume relations

Material	% By Weight	Specific Gravity	% By Volume of Solids (No Voids)
ORE	100%	G_{so}	100%
SCALPED WASTE (e.g. DMS FLOAT)	W_w	G_{sw}	$V_w = \frac{W_w G_{so}}{G_{sw}}$
CONCENTRATE	W_c	G_{sc}	$V_c = \frac{W_c G_{so}}{G_{sc}}$
TAILINGS	W_T	G_{ST}	$V_T = \frac{W_T G_{so}}{G_{ST}}$
SUMMATIONS	100%		100%

$$\frac{H_w}{H_F} = \frac{V_w}{V_F} = \frac{1 - PD}{PD}\, G_{ST}\,(1 - n) - n \quad (6)$$

where H_w, V_w = height and volume of excess water
H_F, V_F = height and volume of settled backfill

If the backfill quantity is specified in terms of W_s = weight of solids per hour (kN/hr) and the pour area is A m^2, the linear filling rate is given as:

$$\text{Filling rate} = \frac{W_s}{G_{ST}\,\gamma_w\,(1\text{-}n)A} \text{ metres/hour} \quad (7)$$

where γ_w = unit weight of water = 9.81 kN/m^3.

In order that the excess water drain through the fill under the gravitational gradient of unity, thus avoiding decant systems, the percolation rate (P) must be equal to or greater than the product of Equations (6) and (7):

$$P \geq \frac{W_s\left[\left(\frac{1\text{-}PD}{PD}\right) G_{ST}\,(1\text{-}n) - n\right]}{G_{ST}\,\gamma_w\,(1\text{-}n)A} \text{ metres/hour} \quad (8)$$

It is apparent, from Equation (8), that the pulp density has a significant effect on the percolation requirement and the pulp density should be maintained as high as practicable in the backfill slurry. Equation (8) yields, for a given pulp density, an hyperbolic relationship between P and A. Large open stopes require only very low percolation rates (typically $P \geq 1.0$ cm/hr for $A = 3000\ m^2$), whereas very high percolation rates are needed for small pour areas ($P \geq 50$ cm/hr for $A = 60\ m^2$ is typical). When A is not a constant (e.g. tapered stopes), it is generally sufficiently accurate to use the average stope area in Equations (7) and (8) to evaluate the required percolation rate. Small stopes can sometimes be backfilled in pairs to increase the pour area and decrease the percolation requirement.

Percolation, Grain Size and Porosity Relations

Hydraulic tailings backfills are generally poorly graded (uniform grain size) and would, therefore, be expected to follow Hazen's formula (Hazen 1892) relating the effective grain size D_{10} (the mean grain diameter that 10% of the material, by weight, is finer than) to the soil permeability. In terms of percolation rate, the formula indicates that

$$P \doteqdot 5000\, D_{10}^2 \text{ cm/hr} \quad (9)$$

for loose tailings, where D_{10} is the effective grain size in mm.

It is further noted, in most soil mechanics texts, that the coefficient of permeability (or P) should be found to decrease exponentially as the void ratio (or porosity) decreases. Thus, a relationship should exist in the form

$$P = P_o / e^{\left(\frac{n_o - n}{\lambda}\right)} \quad \ldots\ldots (10)$$

where e is the Naperian logarith base = 2.718
and P is the percolation rate at porosity, n

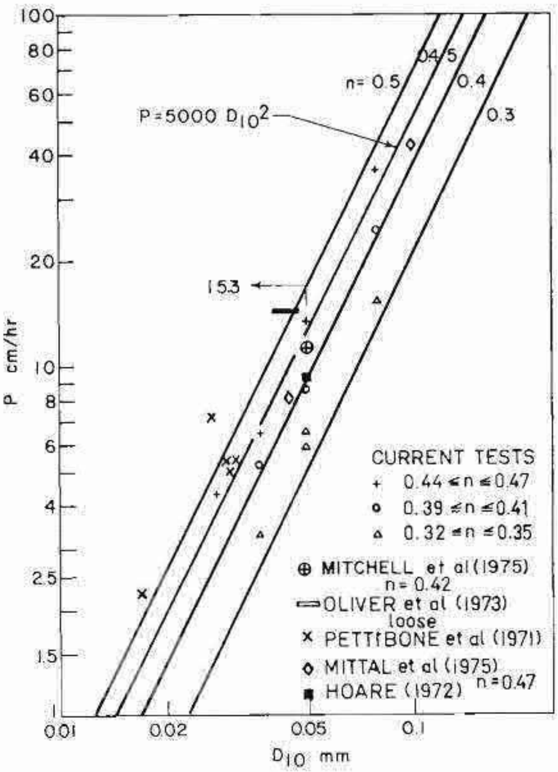


FIGURE 1. Percolation test data.

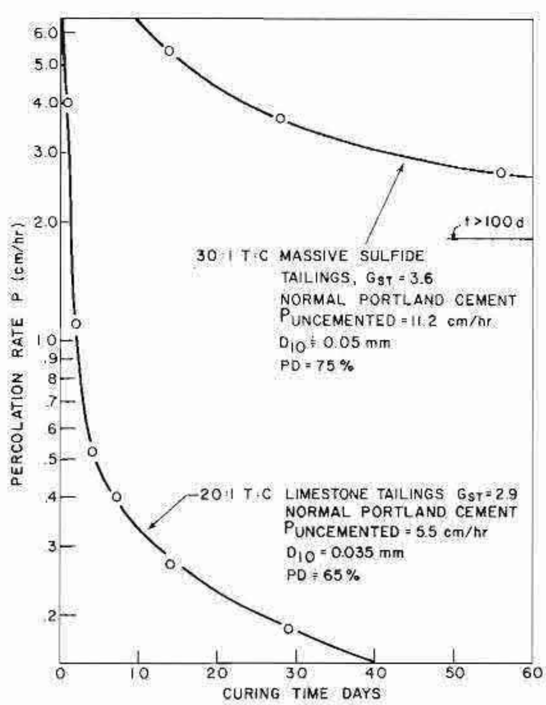


FIGURE 2. Percolation in cemented tailings.

P_o is the percolation rate at $n = n_o$
λ is obtained from experimental data
The values of P_o and n_o may depend somewhat on the grain shape.

Figure 1 shows test data relating D_{10}, P and porosity, n. The percolation rate, P, has been obtained, in most cases, under standard test conditions. These data clearly show that percolation rate does decrease in the expected form with decreasing D_{10} and decreases with decreasing porosity. Using the limited data at $D_{10} = 0.5$ mm, the average value of λ in Equation (10) is found to be 0.16. From these results, an empirical relation can be established as:

$$P \doteq 5000\, D_{10}^2 / e^{\left(\frac{0.45 - n}{0.16}\right)} \text{ cm/hr} \quad \ldots\ldots (11)$$

Knowing the form and having typical values for this relation allows: (1) preliminary calculations to be carried out prior to laboratory testing; (2) planning and correlation of laboratory tests to reduce the over-all testing requirements; and (3) equations to be established for solving complex problems. For example, combining Equations (8) and (11) gives a relationship between stope (or pour) area and D_{10} and, because D_{10} can be related to recovery, R%, a relationship can be established between pour area and the recovery required to give free drainage. For complex geometries (when area and volumes of subsequent pours are neither constant nor in constant ratio), these relationships can be used in planning mining and pouring sequences.

Effects of Cement on Porosity and Drainage

The addition of small quantities of cement to classified hydraulic backfill will not alter the initial porosity significantly. Cementation will, however, decrease the percolation rate due to the formation of cement gel in the void space. The data on Figure 2 show typical limits of this effect for two backfills with similar uncemented percolation rates. In one case, a fivefold decrease was observed after 100 days; in the other case, a tenfold decrease was observed within about 4 days. In the first case, bulk cemented pours were free draining; in the second case, a decant system provided the only way to remove excess water. From a review of available literature, (for example, Thomas, 1976; Weaver and Luka, 1970), it is clear that:
(1) increased cement content decreases the percolation rate;
(2) for a given cement content, the precentage decrease in percolation rate is greater for finer materials;
(3) for a given cement content, the decrease in percolation rate also appears to be dependent on the type of tailings and pulp density of the pour;
(4) in most cases where slimes (minus-0.02-mm material) are included in the classified tailings, decant systems will be required for cemented backfills.

Attention is required to ensure that water ponding is minimized, as ponding will promote cement segregation and reduce the effectiveness of the cement.

Effects of Slimes on Porosity and Drainage

When slimes (minus-0.02-mm sizes) are included in large hydraulic pours there is bound to be segregation in the backfill. Partly due to this segregation and partly due to the natural bulking effect in finer particles, model pours have shown that the initial pour porosity increases (density decreases) once the slimes content exceeds about 15%. Model pours also show that segregated layers of slimes are softer (more compressible and less strong) than the coarser layers, even though the cement content was fairly uniformly distributed by weight. Weak layers are, of course, dangerous to the stability of exposed backfills. However, if a cemented backfill, containing slimes, can be prevented from segregation (by using distribution boxes in the stope, for example), the strength will be higher than an equivalent cemented fill without slimes. In all cases where the primary backfill purpose is to resist deformations (closure and subsidence), slimes should be eliminated and D_{10} should be greater than 0.02 mm. Except in unusual situations, where special control of pouring operations is maintained, the inclusion of fines (slimes, flyash, imported clays, etc.) to improve the strength properties (hence, reduce cement requirements) is not recommended.

Traditionally, slimes elimination has been practised to improve drainage and to reduce slimes problems in decant systems. This practice is supported, in model tests, for improving the over-all performance characteristics (stability and subsidence resistance) of hydraulic backfills. Various commercial additives will assist in preventing segregation (by flocculation of slimes), but these have been found to cause excessive bulking of the backfill, resulting in a softer, more compressible, backfill. These considerations are further discussed by Aitchison *et al.* (1973).

Laboratory Testing of Hydraulic Backfills

Classified tailings are man-made soils and are similar, in physical properties and behavioural characteristics, to natural, uniform fine to medium sands, silty sands and naturally cemented silts (loess). As such, they may be subjected to standard soil mechanics tests. Mine backfilling is, however, a specialized use of soil and, therefore, the details of test procedures and data analyses differ from those applicable to classical soil mechanics problems. Specialized equipment and procedures are warranted in order to provide the required information in an efficient manner. This section provides basic recommendations for backfill testing in an effort to promote standardization in the mining industry as an aid to correlation of site-specific data.

Grain-Size Analyses

Mill tailings grain sizes usually lie between the medium sand fraction ($\leq$ 0.6 mm) and the fine silt fraction ($\geq$ 0.002 mm). The size distribution of the mill tailings represents the backfill plant feed and may be found by wet or dry sieving down to 200 sieve (0.076 mm) or 400 sieve (0.037 mm), followed by hydrometer or cyclosizer analysis of the material passing the finest sieve used. Standard procedures are already available for these tests.

An estimate of the effective size, D_{10} (the diameter in mm that 10% of the material is finer than, by weight), of any proposed recovery (R = per cent classified) can be obtained from the feed grain size distribution by the method shown on Figure 3, where R = 65% yields D_{10} = 0.031 mm. The method assumes a 10% overlap in separation compared to a sieve cut and has proven fairly reliable for predicting the D_{10} value for efficient cyclone separation. Using this method and the empirical relation between D_{10} and percolation rate (see previous section), preliminary drainage calculations can be carried out to provide basic data for backfill plant design. The experimental relation between recovery and percolation must, of course, be obtained from laboratory testing before the backfill design is finalized.

Percolation Tests

Classified tailings for percolation testing should be produced (for a range of recoveries bounding the required recovery, calculated as outlined earlier) by cyclone separation – sieve cutting and other size separation techniques will not produce the same grading as cyclone separation.

Because percolation rate is not a fundamental property, it must be obtained from a test that models the ideal prototype situation – that is, minimal ponding of water on the backfill surface. Thus, the correct percolation rate is equal to the coefficient of permeability and is given as the quantity of water that will flow through unit surface area in unit time under an hydraulic gradient of unity. A standard percolation test should be arranged such that the depth of water overlying the sample is less than 5% of the sample length. A cross-sectional drawing

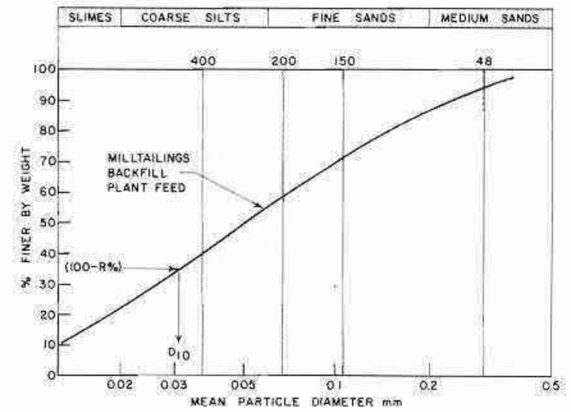


FIGURE 3. Estimation of D_{10} from plant feed recovery.

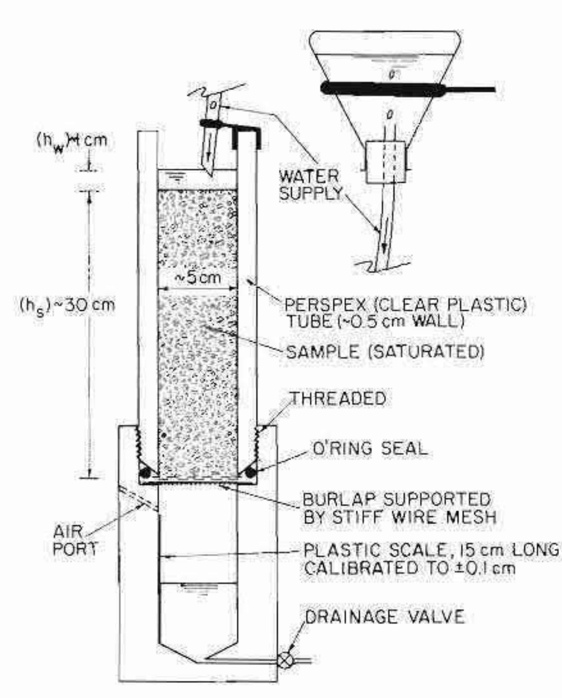


FIGURE 4. Percolation test arrangement.

of the recommended standard test arrangement is shown on Figure 4. A 30-cm sample length is recommended to attain the desired accuracy and a 5-cm tube diameter is recommended to reduce boundary effects (with loose uniform-sized silts and sands larger voids tend to form at the soil-container interface and tend to increase the observed percolation rate in smaller tubes; 5-cm diameter is usually sufficient to negate this effect). Double-thickness burlap is recommended as the drainage filter because other filter cloths, filter papers and porous stones easily become clogged (blind) and yield erroneously low percolation results. The burlap should be supported by a stiff metal screen to maintain a planar sample boundary. Simple visual observation of the height of water in a collecting container is sufficiently accurate and avoids possible error in weighing the quantities of water collected. The following test procedure is recommended.

(1) Collect cyclone sample at the desired pulp density, keeping solids in suspension by agitation or stirring.

(2) Pour mixture into percolation tube using a funnel and

spoon until desired sample length is obtained. Do grain-size distribution and specific gravity test (if required) on excess.
(3) Position water supply tube and wait for sample to settle. Excess overlying water can be siphoned off, if necessary.
(4) Examine sample for any segregation and measure sample mean height. Adjust water supply tube such that $h_w \leq 0.05\ h_s$.
(5) Take periodic readings for a period of not less than 2 hours.
(6) Strike sample tube several blows with a rubber mallet to densify sample, and repeat (4) and (5).
(7) Strike sample tube repeatedly all around with a rubber mallet, and repeat (4) and (5).
(8) Remove water supply tube and wait until surface water layer has percolated into the sample. Pour entire sample into a tare dish and weigh. Oven dry sample for 24 hrs at 105°C and weigh dry solids. Do grain-size analysis on sample.

Note: For cemented backfills the cement should be mixed into the sample in Step (1) above, the readings of Step (5) should be continued for several days, and Steps (6) and (7) would generally be omitted. An identical sample would be used for grain size and specific gravity tests.

The data analysis would provide the following results.
(1) From the grain size, D_{10} could be obtained and the cyclone efficiency would be evaluated with reference to the feed distribution.
(2) By plotting height of water in collecting container vs time, the percolation rate is evaluated, at any stage of the test, as the slope of this relation.
(3) Dry unit weight and porosity are calculated as:

$$\text{dry unit weight } \gamma_d = W_d / h_s A$$

$$\text{porosity } n = 1 - \gamma_d / G_{ST}\ \gamma_w$$

where W_d = dry weight of solids in sample
h_s = height of sample (three values)
A = cross-sectional area of tube
G_{ST} = specific gravity of solids
γ_w = unit weight of water

(4) Assuming that the sample was saturated when removed from the tube, the specific gravity, G_{ST}, may be checked using the relationship:

$$G_{ST} = n \,/\, (1 - n) \left(\frac{W}{W_d} - 1 \right)$$

where W = total weight of sample after removal from tube; other symbols as above.

The recommended test procedure and calculations provide data to establish the relation between porosity and percolation rate. Typical results are shown on Figure 1. Expected variations of percolation rate with depth and with variations in backfill density can then be evaluated.

Non-standard percolation tests can be standardized (corrected) by the following relation:

$$P_{(standardized)} = P_{(measured)} \times \frac{h_s}{h_s + h_w} = P_{(corrected)}$$

It is further recommended that the standardized percolation rate be reduced by 5% for a tube diameter of 3.8 cm and by 25% for a tube diameter of 2.5 cm. Tubes smaller than 2.5 cm I.D. should not be used. Finally, if the temperature of the excess water underground will differ by more than 5% C from the laboratory test temperature, the percolation rate should be corrected as

$$P_{(underground)} = P_{(laboratory)} \times \frac{\mu_{underground}}{\mu_{laboratory}}$$

where $\mu_{underground}$ = viscosity of water at underground temperature
$\mu_{laboratory}$ = viscosity of water at laboratory temperature

Strength Tests

Cemented tailings specimens are generally prepared by casting in a mold and curing in a humid room. In the absence of more elaborate facilities, mixing can be carried out using a laboratory paddle mixer and the mixture (at the correct pulp density and T:C ratio) can be poured or spooned (via funnel if desired) into molds to form test specimens. Cured specimens can be easily extruded from plastic or fibreglass molds providing the mold is coated, inside, with a thin film of vacuum grease (silicon grease) prior to pouring the specimen. A standard recommended cylindrical specimen size for strength testing is 20 cm^2 area (÷ 5 cm diam.) by 10 cm length.

Standard soil mechanics triaxial equipment is suitable for both confined and unconfined strength tests on preformed specimens. This data is generally required to calculate the stability of backfill faces exposed by pillar recovery operations. Standard circular arc and wedge stability calculations are normally employed, although some special calculations are often warranted (e.g. stability of benched ore blocks, etc.).

Although often overlooked, stress-strain properties obtained from triaxial testing should be considered in all stability analyses. Cementation increases the 'brittleness' of backfills, making them more prone to cracking under blast loadings and to rupture under local rock deformation. Thus, forces, displacements and energy dissipation should be considered in backfill design. In many cases, cement contents should be kept below 5% (20:1 T:C) so that the exposed face can yield without rupturing.

Uncemented fills cannot be designed to remain stable at slope angles greater than the natural angle of repose of the tailings and, therefore, require lateral support in underground operations. Remnant pillars are often used to provide this lateral support. Shear box tests are recommended for obtaining the friction angle, ϕ', required in order to calculate the lateral fill pressure and design the remnant pillars. The parameter ϕ' will increase as the porosity decreases (density increases), and samples should be prepared at various densities (as for percolation tests) to obtain the relevant range of values for design.

Compressibility Testing (Load-Density Relations)

Hydraulic backfill will compress non-linearly under load. Typical relationships for a classified backfill (cemented and uncemented) in a rigidly confined compression test (standard soil mechanics oedometer test) are shown on Figure 5. Figure 6 shows the same data in a semi-log space.

As an hydraulic fill is placed with free gravity drainage, the effective vertical stress at any point in the fill is $\sigma'_v = Z\gamma' + iZ\gamma_w = Z\gamma$ (as i = unity), where γ is the total unit weight of the material and Z is the depth of the point

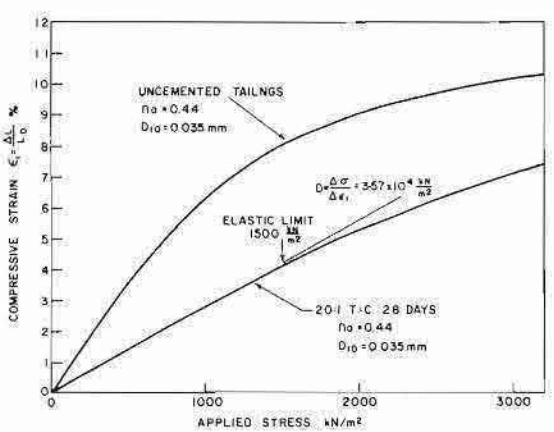


FIGURE 5. Compressibility test results.

below the fill surface. The effect of self-weight consolidation (increase in density with depth) in the backfill can be considered with respect to the data in Figure 6. For example, at a depth of 20 metres in a backfill with γ = 20 kN/m^3, σ_v' = 400 kN/m^2 and, from the initial pour porosity of 0.45 (as obtained from laboratory percolation tests without densification), the porosity at 20 metres depth would be reduced to about 0.43 by self-weight consolidation. The equivalent porosity, on Figure 6, is calculated from the relation $\Delta_n = \Delta\epsilon_1 (1 - n)$. If the same fill was cemented at 20:1 T:C, the porosity would only be reduced to 0.445 under self-weight consolidation. The rate of cement strength gain in cemented fills generally exceeds the rate of stress increase due to filling such that self-weight consolidation does not occur to the same degree as it does in uncemented backfills.

Up to 1500 kN/m^2, the cemented fill behaves nearly linearly with a confined modulus $D = \Delta\sigma_1/\Delta\epsilon_1 = 3.57 \times 10^4$ kN/m^2 (note that $\Delta\epsilon_1 = \Delta L/L_o = \Delta V/V_o$ in this constant-area test). This modulus should be readily related to Young's Modulus, $E = \Delta\sigma_1/\Delta\epsilon_1$, in the triaxial compression tests by the formula

$$D = E\,(1 - \nu)/(1 + \nu)(1 - 2\nu)$$

where $\nu = -\epsilon_3/\epsilon_1$ is the Poisson's ratio in elastic theory. Just as there is a limit to the cement strength in unconfined tests, there is a limit in the oedometer test. Using elastic theory and a maximum distortional strain energy failure criterion, the following relation can be developed:

$$(\sigma_1')_f \ \underset{\text{Test}}{\text{Oedometer}} = (\sigma_1')_f \ \underset{\text{Test}}{\text{Unconfined}} \frac{(1 - \nu)}{(1 - 2\nu)}$$

For a typical range of $0.35 \leq \nu \leq 0.45$, this gives $(\sigma_1')_f$ in the oedometer to be 2.2 to 5.5 times the unconfined compressive strength of an elastic brittle material. The data on Figure 5 show that the cement bonding yields at about 1500 kN/m^2 and the unconfined compressive strength of this mix was found to be about 350 kN/m^2. This would indicate that elastic theory could be used to predict the behaviour of the backfill up to these stress levels.

At high stress levels, the compression of the cemented fill is larger than that of the uncemented fill (on Figure 6 the compression index is noted to be greater for the cemented backfill for $\sigma_v' \geq 2000$ kN/m^2). Indeed, the total compression under large loads will be about the same in uncemented and cemented backfills: for example, using Figure 6 and a backfill of 20-metre depth, the average initial porosity, if uncemented, would be 0.44 and, if loaded to 6000 kN/m^2 (900 psi), $\Delta\epsilon_1$ = 11.6% (ΔH = 2.30 metres); if cemented, the initial porosity is 0.45 and the maximum settlement under 6000 kN/m^2 final stress is $\Delta H = (0.116 - 0.006)\ 20 = 2.2$ metres.

The above relationships may explain why there is no general agreement in the literature as to the benefits of cementing backfills to provide increased resistance to closure or subsidence. Cementation does increase fill support capabilities up to some limit of stress increase, but this limit is sufficiently low (for the usual economical mixes) to be of little practical significance. Larger benefits can be gained, in terms of subsidence or closure resistance, by densification of backfills. In the above case, the maximum settlement under 6000 kN/m^2 final stress could be reduced below 1 metre if the initial porosity could be reduced, by densification, to n_o = 0.40. Vibratory densification is discussed by Nicholson and Wayment (1967).

As most soil mechanics oedometers are not designed for poured samples and also use dead weights for loading, an apparatus has been designed for backfill compression testing and is sketched on Figure 7. Compression results under high stresses are required to evaluate the closure and subsidence resistance of backfills.

The hydraulic loading arrangement shown in Figure 7 can also be used, with a smaller Bellofram, for unconfined testing of formed specimens.

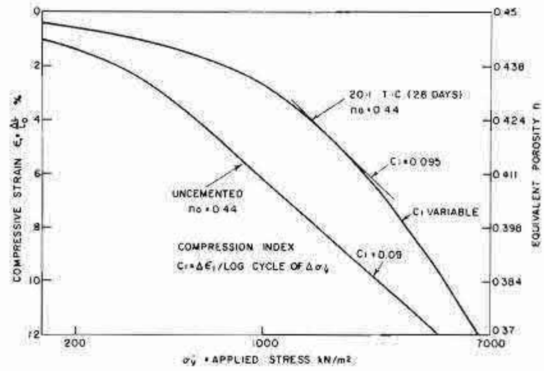


FIGURE 6. Compressibility vs logarithm of pressure.

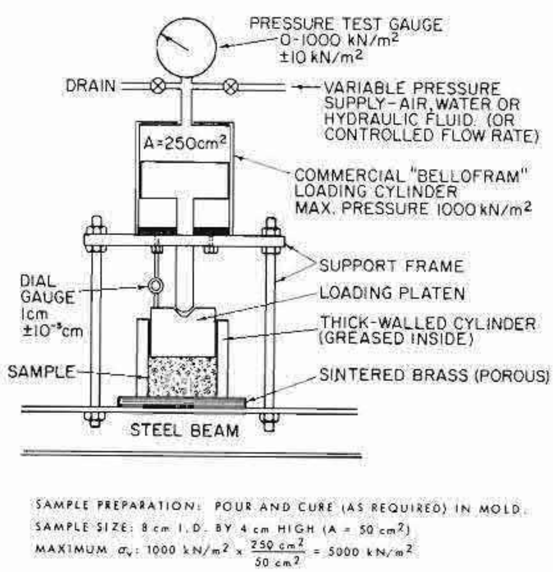


FIGURE 7. Schematic of compression test equipment.

Other Special Tests

Because soils are non-linear and inelastic, it is not generally possible to relate the results of various types of tests in terms of simple moduli. In certain cases, specialized testing may be required. For example, when backfill is required to resist rock distortion (pillar rotation), simple shear test results would be beneficial. For hydraulic backfills of limited pour thickness (e.g. cut-and-fill), air entry tests and moisture retention testing would provide the basic data needed to estimate the internal drainage conditions and the delay time for surface stability. These tests require specialized equipment, although air entry (up to about 0.7 atmosphere) can be measured using the percolation test apparatus shown in Figure 4.

Uncemented backfills must be reasonably well drained or are susceptible to blast liquefaction — an instantaneous loss of strength and stability under vibratory densification. To ensure that this condition cannot develop, it is necessary to ensure that there is insufficient water retained in the fill to saturate it at maximum vibratory density. Relative density tests and moisture-content sampling of the drained backfill pours, prior to adjacent blasting, are recommended in order to avoid possible catastrophic failures due to liquefaction.

Effects of Backfill Design On Mine Layout and Method

The most economical mining method, for a given orebody, is established from a consideration of factors such as: orebody size, shape and orientation; surrounding rock quality; ore grade distribution; and required recovery and production rates (among others). Both primary and secondary mining methods and the sequence of mining must be considered in order to optimize recovery of the orebody. The choice of secondary mining method depends to a large degree on the type and quality of the backfill placed in the primary stopes. This material must perform as planned or ore recovery will be reduced and/or dilution of grade will result. There is a trend toward large bulk mining methods utilizing specialized mechanized equipment. For high-grade orebodies where very high recoveries are necessary, this generally means that primary stope openings must be backfilled with cemented or otherwise stabilized tailings. As noted previously, the backfill strength required to withstand the various applied forces will vary in direct proportion to the exposed height. For lower-grade orebodies or where cementing agents are shown to be uneconomic, the use of remnant rib pillars for fill support and/or post pillars for ground control generally eliminates the need for cementing backfill. In most cases, secondary backfill need not be cemented, and this introduces a significant economic factor.

From a structural approach, and assuming primary mining can be carried out more efficiently than pillar removal, pillar widths should be the minimal required for support and primary stopes should be as large as possible without creating excessive back problems. Modifications to the basic structural approach are derived from considerations of continuous constant production (required for efficient mill operations) and backfilling. At this design stage, preliminary estimates of the classified tailings requirements and properties are necessary for economic analyses, which should include fixed, operating and design costs.

Large primary stope volumes involve large cement costs; if free gravity drainage can be assured, however, the savings in decant costs (and problems) may balance the extra cement costs. If decant systems are necessary for all plausible primary stoping areas, then primary area (and volume) reduction leads to savings in both cement and decant costs. In extreme cases, optimization may result in secondary mining volumes being larger than primary mining volumes. The alternatives for pillar removal (particularly exposed heights of fill, pillar sizes and support requirements) will influence the primary cost analyses and there is, obviously, considerable scope for optimization of the over-all costs. If possible, secondary uncemented backfill pours should be large enough in area to be free draining.

In all cases, it is recommended that a preliminary economic analysis of the more obvious inter-relations between mining and backfilling should be carried out when mining layouts and mining sequences are being considered. An example would be the case where satisfactory recovery is insufficient, without makeup, to fill all openings – by appropriate mine sequencing, filling of the areas most requiring backfill support could result in an economical and satisfactory solution.

A proposed design flow chart is shown on Figure 8.

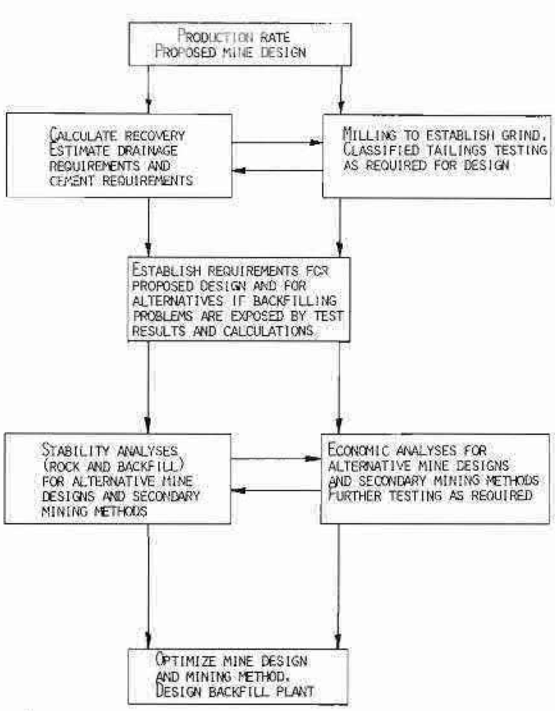


FIGURE 8. Proposed design flow chart for mine backfill design.

Summary and Conclusions

This paper outlines the calculations and general testing requirements for hydraulic backfilling of mines with mill tailings, presents some relationships between the basic parameters and discusses, briefly, the inter-relation of backfill requirements, cement usage, mine layout and mining method.

1. General equations for calculating backfill quantity and percolation requirements as well as relationships between these requirements can, and should, be established, with limited tailings testing, at an early stage of operations planning.
2. Cementation of backfill is primarily useful only if significant heights of backfill are to be exposed at a later mining stage. Uncemented backfills provide comparable confined support characteristics.
3. Percolation testing should be standardized and the percolation-vs-density relation should be established during tests. Specific test recommendations are included in this paper.

It is believed that considerable over-all economies in underground mine operations can be realized if the mining and backfilling alternatives are considered at an early stage of planning. Backfill plant design, primary and secondary mining method selection, and general mine sequencing cannot be rationalized without a knowledge of the backfill requirements and engineering properties.

REFERENCES

AITCHISON, G. D., KURZEME, M., and WILLOUGHBY, D.R. (1973). Geomechanics Considerations in Optimising the Use of Mine Fill. *Proc. Jubilee Symposium on Mine Filling, Australia.*

HOARE, B. (1972). The Disposal of Mine Tailings Materials. *Ph.D. thesis, University of Waterloo, Canada.*

HAZEN, A. (1892). Physical Properties of Sands and Gravels with Reference to their Use in Filtration. *Rept. Mass. State Board of Health, p. 539.*

MITCHELL, R. J., SMITH, J. D., and LIBBY, D. J., Bulkhead Pressures due to Cemented Hydraulic Mine Backfills. *Can. Geot. J., 12:3, pp. 362-371.*

MITTAL, H. K., and MORGENSTERN, N. R. (1975). Parameters for the Design of Tailings Dams. *Can. Geot. J., 12:2, pp. 235-261.*

NICHOLSON, D. E., and WAYMENT, W. R. (1967). Vibratory Compaction of Mine Hydraulic Backfill. *U.S.B.M. RI 6922.*

OLIVER, V. H. R., and RUSSELL, F. M. (1973). The Mufulira Sandplant. *Proc. Jubilee Symposium on Mine Filling, Australia.*

PETTIBONE, H. D., and KEALY, C. D. (1971). Engineering Properties of Mine Tailings. *A.S.C.E., Vol. 97, SM9, pp. 1207-1225.*

THOMAS, E. G. (1976). Fill Technology in Underground Metalliferous Mines. *Australian Mineral Foundation Inc., course notes, October, 1976.*

WEAVER, W. S., and LUKA, R. (1970). Laboratory Studies of Cement-Stabilized Mine Tailings. *CIM Bulletin, Sept. 1970.*

Keywords: Underground mining, Backfill design, Tailings, Mine layout, Drainage, Cement, Slimes, Percolation tests, Strength tests, Compressibility, Hydraulic backfilling.

Robert J. Mitchell

Robert J. Mitchell is a graduate of the University of Waterloo and obtained his Ph.D. from Cambridge University as an Athlone Fellow and N.R.C. special scholar. He has worked in the mining industry as well as with government agencies in geotechnical research, water resources, and materials engineering. He is a designated geotechnical specialist with A.P.E.O., and is regional director and treasurer for the Canadian Geotechnical Society. Dr. Mitchell is currently professor of civil engineering at Queen's University in Kingston and an associate of John D. Smith Engineering Associates Ltd., Kingston, Ontario.

John D. Smith

John D. Smith, born in Hamilton, Ontario, obtained his B.Sc. in mining engineering from Queen's University in 1960 and his M.Sc. in rock mechanics from the same institution in 1962. Since 1962 he has worked as a consulting engineer, specializing in rock mechanics, mine design and instrumentation for both mining and construction companies, as well as government agencies. He is currently president of John D. Smith Engineering Associates Limited and has worked on projects all over Canada and the United States, in Spain and in Ireland. He is a past Secretary of the Metal Mining Division of CIM, a member of the CIM, and of the Canadian National Committee on Rock Mechanics, and a designated consultant and mining specialist with A.P.E.O.

Requirements for Underground Mine Backfill Monitoring

David O. DeGagné[1], Euler De Souza[1], and Jacques Nantel[2]

ABSTRACT

While mine backfill plants are becoming increasingly more automated and instrumented, especially with paste fill operations, underground backfill distribution systems have hardly changed over the years. While fill properties are appropriately measured in the preparation plants, as backfill enters the mine borehole, the technology for control, data gathering and reporting is somewhat lacking.

Mines typically experience problems with borehole and pipeline plugging, pipeline bursting, bulkhead failure and exposed fill sloughing. With adequate instrumentation of the backfill system it is possible to obtain a complete, continuous and detailed picture of the entire filling operation from preparation to post-pour. Good data will, over time, provide the basis for needed improvements of backfill systems and will be essential to safer, more efficient and less costly fill practices.

This paper will review the areas of the filling operations where monitoring would prove critical to eliminate failures such as pipelines, bulkheads, fill exposures, stope walls, etc. The types of monitoring systems currently available will be reviewed and potential areas for development will be highlighted.

INTRODUCTION

In 2000, Canadian mines placed in excess of 71,000 tonnes of backfill underground daily. (Southam, 2001) Increased use of engineered backfill (backfill that is incorporated into the mine design), mounting environmental restrictions, and the use of more complex backfills (such as paste fill) is likely to see increasing volumes of backfill, and of systems requiring greater technical control, being placed in Canadian mines.

Backfill can no longer be simply regarded as a waste stream, but must be treated as an engineered by-product of the mining process and one that is essential for many modern mining methods. As such, it becomes ever more necessary, and technically challenging, to produce and safely deliver such a high quality backfill product from the metallurgical mill to the underground stope. Infrastructure failure or backfill sloughing may create an unsafe working environment and production costs associated with delays in backfilling, clean-ups, infrastructure failures, and dilution can be significant.

Backfill preparation has improved continuously over the past 20 years as operating practices and technology has steadily improved. Today's mining backfill plants are more akin to civil concrete plants with backfill mixture proportioning and concentration being automatically, accurately and continuously monitored. However, once the backfill mixture leaves the preparation plant, few operations regularly monitor the state of their backfill or backfill system beyond visual inspection. This curtails the process, effectively reducing the operator's ability to understand and control the backfill system.

While one of the most invaluable instruments at a mine site is the eyes and ears of an experienced operator, today's mining technology cannot solely rely on the operator's sixth sense and good luck. (Nantel, 1990) Today, mine design and operations require an important component devoted to gathering, analysing and making optimum use of the data. In a few words, monitoring has become an integral part of modern mining.

This paper wishes to address this important area of underground mine backfill and alert the mine operators to existing monitoring instruments and methods now in use o under development. Areas where the technology is lacking will also be highlighted.

[1] Department of Mining Engineering, Queen's University, Kingston, Ontario.
[2] Nantar Engineering Limited, Kingston, Ontario.

THE BACKFILL SYSTEM

A survey conducted by the Department of Mining Engineering at Queen's University of mining operations using backfill, summarized by De Souza et al (2001), saw the participation of 35 mines. The survey showed that backfill was utilized in Canadian mines primarily for hanging wall stability, increased extraction, dilution control, regional support, pillar recovery and as a working floor (Figure 1). For these reasons, the importance of maintaining mine backfill quality cannot be understated in order to ensure personnel safety, protect property and maintain productivity.

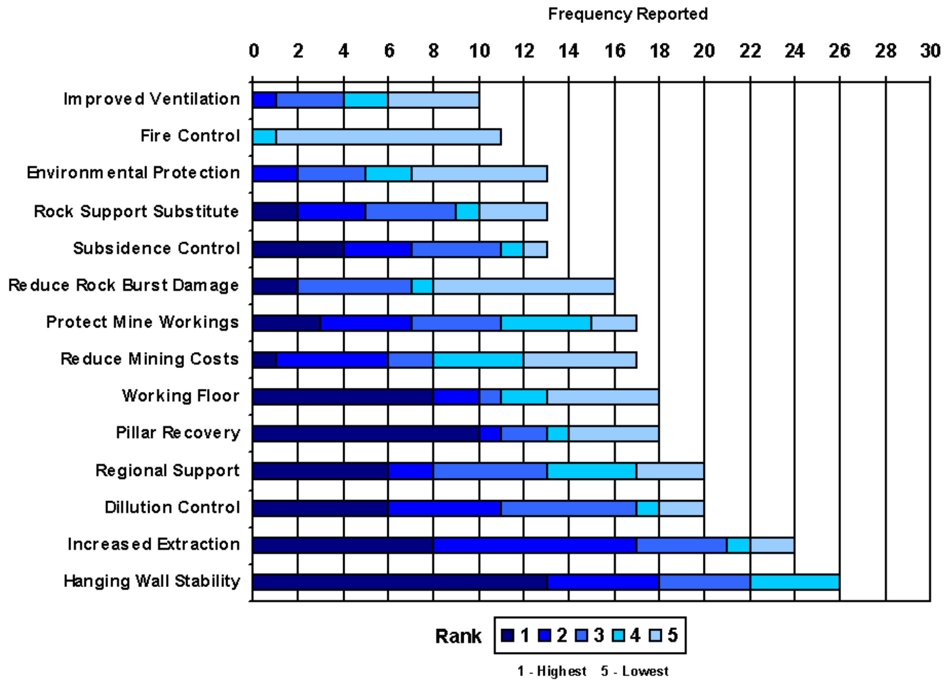


Figure 1. Backfill functions in underground mining.

Over half of the mines responding to the Queen's survey indicated that their operation had experienced some form of backfill system failure within the past 10 years. These failures generally occurred within the distribution system, the stope or at the bulkhead (Figure 2). While the most frequent failures occurred in the distribution system, the potential magnitude of a stope or bulkhead failure should be considered. As detailed by De Souza et al (2001), failures in the distribution system included pipeline and borehole plugs and pipeline bursts, pipe hammering, pump malfunction and plugged sumps. Failures in the stope included exposed backfill sloughing, fill segregation, rat holing, and fill liquefaction. While bulkhead failures may be indirectly caused by stope failures, they may also be caused directly, by poor design or installation, and fail even though the backfill itself has been optimized. While none of these mines reported any serious injuries or any fatalities, significant production and property losses were reported. In any event, the potential for injury or loss of life still remains.

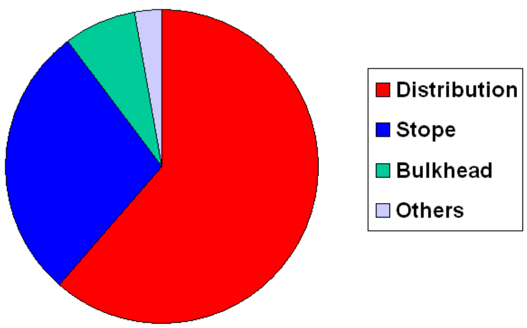


Figure 2. Reported failures in backfill systems.

Examples of such failures can be seen in Figure 3. In the first image (left to right), a considerable length of haulage drift was temporarily closed while backfill, nearly a metre deep, had to be cleaned up. In the middle image, backfill can be seen sloughing into an adjacent opening. The final image, from a research study, was recorded just as a wooden bulkhead failed; the bulkhead was located at the end of a closed drift and was forced to fail using compressed mine air. (Noranda, 1990)

Figure 3. Common backfill system failures. From left to right, plug or burst pipelines, backfill sloughing and critical bulkhead failure.

Such failures may occur due to fill plant or underground operator error, poor engineering design or planning, poor installation or poor maintenance. In all cases, instrumentation and monitoring may have provided sufficient warning to prevent or control the failure and ensure the protection of personnel and equipment. Where poor fill quality may be adopted, consequent ore dilution and loss of structural support may represent considerable economic loss and safety-related problems to mine operations. Operators depend upon the success of backfilling programs to ensure that mine activities run continuously. (Archibald et al, 1993)

The three major failure types (distribution, stope quality and bulkhead) can be seen in Figure 4 along with the main components of a simplified backfill system. Properties that are important to control and be aware of in the backfill system include the mixture proportioning (backfill recipe) and pulp density (solids concentration) at the fill plant; flow velocity and line pressure throughout the distribution system; and the pressure exerted on the bulkhead and the quality (strength, porosity, cement distribution, segregation factor, etc.) of the fill in the stope.

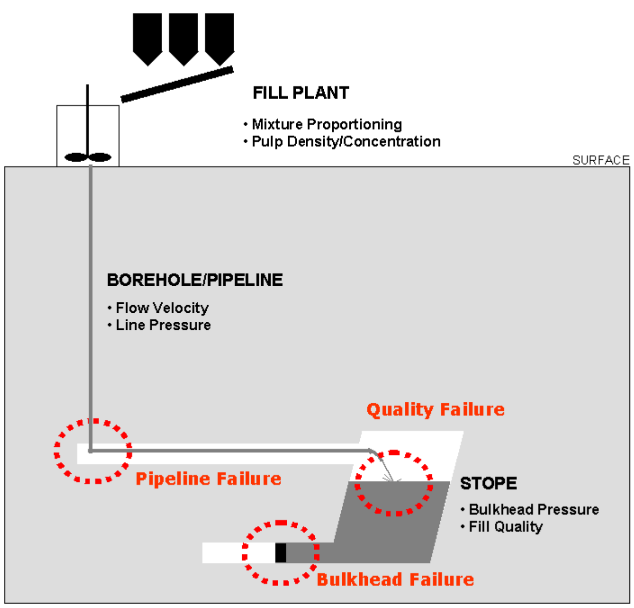


Figure 4. Simplified mine backfill system components.

The **fill plant** is the point where the raw materials of the backfill are proportioned to engineering specifications. Previous authors have been critical of backfill plants, stating that in the mining industry, tight controls on the quality of backfill products produced are minimal relative to those that have been traditionally established by concrete products industries. (Archibald et al, 1993) For example, a small change in the water content of a paste fill can result in considerably large changes in pipeline pressure. Today, however, backfill plants are increasingly well supported by state-of-the-art monitoring instruments and, in the absence of operator error, provide a reliable product.

While such improvements are encouraging, they represent only one part of the equation for complete backfill quality control and system integrity. This is because, as backfill enters the underground mine **distribution system**, the technology for control, data gathering and reporting is lacking in most mining operations. Very few operating mines have instrumented their distribution system for continual monitoring. As such, operators are effectively working blind. Problems are typically identified after they have occurred and that may be sometime after the fact in a large operation. As for fill **quality in the stope**, backfill properties, while routinely measured at the fill plant and in laboratory characterizations, are infrequently measured post-pour in the stope. Operations where in situ testing has been conducted often report significant differences between the backfill properties on surface and those in the stope. Laboratory characterizations represent a snapshot of an operation's backfill properties; daily operation practices and constraints often realize differences in fundamental backfill properties. (Archibald et al, 1993) Geotechnical type monitoring of backfilled stopes and the surrounding rockmass is more common in practice, especially in special cases where a potential for failure is suspected.

The potential for a failure within the backfill system or of the backfill itself exists. The consequences of such a failure can lead to the loss of or damage of property and has the potential to injure or kill operating personnel. The two basic objectives of any mine-monitoring program are to improve mine productivity and to protect mine personnel and property.

REASONS FOR MONITORING

The reasons for monitoring in engineering are well established and apply equally well to backfill operations. The most important were outlined by Franklin (1990). The salient reasons are to protect miners and prevent accidents, to obtain data for design, to verify design and assumptions and to investigate failures. Additionally, De Souza (1998) included:

- Maintain or improve productivity
- Worker confidence
- Model calibration
- Environmental control
- Legislation and legal considerations
- Provide early automated warning
- Public relations
- Research

For a backfill distribution system, Paterson and Cooke (1996) identified the following reasons for monitoring:

- Regulate mixture throughput
- Ensure continuous operations
- Evaluate system performance
- Determine if and where pipeline failure occurs
- Prevent problems, such as blockages.

Reasons for monitoring backfill in situ were identified by Falconbridge (1990):

- Verify design properties
- Monitor changes in fill pillars as they take load of regional ground support
- Monitor damage from blasting and other activities in neighbouring area, to predict ore dilution
- Evaluate merits of new fill types
- Stress monitoring
- Deformation monitoring
- Measure blast vibrations through fill and evaluate liquefaction potential
- Determine the extent of fill sloughing
- Measure the backfill's physical properties after placement

Additionally, backfill instrumentation and monitoring permits the indirect monitoring of the rockmass; effectively making the backfill a sensor of the local rockmass stress and deformation conditions. Reducing the need for manual inspection in remote locations, but remotely monitoring, and therefore reducing the travel time may also attain increases in operator efficiency.

The mining literature is full of examples describing how accidents could have been avoided if an adequate monitoring program had been in place. The high costs associated with injuries, equipment damage, loss of production, delays, and loss of ore reserves are all imperative reasons for a mining company to review and implement a monitoring function. Cost is not usually part of the equation; the question is not how much will it cost to implement a monitoring program, but how much it will cost the mine not to implement one.

Mine operators have moral and financial obligations to their workers and shareholders to operate their mines in the best and most efficient manner. Monitoring of backfill functions and the compilation of resulting data will allow the operators to better understand the mechanisms of the operation and will enable them to verify the assumptions made during the design stages. It becomes an indispensable tool for improving the engineering practice and to maintain control of the backfilling process.

Monitoring directly relates to mine productivity by forecasting problems, preventing dilution of ore and by providing early warning of instability to permit timely planned action and prevent costly delays or shut-down associated with fill or backfill system failure. When installed well ahead of mining, instruments can provide data for design optimization and validation. The instrumentation data is used for back-analysis of initial excavation work where information on rock deformation, induced stresses, loads on pillars and other support systems are assessed during construction and compared with the predictions made by design calculations and numerical modelling. This is the time when the validity of the design is checked and numerical models are calibrated and validated. In this process, the mining methods, layouts, mining sequences and strategies, and support designs can be modified for economy and to guarantee stability. (De Souza, 1998)

When designed to assess backfill failure, monitoring information is used to identify the mechanisms and types of failure, its location, magnitude and direction, and to design applicable remedial work. Monitoring is also necessary for legal reasons; all corporations have a legal obligation to take every precaution reasonable in the circumstances for the protection of the worker. Failure to do so could result in severe penalties. Monitoring is not only designed to provide evidence of compliance with regulations but to demonstrate effectiveness of the company's environmental program. Such environmental issues include seepage and migration of contaminants, groundwater contamination, mine acid drainage, tailings contamination, waste dump stability, crown pillar stability, ground subsidence and reclamation practices.

BACKFILL MONITORING

The Queen's survey revealed that only 10 of the operations employed some form of backfill instrumentation and monitoring program. Of these operations (Figure 5), some form of pressure meters, followed by closure meters, extensometers and piezometers were used as monitoring instruments by the mines. Additionally, 21% of operations reported using other instrument types. These included accelerometers, borehole cameras, thermometers and a ground movement monitor (GMM). Such instruments (Figure 6), and other types, were used to monitor fill pressure, closure, stiffness, deformation, pore-water-pressure and porosity. Some operations also reported measuring bearing capacity, seismic velocity and blast vibrations, post-blast fracture mapping, in situ compressive strength, pH, saturation, and sloughing (through visible inspecting).

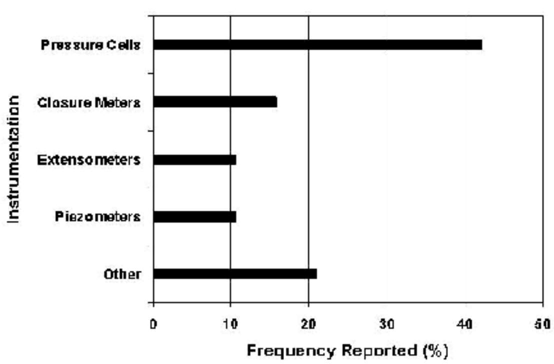


Figure 5. Reported backfill instrumentation.

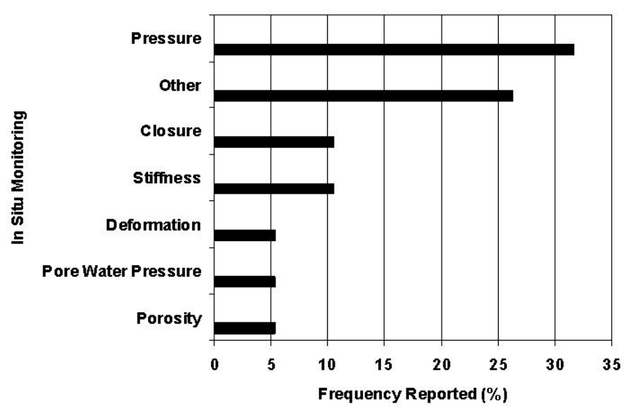


Figure 6. Reported backfill monitoring.

MONITORING and INSTRUMENTATION TECHNOLOGY

The most common backfill instruments associated with backfill include flow meters, extensometers, convergence meters, strain gauges, stressmeters, strain cells, pressure cells, load cells, accelerometers and piezometers. Such backfill and geotechnical instrumentation typically utilizes electrical, vibrating wire, mechanical, hydraulic and optical systems as measuring principles. In general, all components of a monitoring system should be as simple as possible, requiring little or no maintenance; still they must provide accurate data in a relatively short turnaround time. An effective instrument should be of low cost, robust, durable and reliable, of simple design and very easy to install and operate. The degree of design complexity depends on the purpose, the length of time an instrument is needed, and who will read and process the information. (De Souza, 1998) Pipeline based instruments should also be non-intrusive to reduce wear and risk of blockages. (Paterson and Cooke, 1996). A brief summary of these types of monitoring equipment follows in Table 1.

Table 1. Summary of monitoring instruments and practices for mine backfill. After Paterson and Cooke (1996), Falconbridge (1990), De Souza (1998a) and Mackenzie (2001)

DISTRIBUTION	
Flow	• **Bend Meter** - The differential pressure from the inside to the outside measured across a 90° bend can provide the pipeline flow rate. • **Venturi Meter** - Consists of a constricting section of pipe with tapered connections. Based on Bernoulli's equation for head loss, the difference in pressure readings between the two diameter pipelines is proportional to backfill flow. • **Magnetic Flow Meter** - A voltage is induced across the flowing backfill as it moves through a magnetic field. The measured voltage is proportional to the flow. These devices are very common and widely used, but the backfill must be magnetic. • **Ultrasonic Flow Meter** - Generates ultrasonic vibrations using piezoelectric crystals. o **Doppler Flow Meter** - These portable devices are easy to install and work, using the Doppler effect, by transmitting an ultrasonic signal which reflects off of the backfill particles and is measured by a receiving transducer. The change in signal frequency is proportional to backfill velocity. o **Time of Flight Flow Meter** - These devices must be in contact with the backfill and as such are usually supplied built into pipeline flanges. An ultrasonic signal is transmitted in two directions through the backfill and the average of time difference between the signals is proportional to the flow velocity. • **Tracer** - A distinctive material is added to the backfill mixture at a given point and the time required for it to travel a known distance is recorded. • **Trajectory -** The ratio of the horizontal and vertical trajectory lengths provides an approximation of flow velocity. • **PSI-Pill** - Flows within pipelines with diameters as small as 7.6 cm. Measures pressure, at up to every 10 seconds, to provide a pressure trace to assist with the diagnosis of friction losses, freefall regions, impact zones, water hammer and flow velocities
Concentration	• **Marcy Scale** - The backfill is weighed manually using a defined volume. Ensuring a representative sample is critical. • **Intrusive Probe** - Intrusive probes obtain samples at different points along the pipeline. • **Gravimetric Methods** - Concentration is determined by weighing a section of the pipeline. Not a common practice. • **Gamma Ray Densitometer** - A gamma ray source, such as Cesium, radiates a narrow beam of energy through a pipeline to a detector. This energy will be partly absorbed by the backfill in the pipeline and the concentration of the mixture is then a function of the signal output which depends on pipe diameter, gamma-ray intensity and the mass absorption coefficient. • **Counter-Flow Meter** - Consists of a vertical U tube, and pressure gauges on both the up and down pipelines. The weight of the backfill flowing upwards and downwards if effectively determined by measuring pressure differentials and it is possible to calculate the concentration.
Pressure	• **Pressure Transducer** – Depending upon the pressure range, either an electrical transducer (low to medium) or a piezo-electric crystal (high) is used to relate the physical force into a signal. • **PSI Pill** - See description above.

STOPE / BULKHEAD	
Deformation	• **Borehole Extensometer** - measures convergence, settlement, heave and lateral deformations by measuring changes in axial displacement between two or multiple points. Can measure sill deterioration, pillar expansion and vertical face stability. o **Rod Extensometer** - Are anchored into place or mechanically expanded against borehole walls. Provides simple, low-cost deformation measures. o **Wire Extensometer** - Consists of stainless steel wires that are tensioned at a constant force. They are generally more complex and difficult to install than rods, but are often used to monitor multiple points within a borehole. o **Others** - Include magnetic probe extensometers, single/multiple point laser extensometers and Telltale deformation (differential transformer) gauge.
Convergence	• **Closure Meter** - Measures the rate of convergence between two opposing, exposed faces. Usually consists of a spring-loaded potentiometric devices that are anchored to two reference points between which, relative motion is to be measured.
Fill Pressure	• **Total Pressure Cell** - Measures the sum of water pressure and intergranular effective stress. The two basic types are the diaphragm and the hydraulic cells (common). o **Diaphragm Pressure Cell** - Consists of two thin, flexible circular plates sealed by a stiff outer ring. The degree of deflection of the plates can be related to the external pressure as sensed by a strain gauge transducer or vibrating wire transducer. o **Hydraulic (Flat Jack, Glötzl) Cell** - Consists of two thin flexible steel plates weld along their edge and filled with a de-aired fluid. It may use a pneumatic-, pressure- or a vibrating wire- transducer. • **Borerhole Pressure Cell** - Consists of a cylindrical tube covered with a flexible membrane that can be placed into a borehole. A tube connected to the cell allows it to become pressurized and for the pressure to be read using a simple dial gauge or transducer. Measures total fill to determination the elastic modulus of the host rock. • **Deflection** – The deflection of the bulkhead, as measured using a strain gauge or dial gauge can be an indicator of the pressure behind the bulkhead.
Pore Water Pressure	• **Piezometer** o **Pneumatic** - The most common types contain a flexible diaphragm that is protected behind a porous filter. The diaphragm balances the pressures between the pore water pressure and with those of a gas supply. The pressure of this gas is measured to obtain the pore water pressure. o **Vibrating Wire** - A metallic diaphragm is exposed to the pore water pressure and as pressure increases, the diaphragm is pushed in, which causes the tension of a tensioned metal wire to decrease. Pressure can be determined based on the strain on this wire. o **Electrical Resistance** - The deformation of the metal diaphragm is measured using strain gauges.
Temperature	• **Thermal Resistor** - These instruments generate a variable electric current depending upon their environmental temperature.
Sloughing	• **USBM Sonic Probe** - Profiles dimensions of voids that may be created in the fill after production blasting. • **Flashlight Method** - When a flashlight is lowered into a borehole, the light will be reflected up through the hole unless it breaks into a void. • **Sloughmeters** - Consists of an electrified wire secured in backfill. Should the backfill fail, so too will the wire and therefore breaking the electric current. • **Laser Profiling** - Uses a reflecting laser to map surface profile of stope. • **Visual Inspection** – Manual or camera inspection to examine for signs of failure.
Fill Strength	• **Cone Penetrometer** • **Coring** - Process by which a sample of in situ backfill is obtained though overcore drilling. Difficult procedure to perform on materials with low stiffness. • **Borehole Pressure Cell** - See description above.
Liquefaction Potential	• **Accelerometer** - Measure the peak particle velocity of the fill as a shock-wave (blast induced) travels trough the material. This determines the amount of seismic energy that will be transmitted from the rock to the backfill.
Drainage	• **Weirs, ditches or collection containers** - All the water placed into the stope must be accounted for.

An example of how stope instrumentation may be oriented and used is shown in Figure 7.

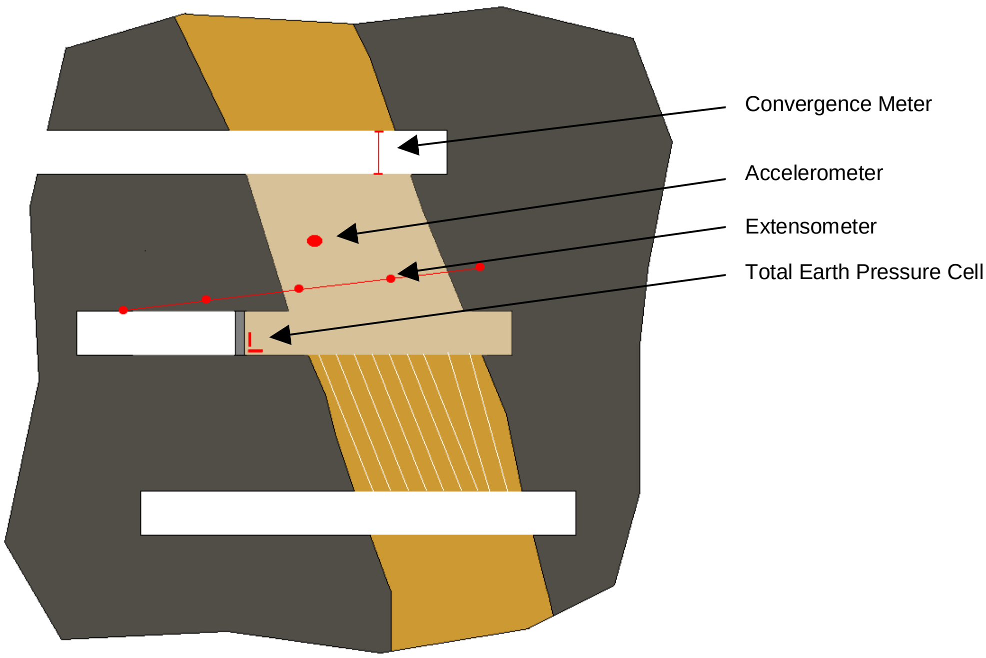


Figure 7. Basic instrumentation of a backfilled stope.

MONITORING SUCCESS

Careful planning of any instrumentation program is essential in order to guarantee successful benefits in terms of mine productivity and safety. A well planned and well-executed instrumentation program can repay its costs many times over and can prevent the high costs associated with injuries, equipment damage and dilution.

In order to ensure successful instrumentation program equipment procurement should be based on quality and reliability, not price alone. The system design should be based on an integrated systems approach (i.e. mixing and matching components should be avoided) and be flexible for future expansion. Personnel must be informed of the importance and purpose of the monitoring and how to work with and beside the system. Additionally, the monitoring program is not intended to replaced visual inspection of critical infrastructure, but to augment it, and manual checks to calibrate and validate instrument readings becomes essential.

Reasons for instrumentation program failure identified by De Souza (1998a) include:

- Inexperienced designers
- Program poorly planned or designed
- Inappropriate instruments
- Poor quality equipment
- Select incorrect properties to monitor and solve problem
- Poor installation
- Poorly calibrated in situ
- Instruments or cables are poorly maintained or protected
- Program is not updated in relations to data gathered
- Poorly informed personnel
- Collected data is improperly measured, analysed or reported

CASE STUDIES IN THE LITERATURE

Two case studies recently published in the literature represent ideal examples of modern backfill monitoring, albeit at two degrees of scale. The first is representative of the state-of-the-art instrumentation available and the extent to which modern operations are taking backfill system monitoring seriously. The second is representative of a successful in situ monitoring program emplaced to ensure that the stope could be safely undercut.

Case 1: Brunswick Mine (Noranda), Bathurst, New Brunswick

Ouellette et al (1999) details the underground monitoring emplaced for Brunswick Mine's recent paste backfill system. The mine has implemented an extensive monitoring system that includes 14 pressure sensors (6 diaphragm, 8 strain gauges) to measure pressure along the pipeline during backfill pours and water flushes. These sensors were instrumental in efforts to depressurize and flush the fill system after two blockages occurred. Mobile cameras, to monitor discharge points and critical and remote system infrastructure, and Doppler flow meters have been installed where cameras are not practical. The entire monitoring system is to be tied into the leaky feeder communication system for remote data acquisition and to allow for sensor portability and flexibility. In addition, during backfilling, manual checks of the pipeline system and the bulkheads are conducted every shift and reported to the backfill plant.

Brunswick's recent backfill system is one of the most advanced and extensive monitoring configurations in place in the Canadian mining industry. It is the authors' opinion that this operation represents the state-of-the-art backfill system and should be used as an example for all mining operations backfilling, particularly those employing paste fills.

Case 2: Garson Mine (INCO), Copper Cliff, Ontario

Ley et al (1998) detail an instrumentation program designed to monitor mining beneath a stope backfilled with paste up to 12.2 meters high. Drill rounds beneath the fill were full width advancing 1.8 metres. Support consisted of a layer of shotcrete to the back and, when backfill was exposed, the walls. After the shotcrete cured, bolts and screens were applied and another layer of shotcrete was sprayed. The undermined stope was continuously monitored using three vertical and one horizontal extensometer(s), a soil temperature probe and two total pressure cells (horizontal and vertical orientations) over a period of 21 months. As the backfill cured, fill pressure decreased, as water reacted with cement or drained off; and temperature increased rapidly, due to cement hydration, and was maintained at elevated temperatures for several months. Ley et al suggested that temperature may even be used as a measurement of fill hydration and thus fill quality. Finally, extensometers revealed that the fill did appear to separate and then compress against the back of the shotcrete shell.

It could be argued that, due to good engineering design, there was no need to monitor, the stope as the support measures implemented were satisfactory. However, through monitoring the researchers were able to visualize what was happening behind the shotcrete shell and would have been forewarned if failure were imminent. Monitoring also provided valuable data that can now be applied to enhance backfill models and prediction and fill temperature may prove to be a useful tool for determining the extent of cement hydration and in helping to explain why in situ backfill strengths (subjected to thermal acceleration due to elevated temperatures) tend to outperform laboratory scale strength tests.

INNOVATIONS and FUTURE DEVELOPMENTS

Drawing on the experience of other industries, mining is beginning to customize monitoring equipment for its own needs. Examples of this are the self-boring pressuremeter and the PSI Pill. The self-boring pressuremeter has been developed due to the difficulty of using pressuremeters in mine backfill in the past. Such pressuremeters were either pushed into the fill or an oversized borehole was first drilled and then the pressuremeter placed within and inflated to match the diameter of the borehole. However, in both cases researchers reported considerable difficulties in obtaining measurements and often, those measured were not necessarily representative of the in situ fill due to influences from the probes themselves. Annor (1990), Scoble et al (1987) Research by Ouellet and Servant (2000) has demonstrated the effectiveness of a self-boring pressuremeter which did not disturb the in situ backfill and provided multiple readings over a reasonable period of time.

Paste Systems Inc. (PSI) of Sudbury Ontario (in association with CAMIRO) has developed, and is testing, a small pill that is battery operated and can record pressure as it travels along a 7.62 cm pipeline before being recovered (Mackenzie, 2001). The resulting pressure trace assists diagnosis of friction losses, freefall regions, impact zones, water hammer and flow velocities. Future pill designs are to include thermocouples, accelerometers, 3-D directional gyros and pipe wear or pipe diameter sensors and the capacity to travel in 5 cm diameter pipelines.

Mine Design Technologies (MDT) of Kingston, Ontario has developed the SMART Contractometer that uses a collapsible structure to measure backfill convergence. It is a six-point fully recessable unit with an integrated electronic readout head. For backfill applications, shear washers are used to ensure that full transfer of convergence is monitored. (Todd, 2001)

As the awareness for the need to actively monitor the backfill system as well underground as we do on surface, then the demand for more customized mine backfill instrumentation will increase and new and innovative instrumentation and practices will result.

Where do we go from here? Mine operators and manufacturers of monitoring equipment have to become experts in what needs to be monitored. Instruments need to be robust, low cost, provide the required sensitivity, can be read remotely and become part of the mine wide monitoring and communication system. Installation of the monitoring equipment has to lend itself to the mining method and be safe in nature.

The authors further propose the development of a computer based Mine Backfill Database similar to the RockPro Rock Mechanics Database developed by ESG Canada of Kingston, Ontario. The ESG database provides the geotechnical engineer with an easy to use tool for recording and reporting all aspects of geotechnical underground observations. The authors propose a parallel package for the mine backfill function. A mine should keep adequate records of all phases of the backfill system and operations. The computer program should include such information as the backfill properties of each filled stope (cross-referenced with preparation plant and pour data), real-time and analysed data from monitoring instruments, incidents of failure, backfill system simulation based on the mine data, etc.

This proposed computer system would incorporate a search routine to arrange and summarise information based on location, date, type of observation, or a combination of parameters. All sorts of types of files could be incorporated to the system: photos, drawings, engineering standards for bulkhead design and so forth.

Finally the program could be tailor made for each operation with the possibility to generate all the reports and output required by the operators. These forms can include for example simple forms given to the workers, mine foreman, engineers, management, governments, etc. The program would have to be compatible with most mine management software packages available today.

CONCLUSIONS

Improvements in backfill operations have a marked influence on the overall efficiency of the entire mining operation. The authors are of the opinion that significant progress needs to happen in the area of backfill monitoring to bring this important aspect of a mining operation in line with what has happened in the area of fill preparation.

There are increasing trends for the utilization of engineered mine backfill and its complexity. No longer can backfill be relegated as a simple waste component of mining. It must be viewed as a product, subject to quality control, designed and manufactured on surface (backfill plant) and delivered (distribution system) to a client (the stope) underground. Along this sequence, information from industry indicates that backfill system failures occur primarily along the distribution system. This is followed by failures occurring within the backfill mass itself, due to poor quality control, and then by bulkhead failures, due to poor design, construction, pouring or fill quality.

In order to provide personnel with backfill operational or engineering design information, it is suggested by the authors that all three of these components be instrumented and monitored. Monitoring protects mine personnel and property; aids in engineering design and prediction; identifies, locates and prevents failures; prevents production downtime and increases operator control, understanding and efficiency. Backfill pressure, closure, stiffness, deformation, pore water pressure and porosity, as well as, bearing capacity, blast vibration response, in situ strength, saturation and sloughing have been reported as being the primary parameters that mines are monitoring. To this end, pressure cells, closure meters, extensometers, piezometers have been identified as instruments commonly used to measure the proceeding parameters; accelerometers and borehole cameras have also be reported in use, but to a lesser extent.

It appears that when backfill monitoring is conducted at an operation, it is primarily focused on the stope. Additionally, modern backfill preparation plants have advanced considerably and often utilize state-of-the-art control and monitoring technology. When one considers that backfill system failures are most frequently associated with the distribution system, it is clear that more attention must be focused on this component of the overall system. This is not to underestimate, however, the importance of plant and stope monitoring, as failures in these components, while less frequent, have the potential for failure on a much larger scale. Few mining operations participating in a Queen’s University survey reported some or most of their backfill properties, especially in situ, crucial to determining the quality of their fill. While it may simply have not been convenient for them to do so, there is a strong possibility that this merely reflects the priority placed on such information or its availability.

In order to be effective, the monitoring system should be reliable, integrated and flexible. Personnel must be informed about the instrumentation and its importance; be properly trained to install and use the equipment and interpret the data. In situ calibration and verification over time, through recalibration or another instrument, is very crucial to obtaining accurate and reliable information.

Recent papers show that there are mines that have established very impressive and effective backfill monitoring programs. In addition to existing technologies adapted from geotechnical applications, innovative technologies have been created specifically for backfill applications. With greater demand for such devices, more useful instrumentation and monitoring practices will develop. The development of an integrated mine backfill database software package similar to those used for general rock mechanics is encouraged. This will provide relevant information both internally for the operation and publicly for research purposes. The authors hope that this paper will lead mining operations to revisit backfill monitoring and consider a holistic approach that provides information encompassing the entire operation. As observed in several areas of engineering, monitoring pays for itself in several ways. Mining activities are no exceptions. If the nineties were the decade of the paste fill, the next big advancement should lie in more advanced backfill monitoring and data analysis. Good monitoring will lead to improved backfill practices and these improvements are a pre-requisite for future mines. The mines most likely to succeed in the future will be using the best technology; a well-monitored backfill system is one such technology.

REFERENCES

ANNOR, A., 1990. Backfill Alternatives in Ontario Mines. Report Canmet Contract No. 23440-6-9011-01-SQ, 249 p.

ARCHIBALD, J.F., LAUSCH, P., and HE, Z.X., 1993. Quality Control Problems Associated with Backfill Use in Mines. CIM Bulletin, 86(972), July-August, 53-57.

DE SOUZA, E., DEGAGNE, D.O., and ARCHIBALD, J.F., 2001. Minefill applications, practices and trends in Canadian mines. MINE FILL 2001, the 7th International Symposium on mining with backfill, Seattle, Washington, USA, September 17-19, 9 p.

DE SOUZA, E., 1998. Keys to successful monitoring of evaporite and coal mines. CIM Bulletin, 91(1020), May, 76-82.

DE SOUZA, E., 1998a. Applied Rock Mechanics Instrumentation. Course Notes. Department of Mining Engineering, Queen's University, Kingston, Ontario.

FRANKLIN, JOHN, 1990. Mine Monitoring Manual. Special Volume 42, Canadian Institute of Mining, Metallurgy and Petroleum, Rock Mechanics and Strata Control Committee, 156 p.

LEY, G.M.M., STEED, C.M., BRONKHORST, GUSTAS, R., 1998. Mining under backfill. CIM Bulletin, 91(1020), May, 65-71.

MACKENZIE, A., 2001. Paste Systems Inc., personal communications.

NANTEL, 1990. Forward Remarks, Mine Monitoring Manual. John Franklin (editor), Special Volume 42, CIM, 156 p.

NORANDA, 1990. In-situ test of backfill bulkhead, 1990-1991, Centre de Technologie and Technik Mining, videocassette.

OUELLET, J. and SERVANT, S., 2000. In-situ mechanical characterization of a paste backfill with a self-boring pressuremeter. CIM Bulletin, 93(1042), July, 110-115.

OUELLETTE, G., MOERMAN, A., and ROGERS, K.., 1999. Paste Backfill at Brunswick – Part II: Underground Construction and Implementation.. Paper #36, 14th CIM Mine Operators Conference, 11 p.

PATTERSON and COOKE, 1996. The Design of Slurry Pipeline Systems. A short course, 14-16 February, Cape Town, South Africa. Chapter 9: Pipeline Instrumentation, 21 p.

SCOBLE, M.J., PICIACCHIA, L. and ROBERT, J.M., 1987. In situ testing in underground backfilled stopes. CIM Bulletin, 90(903), 33-38.

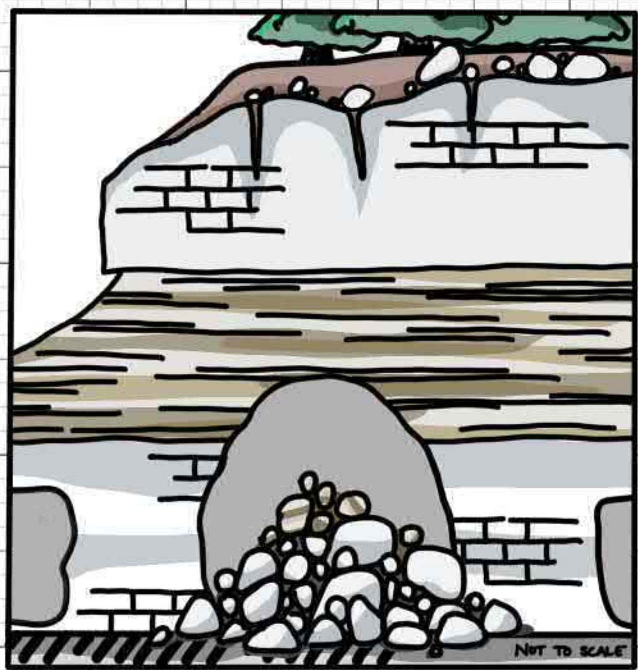
SOUTHAM, 2001. 2001 Mining Sourcebook. Canadian Mining Journal, Backfill Methods, 80-81.

TODD, J., 2001. Mine Design Technologies, personal communication.

Mine Filling Calculations

Streets of West Pryor | J035637.01

MINE FILLING CALCULATIONS

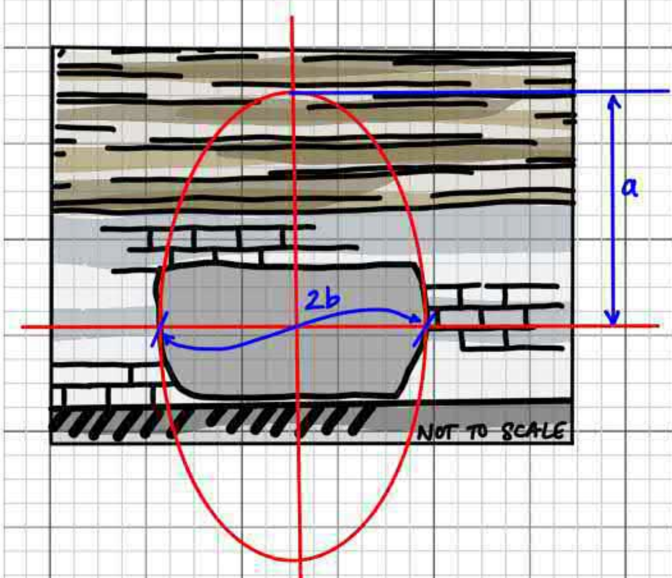


IN GENERAL, ROOF FAILURES CONTINUE TO PROPOGATE LATERALLY AND HORIZONTALLY UNTIL A MORE STABLE CONFIGURATION IS REACHED, GENERALLY AN ARCH.

THE HEIGHT OF THE RESULTING DEBRIS PILE IS DEPENDENT ON:

- THE GEOMETRY OF THE RESULTING ARCH (HOW MUCH DEBRIS IS MADE)
- THE ANGLE OF REPOSE OF THE DEBRIS MATERIAL
- THE BULKING FACTOR

ARCH GEOMETRY : PRESSURE ARCH THEORY



$$a = \frac{(P_v - P_h + \sigma_t)\,b}{2P_h}$$

WHERE: a = ELLIPSE HEIGHT
$b = \frac{1}{2}$ BEAM SPAN
P_v = VERTICAL STRESS
P_h = HORIZONTAL STRESS $(0.7 P_v)$
σ_t = TENSILE STRESS

$$\sigma_t = \frac{\gamma L^2}{2t}$$

WHERE: γ = UNIT WEIGHT
L = BEAM SPAN
t = BEAM THICKNESS

γ = 163 pcf
L = 35 ft
t = 4.5 ft

$$\sigma_t = \frac{163\text{ pcf}\,(35\text{ FT})^2}{2(4.5\text{ FT})} = 22186\text{ psf}$$

Calc. By: ALY	Checked By:	Appvd By:
Date: 12-2-2020	Date:	Date:

Streets of West Pryor | J035637.01

MINE FILLING CALCULATIONS

P_v = VERTICAL STRESS

NOT TO SCALE

		ASSUMED
OVERBURDEN	10.1'	$\gamma = 120\,pcf$
WEA SHALE	8.7'	$\gamma = 140\,pcf$
WINTERSET LIMESTONE	36.1'	$\gamma = 163\,pcf$
STARK SHALE	6.2'	$\gamma = 140\,pcf$
BETHANY FALLS LIMESTONE — ASSUMED BEAM THICKNESS	4.5'	$\gamma = 163\,pcf$

$$P_v = 10.1'(120\,pcf) + 8.7'(140\,pcf) + 36.1'(163\,pcf) + 6.2'(140\,pcf) + 4.5'(163\,pcf) = 9915.8\,psf$$

$$P_h = 0.7 P_v = 6941.1\,psf$$

STRATIGRAPHIC COLUMN BASED ON INFORMATION PROVIDED IN URS REPORT DATED FEBRUARY 7, 2002.

$$a = \frac{(9915.8\,psf - 6941.1\,psf + 22186.1\,psf)\,17.5\,ft}{2(6941.1\,psf)}$$

$= 31.7\ ft$

ROOM HEIGHT 12 ft

ROOF BREAKOUT DEPTH $31.7 - 12 = 19.7\ ft$

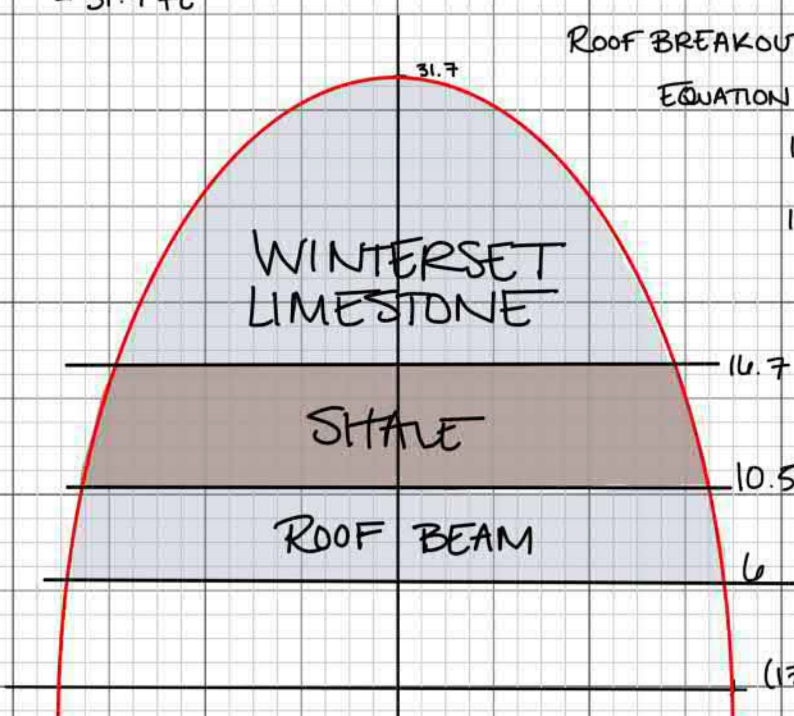
EQUATION OF AN ELLIPSE

$$1 = \frac{x^2}{a^2} + \frac{y^2}{b^2}$$ WHEN CENTERED @ (0,0)

$$1 = \frac{x^2}{(31.7)^2} + \frac{y^2}{(17.5)^2}$$

IN TERMS OF Y

$$x = \sqrt{\left(1 - \frac{y^2}{(17.5)^2}\right)(31.7)^2}$$



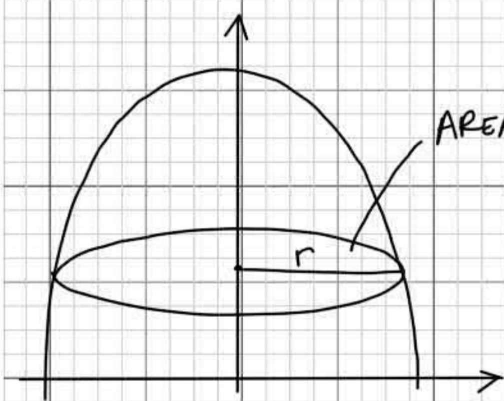
Calc. By: ALY	Checked By:	Appvd By:
Date: 12-2-2020	Date:	Date:

Streets of West Pryor J035637.01

MINE FILLING CALCULATIONS

MATERIAL VOLUME OF A DOME-OUT

ASSUMPTION: THE WINTERSET (WHERE UNWEATHERED & PROTECTED BY THE PRESENCE OF OVERBURDEN) IS A STRONG COMPETENT UNIT. IF MINE FILLING IS PERFORMED, IT IS OUR PROFESSIONAL OPINION THERE WILL BE NO BREAK OUT OF THE WINTERSET.



AREA OF THE DISC $= \pi r^2 = \pi x^2$

FOR VOLUME OF THE SECTION TAKE THE INTEGRAL WITH RESPECT TO THE Y-AXIS

$$V = \int \pi x^2 \, dy$$

$$\hookrightarrow x = \sqrt{\left(1 - \frac{y^2}{17.5^2}\right) 31.7^2}$$

$$V = \int \pi \left(\sqrt{\left(1 - \frac{y^2}{(17.5)^2}\right) 31.7^2} \right)^2 dy \longrightarrow 31.7^2 \pi \left[\int 1 \, dy - \int \frac{y^2}{17.5^2} \, dy \right]$$

FOR THE ROOF BEAM:

$$V = 31.7^2 \pi \left[\int_6^{10.5} 1 \, dy - \frac{1}{17.5^2} \int_6^{10.5} y^2 \, dy \right]$$

$$= 31.7^2 \pi \left(\Big[y \Big]_6^{10.5} - \frac{1}{17.5^2} \left[\frac{y^3}{3} \right]_6^{10.5} \right)$$

$$= 31.7^2 \pi \left(\left[10.5 - 6 \right] - \frac{1}{17.5^2 (3)} \left[10.5^3 - 6^3 \right] \right)$$

$$= 31.7^2 \pi \left(4.5 - 1.025 \right)$$

$$= 10970.74 \text{ ft}^3$$

VOLUME OF SHALE

$$V = 31.7^2 \pi \int_{10.5}^{16.7} 1 - \frac{y^2}{17.5^2} \, dy$$

$$= 7547.18 \text{ ft}^3$$

TOTAL ROCK VOLUME

18517.92 ft

Calc. By: ALY	Checked By:	Appvd By:
Date: 12-2-2020	Date:	Date:

Streets of West Pryor | J035637.01

MINE FILLING CALCULATIONS

EXPANDED VOLUME

ASSUMED BULKING FACTORS

BULKING FACTOR = $\frac{\text{INTACT DENSITY}}{\text{AGGREGATE DENSITY}}$

	INTACT	AGGREGATE	BF
SHALE	$\gamma = 140$ pcf	$\gamma = 125$ pcf	1.12
LIMESTONE	$\gamma = 163$ pcf	$\gamma = 130$ pcf	1.25

EXPANDED ROOF BEAM VOLUME $10970.74\ ft^3 (1.25) = 13713.43\ ft^3$

EXPANDED SHALE VOLUME $7547.18\ ft^3 (1.12) = 8452.84\ ft^3$

$22166.27\ ft^3$ TOTAL

GIVEN THE VOLUME OF THE ROOF BREAK OUT, WHAT HEIGHT OF FILL IS REQUIRED TO ARREST PROPAGATION? (CARRY THE LOAD BACK DOWN TO THE FLOOR)

ASSUMPTION: IF THE VOLUME OF THE FILL + THE EXPANDED VOLUME OF THE BREAKOUT DEBRIS IS EQUAL TO THE VOLUME OF THE ROOM, THE ROOF LOAD WILL BE TRANSLATED TO THE FLOOR

$$V_{DEBRIS} + V_{FILL} = V_{ROOM}$$

VOLUME OF THE ROOM WILL BE APPROXIMATED USING THE AREA BENEATH THE PRESSURE ARCH ASSUMING FAILURE OF ROOF BEAM & SHALE

VOLUME OF THE ROOM

$$V_{ROOM} = 31.7^2 \pi \int_{-6^*}^{16.7} \left(1 - \frac{y^2}{17.5^2}\right) dy$$

*(y=0 REPRESENT THE MID-POINT OF THE PILLAR HEIGHT)

$$= 54916.97\ ft^3$$

$$V_{FILL} = V_{ROOM} - V_{DEBRIS} = 54916.97\ ft^3 - 22166.27\ ft^3 = 32750.7\ ft^3$$

FILL HEIGHT		
	11 ft	$33554.8\ ft^3$
	10.75 ft	$32826.8\ ft^3$
	10.5 ft	$32092.7\ ft^3$

RECOMMENDED MINIMUM WOULD BE 10.75 TO 11 FT (FLOOR TO CEILING)

Calc. By: ALY	Checked By:	Appvd By:
Date: 12-2-2020	Date:	Date:

Contract Documents

Proposal for Underground and Construction Monitoring Services
Filling of Portions of Former Union/Superior Bowen Mine
Lee's Summit, Missouri
Geotechnology Proposal No. P035637.02

		Phase 1												Phase 2		Total Estimated Cost	
		Task 1 Preparation of Submittals for the City of Lee's Summit		Task 2 Preliminary Quality Assurance/Quality Control Plan		Task 3 Finalizing Project Survey		Task 4 Test Holes		Task 5 Engineering Analysis and Reporting		Task 6 Finalization of Design Methodology		Construction Monitoring			
Labor Classification	2020 Rates	Total Hours	Total Cost	Total Hours	Total Cost	Total Hours	Total Cost	Total Hours	Total Cost	Total Hours	Total Cost	Total Hours	Total Cost	Total Hours	Total Cost	Total Hours	Total Cost
Senior Consultant, Group Manager	$190	40	$7,600	40	$7,600		$0		$0		$0		$0		$0	80	$15,200
Senior Project Manager	$185	40	$7,400	40	$7,400	8	$1,480		$0	40	$7,400	40	$7,400	300	$55,500	468	$86,580
Principal Engineer	$170	40	$6,800	40	$6,800		$0	40	$6,800	12	$2,040		$0		$0	132	$22,440
Project Manager	$125		$0		$0		$0	40	$5,000		$0		$0		$0	40	$5,000
Senior Engineer/Geologist	$115		$0		$0		$0		$0		$0		$0		$0	0	$0
Project Engineer/Geologist	$90	40	$3,600	40	$3,600		$0		$0	40	$3,600	40	$3,600		$0	160	$14,400
Engineer/Geologist	$65		$0		$0		$0		$0	20	$1,300		$0	800	$52,000	820	$53,300
CADD	$65		$0		$0	16	$1,040		$0	2	$130		$0		$0	18	$1,170
Project Administrator	$55	2	$110		$0		$0		$0	1	$55		$0		$0	3	$165
Total Direct Labor		162	$25,510	160	$25,400	24	$2,520	80	$11,800	115	$14,525	80	$11,000	1100	$107,500	1721	$ 198,255.00

Other Direct Costs (ODC)		Task 4 Total Units	Task 4 Total Cost	Construction Monitoring Total Units	Construction Monitoring Total Cost	Total Units	Total Cost
Underground Equipment	$20	5	$100	20	$400	25	$500
Total Other Direct Costs			$100		$400		$500

$ 198,755

TERMS FOR GEOTECHNOLOGY'S SERVICES

1 - THE AGREEMENT

a. This AGREEMENT is made by and between: **Geotechnology, Inc.**, hereinafter referred to as GEOTECHNOLOGY, and **Streets of West Pryor, LLC.,** hereinafter referred to as CLIENT.

b. The AGREEMENT between the parties consists of these TERMS, the attached PROPOSAL identified as Proposal No. **P035637.02,** dated **December 2, 2020**, and any exhibits or attachments noted in the PROPOSAL. In the event of a conflict between the TERMS and the PROPOSAL, the provisions of the TERMS shall govern unless the PROPOSAL specifically indicates that it is to govern. Together, these elements will constitute the entire AGREEMENT superseding any and all prior negotiations, correspondence, or agreements either written or oral. Any changes to this AGREEMENT must be mutually agreed to in writing.

c. This proposal is valid for 90 days from **December 2, 2020**.

d. The technical pricing information contained in this PROPOSAL submitted by GEOTECHNOLOGY is to be considered confidential and proprietary and shall not be released or otherwise made available to any third party without the express written consent of GEOTECHNOLOGY.

e. It is intended by the parties to this AGREEMENT that GEOTECHNOLOGY'S services in connection with the project shall not subject GEOTECHNOLOGY'S individual employees, officers or directors to any personal legal exposure for the risks associated with this project. Therefore, and notwithstanding anything to the contrary contained herein, CLIENT agrees that as the CLIENT'S sole and exclusive remedy, any claim, demand or suit shall be directed and/or asserted only against GEOTECHNOLOGY, a Missouri corporation, and CLIENT expressly waives CLIENT's rights against any of GEOTECHNOLOGY'S employees, officers or directors.

2 - STANDARD OF CARE

a. CLIENT recognizes that conditions may vary from those observed at locations where borings, surveys, observations, or explorations are made, and that site conditions may change with time. Data, interpretations, and recommendations by GEOTECHNOLOGY will be based solely on information available to GEOTECHNOLOGY. GEOTECHNOLOGY is responsible for those data, interpretations, and recommendations, but will not be responsible for other parties' interpretations or use of the information developed.

b. GEOTECHNOLOGY offers different levels of services to suit the desires and needs of different clients. Although the possibility of error can never be eliminated, more detailed and extensive services yield more information and reduce the probability of error, but at increased cost. CLIENT has reviewed the scope of services and has determined that it does not need or want a greater level of service than that being provided.

c. The standard of care for all professional engineering and related services performed under this AGREEMENT will be the care and skill ordinarily used by members of the subject profession practicing under similar circumstances at the same time and in the same locality. GEOTECHNOLOGY makes no warranties, express or implied, under this AGREEMENT or otherwise, in connection with any services performed or furnished by GEOTECHNOLOGY.

3 - SITE ACCESS AND SITE CONDITIONS

a. CLIENT will grant or obtain free access to the site for all equipment and personnel necessary for GEOTECHNOLOGY to perform the services set forth in this AGREEMENT. CLIENT will notify any and all possessors of the project site that CLIENT has granted GEOTECHNOLOGY free access to the site.

4 - CHANGED CONDITIONS

a. If, during the course of performance of this AGREEMENT, conditions or circumstances are discovered which were not contemplated by GEOTECHNOLOGY at the commencement of this AGREEMENT, GEOTECHNOLOGY shall notify CLIENT in writing of the newly discovered conditions or circumstances, and CLIENT and GEOTECHNOLOGY shall renegotiate, in good faith, the terms and conditions of this AGREEMENT.

5 - OBSERVATION

a. CLIENT recognizes that unanticipated or changed conditions may be encountered during construction and, principally for this reason, CLIENT shall retain GEOTECHNOLOGY to observe construction when GEOTECHNOLOGY has provided engineering services. CLIENT understands that construction observation is conducted to reduce – not eliminate – the risk of problems arising during construction and that provision of the service does not create a warranty or guarantee of any type. In all cases, contractors shall retain responsibility for the quality and completeness of their work and for adhering to the plans, specifications, and recommendations on which their work is based. Should GEOTECHNOLOGY for any reason not provide construction observation during the implementation of GEOTECHNOLOGY's plans, specifications, and recommendations, or should CLIENT restrict GEOTECHNOLOGY's assignment of observation personnel, CLIENT shall, to the fullest extent permitted by law, waive any claim against GEOTECHNOLOGY, and indemnify, defend, and hold GEOTECHNOLOGY harmless from any claim or liability for injury or loss arising from field problems allegedly caused by findings, conclusions, recommendations, plans, or specifications developed by GEOTECHNOLOGY.

b. If GEOTECHNOLOGY is retained by CLIENT to provide a site representative for the purpose of monitoring specific portions of construction work or other field activities as set forth in the PROPOSAL, then this paragraph applies. For the specified assignment, GEOTECHNOLOGY will report observations and professional opinions to CLIENT. No action of GEOTECHNOLOGY's site representative can be construed as altering any AGREEMENT between CLIENT and others. GEOTECHNOLOGY will report to CLIENT observed conditions related to services for which GEOTECHNOLOGY has been retained to perform which, in GEOTECHNOLOGY's professional opinion, do not conform with plans and specifications. GEOTECHNOLOGY has no right to reject or stop work of any agent of the CLIENT. Such rights are reserved solely for CLIENT. Furthermore, GEOTECHNOLOGY's presence on site does not in any way guarantee the completion or quality of the work of any party retained by CLIENT to provide field or construction-related services.

c. GEOTECHNOLOGY shall not be required to sign any document, no matter by whom requested, that would result in GEOTECHNOLOGY having to certify, guarantee, or warrant the existence of conditions whose existence GEOTECHNOLOGY cannot ascertain. CLIENT agrees not to make resolution of any dispute with GEOTECHNOLOGY or payment of any amount due to GEOTECHNOLOGY in any way contingent upon GEOTECHNOLOGY signing any such document.

d. The use of the word "certify" or "certification" by a registered professional engineer or geologist in the practice of professional engineering constitutes an expression of professional opinion regarding those facts or findings which are the subject of the certification, and does not constitute a warranty or guarantee, either express or implied. The definition and legal effect of any and all certifications shall be limited as stated herein.

6 - JOBSITE

a. Unless specifically set forth in the PROPOSAL, GEOTECHNOLOGY will not be responsible for and will not have control or charge of specific means, methods, techniques, sequences or procedures of construction or other field activities selected by any other person or entity, or safety precautions and programs incident thereto. GEOTECHNOLOGY shall be responsible only for its activities and that of its employees on any site. Neither the professional activities nor the presence of GEOTECHNOLOGY or its employees or its subcontractors on a site shall imply that GEOTECHNOLOGY controls the operations of others, nor shall this be construed to be acceptance by GEOTECHNOLOGY of any responsibility for jobsite safety.

b. Unless indicated otherwise in the PROPOSAL, GEOTECHNOLOGY'S services under this AGREEMENT are limited to geotechnical consulting services and GEOTECHNOLOGY shall have no responsibility to locate, identify, evaluate, treat or otherwise consider or deal with hazardous materials.

c. CLIENT represents that CLIENT has made a reasonable effort to evaluate if hazardous materials are on or near the project site, and that CLIENT has informed GEOTECHNOLOGY of CLIENT's findings relative to the possible presence of such materials.

d. Hazardous materials may exist at a site where there is no reason to believe they could or should be present. GEOTECHNOLOGY and CLIENT agree that the discovery of unanticipated hazardous materials constitutes a changed condition mandating a renegotiation of the scope of work or termination of services. GEOTECHNOLOGY and CLIENT also agree that the discovery of unanticipated hazardous materials may make it necessary for GEOTECHNOLOGY to take immediate measures to protect health and safety. CLIENT agrees to compensate GEOTECHNOLOGY for measures taken to protect health and safety and/or any equipment decontamination or other costs incidental to the discovery of unanticipated hazardous materials.

e. GEOTECHNOLOGY agrees to notify CLIENT when unanticipated hazardous materials or suspected hazardous materials are encountered. CLIENT agrees to make any disclosures required by law to the appropriate governing agencies. CLIENT also agrees to hold GEOTECHNOLOGY harmless for any and all consequences of disclosures made by GEOTECHNOLOGY, which are required by governing law. In the event the project site is not owned by CLIENT, CLIENT recognizes that it is CLIENT's responsibility to inform the property owner of the discovery of unanticipated hazardous materials or suspected hazardous materials.

7 - BILLING AND PAYMENT

a. CLIENT will pay GEOTECHNOLOGY in accordance with the procedures indicated in the PROPOSAL and its attachments. Invoices will be submitted to CLIENT by GEOTECHNOLOGY, and will be due and payable upon presentation. If CLIENT objects to all or any portion of any invoice, CLIENT will so notify GEOTECHNOLOGY in writing within fourteen (14) calendar days of the invoice date, identify the cause of disagreement, and pay when due that portion of the invoice not in dispute. The absence of written notification described above, shall constitute an unqualified acceptance of the invoice amount due and payable, and waiver by CLIENT of all claims with respect thereto.

b. CLIENT recognizes that late payment of invoices results in extra expenses for GEOTECHNOLOGY. GEOTECHNOLOGY retains the right to assess CLIENT interest at the rate of one percent (1%) per month, but not to exceed the maximum rate allowed by law, on invoices which are not paid within thirty (30) days from the date of the invoice. In the event undisputed portions of GEOTECHNOLOGY'S invoices are not paid when due, GEOTECHNOLOGY reserves the right, after seven (7) days prior written notice, to suspend the performance of its services under this AGREEMENT until all past due amounts have been paid in full.

c. If test results that indicate failure of a material to meet the intended specification require retesting of the material after additional work by parties responsible for that material, the cost of retesting will be invoiced to the CLIENT.

d. GEOTECHNOLOGY may elect to adjust its rates under this AGREEMENT to account for changes in overhead rates and salary adjustments no sooner than one year from the date of this AGREEMENT, and no more often than once per year at the end of each subsequent year.

8 - TERMINATION

a. This AGREEMENT may be terminated by either party seven (7) days after written notice in the event of any breach of any provision of this AGREEMENT or in the event of substantial failure of performance by the other party, or if CLIENT suspends the work for more than three (3) months. Both parties shall have the opportunity to initiate a mutually agreeable remedy for failure of performance within fifteen (15) days after notice of termination. In the event of termination, GEOTECHNOLOGY will be paid for services performed prior to the date of termination plus reasonable termination expenses, including, but not limited to the cost of cleanup, demobilization, completing analyses, records, and reports necessary to document job status at the time of termination.

9 - ALLOCATION OF RISK

9.1 LIMITATION OF LIABILITY

a. GEOTECHNOLOGY and CLIENT have evaluated the risks and rewards associated with this project, including GEOTECHNOLOGY'S fee relative to the risks assumed, and agree to allocate certain of the risks, so, to the fullest extent permitted by law, the total aggregate liability of GEOTECHNOLOGY to CLIENT and third parties granted reliance is limited to the greater of $50,000 or GEOTECHNOLOGY'S fee, for any and all injuries, damages, claims, losses, expenses, or claim expenses (including attorney's fees) arising out of GEOTECHNOLOGY'S services or this agreement regardless of cause or causes. Such causes include, but are not limited to, GEOTECHNOLOGY'S negligence, errors, omissions, strict liability, statutory liability, negligent misrepresentation, breach of contract, breach of warranty, or other acts giving rise to liability based on contract, tort or statute. If CLIENT prefers to have higher limits of liability coverage, GEOTECHNOLOGY agrees, upon receipt of CLIENT'S written request at the time of accepting our PROPOSAL, to increase the limits of liability up to a maximum of $1,000,000.00 at an additional cost of 5 percent of our total fee or $1,000.00, whichever is greater.

b. Neither party shall have any liability to the other party for loss of product, loss of profit, loss of use, or any other indirect, incidental, special or consequential damages incurred by the other party.

9.2 INDEMNIFICATION

a. Subject to the provisions of the Limitation of Liability described in 10.1a. above, CLIENT and GEOTECHNOLOGY each agree to indemnify and hold harmless the other party and the other party's officers, directors, partners, employees, and representatives, from and against losses, damages, and judgments, including reasonable attorneys' fees and expenses recoverable under applicable law, but only to the extent they are legally determined to be caused by a negligent act, error, or omission of the indemnifying party or any of the indemnifying party's officers, directors, members, partners, agents, employees, or subconsultants in the performance of services under this AGREEMENT. If claims, losses, damages, and judgments are legally determined to be caused by the joint or concurrent negligence of CLIENT and GEOTECHNOLOGY, they shall be borne by each party in proportion to its negligence.

b. CLIENT shall indemnify and hold harmless GEOTECHNOLOGY, its agents, subcontractors, directors, officers, and employees, from and against any and all claims, suits, liability, damages, injunctive or equitable relief, expenses, including reasonable attorney's fees or other loss arising from damage to subterranean structures or utilities which were not identified or located by CLIENT to GEOTECHNOLOGY in advance of our work or the discovery of unanticipated hazardous materials or suspected hazardous materials, including, but not limited to, any costs created by delay of the project and any costs associated with possible reduction of the property's value.

c. For the purposes of this AGREEMENT only, and except as provided under Paragraph 10.2 (a) above regarding the negligent performance of GEOTECHNOLOGY, CLIENT shall reimburse GEOTECHNOLOGY for or otherwise indemnify, defend, and save GEOTECHNOLOGY, its agents, subcontractors, directors, officers and employees harmless from any and all demands, suits, judgment, expenses, attorney's fees, and losses arising out of or in connection with bodily injury (including death) to persons or damage to property which may arise from the presence or origination of hazardous substances, pollutants, or contaminants on CLIENT'S property, irrespective of whether such materials were generated or introduced before or after execution of this AGREEMENT; provided, however, that nothing hereinabove set forth is intended to shift any responsibility for employee claims that the parties may bear under the Worker's Compensation laws of the state in which the work is to be performed.

d. GEOTECHNOLOGY shall under no circumstances be considered the generator of any hazardous substances, pollutants, or contaminants encountered or handled in the performance of the work. Without contradiction of any assertion by CLIENT or third party liability as described in Paragraph 10.2 (b) above and for the purposes of this AGREEMENT only, it is agreed that any hazardous materials, pollutants, or contaminants generated or encountered in the performance of the work shall be the responsibility of CLIENT.

10 - CONTINUING AGREEMENT

a. The indemnity obligations and limitations of liabilities established throughout this AGREEMENT, regardless of paragraph number, shall survive the assignment, transfer, expiration or termination of this AGREEMENT.

11 - THIRD PARTY RELIANCE UPON REPORTS

a. All Documents are prepared solely for use by CLIENT (and Owner, if applicable) and shall not be provided to any other person or entity without GEOTECHNOLOGY'S written consent. CLIENT shall defend, indemnify and hold harmless GEOTECHNOLOGY, its officers, shareholders and employees, from and against any action or proceeding brought by any person or entity claiming to rely upon information or opinions contained in reports or other documents provided to such person or entity, published, disclosed or referred to without GEOTECHNOLOGY'S written consent.

12 - NON-SOLICITATION OF EMPLOYEES

a. CLIENT recognizes that GEOTECHNOLOGY, as a part of the services covered by this AGREEMENT, may provide one or more of its employees to work with members of CLIENT'S project staff or specifically on a CLIENT'S project. For purposes of this AGREEMENT, an employee of GEOTECHNOLOGY may be a permanent or temporary employee assigned to provide services to CLIENT. CLIENT hereby agrees that CLIENT will not hire, either directly or indirectly, or provide inducement to hire an employee of GEOTECHNOLOGY either as an employee of CLIENT or as an employee of a subcontractor or supplier to CLIENT, such suppliers to include providers of contract labor, during the term of this AGREEMENT and for a period of six months after the termination of this AGREEMENT. Any hiring or inducement to hire any GEOTECHNOLOGY employee during the term of this AGREEMENT and for a period of six months after termination of this AGREEMENT will be subject to a fee equal to 25% of the total fee for services generated by that employee during a nominal 12-month period.

13 - DISPUTES RESOLUTION

a. All claims, disputes, and other matters in controversy between GEOTECHNOLOGY and CLIENT arising out of or in any way related to this AGREEMENT will be submitted to mediation as a condition precedent to litigation. Notwithstanding any other provision of the Agreement, unless prohibited by law, GEOTECHNOLOGY shall have, in addition to any other right or option set forth herein, the right to proceed in creating a lien upon the building or other improvements and upon the real estate on which the building or improvements are situated for the work and labor done and the labor and materials furnished on and to said real estate and to enforce its mechanic's lien pursuant to all rights and remedies available to it under law.

b. If a dispute at law arises from matters related to the services provided under this AGREEMENT and that dispute requires litigation, then:

(1) the claim will be brought and tried in St. Louis County, Missouri and CLIENT waives the right to move the action to any other county or judicial jurisdiction, and

(2) the prevailing party in any arbitration or litigation between GEOTECHNOLOGY and CLIENT shall be entitled to recovery of all reasonable costs incurred, including staff time, court costs, attorneys' fees, expert witness costs, and other claim related expenses. For purposes of this paragraph, a party prevails if (i) the judgment is equal to or in excess of the Plaintiff's last written demand for settlement, the Plaintiff shall also be entitled to recover its costs, expenses and reasonable attorney's fees from Defendant; (ii) the judgment is equal to or less than the Defendant's last written offer of settlement, the Defendant shall be entitled to recover its costs, expenses and reasonable attorney's fees from the Plaintiff; (iii) the judgment is in between the Plaintiff's last written demand for settlement and the Defendant's last offer of settlement, then neither party shall recover any of its costs, expenses or attorney's fees from the other.

14 - GOVERNING LAW AND SURVIVAL

a. The law of the State of Missouri will govern the validity of these TERMS, their interpretation and performance.

b. If any of the provisions contained in this AGREEMENT are held illegal, invalid, or unenforceable, the enforceability of the remaining provisions will not be impaired.

15 - SUCCESSORS AND ASSIGNS

a. This AGREEMENT shall inure to the benefit of and be binding upon the parties hereto and their respective successors and assigns. Neither party may assign its interests herein (unless assignee assumes in writing assignor's obligations hereunder) without the prior written consent of the other party, which consent will not be unreasonably withheld. No assignment shall operate to relieve the assignor of its obligations under the AGREEMENT.

16 - OTHER PROVISIONS

a. It is agreed that this AGREEMENT is entered into by the parties for the sole benefit of the parties to the AGREEMENT, and that nothing in the AGREEMENT shall be construed to create a right or benefit for any third party.

b. Neither party shall hold the other responsible for damages or delay in performance caused by weather and other acts of God, strikes, lockouts, accidents, or other events beyond the reasonable control of the other or the other's employees and agents.

c. The titles used in this AGREEMENT are for general reference only and are not part of the AGREEMENT.

17 - FUTURE SERVICES

a. All future services rendered by GEOTECHNOLOGY at CLIENT'S request for the project described in the PROPOSAL and/or WORK AUTHORIZATION shall be conducted under the terms of this AGREEMENT.

18 - SIGNATURES

a. The parties have read the foregoing, including any attachments thereto, understand completely the terms, and willingly enter into this AGREEMENT that will become effective on the date signed below by CLIENT.

Streets of West Pryor	**Geotechnology, Inc.**
______________________(Signature)	(Signature)
By: ______________________(Print Name)	By: Andrea Prince, PG (Print Name)
Position: ______________________	Position: Senior Project Manager
Date: ______________________	Date: December 2, 2020