FOR REVIEW PURPOSES

NOT TO BE CONSIDERED A FINAL DOCUMENT

REVISED MINE MITIGATION STUDY
MINE FILLING AT PRYOR CROSSING
LEE'S SUMMIT, MISSOURI

Prepared for:

STREETS OF WEST PRYOR, LLC
OVERLAND PARK, KANSAS

Prepared by:

GEOTECHNOLOGY, INC.
OVERLAND PARK, KANSAS

Date:

FEBRUARY 12, 2021

Geotechnology Project No.:

J035637.02

SAFETY
QUALITY
INTEGRITY
PARTNERSHIP
OPPORTUNITY
RESPONSIVENESS

GEOTECHNOLOGY INC
FROM THE GROUND UP

February 12, 2021

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

Re: Revised Mine Mitigation Study
Mine Filling at Pryor Crossing
Lee's Summit, Missouri
Geotechnology Project No. J035637.02

Dear Mr. Olson:

Presented in this revised report are Geotechnology's observations and recommendations regarding the mitigation of the portion of the Union Quarry Mine which underlies the referenced project. It is our understanding this document will be provided to the City of Lee's Summit and a third-party consultant for review.

This report includes three parts: project and background review; review of industry mitigation practices, and the selection of the proposed mitigation practice. In addition, a technical specifications manual for implementation of the chosen methodology is included. Reports by others for the subject area and academic papers pertinent to the project are provided in the Appendices.

Our services were performed in general accordance with Geotechnology's Proposal P035637.02 dated December 19, 2020. We appreciate the opportunity to provide underground services for this project. If you have questions regarding this report, or if we may be of additional service to you, please contact the undersigned.

Respectfully submitted,

GEOTECHNOLOGY, INC.

Andrea Prince, P.G.
Senior Project Manager

Amy Yang
Engineer

ALY/ALP/MHM:aly

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LEE'S SUMMIT, MISSOURI
FEBRUARY 9, 2021 | GEOTECHNOLOGY PROJECT NO. J035637.02

1.0 BACKGROUND AND PROJECT INFORMATION

1.1 Streets of West Pryor Development

Streets of West Pryor, LLC (SWP, LLC) is currently developing a parcel in Lee's Summit, Missouri. The triangular parcel is bounded by NW Lowenstein Drive, NW Pryor Road and Interstate 470 (I-470) as shown in Figure 1. Portions of the property are underlain by mine space. Presently, the Streets of West Pryor development has been limited to the east side of the property in an area not underlain by mine space. It is our understanding SWP, LLC would like to further develop the parcel by constructing the Pryor Crossing subdivision (which consists of single and multifamily residential homes) over the undermined property.

Several reports have been prepared by others over the past 20 years in the vicinity for surface development. A selection of reports written for the subject area and adjacent parcels have been reviewed for the preparation of this report and are included as Appendix A.

1.2 Geologic Conditions

The regional geology generally consists of lower formations of the Kansas City Group, which is characterized by alternating layers of limestone and shale. Geotechnical borings have been performed at multiple locations on the site within the past 17 years. Approximate locations of previous geotechnical borings drilled by others is included as Figure 2. The following stratigraphic descriptions (from the mined unit up to the surface) were excerpted from a URS report[1] for the subject area.

> **Bethany Falls Limestone.** This is the rock unit that has been mined extensively in the Kansas City area. The Bethany Falls is typically a 20 to 25+ foot thick limestone unit that is quite thick bedded in the majority of the lower portion of the unit. The lower 12 +/- feet of this unit is typically mined. The location that normally forms the roof of the mine consists of a thin shale parting that is easy for the drillers to identify. The limestone above this shale seam is often a relatively thick (4 to 6 feet) zone of massive limestone that forms a solid "roof beam" for the mine. The upper 2 to 5 feet of the Bethany Falls consists of a nodular limestone in a greenish gray shale matrix. This zone, called the "peanut rock" by the local miners is a relatively weak zone.

[1] *Lee's Summit, Missouri Mine Evaluation*. Prepared for Lakewood Business Park by URS, dated February 7, 2003.

Galesburg/Stark Shales. The Galesburg Shale transitions from the "peanut rock" and it is often difficult to determine the exact location of this contact. This unit ranges from approximately 3 to 5 feet thick and is often a gray to dark gray clayey material. The overlying Stark Shale is typically a black platy (fissile) shale that ranges from about 2 to 4 feet in thickness. This unit can transmit water horizontally and where fractured, or penetrated by rock bolts drilled into the roof of the mine, can also transmit water vertically. Both the upper and lower contact of this rock unit is quite apparent, due to the unique bedding characteristics.

Winterset Limestone. This unit is one of the thicker limestone units in the Kansas City area. The average thickness ranges from approximately 25 to 35+ feet. The unit consists of two limestone beds, separated by a 1- to 3-foot-thick dark gray shale seam which is located near the central portion. Both upper and lower limestone can contain noduled and seams of blue gray chert.

Fontana Shale. The Fontana Shale is a thin, dark gray clayey unit that ranges in thickness from approximately 1 to 10+ feet. It may have a black zone near the middle, and in some locations may be quite calcareous.

Block Limestone. The Block Limestone is a thin (1 to 7+/- feet thick) limestone which may have a very thin shale seam. The unit may consist of a lower consistent limestone overlain by a variable thickness upper zone. We note that the boring log, discussed below, did not identify this limestone as being present.

Wea Shale. This relatively thick shale was the highest unit identified at the subject site. The Wea Shale can vary from approximately 15 to greater than 30 feet in thickness in the Kansas City area. This shale can vary in color from a light gray to very dark gray to greenish gray, with an occasional maroon zone in the middle portion.

Overburden. A thin veneer of residual clay soils overlies the bedrock units. The thickness of the soils is unknown, but is expected to be relatively thin (5 to 15 feet).

According to the core hole log, the following rock units were present at this location:

Bethany Falls Limestone	Base Elevation 884.7	Thickness 20.5 feet
Stark/Galesburg Shale	Base Elevation 905.2	Thickness 6.2 feet
Winterset Limestone	Base Elevation 911.4	Thickness 36.1 feet
Wea/Fontana Shale	Base Elevation 947.5	Thickness 8.7 feet
Soil Overburden	Base Elevation 956.2	Thickness 10.1 feet

1.3 Mine History

The Bethany Falls Limestone was mined by the Union Quarry company in the subject area starting in 1959. The mine space has been owned and operated by multiple entities during and after completion of underground mining in 1981. For simplicity, the mine space will be herein referred to as the Union Quarry Mine[2]. At completion of underground mining operations, the mine spanned from East Bannister Road to the north to NW Lowenstein Drive. The Union Quarry Mine is bisected by I-470; the mine space will be herein referred to as the north and south sides, respectively. The north and south mine spaces are joined by a series of tunnels (four) which run beneath I-470. Portals were constructed only on the north side of the mine and access to the south side of the mine is from the north side.

The surface over the north and south sides of the mine are currently owned by different parties. Star Excavation operates a quarrying operation on the north side of the mine. The quarrying operation exposed the roof beam on the north side of the mine and excavated portions of the roof beam and pillars for aggregate. The original portal has been excavated and the mine is now accessed through the exposed mine rooms on the edges of the open quarry pit.

Based on reports by others, the north side of the mine appears to have been excavated using random pillar spacing and the pillars have various diameters. The south side was generally mined on a grid pattern and appears to have more uniform pillar dimensions. It is our understanding instabilities reaching the surface have occurred on the north side of the mine. Mine failure does not appear to extend to the ground surface on the south side.

1.4 Mine Failure Modes

Based on our experience with room-and-pillar limestone mines in the Greater Kansas City area, the following issues are common.

Pillar Distress. Good pillar design leaves enough width to transfer the load of the roof beam and overlying rock and soil to the mine floor, while still extracting the maximum amount of aggregate. Pillars become distressed when the present width is too small or too fractured to provide enough strength to carry the load of the roof beam and overburden. Pillar failure generally occurs in two modes:

- Crushing – a pillar with an insufficient diameter may begin to spall (the "sloughing off" of the outermost layer of the pillar) or to fracture vertically. As stresses continue to accumulate the pillar becomes narrower in the middle, taking on an hourglass shape, before finally being crushed under the load of the mine roof and overlying material.

- Punching – results when the pillar detaches from either the floor or the roof. This mode of failure occurs when the pillar is significantly stiffer than the floor or roof. With roof punching, the pillar and underlying floor remain in place while the weaker roof moves downward, likewise with floor punching the roof and pillar together move downward into the softer, weaker floor. Punching is especially a risk when shales in the mine floor or roof swell and/or lose strength due to water infiltration.

[2] Not to be confused with the other Union Quarry mine located in Lenexa, Kansas.

Roof Distress. The two forces of concern in the roof beam span between two pillars are (1) the shear stress that develops between the rock directly above the pillar and the "unsupported" span of the roof beam and (2) the tensile stress experienced by the unsupported beam span as it supports its own weight and a portion of the overburden between the pillars. The ability of the roof to "resist" these stresses depend on the thickness of the roof beam, the presence and condition of the layers comprising the roof beam, the tensile and compressive strength of the rock, and the spacing between pillars. Failure occurs when a change in one or more of these conditions results in a reduction of either the shear or tensile strength. A change in rock strength could be due to one or more of the following:

- Thin beds in the roof beam
- Delamination of the roof beam
- Joints or fractures
- Solutioning of the roof beam
- Weakness in the rock caused by variation in composition, weathering, or swelling in the presence of water
- Increases in the length of the beam span (as a result of further excavation of the mine face or pillar or the loss of a pillar)

Changes to the surface over the mine space (such as development, the placement of fill, significant cuts during grading, or the build-up of either surface or subsurface water) could lead to increased stress and roof failure.

Roof distress can cause joints to open up. Small pockets of rock can break out of the roof. Stress conditions that continue to build can lead to a larger rock fall. Generally, roof failures continue to propagate both laterally and vertically until a more stable stress configuration can be reached, generally an arch. These arches can extend between the pillars and result in the appearance of a dome. This kind of large-scale failure is known as a 'dome-out.' On occasion, when the overlying rock is weak, the dome-out does not resolve the stress condition and failure continues at the precipice of the dome towards the surface. This is called a "chimney" failure and, if the stresses are not resolved, can result in a sinkhole on the surface.

2.0 PRESENT MINE CONDITIONS

2.1 Mine Observation

In late April 2020, a cursory inspection of the mine space was made by Ms. Andrea Prince, R.G. by boat during the lidar survey performed by BHC Rhodes. During this limited mine observation dome-outs were noted in several areas of the mine and the water was observed to be up to 8 feet deep. Water levels could be deeper in other unexplored areas of the mine. Borings performed by others did not note groundwater. A hydrological study of the site has not been performed.

After receiving preliminary survey information from BHC Rhodes, a second reconnaissance of the mine space was conducted. Representatives of Geotechnology were escorted by BHC Rhodes through the mine to confirm the location and extent of dome-outs, as depicted on Figure 3, provided by BHC Rhodes. There are no previously documented surface failures at the site. Surface failures were not observed during Geotechnology's site reconnaissance. Surface failures on the north mine property have been documented. Photographs of the mine entrance and underground space at the time of the second reconnaissance are included in Appendix B.

The mine was accessed through the north side of the mine at the edge of the Star Excavation quarry pit as shown in Photo #1. Star Excavation's quarrying operations revealed pillars are intact where the overlying materials have been removed. The western tunnels, as shown on Figure 3, which were used to enter the south side of the mine, were relatively dry at the time. In general, along the mine face to the west and southwest, the mine floor was relatively dry. Due to deeper water towards the center of the south mine space it was difficult to determine the height of underwater rock piles. The perimeter of the dome-outs is characterized by 4 to 12 inches of roof break-out in the one to three rooms surrounding the dome-outs.

2.2 Stability Analysis

Analyses for stability of the mine space were performed using the assumptions shown in the calculations in Appendix C. Various mine geometry scenarios were considered. Based on the factor of safety (FS) calculations performed, the mine as a FS greater than 2 with the exception of the scenario where the roof beam fails in tension due to the loss of one or more pillars. Based on our calculations, the critical beam span is approximately 50 feet (for FS of 2). Based on our observations of the north and south mine space, pillar loss is unlikely at this site. With the application of engineering controls the surface over the mine is viable for development.

3.0 MINE MITIGATION

3.1 Mine Mitigation Goals

Based on the project requirements, engineering controls will limit both existing and future instability and require minimal maintenance or mine observation. Reinforcement of the mine space, using rock bolts and pillar improvements, would require semi-annual to bi-annual inspection to verify the continued performance of engineering controls and to identify areas which require improvement. We recommend (1) backfilling of the mine area beneath the subject development to limit the need for regular observation and (2) that access to the mine be maintained in the event of future need of observation.

3.2 General Overview of Backfilling

In the case of Kansas City area limestone mines, backfilling has been utilized to limit liability and facilitate redevelopment. Backfilling of portions of mine space has been used to reinforce highways and roadways over mine space. General information regarding mine backfilling methodology is included in Appendix D**.** Supporting documents regarding the use of backfill for limiting surface damage are included in Appendix E.

Backfill is used to translate the load of the roof beam and overburden back down to the mine floor. Complete filling of the mine void laterally and/or vertically is not required in order to achieve effective load distribution. More information about the specific needs of this subject mine space will be discussed in a subsequent section.

Based on our experience in the Kansas City area, backfilling from the mine level using conveyors and dozers is the preferred methodology. However, backfilling is also accomplished from the surface by pumping backfill into the mine space using either hydraulic or pneumatic methods via large diameter holes. This method is commonly referred to as "blind" backfilling. Backfilling materials vary based on the specific engineering needs, material availability, and methodology chosen. Some applications require high strength flowable fill, while other conditions allow the use of waste rock from the mining operation.

3.3 Pryor Crossing Backfilling Challenges

Presented below is a discussion of mine backfilling challenges.

- The only mine opening for the subject property is located on the north side of I-470 in an active quarry.
- The water levels in the mine could be in excess of 8 feet.
- Blind backfilling utilizing hydraulic methods would introduce a substantial volume of water into the mine space. Introduction of additional water and at high volumes could cause water levels to rise such that the mine entrance would become flooded and the mine unpassable. At present, we believe there is enough capacity within the mine space such that the displacement of water by fill is not a concern.
- There is not enough information on pneumatic blind backfilling methods at this time. In addition, backfill suitable for use in pneumatic filling is not economically feasible.
- Traditional blind backfill methodology precludes observation of backfill performance from the underground space. The use of hydraulic or pneumatic methods requires the insertion of a large tube for pumping of material which is difficult to control directionally.

3.4 Non-Traditional Approach

The proposed mine backfilling contractor, Drill Tech Drilling & Shoring, Inc. (Drill Tech), has worked with Ms. Andrea Prince for 15 years. They have many years of experience filling mine spaces across the country. In response to the aforementioned project challenges, Drill Tech has proposed the use of a proprietary device referred to as a "rock slinger". The proposal for the development of the rock slinger and filling of the mine is included in Appendix F. Together, Geotechnology and Drill Tech are proposing the following approach utilizing the rock slinger.

The mine space would be backfilled through approximately 20-inch diameter boreholes drilled into the mine space on a checkerboard pattern across the footprint and extending two rooms laterally beyond the development footprint. The total development area is estimated at approximately 11.2 acres; however, modifications to the mine filling extents will be evaluated based on observed conditions during backfilling operations. A prepared backfill would be placed into the mine space using temporary casing fitted with the rock slinger. The rock slinger, in theory, would place material in a wider arc than could be accomplished via dumping, potentially eliminating the need to drill into every mine room, as illustrated in Figure 4.

An analysis was performed to determine the volume of rubble-debris generated during dome-out failure. In order to self-arrest the dome-out failure and prevent chimney failure, the load of the remaining overburden must be transferred to the mine floor. Based on the assumptions presented in the calculations presented in Appendix G, filling of the mine to within 12 inches of the mine ceiling will self-arrest the dome-out, preventing the formation of a chimney failure.

It should be understood Drill Tech's rock slinger concept has not been used in the Greater Kansas City area and no documentation can be found on similar devices. To finalize the mine filling methodology, we recommend test holes completed with the rock slinger and available prepared aggregate be observed from the mine space to evaluate the geometry of the fill piles.

4.0 QUALITY ASSURANCE/QUALITY CONTROL

Sources regarding industry practice have been included in Appendix H. In general, quality assurance/control measures can be difficult to assign. Recommendations for quality assurance/control measures are outlined below and are included in the *Project Manual* attached as Appendix I.

Surveying. The project surveyor should tie the existing surveys for the surface and subsurface to the same control point. The project surveyor should mark the boring locations as selected by Geotechnology. The boreholes should be numbered and the numbering scheme should be consistent throughout the project.

Borehole Filling. Boreholes should be sized such that temporary casing can be lowered down into the mine. Casing diameter should be selected and the rate of filling controlled such that the pipe does not become clogged with backfill.

Surface Observation. We recommend a representative of Geotechnology be employed on a full-time basis throughout project phases. The representative of Geotechnology should observe the drilling, noting the borehole number, location, date, a general description of the stratigraphy encountered, and the depths of the mine roof and floor.

Aggregate. It is our understanding the prepared backfill is available locally, but could vary in composition. Geotechnology should be notified of upcoming material variations. Samples should be collected from each variable source and tested for gradation. At this time material specifications have not been finalized. Backfill materials should consist of a 2-inch minus, prepared aggregate with less than 20% fines. Organic material should be excluded from the stockpile. Material specifications are subject to change on the basis of the test hole(s) performed and the backfilling operations. Material volumes should be recorded for each borehole and a sample collected for gradation testing. Where samples do not adhere to the gradation requirements, the associated fill pile should be flagged for review.

Mine Filling. The mine space should be filled to within 12 inches of the mine roof, as verified by depth measurements. The base of the fill pile is expected to encompass the entire room filled. A representative of Geotechnology should observe the top of each fill pile with a borehole camera to verify the distance from the mine roof.

Borehole Abandonment. The boreholes should be backfilled to prevent water infiltration into the mine space. The borehole should be plugged with bentonite chips, cement bentonite or bentonite grout, or concrete.

Fill Pile Observation. Each fill pile should be observed from underground. We anticipate observation approximately once every 10 to 15 borehole completions, but more frequent observation will be required during the initial filling. A Geotechnology team should enter the mine space and observe the spread and shape of the fill piles. Use of a boat may be necessary to observe fill piles. Probing by hand at the lateral extents may be required. Piles which have previously been flagged for changes in gradation should be reviewed. Flawed geometry could include piles which have sloughed on one or more sides, or are shorter than recommended. Re-entry to the abandoned boreholes may be required to correct short piles. Additional boreholes between filled rooms may be required if large portions of a fill pile have sloughed or large portions (more than two connecting rooms) are not filled in accordance with specifications.

Reporting. Field notes and laboratory test results shall be shared with the client throughout mine filling operations. Upon completion of the project the quality assurance/quality control documents shall be compiled into a single document for record keeping.

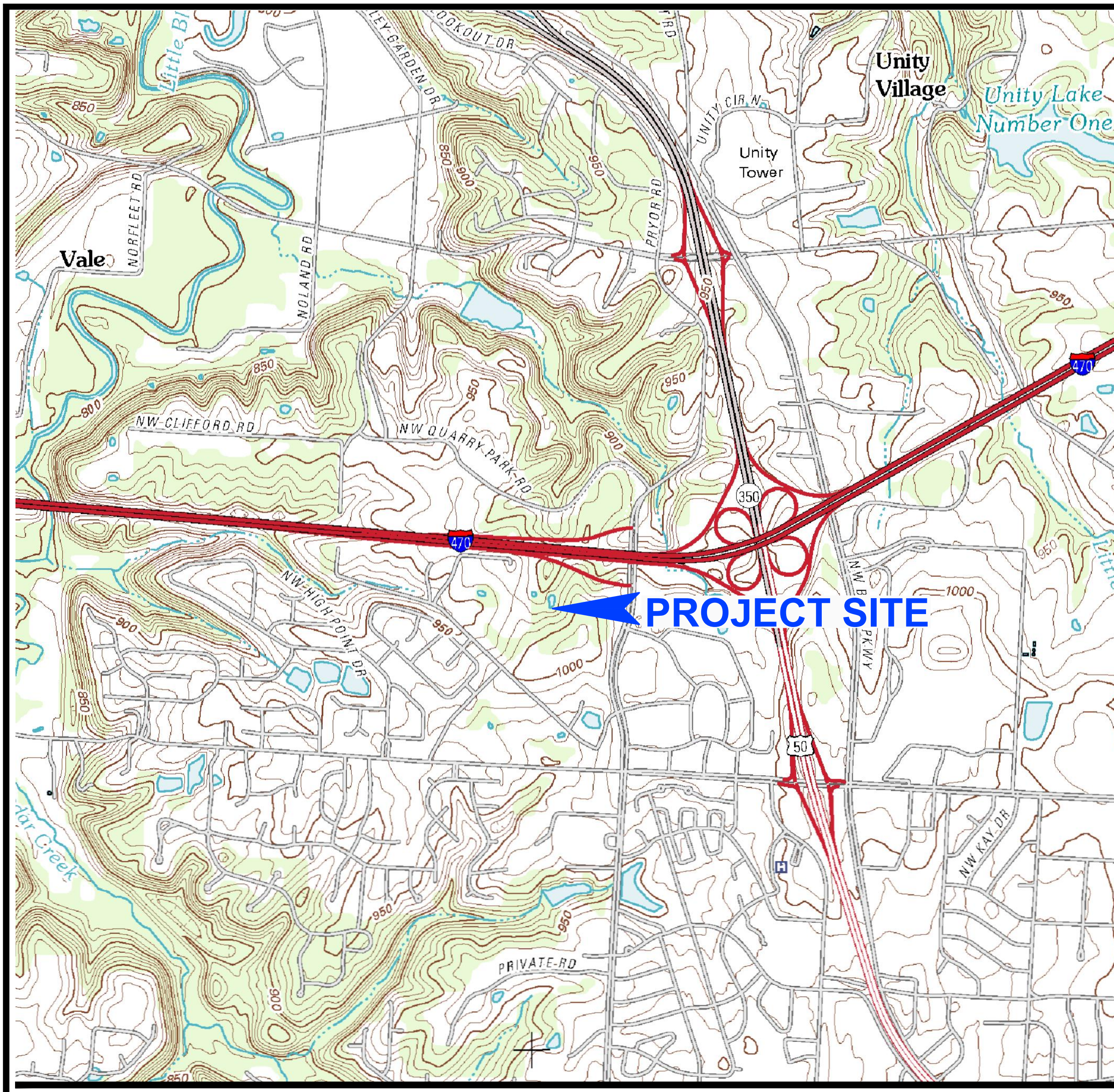
5.0 MINE ACCESS

After backfilling of the mine space beneath the subject developed, we recommend access to the mine space be maintained. While backfilling will have reduced the risks associated with the mine space, access to the mine for observation should be maintained.

6.0 LIMITATIONS

This report has been prepared on behalf of, and for the exclusive use of, the client for specific application to the named project as described herein. If this report is provided to other parties, it should be provided in its entirety with all supplementary information. In addition, the client should make it clear that the information is provided for factual data only, and not as a warranty of subsurface conditions presented in this report.

Geotechnology has attempted to conduct the services reported herein in a manner consistent with that level of care and skill ordinarily exercised by members of the profession currently practicing in the same locality and under similar conditions. The recommendations and conclusions contained in this report are professional opinions. The report is not a bidding document and should not be used for that purpose.



NOTES

1. Plan adapted from 7.5 minute U.S.G.S. maps for Mound City and Dotham, Missouri quadrangles, last revised in 2014.

0 2,000 4,000

SCALE IN FEET

N

Drawn By: ALY	Ck'd By: ALP	App'vd By: MHM
Date: 12-2-20	Date: 12-2-20	Date: 12-18-20

GEOTECHNOLOGY INC
FROM THE GROUND UP

Mine Mitigation Study
Mine Filling at Pryor Crossing
Lee's Summit, Missouri

SITE LOCATION AND TOPOGRAPHY

Project Number J035637.02

FIGURE 1

NOTES

1. Aerial photograph courtesy of Google Earth

2. Boring locations approximated from previous reports prepared by others.

Drawn By: ALY	Ck'd By: ALP	App'vd By: MHM
Date: 12-17-20	Date: 12-18-20	Date: 12-18-20

GEOTECHNOLOGY INC
FROM THE GROUND UP

Mine Mitigation Study
Mine Filling at Pryor Crossing
Lee's Summit, Missouri

AERIAL PHOTOGRAPH OF SITE, LOCATIONS OF BORINGS DRILLED BY OTHERS, AND MINE OUTLINE

Project Number J035367.02	FIGURE 2

MINE COLUMN AND EXTENTS EXHIBIT
PART OF THE SOUTH HALF, SECTION 35, TOWNSHIP 48 NORTH, RANGE 32 WEST, CITY OF LEE'S SUMMIT, JACKSON COUNTY, MISSOURI

Notes by Geotechnology included in blue

GRAPHIC SCALE
100 0 100
1 inch = 100 feet

LEGEND
- Columns
- Wall of Mine
- Dome Out

GENERAL NOTES

1. This is **not** meant to be a geological report or to represent anything other than what we show on the face of this document. Dome out limits are approximate limits based on the fieldwork completed on April 21, 2020. These limits are not meant to indicate areas of the mine that are unstable, only to show areas as indicated. There may be other areas of dome out activity that are not represented on this document.
2. The columns and limits of the mine have been applied an elevation of zero due to a large portion of the mine inundated with water that does not allow for a floor elevation to be obtained with the data collection method used.
3. The aerial image shown on the exhibit is for **informational purposes only.** The image might not represent the actual conditions on the surface. Additional information needs to be obtained to show actual conditions on the surface.
4. The plat boundary show on the exhibit is for **informational purposes only.** This is not a signed and sealed boundary survey.

Rev.	Date	Description	Drawn	Checked

BHC RHODES
Civil Engineering • Surveying • Utilities
7101 College Blvd, Suite 400
Overland Park, Kansas 66210
p. (913) 663-1900 f. (913) 663-1633

Client:
HANEY CO
132ND STREET, SUITE 150,
OVERLAND PARK, KS 66213

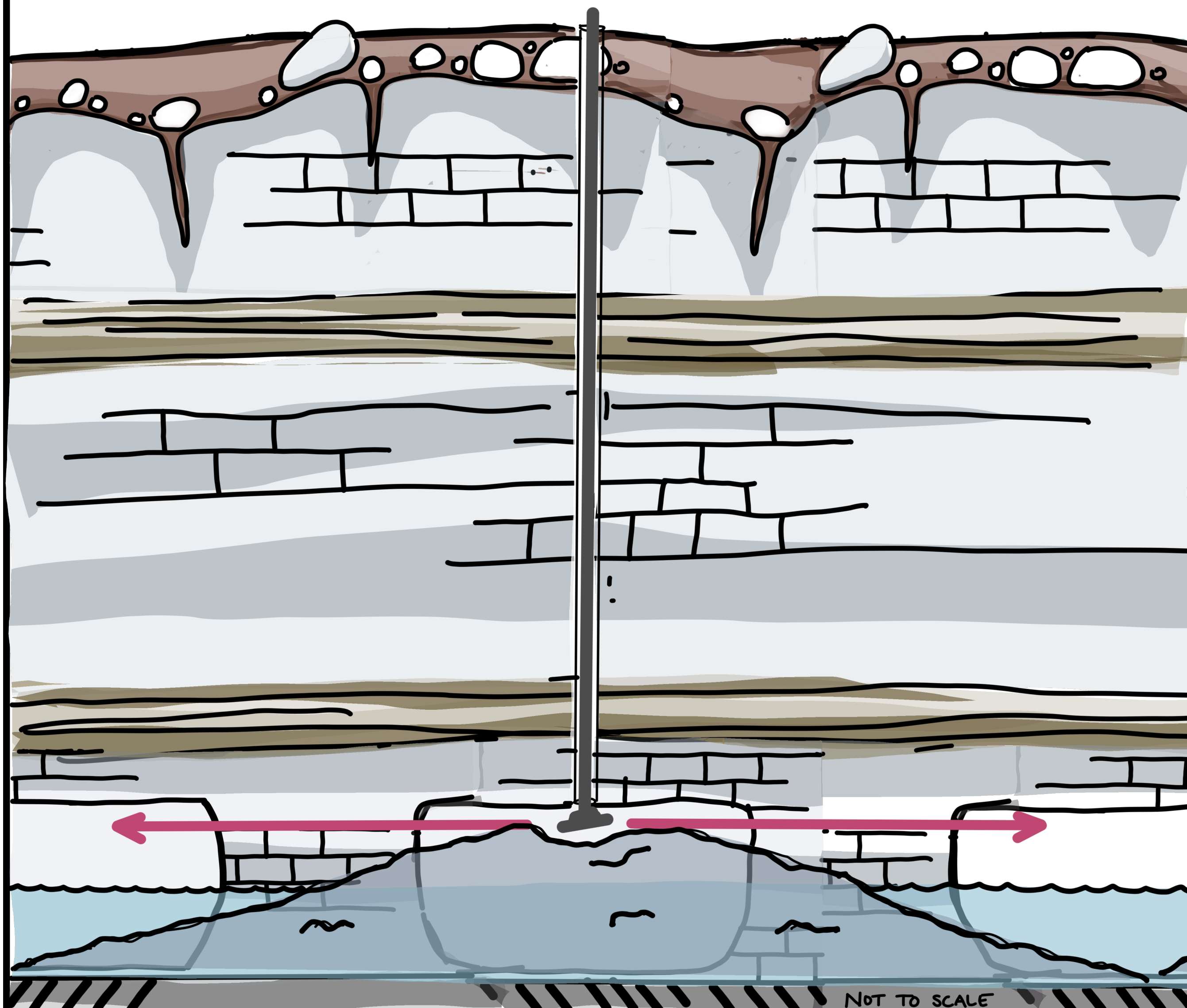
Project:
MINE COLUMN AND EXTENTS EXHIBIT
PART OF THE SOUTH HALF,
SECTION 35,
TOWNSHIP 48 NORTH, RANGE 32 WEST,
LEE'S SUMMIT, JACKSON COUNTY, MISSOURI

Project No:	028440
Field Crew:	MS,TH,ZL
Field Date:	04.21.20
Drawn By:	TAR
Issue Date:	04.27.20

Sheet: 1 OF 1

FIGURE 3

Apr 29, 2020 – 9:00am Plotted by: travis.robertson I:\028440 Streets of West Pryor Mine Scan\DWG\Survey\028440-SRVY-SCAN-XTRT.dwg



Conceptually, the rock slinger will throw material further than could be accomplished by dumping material down the hole.

CONCEPTUAL, PREDICATED ON DRILL TECH DRILLING & SHORING PROPOSAL

Drawn By: ALY	Ck'd By: ALP	App'vd By: MHM
Date: 12-17-20	Date: 12-18-20	Date: 12-18-20

GEOTECHNOLOGY INC
FROM THE GROUND UP

Mine Mitigation Study
Mine Filling at Pryor Crossing
Lee's Summit, Missouri

ROCK SLINGER CONCEPT

Project Number J035637.02	FIGURE 4

APPENDIX A

Reports Prepared by Others

CFS ENGINEERS
cfse.com

GEOTECHNICAL EXPLORATION AND FOUNDATION RECOMMENDATIONS

West Pryor Village
Lee's Summit, Missouri
CFS Project No. 18-5125

Prepared For
Monarch Acquisitions, LLC

June 15, 2018

Prepared by:
Cook, Flatt & Strobel Engineers, P.A.
1100 W. Cambridge Circle Drive Suite 700
Kansas City, Kansas 66103
913.627.9040

SYNOPSIS

An exploration and evaluation of the subsurface conditions have been made on the site of the proposed development located west of Pryor Road in Lee's Summit, Missouri.

Test borings have been drilled and selected soil samples submitted for laboratory tests. The data has been carefully analyzed in light of the project information provided by Monarch Acquisitions, LLC.

The results of the exploration and analysis indicate that conventional spread and continuous wall footings appear to be a suitable type of foundation for the support of the proposed structure. However, when differential settlement is a concern or loading conditions change, alternative foundation approaches may be appropriate.

Detailed analysis of subsurface conditions and pertinent design recommendations are included herein.

Groundwater conditions are not expected to cause any major difficulties. These conditions will be further discussed in the report.

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Geotechnical Exploration And Foundation Recommendations

WEST PRYOR VILLAGE
LEE'S SUMMIT, MISSOURI

Project Number: 18-5125
June 15, 2018

1.0 Introduction

1.1 Authorization

This report presents the results of a geotechnical exploration and foundation analysis for the proposed development, conducted for Monarch Acquisitions, LLC. The work for this project was performed in accordance with our Proposal dated May 17, 2018. Authorization to perform this exploration and analysis was received in the form of a signed copy of that proposal.

1.2 Purpose

The purpose of this exploration was to evaluate the soil and groundwater conditions at the site and to recommend a type and depth of foundation system suitable for the proposed structure as well as to provide criteria for the Architects and Design Engineers to use in preparing the foundation design. Also included were site preparation, earthwork, and pavement design recommendations.

1.3 Scope

The scope of the exploration and analysis included a reconnaissance of the immediate site, the subsurface exploration, field and laboratory testing, and an engineering analysis and evaluation of the foundation materials.

The scope of services did not include any environmental assessment for the presence or absence of wetlands or hazardous or toxic materials in the soil, surface water, groundwater, or air, on or below or around this site. Any statement in this report or on the boring logs regarding odors, colors, or unusual or suspicious items or conditions is strictly for the information of the client.

1.4 General

The general subsurface conditions used in the analysis were based upon interpolation of the subsurface data between the borings. There is a possibility that varying conditions may be encountered between boring locations. If deviations from the noted subsurface conditions are encountered during construction, they should be brought to the attention of the Geotechnical Engineer.

The recommendations submitted for the proposed structure are based on the available soil information and the preliminary design details. Any revision in the plans for the proposed structure, from those described in this report, should be brought to the attention of the Geotechnical Engineer to determine if changes in the foundation recommendations are required.

The Geotechnical Engineer warrants that the findings, recommendations, specifications, and professional advice contained herein have been presented after being prepared in accordance with generally accepted professional engineering practice in the fields of foundation engineering, soil mechanics and engineering geology. No other warranties are implied or expressed.

After the plans and specifications are complete, it is recommended that the Geotechnical Engineer be provided the opportunity to review the final design and specifications, in order to verify that the earthwork and foundation recommendations are properly interpreted and implemented. In particular, treatment of area of fill over 5 feet in depth, deep fills, should be evaluated. Impervious material including shale immediately below footing depth may result in unacceptably wet clay in the bearing zone requiring material be replaced with engineered fill.

2.0 Project Description

It is understood that this project consists of:

1. Two wood framed 4-story slab on grade hotels,
2. One wood framed 4-story slab on grade apartment building,
3. One CMU and Steel framed 60,000 square foot single story grocery store,
4. Several 12,000 or less wood framed single story slab on grade retail buildings,
5. Parking, access roads, and utilities for the above structures.
6. *Potential parking garage which is not included in the recommendations in this report and should be re-evaluated by CFS prior to construction.*

Foundation loads were not provided, but based on the building types and anticipated column spacing, we have assumed that maximum foundation loads will be less than 150 kips for isolated columns and 5 kips per lineal foot (klf) for load bearing walls.

Anticipated site work includes cuts, fills, detention ponds, retaining walls.

2.2 Site Location

The site is located in the southwest quadrant of the intersection of I-470 and NW Pryor Road in Lee's Summit, Missouri. It is located in the southeast corner of Section 35, Township 48N, Range 32W, as described in the PLSS.

2.3 Topography

It is understood that the site is relatively flat. Based on the existing contours shown on the preliminary site plan that was provided to CFS, it is anticipated that only a relatively minor amount of site work will be required in the proposed footprint area of the new building. Finally, it is possible that additional cuts and fill may be required to obtain improved surface drainage.

At the time the borings were performed, the site was a largely vacant, recently cleared tract. The existing ground surface sloped down 40 feet from the center elevation of 1,010 feet to the north and south edges at an approximate 4 percent grade.

2.4 Site Geology

Jackson County is located in the Central Lowland province of the Interior Plains and is near the middle of an approximate 150 mile-wide, north-south trending band of Pennsylvanian-Age Rocks that is located in western Missouri and eastern Kansas. Generally, the rock beds exhibit a subtle prevailing dip to the west-northwest of about 10 feet per mile. The region is underlain by rock units of the Pennsylvanian System, Missourian Series (Kansas City Group, Lansing Group, and Pleasanton Group) in the Time Stratigraphic Unit age classification. The Missouri Geological Survey lists surface material at the project location as residuum from limestone and shale over bedrock chert breccia limestone.

3.0 Site Exploration

3.1 Scope

The field exploration to evaluate the engineering characteristics of the foundation materials included a reconnaissance of the project site, drilling the test borings, performing standard penetration tests, and recovering split barrel samples.

Borings were drilled at 32 locations to depths ranging from three feet to 25 ½ feet below the existing ground surface at locations determined by Cook, Flatt & Strobel Engineers. Boring locations are shown on the accompanying Boring Location Plan (Figure 2, Appendix A).

After completion of the field testing, the excavations were backfilled with the excavated soil.

3.2 Drilling and Sampling Procedures

A drilling rig equipped with a rotary head was used to drill the test borings. Augers were used to advance the holes. Representative samples were obtained employing split-barrel sampling procedures in general accordance with the procedures for "Standard Test Methods for Standard Penetration Test (SPT) and Split-Barrel Sampling of Soils" (ASTM D 1586).

All of the samples recovered were identified, classified, and evaluated by the Geotechnical Engineer.

3.3 Field Tests and Measurements

Penetration Tests – During the soil boring procedure, standard penetration tests (SPT) were performed at pre-determined intervals to obtain the standard penetration value of the soil as outlined in the ASTM D1586 test method. The standard penetration value (N) is defined as the number of blows of a 140-pound hammer, falling thirty (30) inches, required to advance the split-barrel sampler one (1) foot into the soil. The sampler is lowered to the bottom of the previously cleaned drill hole and advanced by blows from the hammer.

The number of blows is recorded for each of three (3) successive increments of six (6) inches penetration. The "N" value is then obtained by adding the second and third incremental numbers. The results of the standard penetration test are shown on the Boring Logs and indicate the relative density of cohesionless soils and comparative consistency of cohesive soils, and thereby provide a basis for estimating the relative strength and compressibility of the soil profile components.

Ground Surface Elevations – The elevation of the ground surface shown on each test boring log was surveyed to within ± one (1) foot.

Boring logs are included in Appendix B. Field logs included visual classification of the materials encountered during drilling, as well as drilling characteristics. Boring logs represent the CFS engineer's interpretation of the field logs combined with laboratory observations and testing of the samples. The stratification boundaries indicated on the boring logs were based on field observations, an extrapolation of information obtained by examining samples from the borings and comparisons of soils and/or bedrock types with similar engineering characteristic. Boundary locations are approximate, and the transitions between soil and bedrock types may be gradual rather than clearly defined.

3.4 Laboratory Testing Program

In addition to the field exploration, a supplemental laboratory testing program was conducted to evaluate additional engineering characteristics of the on-site soils and rock necessary in analyzing the behavior of the foundation systems for the proposed structure.

The laboratory testing program included the following tests:

- Supplementary visual classification (ASTM D2488) of all samples
- Penetrometer tests of all samples
- Water content (ASTM D2216) of all samples
- Atterberg limit tests (ASTM D4318) on selected samples

All phases of the laboratory testing program were conducted in general accordance with applicable ASTM specifications. The results of these tests can be found on the Boring Logs (Appendix B) and in the Laboratory Test Results (Appendix C).

3.5 Subsurface Conditions

3.5.1 General

The types of foundation bearing materials encountered in the test borings have been classified according to the Unified Soil Classification System (USCS). They are described on the Boring Logs. The results of the field tests, water level observations, and laboratory tests are presented on the Boring Logs (Appendix B).

The following presents a general summary of the major strata encountered during our subsurface exploration and includes a discussion of the results of field and laboratory tests conducted. Specific subsurface conditions encountered—including field tests, lab tests, and water level observations—at the boring locations are also presented on the individual boring logs in Appendix B of this report

3.5.2 Overburden Material

A light grass cover and roots up to six (6) inches covered the site. Clay soils generally extend 10 feet below grade transitioning to light brown and gray shale.

3.5.3 Bedrock

Slightly- to un-weathered moderately soft limestone was encountered at the borings below the soil overburden. Based on the published data, in our opinion, the limestone encountered at the borings is the Winterset Limestone of the Dennis Limestone Formation, Kansas City Group, Pennsylvanian System. The Missouri Geological Survey shows bedrock elevations of 840 feet in the project area.

The elevation at the top of limestone was found to vary from 956 to 1,002 feet.

The approximate elevation of the limestone bedrock surface (auger refusal) at boring locations based on elevation data provided by CFS is presented below. The actual elevation could vary over at other locations.

Table 1: Approximate Bedrock Elevation

Boring	Thickness of Overburden Soil (feet)	Approx. Elevation of Limestone Bedrock Surface (feet)
B-1	11	960
B-2	12	957
B-3	15	956
B-4	19	961
B-5	23	961
B-6	24	960
B-7	20	970
B-8	20	965
B-9	16	962
B-10	19	964
B-11	25	966
B-12	19	974
B-13	14	988

B-14	19	975
B-15	19	980
B-16	19	990
B-17	9	1,002
B-18	9	992
B-19	10	1,001
B-20	12	990
B-21	24	962
B-22	18	988
B-23	14	964
B-24	5	984
B-25	19	986
B-26	4	1,000
B-27	15	972
B-28	22	958
B-29	12	987
B-30	13	983
B-31	3	984
B-32	9	956

3.5.4 Groundwater Conditions

Groundwater was not encountered in any of the borings. However, it is not unusual for seasonal groundwater to be encountered on top of the bedrock. The Missouri Geological Survey shows groundwater elevations of 970 feet in the project area.

3.5.5 Refusal Materials

Refusal materials generally consisted of sound limestone.

3.6 Testing Results

Standard Penetration Testing (SPT) was used to evaluate the consistency of the in-situ materials. The N-values for the site's materials were found to range from 3 to over 50 (refusal) blows/foot. Moisture content tests results on the SPT samples ranged from 6 to 33 percent.

Atterberg limits tests were run on three of the samples collected from the SPT sampler. The Plastic Limits were found to range from 20 to 26. The Liquid Limits ranged between 50 and 73, giving the samples Plasticity Indexes ranging between 28 and 47. The cohesive soils were classified using the USCS as CH based on these results.

Representative samples of the soils were placed in sample jars and bags. They are now stored in the laboratory for further analysis if desired. Unless a request to the contrary is received, all samples will be disposed of sixty (60) days from the issuance date of this report.

4.0 Geotechnical Discussion and Recommendations

4.1 Primary Geotechnical Concerns

1. Fat clays with Liquid Limits as high as 70 predominating over the project area are prone to significant variation in volume with changes in moisture and freezing.
2. Shallow rock areas shown in the boring logs may be encountered in excavations.
3. Isolated soft bearing soils may be encountered when excavating footings requiring removal and replacement.

4.2 Foundation Discussion

Subsurface Materials – The bearing capacity of the subsurface materials was evaluated from the results of the field tests. These test results indicate that soils have moderate strength and are uniform in thickness. Beneath the clays, the underlying bedrock limestone has a moderate to high bearing capacity. Project soil conditions and loading are compatible with the use of spread footings.

General – The foundation type selected for the proposed structures is conventional spread and continuous wall footings. Other foundation types including a raft or mat, a grid type beam and slab, and piles are less economical and unnecessary for this project. Conventional spread and continuous wall footings are generally, most economical when the existing soil conditions allow them to be founded at shallow depths. With economy, however, comes a risk of differential settlement that may not be reliably predicted. When elimination of settlement is essential, or loading conditions change, more expensive foundations may be required.

4.3 Foundation Recommendations

4.3.1 Conventional Spread Footings

Conventional spread footings and continuous wall footings are recommended for support of the proposed structure. The conventional spread footings and continuous wall footings should be designed as follows when founded on suitable clay:

Table 1: Conventional Footing Bearing Capacity

Foundation Type	Net Allowable Soil Bearing	Minimum Footing Width	Min. Depth Below Finish Grade for Exterior Ftgs.
Spread footings	3,000 psf	24 inches	36 inches
Continuous wall footings	2,500 psf	16 inches	36 inches

1. Footings should be suitably reinforced to reduce the effects of differential movement that may occur due to variations in the properties of the supporting soils. Top and bottom steel is recommended for continuous wall footings to reduce differential settlement due to possible varying bearing capacities of the existing fill soils. Where footings within a building or fifty (50) feet will bear on both soil and rock, precautions should be taken to control differential settlement. The rock may either be removed then replaced with a closed grade stone (such as MoDOT TYPE 5) or similar compacted soil, transitioning the thickness slowly over at least twenty (20) feet. If compacted structural soil fill is used to back fill the excavation, widening of the excavation one-half (1/2) the depth of the excavation on either side should be performed. All backfill should be compacted to 95% of ASTM D-698 density and +/- 3% of optimum moisture content. As an option, footings may be extended to the underlying rock and backfilled with lean concrete.

2. A representative of the Geotechnical Engineer should test the soils in the footing excavations to verify the design soils bearing pressure. If undercutting of any footing is required to reach design bearing capacity backfill of the undercut footing should be done with a closed grade stone (such as MoDOT TYPE 5 or lean concrete. If compacted structural soil fill is used to back fill the excavation, widening of the excavation one-half (1/2) the depth of the excavation on either side should be performed. All backfill should be compacted to at least 95% of ASTM D-698 density and +/- 3% of optimum moisture content.

3. Every effort should be made to keep the footing excavations dry as the soils will tend to soften when exposed to free water. Footing bottoms should be free of loose soil and concrete should be placed as soon as possible to prevent drying of the foundation soils.

4. Total settlement for the bearing value provided below should not exceed one (1) inch. Differential settlement over fifty (50) feet is estimated to be one-half (1/2) inch or less.

4.4 Lateral Earth Pressures

Lateral earth pressures are determined by multiplying the vertical applied pressure by the appropriate lateral earth pressure coefficient. If the walls are rigidly attached to the structure and not free to rotate or deflect at the top (such as basement walls), CFS recommends designing the walls for the *at-rest* earth pressure coefficient. Walls that are permitted to rotate and deflect at the top (such as retaining walls) can be designed for the *active* lateral earth pressure condition. Horizontal loads acting on shallow foundations are resisted by friction along the foundation base and by *passive* pressure against the footing face that is perpendicular to the line of applied force.

It is recommended that all retaining walls be backfilled with open graded stone (such as No. 57) from two (2) feet behind the wall rising at a 45 degree angle to within two (2) feet of the ground surface. The use of stone to backfill behind the walls will expedite construction, reduce potential settlement, and lower the pressure induced on the wall from the backfill thus potentially reducing the thickness of the walls.

Table 2: Earth Pressure and Friction Coefficients

	Active	Passive	At-Rest	Allowable Base Friction
Open-graded crushed limestone	0.27	3.69	0.43	0.47
In-situ lean clay soils	0.40	2.5	0.68	0.32
In-situ fat clay soils	0.49	2.04	0.66	0.24
Lean clay – conditioned and compacted	0.32	3.12	0.48	0.35
Fat clay – conditioned and compacted	0.45	2.2	0.63	0.27

These earth pressure coefficients do not include the effect of surcharge loads, hydrostatic loading, or a sloping backfill. Nor do they incorporate a factor of safety. Also, these earth pressure coefficients do not account for high lateral pressures that may result from volume changes when expansive clay soils are used as backfill behind walls with unbalanced fill depths. In addition, any disturbed soils that are relied upon to provide some level of passive resistance should be placed in lifts not exceeding six (6) inches in thickness and compacted to a minimum density of 95 percent of the Standard Proctor (ASTM D698) maximum dry density at a moisture content within +- 3 percent of the optimum moisture content. It is recommended that a representative of CFS should verify the compaction of any such materials relied upon to provide passive pressure.

The actual earth pressure on the walls will vary according to material types and backfill materials used and how the backfill is compacted. If the backfill conditions are different than the ones used above, CFS should be notified so the recommendations can be modified. The buildup of water behind a wall will increase the lateral pressure imposed on below-grade walls. Adequate drainage should be provided behind any below grade walls as described in this report. The walls should also be designed for appropriate surcharge pressures such as adjacent traffic, structures, and construction equipment.

Backfill of the below-grade walls may consist of well graded stone or on-site clay soils compacted to at least 95% of optimum dry density at a moisture content within 3% of optimum. We advise performing field density tests on the backfill to monitor compliance with the recommendations provided. Care should be exercised during the backfilling operation to prevent overstressing and damaging the walls. Heavy compactors and grading equipment should not be allowed to operate within 5 to 10 feet of the walls during backfilling to avoid developing excessive temporary or long-term lateral soil pressures.

4.5 Engineering Analysis – Seismic

The Seismic Site Class was determined by the General Procedure in accordance with Section 1613.3.2 of the 2012 International Building Code. The soil properties were evaluated for the top 100 feet of the profile. The generalized profile at this site consists of soil to a depth of ten feet where shale was encountered. The seismic properties of the soil were interpolated from the standard penetration test values. A Seismic Site Class "C" was determined for this site. In addition, there is no significant risk of liquefaction or mass movement of the on-site soils due to a seismic event.

Higher soil shear wave velocities, and therefore a better Site Class, are sometimes obtained by direct testing of the subsurface materials. However, the cost of completing a site-specific seismic study including seismic shear wave velocity testing was beyond the scope of this exploration, however CFS can provide this service if requested.

5.0 Earthwork Discussion and Recommendations

5.1 Discussion – Earthwork

Groundwater – Groundwater that was not encountered in site in borings. Normal seasonal weather conditions should be anticipated and planned for during earthwork. It is recommended that the Contractor determine the actual groundwater levels at the site at the time of the construction activities to assess the impact groundwater may have on construction. Water should not be allowed to collect in the foundation excavation, on floor slab areas, or on prepared subgrades of the construction area either during or after construction. Undercut or excavated areas should be sloped toward one corner to facilitate removal of collected rainwater, groundwater, or surface runoff. Positive site drainage should be provided to reduce infiltration of surface water around the perimeter of the building and beneath the floor slabs. The grades should be sloped away from the building and surface drainage should be collected and discharged such that water is not permitted to infiltrate the backfill and floor slab areas of the building.

Suitable Fill Material – All structural fill should be free of debris and defined by ASTM 2487 as CH, CL, ML, GW, GP, SM, SW, SC, and SP. The onsite soils do meet this requirement; however CH soils should NOT be used as structural fill within two (2) feet of the finished grade under the building slab and five (5) feet outside the building perimeter. Fat clays (CH) with Liquid Limits of greater than 55 should not be used in the upper one (1) foot beneath the pavement without being treated with lime, fly ash or cement as outlined later in this report in Section 6.4.

Depending upon the amount of weather in months preceding construction, it may be necessary to moisture adjust the soils prior to their being able to be properly compacted.

Unsuitable Fill Material – The topsoil contains organic material and is unsuitable for use as structural fill. Unsuitable materials are those defined by ASTM 2487 as MH, OL, OH, and PT.

Retaining Walls, Utility Trenches and Paved Areas – Subsurface materials in the area of retaining walls, utility trenches, and paved areas may encounter varying subsurface conditions which should be revaluated.

5.2 Recommendations – Earthwork

1. The grass and topsoil should be stripped from all structural areas and be stockpiled for later use in landscape areas or be discarded.
2. The surface of the site should be proof compacted to detect and compact any localized soft areas at the surface of the site.
3. Structural fill materials should be free of organic matter. Moisture contents should be within 0% and +4% of the optimum for soils with a liquid limit of greater than 40, and +/-3% of the optimum for soils with a liquid limit of less than 40. Maximum dry density and optimum moisture content should be determined by the Standard Proctor test (ASTM D 698).
4. Fill should be placed in six (6) inch lifts (compacted thickness) in mass fill areas and as needed to obtain proper compaction in utility trenches and behind walls.
5. Structural fill should extend a minimum of five (5) feet outside the building line. The top of slopes should also be a minimum of ten (10) feet outside the building line.
6. The site should be graded such that positive drainage (normally 2% minimum) is provided away from the building.
7. A representative of the Geotechnical Engineer should monitor filling operations. A sufficient number of density tests should be taken to verify that the specified compaction is obtained. See Table below for required testing frequency.

Table 3: Density Testing Frequency

Location or Area	Standard Proctor Density (ASTM D 698)	Testing Frequency One per lift per …
Structures and Walkways	95%	20,000 sf
Retaining Walls	95%	150 lf
Trenches	95%	150 lf
Lawn or Unimproved Areas	92%	20,000 sf
Building and Pavement Subgrades	95%	10,000 sf
Out-Parcels	95%	20,000 sf

5.3 Recommendations – Slab-on-Grade

After completion of earthwork in accordance with Section 5.2 above, a minimum 6-inch thick mat of well-graded crushed stone—equivalent to MoDOT Type 5 or ASTM C-33 No. 57 stone (1"minus)—should be placed beneath the floor slab. The granular layer will ease construction, provide capillary break, aid in drainage, and reduce slab curling due to differential cure. Prior to placement of concrete, the granular material should be compacted to a minimum dry density of 95 percent of the maximum dry density as determined by ASTM D 698 at moisture contents within ±3 percent of the optimum moisture content.

Proof-rolling and/or re-compaction of the subgrade soils should be accomplished just prior to placement of the base stone to identify soft/unstable soils and disturbance from utility excavations. Unsuitable soils should be removed and replaced with engineered controlled fill. Moisture conditioning of the upper soils may also be required prior to placement of the base stone and slab on grade. It is very important that the moisture content of the subgrade soils be maintained until concrete is placed. Rutted subgrade should be repaired prior to placement of base rock to avoid a potential water trap and subsequent subgrade heave.

The upper eighteen (18) inches below the six (6) inch (minimum) stone layer outlined above should consist of low volume change (LVC) material. Including the stone layer, there should be a minimum of 24 inches of LVC below all slabs-on-grade.

Low Volume Change (LVC) material is defined as soils or stone with liquid limits less than 45 and a plasticity index below 25. The on-site soils **do not** meet these requirements. Well graded stone such as MoDOT TYPE 5 or limestone screenings may be used as LVC material. It is not recommended to use limestone screenings in the winter months, as damage may occur from the screenings freezing. "Buckshot" (ag-lime waste byproduct) is not an approved LVC material.

As a substitute to the placement of LVC beneath the slab-on grade, the on-site clays can be mixed and compacted with 13% to 15% by weight class "C" fly-ash or 5% by weight hydrated lime for a minimum depth of eighteen (18) inches provided the minimum layer of stone is also provided. Five percent (5%) by weight Type 1/2 Portland cement for a minimum depth of twelve (12) inches may be substituted for the eighteen (18) inches of fly ash or lime treated or LVC soil with a minimum six (6) inch layer of stone. . Fly-ash should not be used during the winter months, or during any time where the ambient temperature may drop below 40°F before curing of the fly-ash can occur. See Appendix D for guidelines related to fly-ash stabilization of soils.

The LVC material or soil treatment should extend ten (10) feet outside the building walls plus all sidewalks and entries to the building.

Every floor slab-on-ground (on-grade or below-grade) should have a vapor retarder under the concrete that meets the requirements of ASTM E1745, or E1993, installed in accordance with the recommendations of ACI 302.2R. The slab designer should refer to ACI 302 and/or ACI 360 for procedures regarding the use and placement of a vapor retarder.

To reduce the effects of differential movement, slabs-on-grade should not be rigidly connected to columns, walls, or foundations unless it is designed to withstand the additional resultant forces. Floor slabs should not extend beneath exterior doors or over foundation grade beams, unless saw cut at the beam after construction. Expansion joints may be used to allow unrestrained vertical movement of the slabs. The floor slabs should be designed to have an adequate number of joints to reduce cracking resulting from differential movement and shrinkage. We suggest joints be provided on a minimum spacing of twelve (12) feet on center. For additional recommendations refer to the ACI Design Manual. The requirements for the slab reinforcement should be established by the designer based on experience and the intended slab use.

For prepared subgrade as recommended and placed on properly compacted LVC soil and six (6) inch stone base, a modulus of subgrade reaction, k value, of 100 (psi/in) may be used for slab design. If stone is used in the upper 24 inches for the LVC, a modulus of subgrade reaction, k value, of 200 (psi/in) may be used for slab design.

5.4 Excavations and Trenches

All temporary slopes and excavations should conform to Occupational Safety and Health Administration (OSHA) Standards for the Construction Industry (29 CFR Part 1926, Subpart P). Excavations at this site are *expected* to be made in "Type A" clayey soil. Soil types should be verified in the field by a competent individual.

Excavations through the very hard limestone and shale bedrocks may be necessary. The Boring Logs (Appendix B) and the Boring Location Plan (Figure 2, Appendix A) should be consulted in estimating the amount of rock to be excavated.

All excavations should be kept dry during subgrade preparation. Storm water runoff should be controlled and removed to prevent severe erosion of the subgrade and eliminate free standing water. Subgrade that has been rendered unsuitable from erosion or excessive wetting should be removed and replaced with controlled fill.

Trenches should be excavated so that pipes and culverts can be laid straight at uniform grade between the terminal elevations. Trench width should provide adequate working space and sidewall clearances. Trench subgrade should be removed and replaced with controlled fill if found to be wet, soft, loose, or frozen. Trench sub-grades should be compacted above 95% of the maximum dry density in accordance with ASTM D 698 at moisture contents between -3% to +3% of the optimum moisture content.

Granular bedding materials for pipes, such as well-graded sand or gravel, may be used provided that the bottom of the trench is graded so that water flows away from structure

Bedding material should be graded to provide a continuous support beneath all points of the pipe and joints. Embedment material should be deposited and compacted uniformly and simultaneous on each side of the pipe to prevent lateral displacement. Compacted control fill material will be required for the full depth of the trench above the embedment material except in area landscape area with the compaction may be reduced to 90% Standard Proctor ASTM D 698. No backfill should be deposited or compacted in standing water.

Precautions should be taken by the contractor to avoid undermining the newly constructed foundations. Shoring and excavations supports should be designed to account for the existing structure loads.

Permanent slopes greater than 3 horizontal to 1 vertical should not be used.

5.5 Drainage

The site should be graded so that surface water flows away from the building. Where sidewalks or paving do not immediately adjoin the structure, protective slopes of at least 5% for a minimum of 10 feet from the perimeter walls are recommended. Roof drains and downpours should also be directed away from the building. Open-graded stone is not recommended for use under sidewalks unless the stone is adequately drained to prevent collection of water under the walks.

The sites should also be graded to avoid water flows, concentrations, or pools behind retaining walls. If swales are designed at the top of the walls, proper line and slope should be considered to avoid any flow down behind walls. Special attention is needed for sources of storm water from building roofs, gutter downspouts, and paved areas draining to one point.

Perforated plastic pipes should be placed on the backfilled side of the walls near the bottom and day-lighted. Six inches of open graded crushed rock wrapped with geo-textile fabric should be placed behind the walls up to a depth of two feet below the finished grade. As an alternative to the open graded crushed rock, a manufactured geo-composite sheet drain such as Mirafi G100N, Contech C-Drain, or equivalent, may be used in conjunction with the perforated pipe.

5.6 Landscaping

Landscaping and irrigation should be limited adjacent to buildings and pavements to reduce the potential for large moisture changes. Trees and large bushes can develop intricate root systems that can draw moisture from the subgrade, resulting in shrinkage of the bearing material during dry periods of the year. Desiccation of bearing material below foundations may result in foundation settlement.

Landscaped areas near pavements and sidewalks should include a drainage system that prevents over saturation of the subgrade beneath asphalt and concrete surfaces. Drainage systems in irrigation areas should be incorporated into the storm drain system.

6.0 Pavement Recommendations

The American Association of State Highway and Transportation Officials (AASHTO) methods for design of flexible and rigid pavements were adopted in our design analyses. The light duty pavement design for the parking lots was based on the assumed traffic load of 2,000 ESAL's per year for 20 years and 10,000 ESAL's per year for heavy duty pavement.

The pavement subgrade is assumed to be on-site soils consisting of low plasticity clays. The assumed CBR value for the clayey soil that has been used in our design is 3.0. The use and placement of Tensar grid should be in accordance with manufactures specifications.

The performance of pavements is greatly dependent upon proper drainage. Proper sloping of the pavement at ¼ inch per foot or more should be provided.

6.1 Recommended Pavement Thickness

The pavement sections presented below are considered typical and minimum for the report basis parameters. The client should be aware that thinner pavement sections might result in increased maintenance costs and lower than anticipated pavement life. The pavement area subgrade consists of a moisture sensitive soil; yearly maintenance of the pavement will increase the pavement life.

CFS's preferred asphalt option is option 1 or 1A. Due to the moisture sensitivity of the soils and the increase in ESALs, the use of grid provides an increase safety factor from failure. For Light Duty, Option One has ESAL=63,000 and SN=2.83 and Option 3 ESAL=34,000 and SN=2.48. Similarly for Heavy Duty, Option One ESAL=374,000 and SN=3.57 and Option 3 is ESAL=220,000 and SN=3.30. Consideration should also be given to increasing the pavement thickness in the front third of the parking stalls where most of the customers will park.

In areas that will experience heavy parking volumes (typically those near to the building), an increase of one (1) inch of base is recommended (Option 1A). In these areas, the use of a grid provides the greatest benefit. As the pavement depresses due to consolidation, stripping, or densification of the subgrade with time and automobile wheel loads, water starts to stand in the depression. As is stands it will eventually reach the subgrade and weaken the soils. The use of the grid helps reduce the wheel load depressions.

The use of concrete pavement is also recommended for these soils.

Table 4: Light Duty Pavement Thicknesses (Parking lots)

Asphalt Pavement	Option 1	Option 1A	Option 2	Option 3	Option 4
APWA Type 3-01 AC Surface	3.0"	2.0"	2.0"	2.0"	---
APWA Type 1-01 AC Base	---	---	4.0"	---	---
APWA Type 2-01 AC Base	---	2.0"	---	2.0"	---
Tensar Triax TX 5 Geogrid	yes	yes	no	no	no
Concrete	---	---	---	---	5.0"
Gravel Base (MoDOT Type 5 Type 5or equivalent)	6.0"	6.0"	---	6.0"	4.0"

Table 5: Heavy Duty Pavement Thicknesses (Truck areas and drives)

Asphalt Pavement	Option 1	Option 2	Option 3	Option 4
APWA Type 3-01 AC Surface	2.0"	2.0"	2.0"	---
APWA Type 1-01 AC Base	---	6.0"	4.0"	---
APWA Type 2-01 AC Base	3.0"	---	---	---
Tensar Triax TX 5 Geogrid	yes	no	no	no
Concrete	---	---	---	7.0"
Gravel Base (MoDOT Type 5 Type 5 or equivalent)	6.0"	---	6.0"	4.0"

Note: If base is to be placed in the fall and surface in the spring, then APWA Type 2-01 is recommended to improve performance of base due to lower permeability.

6.2 Asphalt Pavement Construction

The granular base course should be built at least 2 feet wider than the pavement on each side to support the tracks of the slip form paver. This extra width is structurally beneficial for wheel loads applied at pavement edge.

Asphalt cement (bitumen) used in the manufacture of asphalt pavement should conform to the Performance Grading system. In the project area, the provincial grade asphalt binder course is PG 64-22. The asphaltic mix for conventional roadway should be designed for 4% air voids. During production, the voids can be expected to vary ±1% of the design value of 4%. Under these conditions, the minimum allowable VMA for base and surface course shall be 12% and 14%, respectively.

Immediately after spreading, each course of the pavement mixture should be compacted by rolling. The initial or "breakdown" rolling shall be accomplished with a steel-wheeled vibratory roller. The motion of the roller should be slow enough at all times to avoid displacement of the hot mixture. The surface of the mixture after compaction should be smooth and true to established section and grade. The completed asphalt concrete paving should have a density equal to or greater than 95% for the base and 96% for the surface of theoretical density.

All asphaltic concrete mix designs and Marshall Characteristics should be submitted to our office and reviewed in order to determine if they are consistent with the recommendations given in this report. All materials to be employed and field operations required in connection with the pavement reconstruction should follow requirements and procedural details as per APWA 2001. In addition, representative of CFS should observe and monitor the pavement construction to assure satisfactory compliance with our engineering recommendations.

6.3 Concrete Pavement Construction

The pavement on this site will be subjected to freeze-thaw cycles. Sufficient air entrainment in the range of 6% to 8% is required to provide freeze-thaw durability in the concrete. Concrete with a 28-day specified compressive strength of 4,000 psi is recommended. The concrete mix should contain at least 564 pounds of concrete per cubic yard. A mixture with a maximum slump of 4 inch +/- 1inch is acceptable. If a water-reducing admixture is specified, slump can be higher. For better performance and crack control, synthetic fiber reinforcement such as Fibermesh® 300 is recommended for the concrete instead of welded wire mesh. Add synthetic fiber reinforcement to concrete mixture in accordance with manufacturer's instructions.

6.4 Pavement Subgrade Preparation

Prior to placement of granular base or asphalt, proof roll and re-compact the exposed surfaces up to a minimum lateral distance of two (2) feet outside the pavement Any localized soft, wet, or loose areas identified during the proof rolling should be repaired prior to paving. Fill material should be placed in loose lifts up to a maximum of eight (8) inches in thickness and compacted to at least 95% of the maximum dry density in accordance with ASTM D698 at moisture contents outlined in the Earthwork section. Construction traffic, including foot traffic, should be minimized to prevent unnecessary disturbance of the pavement subgrade. Disturbed areas, as verified by CFS's geotechnical engineer, should be removed and replaced with properly compacted material.

Fat clays (CH) with Liquid Limits of greater than 55 should not be used in the upper one (1) foot beneath the pavement without being treated with a nine (9) inch layer of lime, fly ash or cement as outlined previously in Section 5.3 in this report. Consideration should be given to treating all non-LVC clays so as to extend the life of the pavement, improve performance and reduce maintenance costs.

The granular base should be placed in loose lifts up to a maximum of twelve (12) inches in thickness and compacted to at least 98% of the maximum dry density in accordance with ASTM D698.

If open graded stone is used under the pavement, the pavement subgrade should be graded to provide positive drainage of the granular base section. Provision should be made to provide drainage into the storm water system. The use of a granular blanket drain near storm water inlets that provides weep holes from the drain to the inlets is recommended.

7.0 General Comments

When the plans and specifications are complete, or if significant changes are made in the character or location of the proposed structure, a consultation should be arranged to review the changes with respect to the prevailing soil conditions. At that time it may be necessary to submit supplementary recommendations.

It is recommended that the services of Cook, Flatt & Strobel Engineers be engaged to test and evaluate the compaction of any additional fill materials and to test and evaluate the bearing value of the soils in the footing excavations.

Respectfully submitted,

COOK, FLATT & STROBEL ENGINEERS, P.A.

William J. Stafford

STATE OF MISSOURI
WILLIAM
J. STAFFORD
NUMBER
E-29183
6/15/18
REGISTERED PROFESSIONAL ENGINEER

William J. Stafford, P.E.
Senior Geotechnical Engineer

Patrick Doyle, P.E.
Project Engineer

Appendix A: Figures

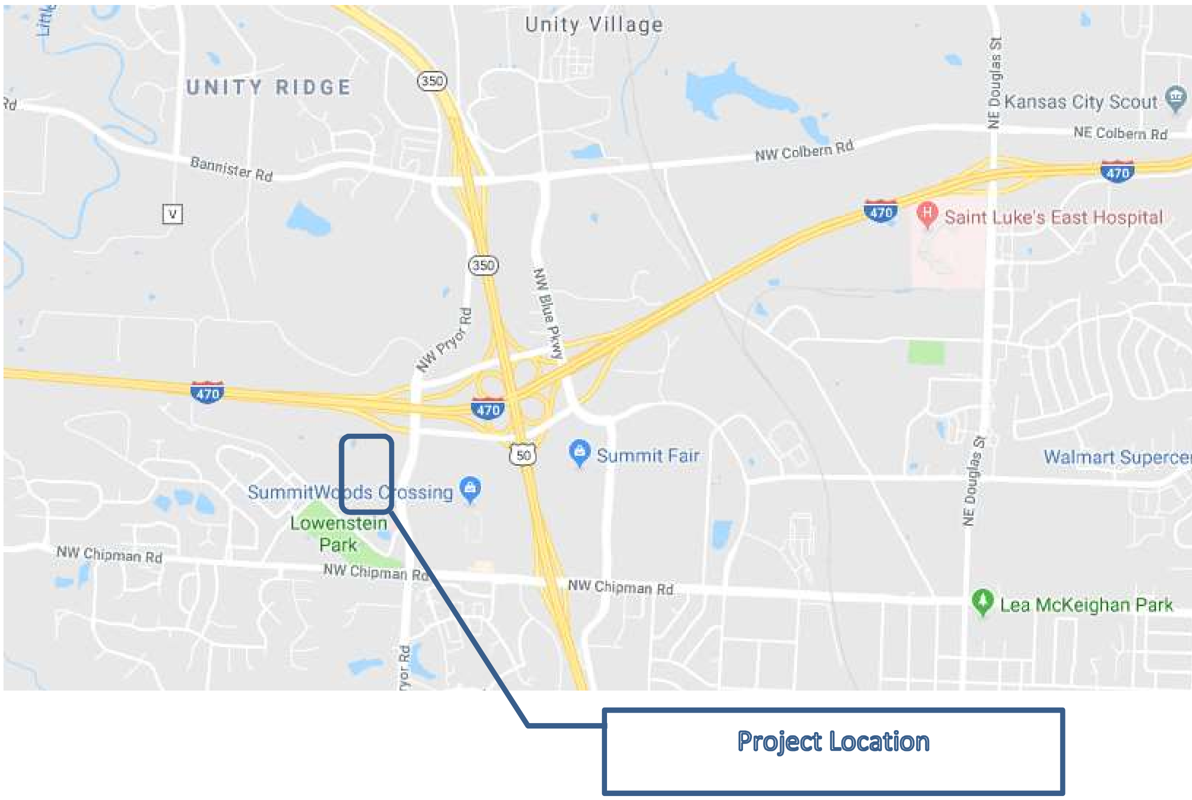

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Figure 1. Site Location Map
West Pryor Village
Lee's Summit, Missouri

Drawn By: PD

Project #: 18-5125

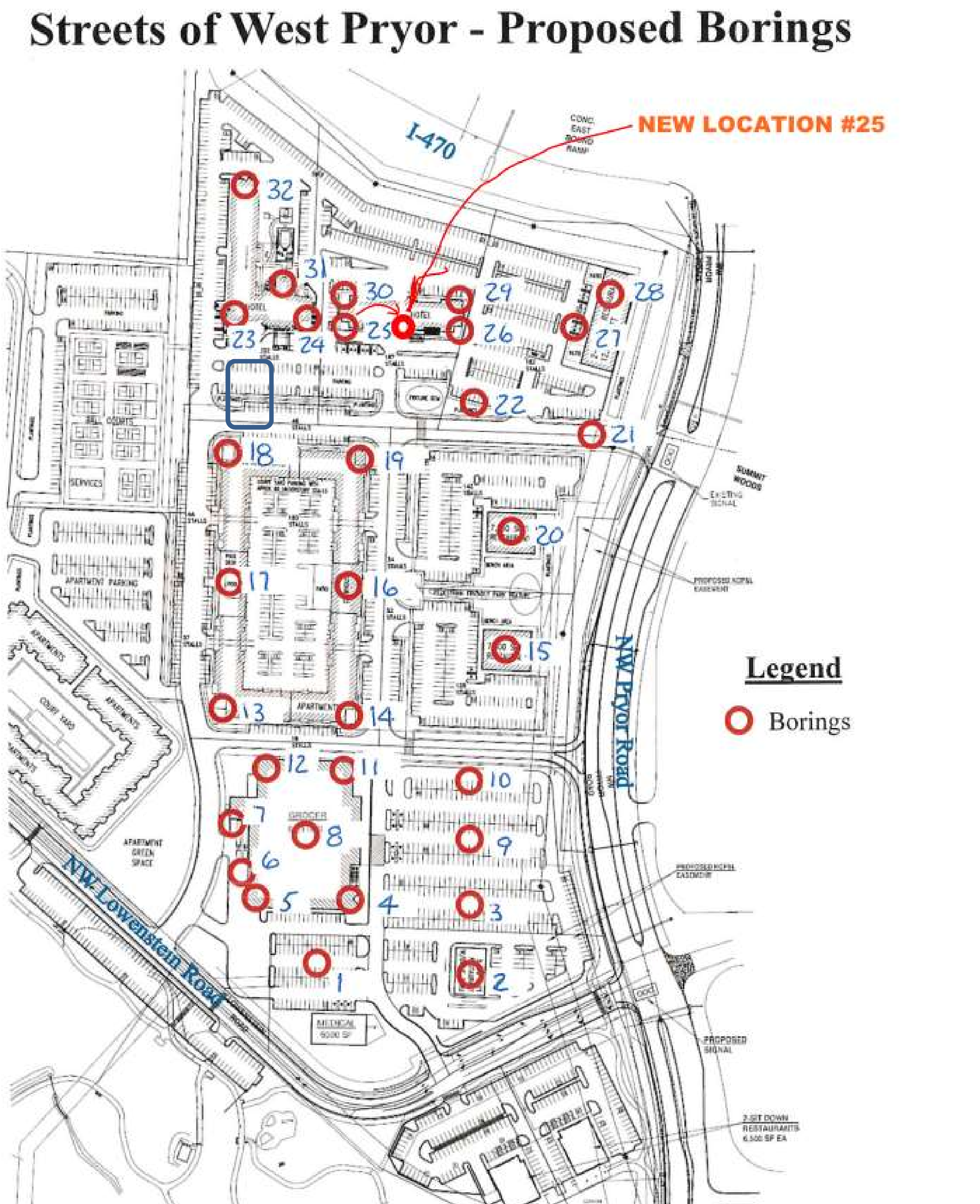

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Figure 2. Boring Location Map
West Pryor Village
Lee's Summit, Missouri
Drawn By: PD
Project #: 18-5125

Appendix B: Boring Logs

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BORING NUMBER B-1

CLIENT Monarch Acquisitions LLC

PROJECT NAME West Pryor Village

PROJECT NUMBER 18-5125

PROJECT LOCATION Lee's Summit, MO

DATE STARTED 6/6/18 **COMPLETED** 6/6/18

GROUND ELEVATION 970.75 ft MSL **HOLE SIZE** 6

DRILLING CONTRACTOR RC Drilling

GROUND WATER LEVELS:

DRILLING METHOD HSA

AT TIME OF DRILLING ---

LOGGED BY Luke **CHECKED BY** HM

AT END OF DRILLING ---

NOTES

AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 970.3
						Brown moist firm CLAY
	SS 1	89	2-2-3 (5)	PP = 2 tsf MC = 28.8%		
5	SS 2	89	3-4-5 (9)	PP = 2.25 tsf MC = 24.6%		
	SS 3	100	4-4-5 (9)	PP = 2.75 tsf MC = 26.5%		
10	SS 4	100	3-4-6 (10)	PP = 2.75 tsf MC = 25.9%		10.0 960.8
						10.5 Grey moist Firm Weathered LIMESTONE 960.3

Auger Refusal at 10.5

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BORING NUMBER B-2

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/7/18 **COMPLETED** 6/7/18
GROUND ELEVATION 968.6 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 968.1
						Brown moist firm CLAY
	SS 1	56	3-2-2 (4)	PP = 3.75 tsf MC = 22.2%		
5	SS 2	67	2-3-3 (6)	PP = 2 tsf MC = 30%		
						6.0 962.6
						Brown moist stiff CLAY
	SS 3	100	8-3-4 (7)	PP = 3.5 tsf MC = 24.9%		
10	SS 4	100	3-4-7 (11)	PP = 3 tsf MC = 32.1%		
						10.5 958.1
						Grey moist Firm Weathered LIMESTONE
						11.2 957.4
						LIMESTONE grey moderately soft rock
						11.7 956.9

Auger Refusal at 11.7 feet

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BORING NUMBER B-3

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/7/18 **COMPLETED** 6/7/18
GROUND ELEVATION 971.7 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 971.2
						Brown moist firm CLAY
	SS 1	56	5-3-2 (5)	PP >4.5 tsf MC = 20%		
	SS 2	67	3-3-3 (6)	PP = 4 tsf MC = 23%		
5						5.0 966.7
						Brown moist stiff CLAY
	SS 3	100	4-4-5 (9)	PP = 3.25 tsf MC = 24.6%		
						8.5 963.2
	SS 4	100	5-29-10 (39)	PP = 2.25 tsf MC = 30.8%		9.0 Brown moist hard CLAY 962.7
10						Grey moist Firm Weathered LIMESTONE
						12.0 959.7
						Grey moist soft SHALE
15	SS 5	78	20-21-24 (45)	PP >4.5 tsf MC = 16%		14.7 957.0
						15.3 LIMESTONE grey moderately soft 956.4

Auger refusal at 15.3

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BORING NUMBER B-4

CLIENT Monarch Acquisitions LLC

PROJECT NAME West Pryor Village

PROJECT NUMBER 18-5125

PROJECT LOCATION Lee's Summit, MO

DATE STARTED 6/6/18 **COMPLETED** 6/6/18

GROUND ELEVATION 979.29 ft MSL **HOLE SIZE** 6

DRILLING CONTRACTOR RC Drilling

GROUND WATER LEVELS:

DRILLING METHOD HSA

AT TIME OF DRILLING ---

LOGGED BY Luke **CHECKED BY** HM

AT END OF DRILLING ---

NOTES

AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL 978.8
	SS 1	33	5-4-3 (7)	PP = 4.5 tsf MC = 20.2%		Brown moist stiff CLAY
5	SS 2	78	4-3-3 (6)	PP >4.5 tsf MC = 19.5%		Brown moist firm CLAY
	SS 3	89	3-3-6 (9)	PP = 2.25 tsf MC = 29.3%		Brown moist stiff CLAY
10	SS 4	100	3-4-5 (9)	PP = 2.25 tsf MC = 25.9%		
15	SS 5	100	4-4-6 (10)	PP = 2.5 tsf MC = 26.2%		
					18.0	961.3
					18.5	LIMESTONE grey moderately soft 960.8

Auger refusal at 18.5

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BORING NUMBER B-5

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/7/18 **COMPLETED** 6/7/18
GROUND ELEVATION 984.04 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 983.5
						Brown moist stiff CLAY
	SS 1	56	4-4-5 (9)	PP = 4.5 tsf MC = 20.2%		
5	SS 2	56	5-5-5 (10)	PP = 4 tsf MC = 22%		
	SS 3	78	4-5-6 (11)	PP = 3.25 tsf MC = 24.4%		
10	SS 4	100	3-3-5 (8)	PP = 2.5 tsf MC = 26.3%		
						Brown moist very stiff CLAY
15	SS 5	100	4-5-8 (13)	PP = 3.5 tsf MC = 27.4%		
20	SS 6	100	6-6-8 (14)	PP = 3.5 tsf MC = 23.8%		
						21.1 LIMESTONE grey moderately soft 962.9
						22.8 961.2

Auger refusal at 22.8

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BORING NUMBER B-6

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/7/18 **COMPLETED** 6/7/18
GROUND ELEVATION 984.34 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						
					0.5	TOPSOIL 983.8
	SS 1	56	5-5-6 (11)	PP >4.5 tsf MC = 23%		Brown moist stiff CLAY
5	SS 2	67	6-5-5 (10)	PP = 4.25 tsf MC = 25%		
	SS 3	100	4-6-7 (13)	PP = 3.5 tsf MC = 22.7%		Brown moist very stiff CLAY
10	SS 4	100	5-7-8 (15)	PP = 4 tsf MC = 24.7%		
					13.5	970.8
15	SS 5	100	10-14-15 (29)	PP >4.5 tsf MC = 20.3%		Grey moist soft SHALE
20	SS 6	100	20-25-34 (59)	PP >4.5 tsf MC = 19.2%		
					23.8	960.5
	SS 7	100	23-50/1"	MC = 16.2%	24.2	LIMESTONE grey moderately soft 960.1
						Auger refusal at 24.2

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BORING NUMBER B-7

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/6/18 **COMPLETED** 6/6/18
GROUND ELEVATION 989.7 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL 989.2
	SS 1	67	3-3-3 (6)	PP = 4 tsf MC = 22%		Brown moist firm CLAY
						Brown moist stiff CLAY
5	SS 2	89	3-4-4 (8)	PP = 4.25 tsf MC = 22.1%		
	SS 3	100	3-5-6 (11)	PP = 2.25 tsf MC = 24.9%		
10	SS 4	100	5-6-8 (14)	PP = 3 tsf MC = 20.8%	10.0	Brown moist very stiff CLAY 979.7
						Grey moist soft SHALE
15	SS 5	100	18-27-50 (77)	PP >4.5 tsf MC = 16.6%		
	SS 6	100	24-50/6"	PP >4.5 tsf MC = 17.6%	19.5	970.2

Spoon refusal at 19.5

BORING NUMBER B-8

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CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/6/18 **COMPLETED** 6/6/18
GROUND ELEVATION 984.64 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL (984.1)
						Brown moist stiff CLAY
	SS 1	44	5-5-5 (10)	PP >4.5 tsf MC = 33.2%		
5	SS 2	78	4-5-7 (12)	PP = 4.5 tsf MC = 23.5%		
	SS 3	100	5-4-7 (11)	PP = 3 tsf MC = 23%		
					9.0	(975.6)
10	SS 4	100	5-6-8 (14)	PP = 3.25 tsf MC = 24.9%		Grey moist soft SHALE
15	SS 5	100	14-22-50 (72)	PP >4.5 tsf MC = 16.8%		
20	SS 6	100	23-26-50 (76)	PP >4.5 tsf MC = 15.7%	20.0	(964.6)

Boring terminated at 20 ft

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BORING NUMBER B-9

CLIENT Monarch Acquisitions LLC

PROJECT NAME West Pryor Village

PROJECT NUMBER 18-5125

PROJECT LOCATION Lee's Summit, MO

DATE STARTED 6/6/18 **COMPLETED** 6/6/18

GROUND ELEVATION 977.32 ft MSL **HOLE SIZE** 6

DRILLING CONTRACTOR RC Drilling

GROUND WATER LEVELS:

DRILLING METHOD HSA

AT TIME OF DRILLING ---

LOGGED BY Luke **CHECKED BY** HM

AT END OF DRILLING ---

NOTES

AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	976.8
	SS 1	44	3-3-2 (5)	PP = 3 tsf MC = 27.3%		Brown moist firm CLAY	
						Brown moist stiff CLAY	
5	SS 2	89	3-4-5 (9)	PP = 3.5 tsf MC = 23.9%			
	SS 3	100	4-5-5 (10)	PP = 4.25 tsf MC = 24.2%			
10	SS 4	100	5-5-6 (11)	PP >4.5 tsf MC = 23.7%		Grey moist soft SHALE	
					13.5		963.8
	SS 5	100	6-50/4"	PP >4.5 tsf MC = 21%	14.0	Grey moist soft SHALE	963.3
15						Grey moist Firm Weathered LIMESTONE	
					15.7		961.6

Auger refusal at 15.7

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BORING NUMBER B-10

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/1/18 **COMPLETED** 6/1/18
GROUND ELEVATION 982.85 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 982.4
						Brown moist very stiff CLAY
	SS 1	67	3-5-8 (13)	PP >4.5 tsf MC = 19.6%		
5	SS 2	89	7-9-9 (18)	PP >4.5 tsf MC = 19.3% LL = 55 PL = 20		
	SS 3	100	5-7-7 (14)	PP >4.5 tsf MC = 21.9%		
10	SS 4	100	7-6-8 (14)	PP >4.5 tsf MC = 22.3%		
						12.0 970.9
						Grey moist soft SHALE
15	SS 5	33	17-19-50 (69)	PP >4.5 tsf MC = 14.5%		
						18.5 964.4
	SS 6	100	50/4"	PP = 3.5 tsf MC = 15.3%		18.8 LIMESTONE grey moderately soft 964.0
						Spoon refusal at 18.83

GENERAL BH / TP / WELL - GINT STD US.GDT - 6/14/18 16:32 - C:\USERS\PAT\~CFSE\WORK\PROJECT\185125 PRYOR CROSSING, LEE_S SUMMIT\PRYOR.GPJ

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BORING NUMBER B-11

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/1/18 **COMPLETED** 6/1/18
GROUND ELEVATION 990.51 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 990.0
						Brown moist stiff CLAY
	SS 1	78	3-4-4 (8)	PP = 3 tsf MC = 28.7%		
5	SS 2	100	3-4-6 (10)	PP = 2.5 tsf MC = 26.8%		
	SS 3	100	4-5-7 (12)	PP = 3.5 tsf MC = 24.4%		
						8.5 982.0
						Brown moist very stiff CLAY
10	SS 4	100	5-8-9 (17)	PP >4.5 tsf MC = 25.1%		9.3 981.3
						Grey moist soft SHALE
15	SS 5	100	11-13-17 (30)	PP >4.5 tsf MC = 17.1%		
20	SS 6	100	19-27-50/4"	PP >4.5 tsf MC = 16%		
	SS 7	50	30-50/6"	MC = 11.6%		24.5 966.0
						Spoon refusal at 24.5

(Continued Next Page)

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BORING NUMBER B-11

CLIENT Monarch Acquisitions LLC

PROJECT NAME West Pryor Village

PROJECT NUMBER 18-5125

PROJECT LOCATION Lee's Summit, MO

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION

BORING NUMBER B-12

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CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/5/18 **COMPLETED** 6/5/18
GROUND ELEVATION 992.87 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 992.4
						Brown moist stiff CLAY
	SS 1	89	4-5-6 (11)	PP >4.5 tsf MC = 21.6%		
5	SS 2	100	5-6-7 (13)	PP >4.5 tsf MC = 17.5%		5.5 987.4
						Grey moist soft SHALE
	SS 3	67	7-8-8 (16)	PP >4.5 tsf MC = 16.8%		
10	SS 4	78	9-9-11 (20)	PP >4.5 tsf MC = 17%		
15	SS 5	100	24-27-50 (77)	PP >4.5 tsf MC = 13.3%		
	SS 6	100	50/6"	PP >4.5 tsf MC = 11.2%		19.0 973.9
						Spoon refusal at 19

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BORING NUMBER B-13

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/1/18 **COMPLETED** 6/1/18
GROUND ELEVATION 1002 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	1001.5
						Brown moist stiff CLAY	
	SS 1	67	3-3-4 (7)	PP = 3.5 tsf MC = 30.2%			
5	SS 2	100	4-6-6 (12)	PP = 3.5 tsf MC = 24.4%			
	SS 3	100	3-4-5 (9)	PP = 3.5 tsf MC = 25.7% LL = 73 PL = 26			
					8.5	Grey moist soft SHALE	993.5
10	SS 4	67	7-7-6 (13)	PP = 3.25 tsf MC = 16.8%			
					13.2		988.8
					13.7	LIMESTONE grey moderately soft	988.3

Auger refusal at 13.7

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BORING NUMBER B-14

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/1/18 **COMPLETED** 6/1/18
GROUND ELEVATION 993.91 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL 993.4
	SS 1	67	5-6-7 (13)	PP >4.5 tsf MC = 20.1%		Brown moist very stiff CLAY
					3.5	990.4
5	SS 2	78	5-6-6 (12)	PP >4.5 tsf MC = 20.2%		Brown moist stiff CLAY
					6.0	987.9
	SS 3	100	7-7-12 (19)	PP >4.5 tsf MC = 20.8%		Grey moist soft SHALE
10	SS 4	89	7-8-9 (17)	PP >4.5 tsf MC = 19.3%		
15	SS 5	100	17-25-50 (75)	PP >4.5 tsf MC = 16.8%		
	SS 6	100	50/6"	PP >4.5 tsf MC = 10.8%	19.0	974.9
						Spoon refusal at 19

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BORING NUMBER B-15

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/31/18 **COMPLETED** 5/31/18
GROUND ELEVATION 998.59 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 998.1
						Brown moist stiff CLAY
	SS 1	78	3-4-6 (10)	PP = 2.5 tsf MC = 28%		
5	SS 2	100	3-5-4 (9)	PP = 2.25 tsf MC = 26.2%		
	SS 3	100	4-5-6 (11)	PP = 2 tsf MC = 24.4%		
						8.5 990.1
						Brown moist very stiff CLAY
10	SS 4	78	6-9-5 (14)	PP = 3.5 tsf MC = 25.7%		9.3 989.3
						Grey moist soft SHALE
15	SS 5	100	50/5"	PP >4.5 tsf MC = 10.3%		
	SS 6	100	50/3"	PP >4.5 tsf MC = 10.3%		18.8 979.8
						Auger refusal at 18.75

GENERAL BH / TP / WELL - GINT STD US.GDT - 6/14/18 16:32 - C:\USERS\PATI~CFSE\WORK\PROJECT\185125 PRYOR CROSSING, LEE_S SUMMIT\PRYOR.GPJ

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BORING NUMBER B-16

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/4/18 **COMPLETED** 6/4/18
GROUND ELEVATION 1008.67 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 1008.2
						Brown moist stiff CLAY
	SS 1	78	4-5-6 (11)	PP = 2.5 tsf MC = 31.9%		
5	SS 2	100	4-5-7 (12)	PP = 3 tsf MC = 22%		5.5 1003.2
						Grey moist Firm Weathered LIMESTONE
	SS 3	100	4-50/4"	PP = 4.5 tsf MC = 17.6%		
10	SS 4	78	8-9-11 (20)	PP >4.5 tsf MC = 15.7%		9.3 999.4
						Grey moist soft SHALE
15	SS 5	100	4-50/6"	PP >4.5 tsf MC = 12.8%		
	SS 6	100	50/6"	PP >4.5 tsf MC = 12.1%		19.0 989.7
						Spoon refusal at 19

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BORING NUMBER B-17

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/4/18 **COMPLETED** 6/4/18
GROUND ELEVATION 1011.04 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	1010.5
	SS 1	78	4-6-8 (14)	PP = 3.5 tsf MC = 21.8%	3.5	Brown moist very stiff CLAY	1007.5
5	SS 2	100	4-5-6 (11)	PP = 3.5 tsf MC = 22.6%		Brown moist stiff CLAY	
	SS 3	100	50/4"	PP = 4 tsf MC = 27.5%	7.0		1004.0
					8.8	Grey moist Firm Weathered LIMESTONE	1002.2

Auger refusal at 8.83

BORING NUMBER B-18

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CLIENT Monarch Acquisitions LLC

PROJECT NAME West Pryor Village

PROJECT NUMBER 18-5125

PROJECT LOCATION Lee's Summit, MO

DATE STARTED 6/5/18 **COMPLETED** 6/5/18

GROUND ELEVATION 1001.04 ft MSL **HOLE SIZE** 6

DRILLING CONTRACTOR RC Drilling

GROUND WATER LEVELS:

DRILLING METHOD HSA

AT TIME OF DRILLING ---

LOGGED BY Luke **CHECKED BY** HM

AT END OF DRILLING ---

NOTES

AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL 1000.5
						Brown moist firm CLAY
	SS 1	67	3-3-4 (7)	PP = 2.5 tsf MC = 21.8%		
5	SS 2	78	4-3-4 (7)	PP = 2.5 tsf MC = 20.9%	5.5	995.5
						Grey moist soft SHALE
	SS 3	100	9-28-50 (78)	PP >4.5 tsf MC = 5.5%		
	SS 4	100	50/6"	PP >4.5 tsf MC = 12.3%	9.0	992.0
						Spoon refusal at 9

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BORING NUMBER B-19

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/4/18 **COMPLETED** 6/4/18
GROUND ELEVATION 1011.23 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL 1010.7
						Brown moist stiff CLAY
	SS 1	78	4-6-7 (13)	PP = 3 tsf MC = 21.7%		
5	SS 2	100	4-5-6 (11)	PP = 3.5 tsf MC = 22.1%		
	SS 3	100	5-5-6 (11)	PP >4.5 tsf MC = 18.5%		
	SS 4	100	50/6"		8.8	1002.5
						Grey moist Firm Weathered LIMESTONE
					9.8	1001.4

Auger refusal at 9.8

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BORING NUMBER B-20

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/31/18 **COMPLETED** 5/31/18
GROUND ELEVATION 1002.49 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	1002.0
	SS 1	100	4-5-7 (12)	PP = 2.5 tsf MC = 27%		Brown moist stiff CLAY	
					3.5		999.0
5	SS 2	100	4-5-8 (13)	PP = 3.25 tsf MC = 27%		Brown moist very stiff CLAY	
					6.8		995.7
	SS 3	100	7-13-11 (24)	PP >4.5 tsf MC = 19.4%		Grey moist soft SHALE	
10	SS 4	100	8-10-8 (18)	PP >4.5 tsf MC = 18.2%			
					11.0		991.5
						Grey moist Firm Weathered LIMESTONE	
					12.1		990.4

Auger refusal at 12.1

BORING NUMBER B-21

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CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/1/18 **COMPLETED** 6/1/18
GROUND ELEVATION 985.55 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	985.1
						Brown moist stiff CLAY	
	SS 1	67	5-5-5 (10)	PP = 3.75 tsf MC = 24.4%			
5	SS 2	78	4-5-6 (11)	PP = 3.75 tsf MC = 22.8%		Brown moist very stiff CLAY	
					6.0		979.6
					6.8		978.8
	SS 3	67	6-6-7 (13)	PP >4.5 tsf MC = 20.7%		Grey moist soft SHALE	
10	SS 4	67	6-14-16 (30)	PP >4.5 tsf MC = 31.2%			
15	SS 5	100	23-25-22 (47)	PP >4.5 tsf MC = 16%			
20	SS 6	100	16-25-50 (75)	PP >4.5 tsf MC = 15.9%			
	SS 7	100	50/6"	PP >4.5 tsf MC = 11.1%	24.0		961.6
						Spoon refusal at 24	

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BORING NUMBER B-22

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/31/18 **COMPLETED** 5/31/18
GROUND ELEVATION 1005.78 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 1005.3
						SANDSTONE grey moderately soft
	SS 1	67	4-6-8 (14)	PP = 3.5 tsf MC = 26.6%		
5	SS 2	100	24-30-8 (38)	PP >4.5 tsf MC = 27.4%		5.0 1000.8
						Brown moist very stiff CLAY
	SS 3	100	5-9-10 (19)	PP >4.5 tsf MC = 19.3%		6.8 999.0
						Grey moist soft SHALE
10	SS 4	100	5-5-10 (15)	PP >4.5 tsf MC = 22.6%		
15	SS 5	100	17-22-50 (72)	PP >4.5 tsf MC = 18.4%		
						17.8 988.0
						18.1 Grey moist Firm Weathered LIMESTONE 987.7

Auger refusal at 18.1

BORING NUMBER B-23

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CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/4/18 **COMPLETED** 6/4/18
GROUND ELEVATION 978.03 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL 977.5
					0.7	Brown moist stiff CLAY 977.4
					1.0	Grey moist Firm Weathered LIMESTONE 977.0
	SS 1	78	4-4-5 (9)	PP = 2.5 tsf MC = 19.9%		Grey moist soft SHALE
5	SS 2	78	4-5-5 (10)	PP = 4 tsf MC = 16.9%		
	SS 3	100	20-29-28 (57)	PP >4.5 tsf MC = 11.7%		
10	SS 4	100	18-24-50 (74)	PP >4.5 tsf MC = 12.3%		
	SS 5	100	50/4"	PP >4.5 tsf MC = 7%	13.8	964.2
						Spoon refusal at 13.83

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BORING NUMBER B-24

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/5/18 **COMPLETED** 6/5/18
GROUND ELEVATION 989.15 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	988.7
					1.8	Brown moist very stiff CLAY	987.4
	SS 1	67	4-7-7 (14)	PP >4.5 tsf MC = 15.8%	3.5	Grey moist soft SHALE	985.7
					4.2	Brown moist hard CLAY	985.0
	SS 2	100	8-50/5"	PP >4.5 tsf MC = 12.9%	4.8	LIMESTONE grey moderately soft	984.4

Auger refusal at 4.8

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BORING NUMBER B-25

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/5/18 **COMPLETED** 6/5/18
GROUND ELEVATION 1005 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0							
0.5						TOPSOIL	1004.5
	SS 1	56	4-4-4 (8)	PP = 3 tsf MC = 25.5% PP = 3 tsf MC = 20.8%		Brown moist stiff CLAY	
5	SS 2	100	4-4-5 (9)	PP = 3 tsf MC = 20%			
6.8	SS 3	39	23-10-5 (15)	PP >4.5 tsf MC = 13.4%			998.3
7.5						Grey moist Firm Weathered LIMESTONE	997.5
						Grey moist soft SHALE	
10	SS 4	100	6-6-8 (14)	PP >4.5 tsf MC = 10.7%			
15	SS 5	100	22-50/6"	PP >4.5 tsf MC = 7.5%			
19.3	SS 6	100	18-50/3"	PP >4.5 tsf MC = 11.3%			985.8

Spoon refusal at 19.25

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BORING NUMBER B-26

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/30/18 **COMPLETED** 5/30/18
GROUND ELEVATION 1004.29 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	1003.8
						Grey moist soft SHALE	
	SS 1	61	10-9-13 (22)	PP = 1.5 tsf MC = 21.6%	1.8		1002.5
						Grey moist Firm Weathered LIMESTONE	
	SS 2	100	50/3"	PP = 3 tsf MC = 16.1%	4.0		1000.3
						Auger refusal at 4	

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BORING NUMBER B-27

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/30/18 **COMPLETED** 5/30/18
GROUND ELEVATION 986.67 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0.5						TOPSOIL	986.2
1.8	SS 1	78	2-2-1 (3)	PP = 2 tsf MC = 24%		Brown moist soft CLAY	984.9
2.5						Grey moist soft SHALE	984.2
6.0	SS 2	78	1-2-4 (6)	PP = 1.25 tsf MC = 28.6%		Brown wet moist CLAY	980.7
7.5	SS 3	89	3-6-9 (15)	PP = 3.5 tsf MC = 21.9%		Grey moist soft SHALE	979.2
8.5						Brown wet moist CLAY	978.2
14.9	SS 4	100	16-21-50/4"	PP >4.5 tsf MC = 17.7%		Grey moist soft SHALE	971.8
	SS 5	100	20-26-50/5"	PP >4.5 tsf MC = 14.9%			

Auger refusal at 14.92

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BORING NUMBER B-28

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/31/18 **COMPLETED** 5/31/18
GROUND ELEVATION 979.77 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0						0.5 TOPSOIL	979.3
	SS 1	67	4-5-6 (11)	PP = 3 tsf MC = 24.5%		Brown moist stiff CLAY	
						3.5	976.3
5	SS 2	100	5-6-8 (14)	PP = 2.5 tsf MC = 26.1%		Grey moist very stiff CLAY	
						6.0	973.8
	SS 3	100	4-5-6 (11)	PP = 2.5 tsf MC = 26.1%		Brown moist stiff CLAY	
						9.3	970.5
10	SS 4	56	7-8-11 (19)	PP >4.5 tsf MC = 24.1%		Grey moist soft SHALE	
15	SS 5	100	15-23-32 (55)	PP >4.5 tsf MC = 15.7%			
20	SS 6	100	19-16-50 (66)	PP >4.5 tsf MC = 16.4%			
						21.0	958.8
						LIMESTONE grey moderately soft	
						22.2	957.6

Auger refusal at 22.2

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BORING NUMBER B-29

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/30/18 **COMPLETED** 5/30/18
GROUND ELEVATION 999.4 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0						0.5 TOPSOIL	998.9
	SS 1	33	3-3-2 (5)	PP = 1.5 tsf MC = 20.5%		Brown moist firm CLAY	
						2.5	996.9
5	SS 2	89	2-4-4 (8)	PP = 2.5 tsf MC = 27.5%		Grey moist stiff CLAY	
						6.0	993.4
	SS 3	67	10-2-1 (3)	PP = 2.5 tsf MC = 20.2%		Brown moist firm CLAY	
						8.5	990.9
						Grey moist soft SHALE	
						9.3	990.2
10	SS 4	89	7-8-8 (16)	PP = 3 tsf MC = 22.7%		Brown moist moderately soft SANDSTONE	
						10.3	989.1
						Grey moist Firm Weathered LIMESTONE	
						12.3	987.1

Auger refusal at 12.3

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BORING NUMBER B-30

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 5/30/18 **COMPLETED** 5/30/18
GROUND ELEVATION 995.24 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0						0.5 TOPSOIL 994.7
						Brown moist stiff CLAY
	SS 1	89	3-3-5 (8)	PP = 25 tsf MC = 25%		
5	SS 2	56	3-3-7 (10)	PP = 2.5 tsf MC = 23.5% LL = 50 PL = 22		
	SS 3	67	3-6-9 (15)	PP = 3 tsf MC = 25.2%		
10	SS 4	100	3-11-15 (26)	PP = 2 tsf MC = 32.3%		10.3 985.0
						Grey moist Firm Weathered LIMESTONE
						12.5 982.7

Auger refusal at 12.3

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BORING NUMBER B-31

CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/4/18 **COMPLETED** 6/4/18
GROUND ELEVATION 986.97 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION	
0					0.5	TOPSOIL	986.5
					1.4	Brown moist very hard CLAY	985.6
	SS 1	50	3-50	PP = 2 tsf	2.8	Grey moist Firm Weathered LIMESTONE	984.2

Auger refusal at 2.8 ft

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BORING NUMBER B-32

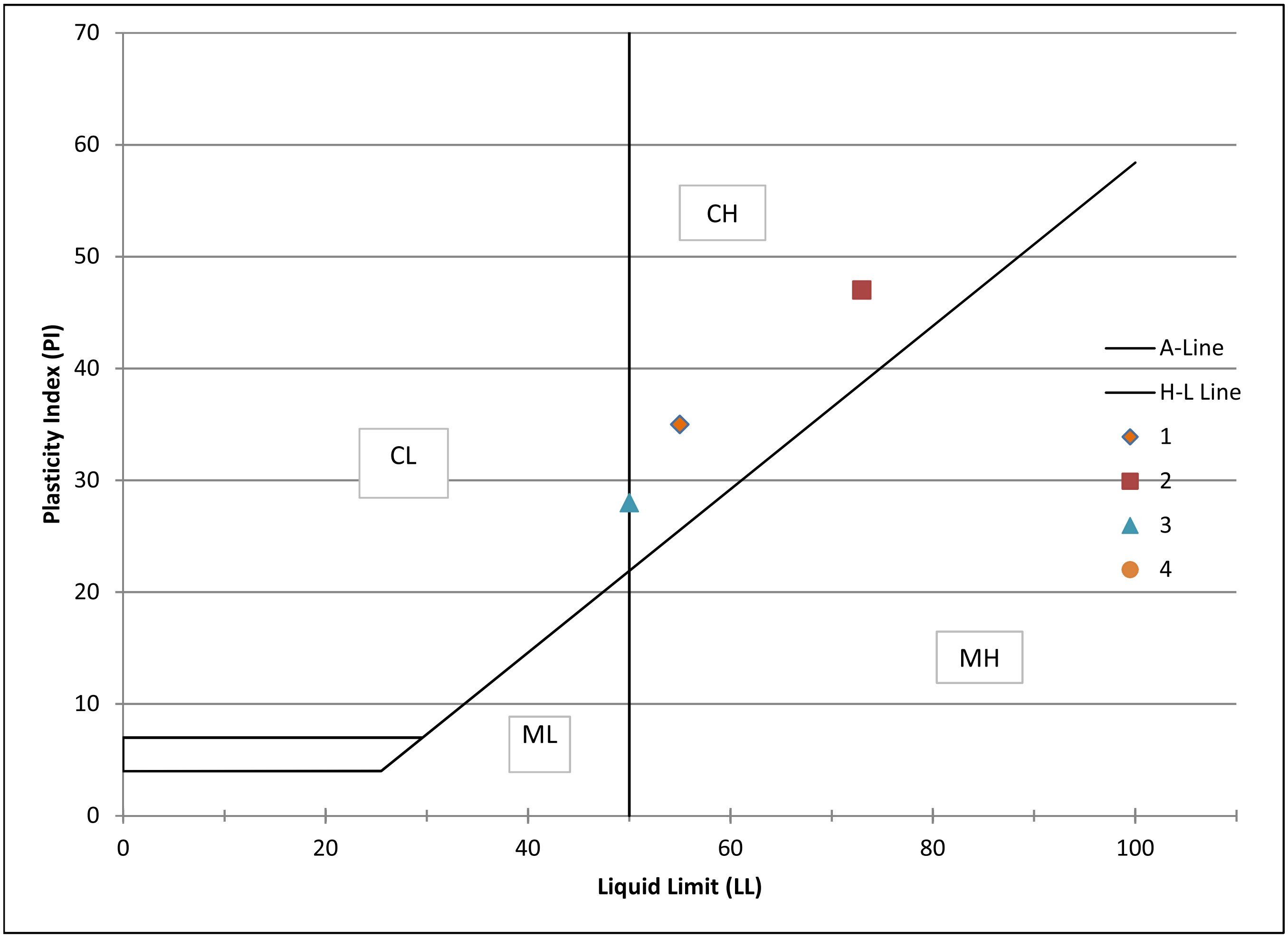
CLIENT Monarch Acquisitions LLC
PROJECT NAME West Pryor Village
PROJECT NUMBER 18-5125
PROJECT LOCATION Lee's Summit, MO
DATE STARTED 6/5/18 **COMPLETED** 6/5/18
GROUND ELEVATION 964.65 ft MSL **HOLE SIZE** 6
DRILLING CONTRACTOR RC Drilling
GROUND WATER LEVELS:
DRILLING METHOD HSA
AT TIME OF DRILLING ---
LOGGED BY Luke **CHECKED BY** HM
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	RECOVERY %	BLOW COUNTS (N VALUE)	TESTS	GRAPHIC LOG	MATERIAL DESCRIPTION
0					0.5	TOPSOIL 964.2
						Brown moist stiff CLAY
	SS 1	67	5-6-7 (13)	PP >4.5 tsf MC = 20.2%		
					3.5	961.2
						Grey moist soft SHALE
5	SS 2	100	6-7-9 (16)	PP >4.5 tsf MC = 15.2%		
	SS 3	100	13-23-50 (73)	PP >4.5 tsf MC = 13.9%		
	SS 4	100	50/6"	PP >4.5 tsf MC = 10.8%	9.0	955.7
						Spoon refusal at 9

Appendix C: Laboratory Test Results

Liquid Limit, Plastic Limit, & Plasticity Index
Test Report

Project:	Street West of Pryor	Job No.:	='Sample 1'!G6:H6
Location:	Lee's Summit	Sample Date:	6/1/2018
Tested By:	AB	Test Date:	6/12/2018



	Boring / Sample	Material Description	LL	PL	PI	USCS
1	10 / 2	Brown CLAY	55	20	35	CH
2	13 / 3	Light brown CLAY	73	26	47	CH
3	30 / 2	Gray CLAY	50	22	28	CL-CH
4						

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Water Content Determination

Project: Pryor
Location: Lee's Summit
Project #: 18-5125
Date: 13 June 2018

Test Performed By: AB
Checked By: PD

Boring	Sample	Can #	Wt. of Can	Wt can+wet	Wt can+dry	Wt water	Wt dry	M/C (%)
B-1	1		2303	6820	5811	1009	3508	28.8%
B-1	2		1569	8063	6782	1281	5213	24.6%
B-1	3		2276	8998	7588	1410	5312	26.5%
B-1	4		2250	8078	6878	1200	4628	25.9%
B-2	1		2232	7996	6950	1046	4718	22.2%
B-2	2		2234	8147	6783	1364	4549	30.0%
B-2	3		2258	6451	5614	837	3356	24.9%
B-2	4		2268	7689	6371	1318	4103	32.1%
B-3	1		2246	8007	7046	961	4800	20.0%
B-3	2		2269	8842	7615	1227	5346	23.0%
B-3	3		2248	8833	7533	1300	5285	24.6%
B-3	4		2240	8031	6668	1363	4428	30.8%
B-3	5		1653	8867	7872	995	6219	16.0%
B-4	1		2278	5608	5049	559	2771	20.2%
B-4	2		2282	6172	5537	635	3255	19.5%
B-4	3		1582	6852	5658	1194	4076	29.3%
B-4	4		2243	6442	5578	864	3335	25.9%
B-4	5		1565	8607	7144	1463	5579	26.2%
B-5	1		1486	4547	4032	515	2546	20.2%
B-5	2		2292	7050	6192	858	3900	22.0%
B-5	3		2297	6447	5634	813	3337	24.4%
B-5	4		2243	5950	5179	771	2936	26.3%
B-5	5		2265	8392	7074	1318	4809	27.4%
B-5	6		2242	8577	7360	1217	5118	23.8%
B-6	1		2254	9130	7843	1287	5589	23.0%
B-6	2		2254	7509	6459	1050	4205	25.0%
B-6	3		1467	6456	5532	924	4065	22.7%
B-6	4		1543	6980	5904	1076	4361	24.7%
B-6	5		2279	7496	6617	879	4338	20.3%
B-6	6		2282	6303	5655	648	3373	19.2%
B-6	7		1545	6837	6101	736	4556	16.2%
B-7	1		2235	8344	7241	1103	5006	22.0%
B-7	2		2249	7243	6339	904	4090	22.1%
B-7	3		2237	6334	5518	816	3281	24.9%
B-7	4		2267	7534	6628	906	4361	20.8%
B-7	5		1454	9619	8454	1165	7000	16.6%
B-7	6		1624	7783	6861	922	5237	17.6%
B-8	1		2254	7491	6187	1304	3933	33.2%

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Water Content Determination

Project: Pryor
Location: Lee's Summit
Project #: 18-5125
Date: 13 June 2018
Test Performed By: AB
Checked By: PD

Boring	Sample	Can #	Wt. of Can	Wt can+wet	Wt can+dry	Wt water	Wt dry	M/C (%)
B-8	2		2261	7465	6474	991	4213	23.5%
B-8	3		1563	7690	6545	1145	4982	23.0%
B-8	4		1567	5458	4682	776	3115	24.9%
B-8	5		1589	8713	7689	1024	6100	16.8%
B-8	6		1577	5483	4953	530	3376	15.7%
B-9	1		2286	7727	6560	1167	4274	27.3%
B-9	2		1568	6233	5333	900	3765	23.9%
B-9	3		1609	7580	6418	1162	4809	24.2%
B-9	4		1609	6909	5894	1015	4285	23.7%
B-9	5		1552	5361	4699	662	3147	21.0%
B-10	1		2260	6876	6120	756	3860	19.6%
B-10	2		2249	5852	5268	584	3019	19.3%
B-10	3		2234	8488	7363	1125	5129	21.9%
B-10	4		2252	8240	7149	1091	4897	22.3%
B-10	5		1469	8801	7874	927	6405	14.5%
B-10	6		2274	5865	5388	477	3114	15.3%
B-11	1		2237	6967	5912	1055	3675	28.7%
B-11	2		1451	5897	4956	941	3505	26.8%
B-11	3		2298	8521	7302	1219	5004	24.4%
B-11	4		1560	8541	7141	1400	5581	25.1%
B-11	5		2251	7446	6687	759	4436	17.1%
B-11	6		1544	4733	4292	441	2748	16.0%
B-11	7		2199	6387	5951	436	3752	11.6%
B-12	1		2290	5100	4601	499	2311	21.6%
B-12	2		2295	8508	7583	925	5288	17.5%
B-12	3		2235	7167	6458	709	4223	16.8%
B-12	4		2260	7216	6497	719	4237	17.0%
B-12	5		2268	7573	6951	622	4683	13.3%
B-12	6		2277	6268	5866	402	3589	11.2%
B-13	1		1567	6822	5604	1218	4037	30.2%
B-13	2		2276	8446	7234	1212	4958	24.4%
B-13	3		1585	7903	6611	1292	5026	25.7%
B-13	4		1570	6686	5952	734	4382	16.8%
B-14	1		2267	7670	6766	904	4499	20.1%
B-14	2		2245	7676	6762	914	4517	20.2%
B-14	3		2256	7185	6337	848	4081	20.8%
B-14	4		1570	6678	5850	828	4280	19.3%
B-14	5		2254	6538	5921	617	3667	16.8%

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Water Content Determination

Project: Pryor
Location: Lee's Summit
Project #: 18-5125
Date: 13 June 2018

Test Performed By: AB
Checked By: PD

Boring	Sample	Can #	Wt. of Can	Wt can+wet	Wt can+dry	Wt water	Wt dry	M/C (%)
B-14	6		1486	5064	4714	350	3228	10.8%
B-15	1		2286	6056	5232	824	2946	28.0%
B-15	2		2222	5419	4755	664	2533	26.2%
B-15	3		2291	7149	6195	954	3904	24.4%
B-15	4		2243	5311	4684	627	2441	25.7%
B-15	5		2240	6605	6197	408	3957	10.3%
B-15	6		2236	4538	4323	215	2087	10.3%
B-16	1		2285	6554	5521	1033	3236	31.9%
B-16	2		1468	6893	5916	977	4448	22.0%
B-16	3		2290	8010	7156	854	4866	17.6%
B-16	4		2243	7050	6396	654	4153	15.7%
B-16	5		2297	5601	5225	376	2928	12.8%
B-16	6		2242	6320	5879	441	3637	12.1%
B-17	1		1566	4373	3871	502	2305	21.8%
B-17	2		2272	6801	5965	836	3693	22.6%
B-17	3		1450	6608	5497	1111	4047	27.5%
B-18	1		2278	5712	5098	614	2820	21.8%
B-18	2		2243	7668	6732	936	4489	20.9%
B-18	3		2267	5843	5658	185	3391	5.5%
B-18	4		2275	5526	5170	356	2895	12.3%
B-19	1		1584	5970	5188	782	3604	21.7%
B-19	2		2252	5907	5246	661	2994	22.1%
B-19	3		2258	5898	5331	567	3073	18.5%
B-20	1		2273	6377	5505	872	3232	27.0%
B-20	2		2249	5918	5137	781	2888	27.0%
B-20	3		2234	8579	7549	1030	5315	19.4%
B-20	4		2248	7031	6295	736	4047	18.2%
B-21	1		2277	7455	6441	1014	4164	24.4%
B-21	2		2260	9395	8071	1324	5811	22.8%
B-21	3		2262	7007	6193	814	3931	20.7%
B-21	4		2269	6543	5527	1016	3258	31.2%
B-21	5		1467	4809	4349	460	2882	16.0%
B-21	6		1569	3955	3627	328	2058	15.9%
B-21	7		2250	5675	5333	342	3083	11.1%
B-22	1		2232	6377	5505	872	3273	26.6%
B-22	2		2287	5918	5137	781	2850	27.4%
B-22	3		2220	8579	7549	1030	5329	19.3%
B-22	4		2271	6968	6102	866	3831	22.6%

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Water Content Determination

Project: Pryor
Location: Lee's Summit
Project #: 18-5125
Date: 13 June 2018

Test Performed By: AB   Checked By: PD

Boring	Sample	Can #	Wt. of Can	Wt can+wet	Wt can+dry	Wt water	Wt dry	M/C (%)
B-22	5		2287	7031	6295	736	4008	18.4%
B-23	1		2241	6517	5807	710	3566	19.9%
B-23	2		2255	6987	6303	684	4048	16.9%
B-23	3		1559	5151	4776	375	3217	11.7%
B-23	4		2222	5601	5232	369	3010	12.3%
B-23	5		2256	4583	4430	153	2174	7.0%
B-24	1		2249	7788	7031	757	4782	15.8%
B-24	2		2272	6960	6425	535	4153	12.9%
B-25	1		1573	5691	4855	836	3282	25.5%
B-25	1		2271	4852	4408	444	2137	20.8%
B-25	2		2289	5012	4558	454	2269	20.0%
B-25	3		2271	6257	5787	470	3516	13.4%
B-25	4		1621	6198	5757	441	4136	10.7%
B-25	5		1544	4723	4502	221	2958	7.5%
B-25	6		2282	6111	5723	388	3441	11.3%
B-26	1		2276	6826	6018	808	3742	21.6%
B-26	2		2243	6910	6262	648	4019	16.1%
B-27	1		2293	4952	4437	515	2144	24.0%
B-27	2		2295	5512	4796	716	2501	28.6%
B-27	3		2277	5262	4726	536	2449	21.9%
B-27	4		2289	5251	4806	445	2517	17.7%
B-27	5		2243	5767	5309	458	3066	14.9%
B-28	1		2277	4806	4309	497	2032	24.5%
B-28	2		1455	4437	3820	617	2365	26.1%
B-28	3		2247	7167	6148	1019	3901	26.1%
B-28	4		2291	5372	4774	598	2483	24.1%
B-28	5		1622	5199	4713	486	3091	15.7%
B-28	6		2254	6142	5594	548	3340	16.4%
B-29	1		2292	4798	4372	426	2080	20.5%
B-29	2		2272	6165	5325	840	3053	27.5%
B-29	3		2273	7030	6231	799	3958	20.2%
B-29	4		2285	5558	4953	605	2668	22.7%
B-30	1		1653	5116	4424	692	2771	25.0%
B-30	2		2282	8541	7349	1192	5067	23.5%
B-30	3		2235	7112	6129	983	3894	25.2%
B-30	4		2258	5828	4956	872	2698	32.3%
B-31	1					0	0	
B-32	1		2220	7667	6753	914	4533	20.2%

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Water Content Determination

Project: Pryor
Location: Lee's Summit
Project #: 18-5125
Date: 13 June 2018
Test Performed By: AB
Checked By: PD

Boring	Sample	Can #	Wt. of Can	Wt can+wet	Wt can+dry	Wt water	Wt dry	M/C (%)
B-32	2		2244	6422	5870	552	3626	15.2%
B-32	3		2236	8437	7682	755	5446	13.9%
B-32	4		2286	7268	6783	485	4497	10.8%

Appendix D: Fly Ash, Lime and Cement Specifications

GUIDELINE FOR FLY ASH OR CEMENT STABILIZATION

Fly ash or cement stabilized soils should not be constructed without the presence of the geotechnical engineer's designated representative.

MATERIALS. The fly ash material used in stabilization should meet the physical characteristics of ASTM D 5239 6.4, with a minimum compressive strength of 500 psi at 7-days. The fly ash material should meet the chemical requirements of ASTM C 618, Class C. The source material should be identified and approved by the geotechnical engineer prior to delivery to the site.

Fly ash should be kept free from moisture prior to use. Fly ash stored on the project site should be placed in weatherproof bins or buildings with adequate protection from ground dampness.

CONSTRUCTION. The fly ash stabilized soil should be constructed as described herein. The fly ash should be spread uniformly across the prepared soil surface at the full application rate by using an agricultural seed or fly ash spreader or other equipment acceptable to the geotechnical engineer's designated representative.

Fly ash stabilized material should be placed in approximately horizontal layers not to exceed 12 inches in uncompacted thickness.

Subgrade Preparation. Prior to the beginning of fly ash treatment, the Contractor should construct the subgrade to an elevation that will provide a subgrade surface conforming to the contract documents upon completion of the fly ash treatment.

The fly ash should be spread with an approved spreader and added as a percentage by weight. The amount of fly ash should be approved by the geotechnical engineer and be based on laboratory testing with the soil materials.

If moisture content of the soil exceeds the specified limits, additional fly ash may be added to lower the moisture content. Lowering moisture contents by aeration after the application of fly ash should not be permitted.

Fly ash should be spread only on those areas where mixing operations can be completed during the same working day.

Weather Conditions. *Fly ash should not be applied when the atmospheric temperature is less than 40°F.*

When the temperature is predicted to drop below 40°F after completion of the fly ash treatment, the treated areas should be protected against freezing by a sufficient covering of straw, or by other approved methods, until the course has cured. Any areas of completed base course that are damaged by freezing, rainfall, or other weather conditions should be repaired by the contractor.

GUIDELINE FOR FLY ASH OR CEMENT STABILIZATION

No fly ash should be applied to soils that are frozen or contain frost, or when the underlying material is frozen. If the temperature falls below 40°F, completed fly ash-treated areas should be protected against the detrimental effects of freezing.

The fly ash should be distributed at a uniform rate and in such a manner to prevent the scattering of fly ash by wind. Fly ash should not be added when wind or weather conditions are not favorable in the opinion of the geotechnical engineer's designated representative. A motor grader should not be used to spread the fly ash.

Mixing. Mixing should begin within 1 hour of distribution of the fly ash. The fly ash, soil, and required water should be thoroughly mixed, blended, and pulverized by approved road mixers or by a depth-controlled rotary tiller. Except as provided hereinafter, the Contractor should continue mixing and applying water until all material will pass a 1-inch screen. Scarifying and mixing should be controlled to provide uniform depth within 0.1 ft of the depth specified. If, in the opinion of the geotechnical engineer's designated representative the material was mixed to a depth greater than indicated on the drawing or as specified herein, additional fly ash should be added to achieve the desired application rate. If in the opinion of the geotechnical engineer's designated representative, the material was mixed to a depth less than indicated on the drawing or specified, the material should be remixed.

Moisture content of the mixture should be determined in preparation for final mixing. Moisture in the mixture following final mixing should not be less than the water content determined to be optimum based on dry weight of soil and should not exceed the optimum water content by more than 5 percentage points. Water may be added in increments as large as the equipment will permit; however, such increment of water should be partially incorporated in the mix to avoid concentration of water near the surface. After the last increment of water has been added, mixing should be continued until the water is uniformly distributed throughout the full depth of the mixture, including satisfactory moisture distribution along the edges of the section.

Compaction. Compaction should begin within 2 hours of the start of mixing. The fly ash stabilized subgrade should be compacted in accordance with the requirements for controlled fill. The compaction should be a minimum of 95% of the maximum density in accordance with ASTM D698 and within –2% to +3% of the optimum moisture content of the fly ash-stabilized soil.

The compaction should be achieved using a vibratory pad-foot roller or other equipment approved by the geotechnical engineer's representative.

Protection and Curing. The Contractor should protect the finished treated subgrade from rapid drying, for 3 days, by sprinkling with water as often as is necessary to prevent drying of the surface of the fly ash-treated subgrade, or by application of the overlying base course. The Contractor should not allow any vehicles or operations that will distort the surface to the extent that proper curing will be affected on the treated subgrade during the curing period.

GUIDELINE FOR IN-PLACE LIME STABILIZATION
Reference: Standard Specification and Construction Manual – KDOT 2015

Lime stabilized soils should not be constructed without the presence of the geotechnical engineer’s designated representative.

MATERIALS. The lime material used in stabilization should meet the chemical and physical characteristics of ASTM C977. Hydrated lime should be kept free from moisture prior to use. Lime stored on the project should be placed in weatherproof bins or buildings with adequate protection from ground dampness.

CONSTRUCTION. The lime stabilized soil should be constructed as described herein. The lime should be spread uniformly across the prepared soil surface at the full application rate by using an agricultural seed or lime spreader or other equipment acceptable to the geotechnical engineer’s designated representative.

Lime stabilized material should be placed in approximately horizontal layers not to exceed 8 inches in uncompacted thickness.

Subgrade Preparation. Prior to the beginning of lime treatment, the Contractor should construct the subgrade to an elevation which will provide a subgrade surface conforming to the contract documents upon completion of the lime treatment. The subgrade should be scarified to a minimum depth of 4 inches and a maximum depth of approximately 1 inch less than the specified depth of lime treatment. Positive depth control equipment should be used to scarify the subgrade, do not use plow or disc. Proper drainage should be maintained at all times.

Lime should be spread only on those areas where mixing operations can be completed during the same working day. Mixing and spreading should not be performed during freezing temperatures. When the temperature is below 40 degrees F, the completed base course should be protected against freezing by a sufficient covering of straw, or by other approved methods, until the course has dried out. Any areas of completed base course that are damaged by freezing, rainfall, or other weather conditions should be repaired by the contractor. Lime should not be applied when the atmospheric temperature is less than 40 degrees F. No lime should be applied to soils that are frozen or contain frost, or when the underlying material is frozen. If the temperature falls below 35 degrees F, completed lime-treated areas should be protected against any detrimental effects of freezing.

The lime should be spread with an approved spreader and added as a percentage by weight. Apply hydrated lime to scaridfied areas as a slurry. The application and mixing of the hydrated lime slurry shall result in a uniform lime concentration. The amount of lime should be approved by the geotechnical engineer and based on laboratory testing with the soil materials.

The lime should be distributed at a uniform rate and in such a manner to prevent the scattering of lime by wind. If the application rate is not shown in contractor documents, assume a rate of 5% of the weight of soil. Lime should not be added when wind or weather conditions are not favorable in the opinion of the geotechnical engineer's designated representative. A motor grader should not be used to spread the lime.

Preliminary Mixing. The lime, material, and required water should be thoroughly mixed, blended, and pulverized by approved road mixers or by a depth-controlled rotary tiller. Except as provided hereinafter, the Contractor should continue mixing and applying water until 95% of the material passes a 2-inch screen as determined by the geotechnical engineer. Scarifying and mixing should be controlled to provide uniform depth within 0.1 ft of the depth specified. If, in the opinion of the geotechnical engineer's designated representative the material was mixed to a depth greater than indicated on the drawing or as specified herein, additional lime should be added to achieve the desired application rate. If in the opinion of the geotechnical engineer's designated representative, the material was mixed to a depth less than indicated on the drawing or specified, the material should be remixed.

Moisture content of the mixture should be determined in preparation for final mixing. Moisture in the mixture following final mixing should not be less than the water content determined to be optimum based on dry weight of soil and should not exceed the optimum water content by more than 5 percentage points. Water may be added in increments as large as the equipment will permit; however, such increment of water should be partially incorporated in the mix to avoid concentration of water near the surface. After the last increment of water has been added, mixing should be continued until the water is uniformly distributed throughout the full depth of the mixture, including satisfactory moisture distribution along the edges of the section. A minimum of 2 passes should be performed with the mixer traveling in the primary direction.

Aging. Seal the mixture to prevent moisture loss by lightly rolling with a pneumatic-tired roller and blade the surface to shed water. Maintain the mixture in the sealed condition for a minimum of 24 hours prior to final misxing. Keep the surface moist by spraying water. If the final mixing is not completed with 14 days of preliminary mixing, add 1% lime by weight of raw soil in the final mixing operation.

Final Mixing. After initial mixing and ageing a minimum of 24 hours, the mixture should be re-mixed to the specified depth until 95% of the mixture passes the 1 ½ inch sieve and 40% passes the No. 4 sieve as determined by the geotechnical engineer. To breakdown particle size periodic mixing is allowed over an interval of time. Bring the mixture moisture content required for compaction with a minimum of 3% above optimum of the proctor density of the lime treated soil.

Compaction. The lime stabilized subgrade should be compacted in accordance with the requirements for controlled fill. The compaction should be a minimum of 95% of the maximum density in accordance with ASTM D698 and a moisture content with a minimum of +3% of the optimum moisture content of the lime-stabilized soil.

Protection and Curing. The Contractor should protect the finished treated subgrade from rapid drying, for 7 days, by sprinkling with water as often as is necessary to prevent drying of the surface of the lime-treated subgrade. When an overlying base course is to be constructed the engineer may reduce the curing period to when the lime treated subgrade gains sufficient strength to support the construction and hauling equipment. Alternatively, an asphalt prime coat may be applied instead of keeping the finishe surface moist. If the asphalt prime coat is used, SS-1, CSS-1 or MC-250 should be applied at a rate of 0.22 gallons per square yard to achieve a minimum of 0.13 gallons per square yard residue. The use of a liquid membrane forming compound is also an acceptable curing medium. Multiple light applications may be required to obtain the specified rate of application without run-off.

The Contractor should not allow any vehicles or operations which will distort the surface to the extent that proper curing will be affected on the treated subgrade during the curing period.

GUIDELINE FOR CEMENT STABILIZATION

Cement stabilized soils should not be constructed without the presence of the geotechnical engineer's designated representative.

MATERIALS. The material used in stabilization should meet the chemical and physical characteristics of Type I cement ASTM C150. Cement should be kept free from moisture prior to use. Cement stored on the project should be placed in weatherproof bins or buildings with adequate protection from ground dampness.

CONSTRUCTION. The cement stabilized soil should be constructed as described herein. The cement should be spread uniformly across the prepared soil surface at the full application rate by using an agricultural seed spreader, mechanical bulk cement spreader, or other equipment acceptable to the geotechnical engineer's designated representative.

Cement stabilized material should be placed in approximately horizontal layers not to exceed 8 inches in uncompacted thickness.

Subgrade Preparation. Prior to the beginning of cement treatment, the Contractor should construct the subgrade to an elevation which will provide a subgrade surface conforming to the contract documents upon completion of the cement treatment.

The clay soils should be scarified and pulverized prior to application of the cement. A disc should be used to break up the surface of the material to be stabilized. The mixer or tiller should be used for the full depth of stabilization to break up the clay.

Application. Cement should be spread only on those areas where mixing operations can be completed during the same working day. Mixing and spreading should not be performed during freezing temperatures. When the temperature is below 40 degrees F, the completed stabilized fill should be protected against freezing by a sufficient covering of straw, or by other approved methods. Any areas of completed stabilized subgrade course that are damaged by freezing, rainfall, or other weather conditions should be repaired by the contractor.

The cement should be applied with an approved spreader at an application rate that has been established by the geotechnical engineer, based on laboratory tests with the site soils.

The cement should be distributed at a uniform rate and in such a manner to prevent the scattering of cement by wind. Cement should not be added when wind or weather conditions are not favorable in the opinion of the geotechnical engineer's designated representative. A motor grader should not be used to spread the cement.

Mixing. The cement, material, and required water should be thoroughly mixed, blended, and pulverized by approved road mixers or by a depth-controlled rotary tiller. Except as provided hereinafter, the Contractor should continue mixing and drying the soil until all material will pass a 1-inch screen. Scarifying and mixing should be controlled to provide uniform depth within 0.1 ft of the depth specified. If, in the opinion of the geotechnical engineer's designated representative the material was mixed to a depth greater than indicated on the drawing or as specified herein, additional cement should be added to achieve the desired application rate. If in the opinion of the geotechnical engineer's designated representative, the material was mixed to a depth less than indicated on the drawing or specified, the material should be remixed.

Moisture content of the mixture should be determined in preparation for final mixing. Moisture in the mixture following final mixing should not be less than the water content determined to be optimum based on dry weight of soil and should not exceed the optimum water content by more than 5 percentage points. Water may be added in increments as large as the equipment will permit; however, such increment of water should be partially incorporated in the mix to avoid concentration of water near the surface. After the last increment of water has been added, mixing should be continued until the water is uniformly distributed throughout the full depth of the mixture, including satisfactory moisture distribution along the edges of the section.

Compaction. The cement stabilized subgrade should be compacted in accordance with the requirements for controlled fill. The compaction should be a minimum of 95% of the maximum density in accordance with ASTM D698 and within +0% to +5% of the optimum moisture content of the cement-stabilized soil.

Not more than 60 minutes should elapse between the time of final mixing and the beginning of compaction.

Protection and Curing. The Contractor should protect the finished treated subgrade from rapid drying, for 7 days, by sprinkling with water as often as is necessary to prevent drying of the surface of the cement-treated subgrade, or by application of the overlying base course. The Contractor should not allow any vehicles or operations which will distort the surface onto the treated surface during the curing period.

KAW VALLEY ENGINEERING, INC.

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Address: 14700 West 114th Terrace
Lenexa, KS 66215

By Email to olsonmonarch@gmail.com

August 4, 2017 **A14D7067**

Mr. Dave Olson
Pryor Crossing, LLC
9400 Reeds Road, Suite 100
Overland Park, KS 66207

RE: ACADEMY SPORTS / SAM'S CLUB
I-470 & NORTHWEST PRYOR ROAD
LEE'S SUMMIT, MISSOURI

Dear Mr. Olson:

Please find attached copies of the Boring Logs and Boring Location Map documenting the field work performed during the July 2017 field effort at the Academy Sports/Sam's Club site. Kaw Valley Engineering, Inc. (KVE) was able to re-enter the pre-drilled borings numbered B-1, B-3 through B-8, and B-16 through B-20. These borings are labeled with an extension of "17" on the attached Boring Location Map. Borings B-21-17 and B-22-17 were drilled by KVE from the surface to total depth during this field effort. Borings B-2, and B-9 through B-15, were not re-entered by KVE and are marked with an "X" on the Boring Location Map.

To the best of our knowledge and subject to the limitations of the fieldwork conducted on the borings described herein, in KVE's opinion, the mine was located in B-6-17, B-7-17, B-8-17 and B-21-17; and not located in B-1-17, B-3-17 through B-5-17, B-16-17 through B-20-17, or B-22-17.

Please feel free to contact this office with any questions regarding this project.

Respectfully submitted,
Kaw Valley Engineering, Inc.

L. Kristopher Moore
Geotechnical/Environmental Coordinator

LKM:lam

Attachments: Fourteen (14) Boring Logs
One (1) Boring Location Map

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LOG OF BORING B-1-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/19/17 - 7/20/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:

SURFACE ELEVATION: 996.6'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored.
	101.5												895.1' LIMESTONE:
	108.5												888.1' SHALE: Gray
	116.1												880.5' LIMESTONE:
	116.9												879.7' SHALE: Gray
	118.5												878.1'

BORING TERMINATED AT 118.5'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-3-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/19/17 - 7/19/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 996.3'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–101.7												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored.
	101.7												894.6'
													LIMESTONE: Gray and light gray
	109.8												886.5'
													SHALE: Gray
	116.2												880.1'
													SHALE: Dark gray; calcareous
	117.8												878.5'

Depth scale: 10, 20, 30, 40, 50, 60, 70, 80, 90, 100, 110, 120, 130

BORING TERMINATED AT 117.8'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-4-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/18/17 - 7/18/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 997.7'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–101.6												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored.
	101.6												896.1'
	101.6–113												LIMESTONE:
	113												884.7'
	113–119.1												SHALE:
	119.1												878.7'

BORING TERMINATED AT 119.1'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-5-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/18/17 - 7/18/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 997.2'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
													Existing boring was crooked, unable to proceed. 992.2'

BORING TERMINATED AT 5.0'

Depth scale (ft): 10, 20, 30, 40, 50, 60, 70, 80, 90, 100, 110, 120, 130

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-6-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/13/17 - 7/13/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:

SURFACE ELEVATION: 998.9'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–115.1												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored. It is KVE's opinion that the mine or a void is present in this boring due to the loss of circulation of drilling fluid. 883.8'
	115.1–119.0												SHALE: Gray; no water recirculation 879.9'

Depth scale: 10, 20, 30, 40, 50, 60, 70, 80, 90, 100, 110, 120, 130

BORING TERMINATED AT 119.0'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-7-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/18/17 - 7/18/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:

SURFACE ELEVATION: 997.6'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–5												Boring was predrilled to an unknown depth. Unable to advance drill stem down hole. A measuring tape was lowered in the boring and extended 130 feet. Water was likely flowing in mine and extending the tape measure. 992.6'
	10–130												
													EXACT DEPTH OF BORING WAS NOT DETERMINED

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-8-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/14/17 - 7/14/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:

SURFACE ELEVATION: 1000.5'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–101.5												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored. 899.0'
	101.5–103.6												LIMESTONE: No water recirculation 896.9'
	103.6–108.6												MINE: Added an additional 5 feet of drill stem. No rotation to verify 891.9'

BORING TERMINATED AT 108.6'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-16-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/12/17 - 7/12/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 1002.5'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–101.8												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored. (900.7')
	101.8–117.0												LIMESTONE: (885.5')
	117.0–119.0												SHALE: (883.5')

Depth scale: 10, 20, 30, 40, 50, 60, 70, 80, 90, 100, 110, 120, 130

BORING TERMINATED AT 119.0'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-17-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/12/17 - 7/12/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 1000.8'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–101.5												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored.
	101.5												899.3'
	101.5–114												LIMESTONE: Gray
	114												886.8'
	114–119												SHALE: Gray
	119												881.8'

Depth scale: 10, 20, 30, 40, 50, 60, 70, 80, 90, 100, 110, 120, 130

BORING TERMINATED AT 119.0'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-18-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/11/17 - 7/11/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 1000.4'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–101.8												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored.
	101.8												898.6'
													LIMESTONE:
	114.0												886.4'
													SHALE: Gray; with limestone seams
	119.2												881.2'

Depth scale: 10, 20, 30, 40, 50, 60, 70, 80, 90, 100, 110, 120, 130

BORING TERMINATED AT 119.2'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-19-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/17/17 - 7/17/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 1007.3'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–102												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored.
	102												905.3'
Limestone	102–119												LIMESTONE:
	119												888.3'

BORING TERMINATED AT 119.0'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-20-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/17/17 - 7/17/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:

SURFACE ELEVATION: 1006.6'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
	0–102												Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored. 904.6'
	102–117												LIMESTONE: Broke tricone 889.6'

BORING TERMINATED AT 117.0'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-21-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/20/17 - 7/21/17
DRILLING METHOD(S): 4" CFA AND NQ2
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Scott Bishop

GROUNDWATER INFORMATION:

SURFACE ELEVATION: 1001.3'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
													TOPSOIL (12") 1000.3'
	10												FAT CLAY: Brown; stiff; moist 990.3'
													SHALE: Olive, light brown, and brown; weathered 986.6'
													LIMESTONE: Brown; weathered 985.8'
	20												LIMESTONE: Light gray; slightly weathered 981.6'
	30												SHALE: Gray
	40												957.9'
	50												LIMESTONE: With shale seams 949.1'
	60												SHALE: Gray 940.3'
													LIMESTONE 936.2'
													SHALE: Gray 935.3'
	70												LIMESTONE 934.1'
													SHALE 932.4'
	80												LIMESTONE: With shale seams
	90												908.1'
													SHALE: Black 907.1'
													SHALE: Gray 904.6'
	100												LIMESTONE 896.6'
	110		BORING TERMINATED AT 104.7'										HIT VOID OR MINE AT 104.7'
	120												
	130												

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

LOG OF BORING B-22-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/24/17 - 7/25/17
DRILLING METHOD(S): 4" CFA AND NQ2
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott

GROUNDWATER INFORMATION:

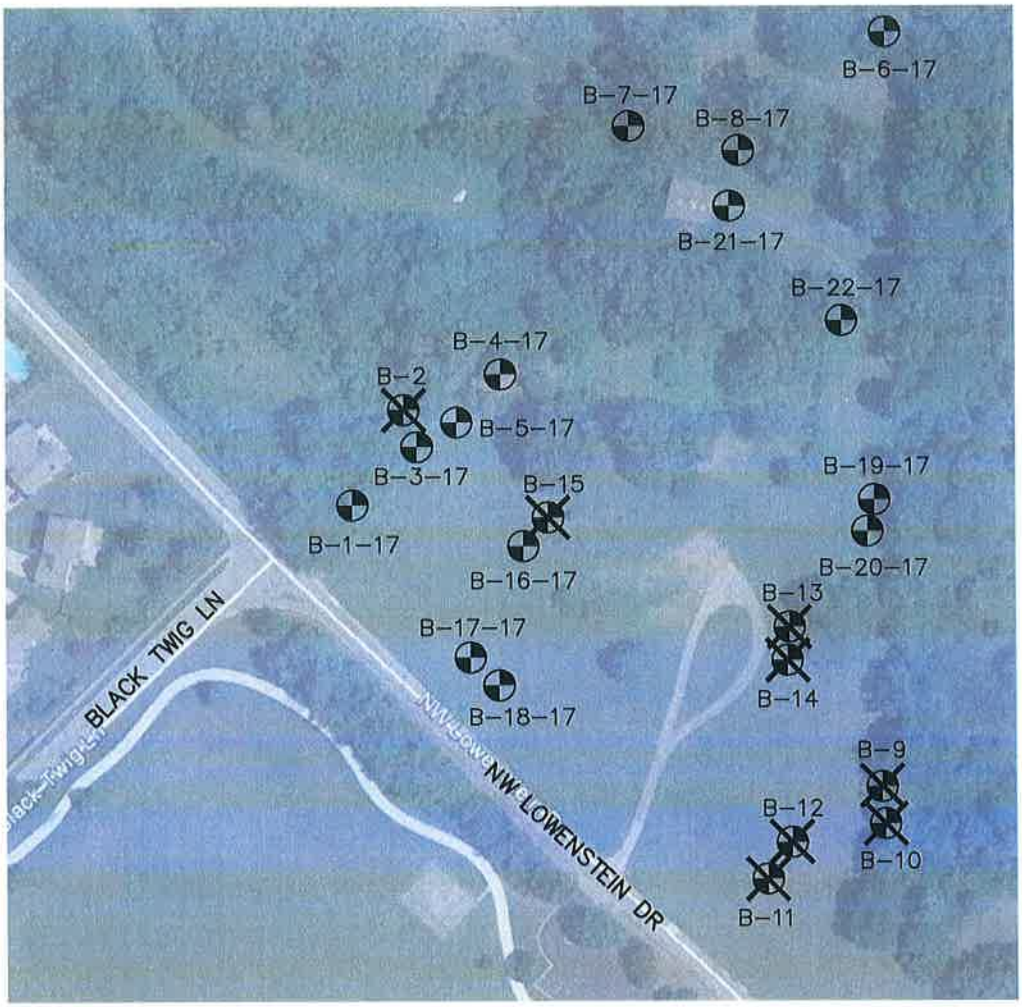
SURFACE ELEVATION: 1008.2'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
													TOPSOIL (12") 1007.2'
													FAT CLAY: Red/brown; stiff; moist 999.7'
	10												LIMESTONE: Brown; broken 998.7'
													SHALE: Light brown; slightly moist; highly weathered 987.9'
	20												LIMESTONE: Light gray 982.9'
	30												SHALE: Gray 943.0'
	70												LIMESTONE: Gray 937.5'
													SHALE: Gray; with limestone seams 934.2'
	80												LIMESTONE: Gray with shale seams 911.6'
	100												SHALE: Gray 910.7'
													SHALE: Dark gray 909.2'
													SHALE: Gray 902.1'
	110												LIMESTONE 890.0'
	120		BORING TERMINATED AT 118.2'										BOTTOM OF BORING AT 118.2'

Depth scale: 10, 20, 30, 40, 50, 60, 70, 80, 90, 100, 110, 120, 130

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:



LEGEND:

B-18-17

RE-ENTERED IN 2017

NOT RE-ENTERED

BORING LOCATIONS

PLATE 1
1 OF 2

ACADEMY SPORTS / SAM'S CLUB I-470 & NORTHWEST PRYOR ROAD LEE'S SUMMIT, MISSOURI		KAW VALLEY ENGINEERING
APPROVED BY: LKM	NOT TO SCALE	A14G7067

BORING LOCATION AND ELEVATION

B–1–17 –	NORTHING=1006185.0595	EASTING=2812141.2953	ELEV=996.61
B–2 –	NORTHING=1006259.3700	EASTING=2812181.1302	ELEV=993.67
B–3–17 –	NORTHING=1006230.3694	EASTING=2812191.2526	ELEV=996.26
B–4–17 –	NORTHING=1006286.8700	EASTING=2812256.1412	ELEV=997.71
B–5–17 –	NORTHING=1006249.4403	EASTING=2812222.0137	ELEV=997.16
B–6–17 –	NORTHING=1006554.3160	EASTING=2812554.9891	ELEV=998.86
B–7–17 –	NORTHING=1006480.7790	EASTING=2812356.2555	ELEV=997.63
B–8–17 –	NORTHING=1006461.9552	EASTING=2812441.2209	ELEV=1000.48
B–9 –	NORTHING=1005965.0150	EASTING=2812550.8260	ELEV=995.96
B–10 –	NORTHING=1005936.0757	EASTING=2812553.3286	ELEV=994.05
B–11 –	NORTHING=1005893.1673	EASTING=2812461.9669	ELEV=992.12
B–12 –	NORTHING=1005921.6942	EASTING=2812480.8347	ELEV=993.49
B–13 –	NORTHING=1006089.5918	EASTING=2812479.4193	ELEV=1003.80
B–14 –	NORTHING=1006063.6427	EASTING=2812477.7225	ELEV=1002.32
B–15 –	NORTHING=1006175.0857	EASTING=2812292.4965	ELEV=1002.60
B–16–17 –	NORTHING=1006152.8191	EASTING=2812273.3292	ELEV=1002.48
B–17–17 –	NORTHING=1006066.1171	EASTING=2812232.3483	ELEV=1000.79
B–18–17 –	NORTHING=1006044.2307	EASTING=2812254.5005	ELEV=1000.36
B–19–17 –	NORTHING=1006188.7714	EASTING=2812545.4507	ELEV=1007.27
B–20–17 –	NORTHING=1006164.1005	EASTING=2812540.0606	ELEV=1006.55
B–21–17 –	NORTHING=1006418.3910	EASTING=2812433.8902	ELEV=1001.31
B–22–17 –	NORTHING=1006328.6944	EASTING=2812520.4875	ELEV=1008.16

PLATE 1
2 OF 2

ACADEMY SPORTS / SAM'S CLUB
I-470 & NORTHWEST PRYOR ROAD
LEE'S SUMMIT, MISSOURI

KAW VALLEY ENGINEERING

APPROVED BY: LKM	NOT TO SCALE	A14G7067

CFS ENGINEERS
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One Vision. One Team. One Call.

April 1, 2016

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Kansas City, Missouri
Lawrence, Kansas
Topeka, Kansas
Branson, Missouri
Springfield, Missouri
Jefferson City, Missouri

Pryor Crossing, LLC
9400 Reeds Road, Suite 100
Overland Park, Kansas 66207

Attn: Dave Olson

Re: Drilling Services
Pryor Road
Lee's Summit, Missouri

To Whom It May Concern:

CFS Engineers have reviewed and logged the results from air procession, rotary drilling at the proposed Pryor Crossing project off of Lowenstein Road in Lee's Summit, Missouri. The boring logs and a map of the borings locations are attached.

General findings include underground voids indicative of previous mining practices. The voids were encountered at an elevation of approximately 895 to 897 feet in limestone. "Dome out," voids are the result of the roof of the limestone ceiling of the mine shaft caving in up to the shale layer. "Dome out," voids in shale were encountered at an elevation of approximately 900 to 907 feet above sea level. Three (3) borings did not encounter voids during drilling and extended to the full 84 foot depth. These borings included: B-10 terminated at an elevation 883 feet, B-13 terminated at an elevation of 895 feet, and B-14 terminated at an elevation of 902 feet. Please note that this does not necessarily indicate that there is not a mine under these borings.

Additional information detailing the type of materials encountered and depths can be found in the attached boring logs.

We appreciate the opportunity to provide this service to Pryor Crossing, LLC.

Respectfully,
Cook, Flatt & Strobel Engineers, P.A.

Justin L. Clay, P.E.
Project Engineer

Holly Haywood, E.I.
Staff Engineer

Kenneth M. Blair, P.E.
Chairman

Robert S. Chambers, P.E.
President-Treasurer

Sabin A. Yañez, P.E.
Senior Vice President-Secretary

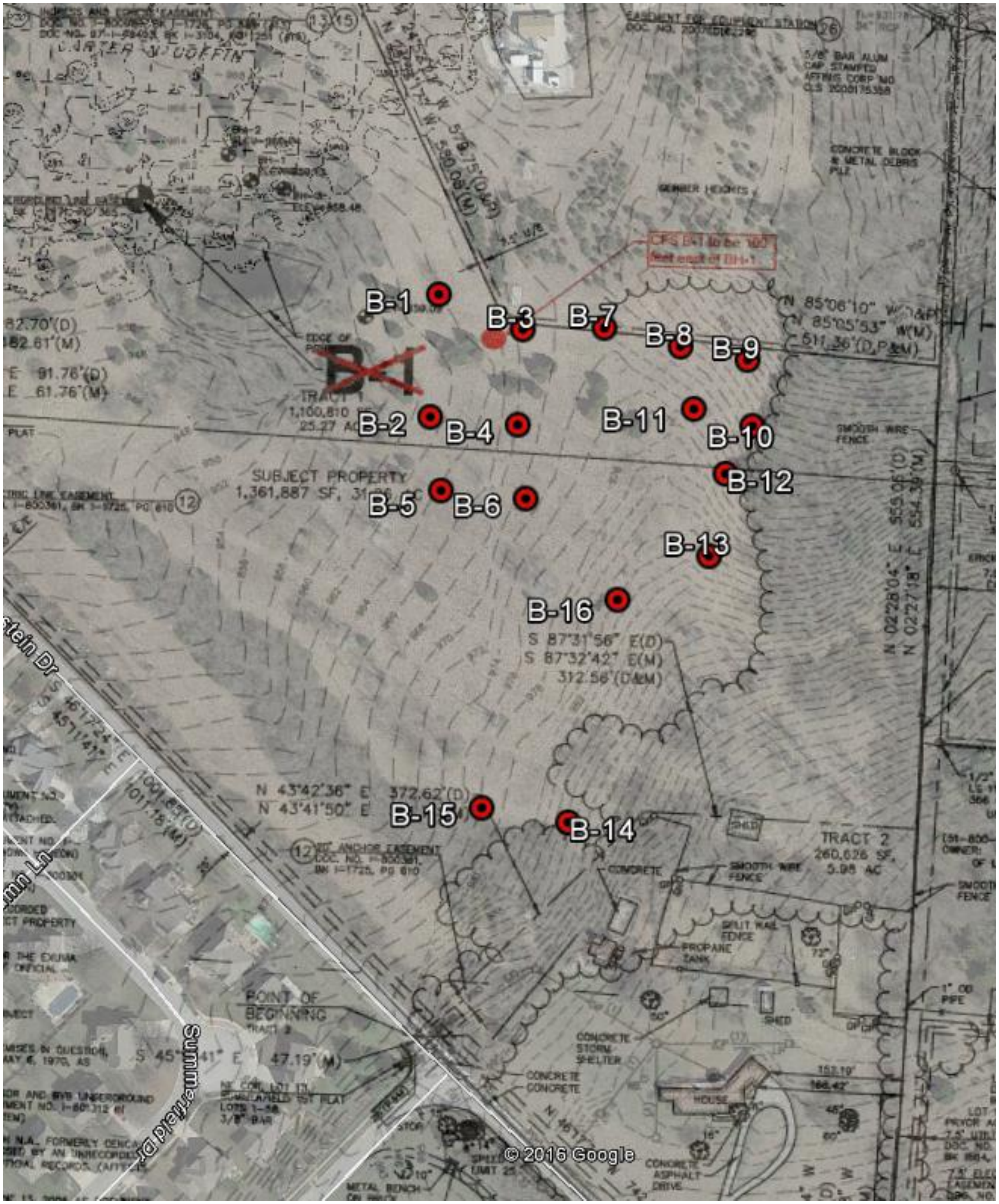
Kevin K. Holland, P.E.
Vice President

Daniel W. Holloway, P.E.
Vice President

Charles C. LePage, P.E.
Vice President

Lance W. Scott, P.E.
Vice President

Associates
Aaron Gaspers, P.E.
Mike Morrissey, P.E.
Gene Petersen, P.E.
Todd Polk, P.E.
William Stafford, P.E.
Richard Walker, P.E.
Lucas Williams, P.E.



CFS ENGINEERS
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Figure 1. Boring Locations
Pryor Road Drilling
Lee's Summit, Missouri

Drawn By: HH | Project #: 16-5078

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CFS Engineers
1100 W Cambridge Cir. Dr. Ste. 700
Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-1

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 965 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
11.0			LIMESTONE	954.0
16.0			SHALE	949.0
28.0			LIMESTONE	937.0
56.0			SHALE	909.0
66.0			LIMESTONE	899.0
69.0			VOID @ 69 feet	896.0

Depth scale: 0, 10, 20, 30, 40, 50, 60

Bottom of borehole at 69.0 feet.

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Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-2

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 961 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	
0			OVERBURDEN	
10		11.0	LIMESTONE	950.0
		16.0	SHALE	945.0
20				
30		29.0	LIMESTONE	932.0
40				
50		54.0	SHALE	907.0
		56.0	VOID (DOME OUT) @ 56 feet	905.0

Bottom of borehole at 56.0 feet.

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Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-3

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 969 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
18.0			LIMESTONE	951.0
29.0			SHALE	940.0
33.0			LIMESTONE	936.0
62.0			SHALE	907.0
65.0			LIMESTONE	904.0
74.0			VOID @ 74 feet	895.0

Depth scale: 0, 10, 20, 30, 40, 50, 60, 70

Bottom of borehole at 74.0 feet.

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BORING NUMBER B-4

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 968 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	
0			OVERBURDEN	
14.0			LIMESTONE	954.0
18.0			SHALE	950.0
31.0			LIMESTONE	937.0
56.0			SHALE	912.0
62.0			LIMESTONE	906.0
71.0			VOID @ 71 feet	897.0

Bottom of borehole at 71.0 feet.

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BORING NUMBER B-5

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 964 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0 – 12.0			OVERBURDEN	952.0
12.0 – 19.0			LIMESTONE	945.0
19.0 – 28.0			SHALE	936.0
28.0 – 56.0			LIMESTONE	908.0
56.0 – 62.0			SHALE	902.0
62.0 – 68.0			LIMESTONE	896.0

VOID @ 68 feet

Bottom of borehole at 68.0 feet.

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Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-6

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 970 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
12.0			SHALEY CLAY	958.0
19.0			LIMESTONE	951.0
25.0			SHALE	945.0
36.0			LIMESTONE	934.0
62.0			SHALE	908.0
66.0			LIMESTONE	904.0
74.0				896.0

VOID @ 74 feet

Bottom of borehole at 74.0 feet.

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Telephone: 913-627-9040

BORING NUMBER B-7

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 971 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
17.0			SHALEY CLAY	954.0
20.0			LIMESTONE	951.0
25.0			SHALE	946.0
39.0			LIMESTONE	932.0
65.0			SHALE	906.0
69.0			LIMESTONE	902.0
76.0			VOID @ 76 feet	895.0

Depth scale: 0, 10, 20, 30, 40, 50, 60, 70

Bottom of borehole at 76.0 feet.

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Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-8

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 968 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
11.0			SHALEY CLAY	957.0
17.0			LIMESTONE	951.0
29.0			SHALE	939.0
36.0			LIMESTONE	932.0
61.0			SHALE	907.0
63.0			VOID (DOME OUT) @ 63 feet	905.0

Depth scale: 10, 20, 30, 40, 50, 60

Bottom of borehole at 63.0 feet.

CFS Engineers
1100 W Cambridge Cir. Dr. Ste. 700
Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-9

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 964 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
6.0			SHALEY CLAY	958.0
12.0			LIMESTONE	952.0
18.0			SHALE	946.0
28.0			LIMESTONE	936.0
56.0			SHALE	908.0
61.0			LIMESTONE	903.0
68.0			VOID @ 68 feet	896.0

Bottom of borehole at 68.0 feet.

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Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-10

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 967 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0				
2.0			OVERBURDEN	965.0
8.0			SHALEY CLAY	959.0
20.0			LIMESTONE	947.0
23.0			SHALE	944.0
61.0			LIMESTONE	906.0
63.0			SHALE	904.0
84.0			LIMESTONE	883.0

Bottom of borehole at 84.0 feet.

CFS Engineers
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Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-11

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 972 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	
0			OVERBURDEN	
2.0			SHALEY CLAY	970.0
14.0			SHALE	958.0
21.0			LIMESTONE	951.0
25.0			SHALE	947.0
38.0			LIMESTONE	934.0
65.0			VOID @ 65 feet	907.0

Bottom of borehole at 65.0 feet.

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Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-12

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 972 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
3.0			SHALEY CLAY	969.0
7.0			SHALE	965.0
19.0			LIMESTONE	953.0
26.0			SHALE	946.0
38.0			LIMESTONE	934.0
64.0			SHALE	908.0
66.0			LIMESTONE	906.0
77.0			VOID @ 77 feet	895.0

Bottom of borehole at 77.0 feet.

CFS cfse.com
CFS Engineers
1100 W Cambridge Cir. Dr. Ste. 700
Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-13

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 979 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	
0			OVERBURDEN	
2.0			SHALEY CLAY	977.0
12.0			SHALE	967.0
29.0			LIMESTONE	950.0
73.0			SHALE	906.0
75.0			LIMESTONE	904.0
84.0				895.0

Bottom of borehole at 84.0 feet.

GENERAL BH / TP / WELL - GINT STD US.GDT - 3/31/16 20:03 - C:\USERS\HOLLY\GOOGLE DRIVE\ACTIVE GEOTECH PROJECTS\PRYOR ROAD DRILLING\BORING LOGS.GPJ

CFS cfse.com

CFS Engineers
1100 W Cambridge Cir. Dr. Ste. 700
Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-14

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 986 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	Elevation
0			OVERBURDEN	
6.0			SHALEY CLAY	980.0
12.0			SHALE	974.0
33.0			LIMESTONE	953.0
38.0			SHALE	948.0
48.0			LIMESTONE	938.0
78.0			SHALE	908.0
82.0			LIMESTONE	904.0
84.0				902.0

Bottom of borehole at 84.0 feet.

CFS cfse.com

CFS Engineers
1100 W Cambridge Cir. Dr. Ste. 700
Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-15

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 977 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	
0			OVERBURDEN	
		6.0	SHALEY CLAY	971.0
10				
		16.0	SHALE	961.0
20				
		23.0	LIMESTONE	954.0
30				
		33.0	SHALE	944.0
40				
		43.0	LIMESTONE	934.0
50				
60				
		65.0	SHALE	912.0
70		70.0	LIMESTONE	907.0
		77.0		900.0

VOID @ 77 feet

Bottom of borehole at 77.0 feet.

CFS cfse.com
CFS Engineers
1100 W Cambridge Cir. Dr. Ste. 700
Kansas City, KS 66103
Telephone: 913-627-9040

BORING NUMBER B-16

CLIENT Pryor Crossing, LLC
PROJECT NAME Pryor Road
PROJECT NUMBER 16-5078
PROJECT LOCATION Lee's Summit, Missouri
DATE STARTED 3/29/16 **COMPLETED** 3/29/16
GROUND ELEVATION 981 ft **HOLE SIZE** 4
DRILLING CONTRACTOR PD&B
GROUND WATER LEVELS:
DRILLING METHOD SSA
AT TIME OF DRILLING ---
LOGGED BY Roy **CHECKED BY** HH
AT END OF DRILLING ---
NOTES
AFTER DRILLING ---

DEPTH (ft)	SAMPLE TYPE NUMBER	GRAPHIC LOG	MATERIAL DESCRIPTION	
0			OVERBURDEN	
12.0			SHALEY CLAY	969.0
17.0			SHALE	964.0
30.0			LIMESTONE	951.0
36.0			SHALE	945.0
48.0			LIMESTONE	933.0
72.0			SHALE	909.0
77.0			LIMESTONE	904.0
84.0				897.0

Bottom of borehole at 84.0 feet.

LOG OF BORING B-20-17

Kaw Valley Engineering, Inc.
14700 W 114th Terrace
Lenexa, Kansas 66215
Telephone: (913)894-5150
Fax: (913)894-5977

CLIENT: Pryor Crossing, LLC
PROJECT: Academy Sports
NUMBER: A14D7067
LOCATION: SWQ NW Pryor RD & I-470 - Lee's Summit, MO
DATE(S) DRILLED: 7/17/17 - 7/17/17
DRILLING METHOD(S): 2-15/16" Tricone Hollow Stem Casing
DRILL RIG: CME 55
DRILL RIG OPERATOR: Les Scott
LOGGED BY: Les Scott
GROUNDWATER INFORMATION:
SURFACE ELEVATION: 1006.6'

SOIL SYMBOL	DEPTH (FT)	SAMPLES	N: BLOWS/FT P: TONS/SQ FT T: BLOWS/SIX INCHES REC: % RQD: %	RECOVERY (IN)	MOISTURE CONTENT (%)	ATTERBERG LIMITS LIQUID LIMIT LL	PLASTIC LIMIT PL	PLASTICITY INDEX PI	DRY DENSITY POUNDS/CUBIC FT	COMPRESSIVE STRENGTH (POUNDS/SQ FT)	CONFINING PRESSURE (POUNDS/SQ IN)	MINUS NO. 200 SIEVE (%)	DESCRIPTION OF STRATUM
													Boring was predrilled. KVE cased the boring with 3.25 inch ID hollow stem auger and continued to the depths explored.
	10												
	20												
	30												
	40												
	50												
	60												
	70												
	80												
	90												
	100												904.6' LIMESTONE: Broke tricone
	110												889.6'
	120												
	130												

Data above the 101 feet of depth level taken from the Pexco drill log dated 6/19/17.

Over Burden and Clay

Rock

Shale

Rock

Shale

Elevation 944.00

Rock

Shale

BORING TERMINATED AT 117.0'

N - STANDARD PENETRATION TEST RESISTANCE
P - POCKET PENETROMETER RESISTANCE
T - BLOWS PER SIX INCHES
REC - ROCK CORE RECOVERY
RQD - ROCK QUALITY DESIGNATION

REMARKS:

№ 15878

P D & B, Inc.
9900 Skyview Lane – Lenexa, Kansas 66220 (913)907-5022 / Fax (913) 273-0661

Driller ROY SIMPSON Drill # 66 Engine Hrs Start 5454 Stop 5466

DRILLING CUSTOMER Name PRYER CROSSING **Customer DRILL LOG or SHOT Number** ______

Fuel supplied by (circle one) PEXCO --- Quarry ---Other - ______ **Fuel used for this drill log** 120 LESS SUMMIT

JOB or P O NUMBER ______ **DRILLING LOCATION** TEST AREA **Mobilization Charge (circle one) YES NO**

DATE DRILLED 3/29/16 **DATE SHOT** ______ **Hammer Hrs Start** 2458.2 **Stop** ______

SPACING OF HOLES ______ **BURDEN OF HOLES** ______ **DIAMETER OF BIT** 4"

TOTAL DRILL FOOTAGE 1124' **# OF HOLES DRILLED** 16 **TOTAL DRILL HOURS** 10

#3
16 = OB
17 - 19 = ROCK
19 - 22 = SHALE
22 - 56 = ROCK
56 - 62 = SHALE
62 - 68 = ROCK
68

#4
14 = (OB)
14 - 18 = ROCK
18 - 31 = SHALE
31 - 56 = ROCK
~~56 = SHALE~~
56 - 62 = ROCK
62 - 65 = SHALE
65 - 71 = ROCK
71

#3
18 = (OB)
18 - 25 = ROCK
25 - 33 = SHALE
33 - 50 = ROCK
50 - 62 = ROCK
62 - 65 = SHALE
65 - 74 = ROCK
74

#2
11 = (OB)
11 - 16 = ROCK
16 - 29 = SHALE
29 - 43 = ROCK
~~43 - 54~~
43 - 54 = ROCK
54 - 56 = SHALE
56

#1
11 = (OB)
11 - 16 = ROCK
16 - 28 = HARD SHALE
28 - 45 = ROCK
45 - 56 = ROCK
56 - 66 = SHALE
66 - 67 = ROCK
67

SOUTH

#9
6 = (OB)
6 - 12 = CLAY
12 - 18 = ROCK
18 - 28 = SHALE
28 - 56 = ROCK
56 - 61 = SHALE
61 - 68 = ROCK
68

#8
11 = (OB)
11 - 17 = CLAY
17 - 29 = ROCK
29 - 36 = SHALE
36 - 50 = ROCK
50 - 61 = ROCK
61 - 63 = SHALE
63

#7
17 = (OB)
17 - 28 = CLAY
28 - 25 = ROCK
25 - 39 = SHALE
39 - 54 = ROCK
54 - 65 = ROCK
65 - 69 = SHALE
69 - 76 = ROCK
76

#6
12 = (OB)
12 - 19 = CLAY
19 - 25 = ROCK
25 - 36 = SHALE
36 - 62 = ROCK
62 - 66 = SHALE
66 - 74 = ROCK
74

5 3 1
6 4 3
2
#10 #9

SOUTH

Hole #	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Row Number	#10			#11			#12	3 = (OB)			#13				#12
Depth of Drilling	2 = (OB) 2 - 8 = CLAY			2 = (OB) 2 - 14 = CLAY 14 - 21 = HARD SHALE				3 - 7 = CLAY 7 - 19 = SHALE			2 = (OB) 2 - 12 = CLAY 12 - 29 = SHALE			#16	#13
Hole #	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Row Number	8 - 20 = ROCK 20 - 23 = SHALE			21 - 25 = ROCK 25 - 38 = SHALE				19 - 26 = ROCK 26 - 38 = SHALE			29 - 73 = ROCK 73 - 75 = SHALE			#15	#14
Depth of Drilling	23 - 61 = ROCK 61 - 63 = SHALE			38 - 65 = ROCK				38 - 64 = ROCK 64 - 66 = SHALE			75 - 84 = ROCK				
Hole #	31	32	33	34 65	35	36	37	38	39	40	84 41	42	43	44	45
Row Number	63 - 84 = ROCK 84							66 - 77 = ROCK 77							
Depth of Drilling															
Hole #	46 #14	47	48	49	50	51	52 #16	53	54	55	56	57	58	59	60
Row Number	6 = (OB)			#15 6 = (OB)				12 = (OB)							
Depth of Drilling	6 - 12 = CLAY 12 - 33 = SHALE			6 - 16 = CLAY 16 - 23 = SHALE 23 - 33 = ROCK			12 - 17 = CLAY 17 - 30 = SHALE								
Hole #	61	62	63	64	65	66	67 30 - 36 = ROCK	68	69	70	71	72	73	74	75
Row Number	33 - 38 = ROCK 38 - 48 = SHALE			33 - 43 = SHALE 43 - 50 = ROCK			36 - 48 = SHALE 48 - 54 = ROCK								
Depth of Drilling	48 - 72 = ROCK 72			50 - 65 = ROCK 65 - 72 = SHALE			54 - 72 = ROCK 72 - 77 = SHALE								
Hole #	76	77	78	79 72 - 77 = ROCK	80	81	82 77 - 84 = ROCK	83	84	85	86	87	88	89	90
Row Number	84			77			84								
Depth of Drilling															

20076

The PEXCO Company, LLC

Driller ROY SIMPSON Drill # 224 Engine Hrs Start 1295 Stop 1318

DRILLING CUSTOMER Name PRYOR CROSSING **Customer DRILL LOG or SHOT Number**

Fuel supplied by (circle one) PEXCO --- Quarry ---Other - **Fuel used for this drill log** 98

JOB or P O NUMBER **DRILLING LOCATION** LEE'S SUMMIT **Mobilization Charge (circle one) YES NO**

DATE DRILLED 6/19-6/20-6/21 **DATE SHOT** **Hammer Hrs Start** 526 **Stop** 586

SPACING OF HOLES **BURDEN OF HOLES** **DIAMETER OF BIT** 3½"

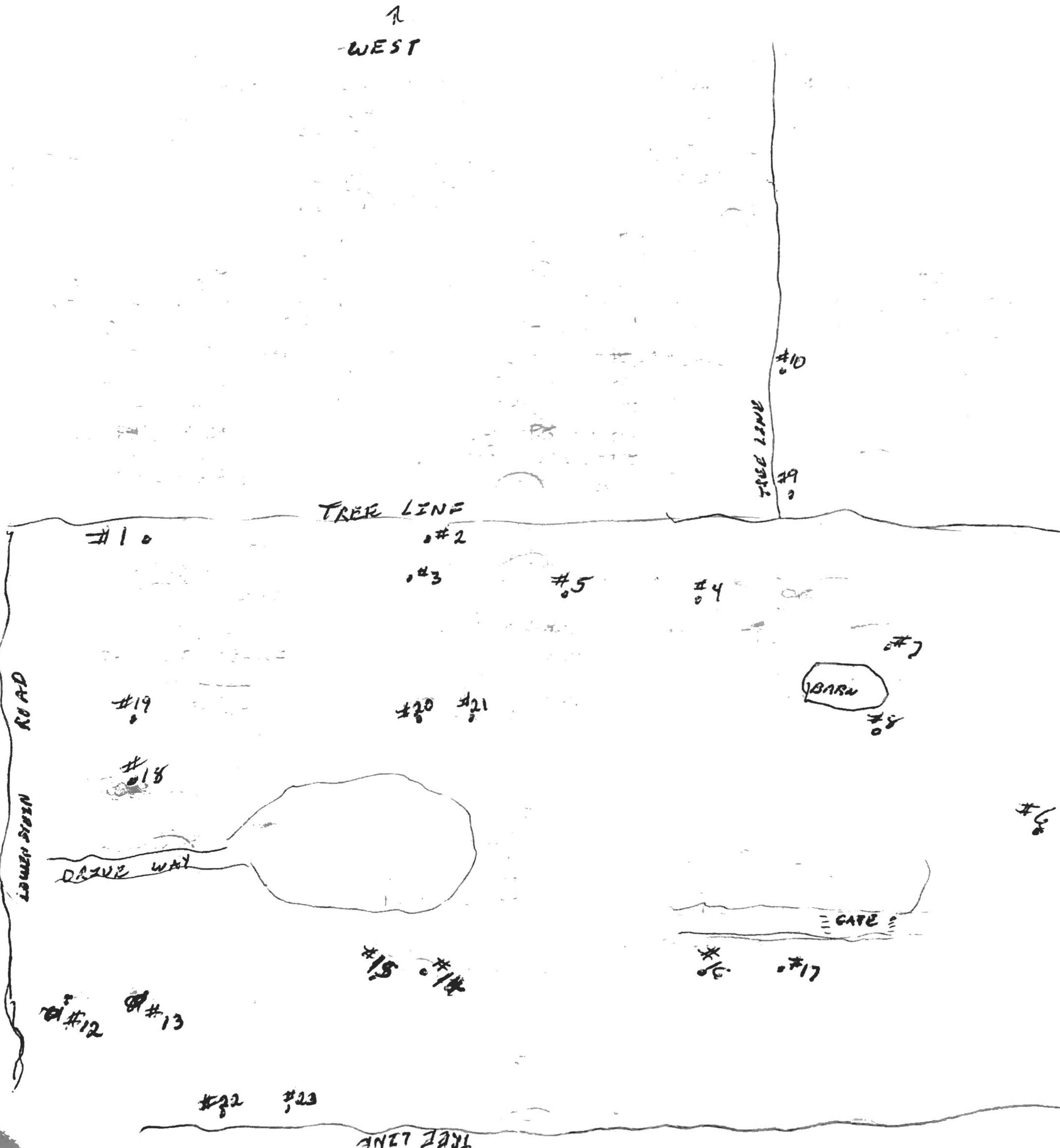
TOTAL DRILL FOOTAGE 2219' **# OF HOLES DRILLED** 23 **TOTAL DRILL HOURS**

Bits worn out or broken on shot **Striker Bars or hammers worn or broken** **Steel worn or broken**

Refer to Kaw Valley Letter Dated August 4, 2017 for extension of these borings beyond 101 feet in depth.

Kaw Valley Boring Number is list below as B-X-17

	B-1-17	B-2	B-3-17	B-4-17	B-5-17	B-6-17	B-7-17	B-8-17	not used			B-11-17	B-12-17	B-13-17	B-14-17
Hole #	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Row Number	1	2													
Depth of Drilling	101	94	101	101	101	101	101	101	84	61	61	101	101	101	101
Hole #	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Row Number	101	101	101	101	101	101	101	101							
Depth of Drilling	B-20-17	B-19-17	B-18-17	B-17-17	B-16-17	B-15-17	B-10-17	B-9-17							
Hole #	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45
Row Number															
Depth of Drilling															
Hole #	46	47	48	49	50	51	52	53	54	55	56	57	58	59	60
Row Number															
Depth of Drilling															
Hole #	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75
Row Number															
Depth of Drilling															
Hole #	76	77	78	79	80	81	82	83	84	85	86	87	88	89	90
Row Number															
Depth of Drilling															


WEST
TREE LINE
TREE LINE
#10
#9
#1
#2
#3
#5
#4
#7
BARN
#8
#19
#20
#21
#18
#6
ROAD
DRIVE WAY
GATE
#15
#14
#16
#17
#12
#13
#22
#23
TREE LINE

6-19-17

#1 0-9 = (OB)
9-40 = SHALE
40-44 = ROCK
44-54 = SHALE
54-84 = ROCK
84-88 = SHALE
88-101 = ROCK

#2 0-7 = (OB)
7-14 = CLAY
14-38 = SHALE
38-43 = ROCK
43-53 = SHALE
53-81 = ROCK
81-87 = SHALE
87-94 = ROCK
X FELL OUT

#3 0-8 = (OB)
8-14 = CLAY
14-41 = (GRAY) SHALE
41-45 = ROCK
45-55 = SHALE
55-84 = ROCK
84-88 = SHALE
88-101 = ROCK

#4 0-10 = (OB)
10-13 = BROWN CLAY
13-42 = SHALE (GRAY)
42-47 = ROCK
47-57 = SHALE
57-86 = ROCK
86-90 = SHALE
90-101 = ROCK

#5 0-10 = (OB)
10-15 = (BROWN) CLAY
15-42 = SHALE
42-46 = ROCK
46-56 = SHALE
56-86 = ROCK
86-92 = SHALE
92-101 = ROCK

#6 0-7 = (OB)
7-14 = CLAY
14-17 = ROCK
17-46 = SHALE
46-49 = ROCK
49-59 = SHALE
59-88 = ROCK
88-92 = SHALE
92-101 = ROCK
X FELL OUT

#7 0-11 = (OB)
11-45 = SHALE
45-50 = ROCK
50-60 = SHALE
60-89 = ROCK
89-92 = SHALE
92-101 = ROCK
X FELL OUT

#8 0-14 = (OB)
14-60 = SHALE
60-90 = ROCK
90-95 = SHALE
95-101 = ROCK

#9 0-9 = (OB)
9-14 = SHALE
14-17 = ROCK
17-35 = SHALE
35-58 = ROCK
58-60 = SHALE
60-84 = ROCK

#10 0-10 = (OB)
10-14 = SHALE
14-16 = ROCK
16-30 = SHALE
30-55 = ROCK
55-59 = SHALE
59-61 = ROCK
X FELL OUT

#11 0-10 = (OB)
10-26 = SHALE
26-50 = ROCK
50-55 = SHALE
55-61 = ROCK
X FELL OUT

6/20/17

#12 0-9=(OB)
9-14 = BROWN CLAY
14-36 = GRAY SHALE
36-40 = ROCK
40-47 = SHALE
47-79 = ROCK
79-86 = SHALE
86-101 = ROCK

#13 0-6=(OB)
6-12 = BROWN CLAY
12-36 = SHALE
36-41 = ROCK
41-48 = SHALE
48-80 = ROCK
80-87 = SHALE
87-101 = ROCK

#14 0-9=(OB)
9-16 = CLAY
16-48 = SHALE
48-53 = ROCK
53-59 = SHALE
59-92 = ROCK
92-98 = SHALE
98-101 = ROCK

#15 0-9=(OB)
9-14 = CLAY
14-18 = ROCK
18-48 = SHALE
48-52 = ROCK
52-57 = SHALE
57-91 = ROCK
91-96 = SHALE
96-101 = ROCK

#16 0-14=(OB)
14-18 = CLAY
18-23 = ROCK
23-52 = SHALE
52-55 = ROCK
55-62 = SHALE
62-94 = ROCK
94-99 = SHALE
99-101 = ROCK

#17 0-7=(OB)
7-14 = CLAY
14-20 = SHALE
20-24 = ROCK
24-53 = SHALE
53-56 = ROCK
56-63 = SHALE
63-96 = ROCK
96-99 = SHALE
99-101 = ROCK

#18 0-11=(OB)
11-17 = CLAY
17-45 = SHALE
45-48 = ROCK
48-55 = SHALE
55-88 = ROCK
88-94 = SHALE
94-101 = ROCK

#19 0-11=(OB)
11-12 = CLAY
12-14 = ROCK
14-43 = SHALE
43-47 = ROCK
47-56 = SHALE
56-88 = ROCK
88-92 = SHALE
92-101 = ROCK

#20 0-9=(OB)
9-15 = CLAY
15-18 = ROCK
18-45 = SHALE
45-50 = ROCK
50-58 = SHALE
58-89 = ROCK
89-93 = SHALE
93-101 = ROCK

#21 0-9=(OB)
9-15 = CLAY
15-19 = ROCK
19-45 = SHALE
45-50 = ROCK
50-60 = SHALE
60-90 = ROCK
90-96 = SHALE
96-101 = ROCK

#22 0-8=(OB)
8-12 = CLAY
12-37 = SHALE
37-41 = ROCK
41-47 = SHALE
47-81 = ROCK
81-86 = SHALE
86-101 = ROCK

#23 0-7 = OB
7-11 = ROCK
11-39 = SHALE
39-43 = ROCK
43-50 = SHALE
50-83 = ROCK
83-90 = SHALE
90-101 = ROCK

APPENDIX B

Photographic Log

Project:	Mine Filling at Pryor Crossing	J035637.02
Site Location:	Star Excavation Quarry - Pit North Side of Mine	Date: 5-8-2020
Description: Base of the quarry pit where the mine space can be accessed. **Photo #1: Quarry Pit**		

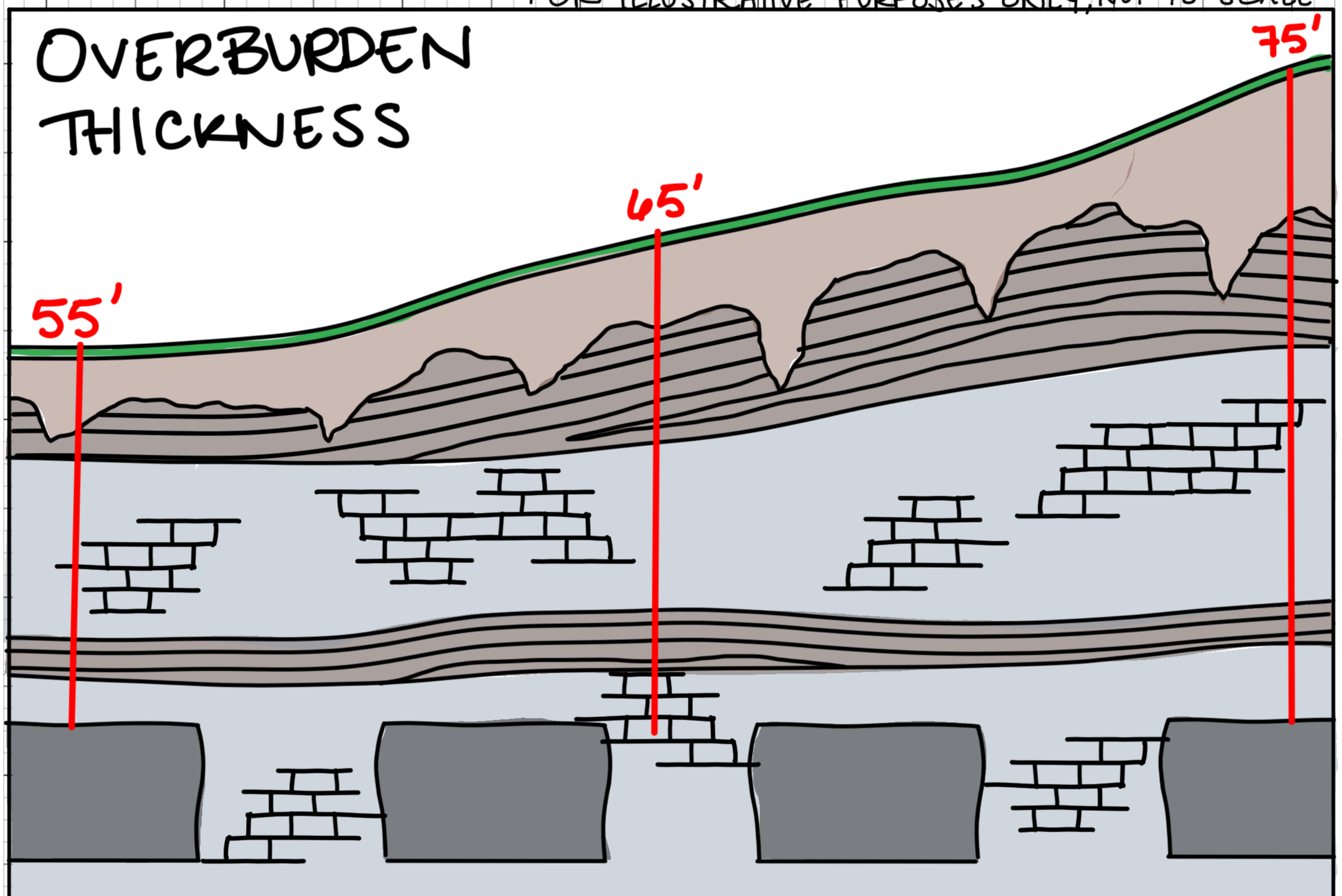
Project:	Mine Filling at Pryor Crossing	J035637.02
Site Location:	South Side of the Mine	Date: 5-8-2020
Description: Small dome-out and surrounding roof fall. Large loose rock on the roof. **Photo #2: Dome-Out**		

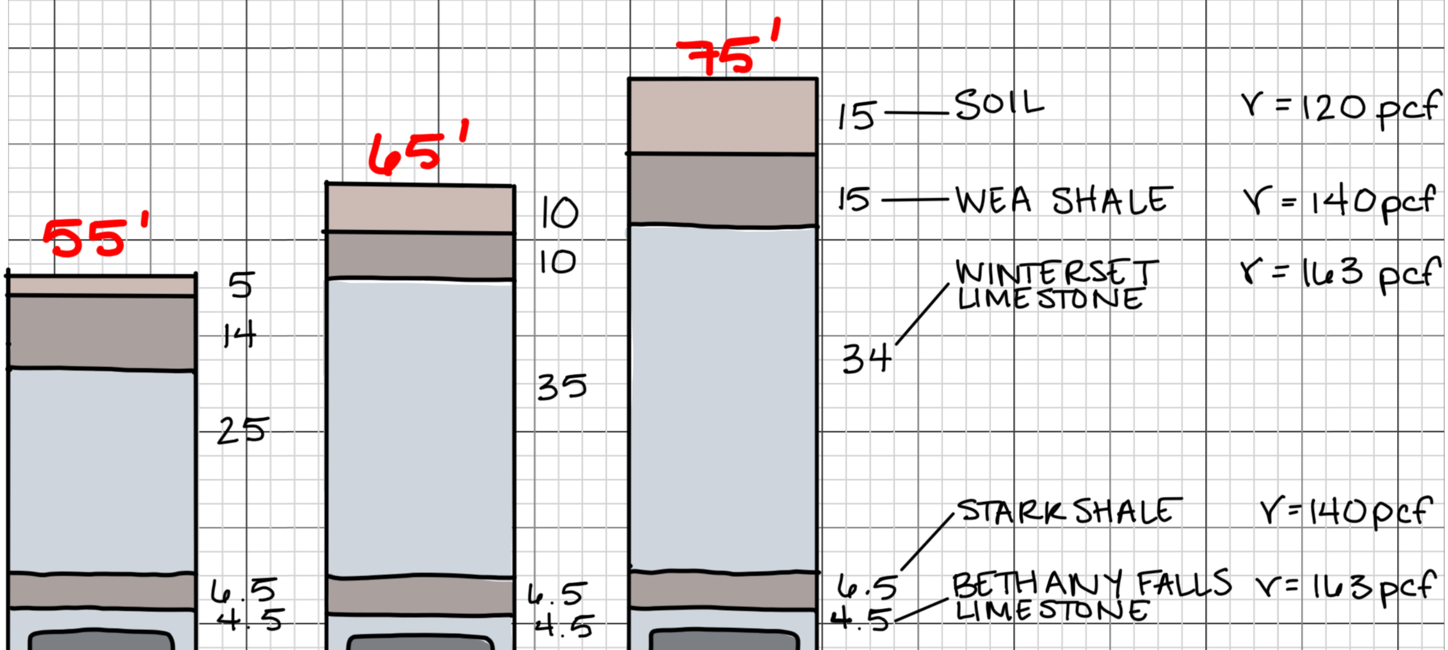
<u>APPENDIX C</u>

Calculations for the General Stability of the Mine Space

MINE MITIGATION STUDY	J035637.02
MINE STABILITY CALCULATIONS	

FOR ILLUSTRATIVE PURPOSES ONLY, NOT TO SCALE





Calc. By: alp	Checked By: ALP	Appvd By: ALP
Date: 1/9/2021	Date: 1/13/2021	Date: 1/13/2021

	MINE MITIGATION STUDY	J035637.02
	MINE STABILITY CALCULATIONS	

PILLAR STRESS FROM NIOSH INFORMATION CIRCULAR 9526 EQ1

$$\sigma_p = \underbrace{\gamma h}_{\sigma_v}\left(\frac{C_1 C_2}{wL}\right)$$

WHERE: γ = UNIT WEIGHT

h = THICKNESS OF OVERBURDEN

w, L = ROOM DIMENSIONS

C_1 = HEADING DISTANCE

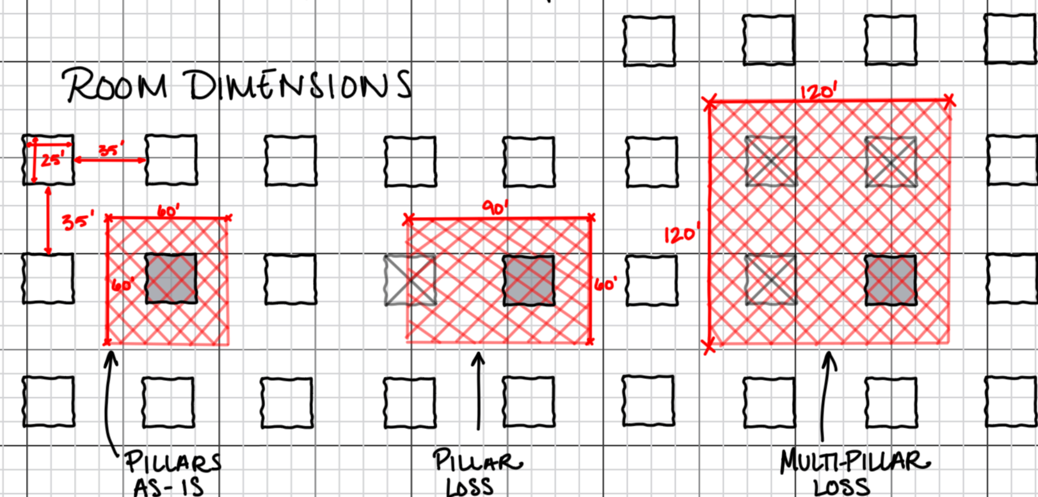
C_2 = CROSS CUT DISTANCE

$\sigma_v(h=55) = 120\,pcf(5ft) + 140\,pcf(14ft + 6.5ft) + 163\,pcf(25ft + 4.5ft) = 8278.5\,psf$

$\sigma_v(h=65) = 120\,pcf(10ft) + 140\,pcf(10ft + 6.5ft) + 163\,pcf(35ft + 4.5ft) = 9948.5\,psf$

$\sigma_v(h=75) = 120\,pcf(15ft) + 140\,pcf(15ft + 6.5ft) + 163\,pcf(34ft + 4.5ft) = 10485.5\,psf$

ROOM DIMENSIONS



h = 55 ft

$\sigma_p\,(AS\text{-}IS) = \frac{8278.5\,lb}{ft^2}\,\frac{(60ft)^2}{(25ft)^2} = 47684.16\,psf$ 331.14 psi

$\sigma_p\,(PILLAR\ LOSS) = \frac{8278.5\,lb}{ft^2}\,\frac{60ft(90ft)}{(25ft)^2} = 71526.24\,psf$ 496.71 psi

$\sigma_p\,(MULTI\text{-}PILLAR) = \frac{8278.5\,lb}{ft^2}\,\frac{(120ft)^2}{(25ft)^2} = 190736.64\,psf$ 1324.56 psi

h = 65 ft

$\sigma_p\,(AS\text{-}IS) = \frac{9948.5\,lb}{ft^2}\,\frac{(60ft)^2}{(25ft)^2} = 57303.36\,psf$ 397.94 psi

$\sigma_p\,(PILLAR\ LOSS) = \frac{9948.5\,lb}{ft^2}\,\frac{60ft(90ft)}{(25ft)^2} = 85955.04\,psf$ 596.91 psi

$\sigma_p\,(MULTI\text{-}PILLAR) = \frac{9948.5\,lb}{ft^2}\,\frac{(120ft)^2}{(25ft)^2} = 229213.44\,psf$ 1591.76 psi

h = 75 ft

$\sigma_p\,(AS\text{-}IS) = \frac{10485.5\,lb}{ft^2}\,\frac{(60ft)^2}{(25ft)^2} = 60396.48\,psf$ 419.42 psi

$\sigma_p\,(PILLAR\ LOSS) = \frac{10485.5\,lb}{ft^2}\,\frac{60ft(90ft)}{(25ft)^2} = 90594.72\,psf$ 629.13 psi

Calc. By: aly	Checked By: ALP	Appvd By: ALP
Date: 1/9/2021	Date: 1/13/2021	Date: 1/13/2021

MINE MITIGATION STUDY | J035637.02

MINE STABILITY CALCULATIONS

$$\sigma_p(\text{MULTI-PILLAR}) = \frac{10485.5\text{ lb}}{\text{ft}^2}\frac{(120\text{ ft})^2}{(25\text{ ft})^2} = 241585.92\text{ psf}$$ 1677.68 psi

PILLAR STRENGTH FROM NIOSH INFORMATION CIRCULAR 9526 EQ 3

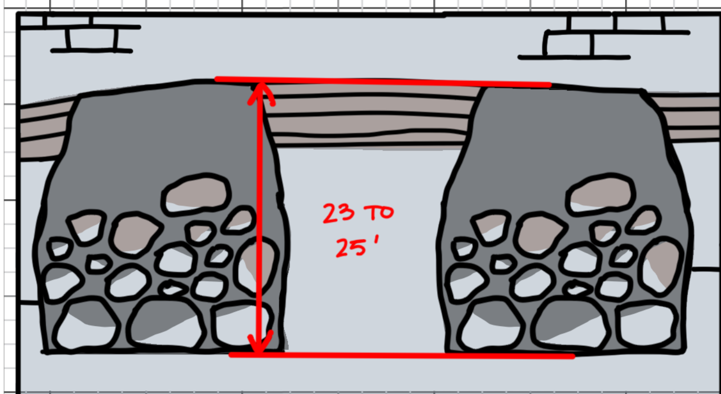
$$S = k\left(\frac{w^{0.3}}{h^{0.59}}\right)$$ WHERE: $k = 0.92$ UCS   USC = 10k psi

w = PILLAR WIDTH 25 ft

h = PILLAR HEIGHT VARIES FROM 12 – 14 ft

$$S(h=12\text{ ft}) = 0.92\left(\frac{10000\text{ lb}}{\text{in}^2}\right)\frac{(25\text{ ft})^{0.3}}{(12\text{ ft})^{0.59}} = 5577\text{ psi}$$

$$S(h=14\text{ ft}) = 0.92\left(\frac{10000\text{ lb}}{\text{in}^2}\right)\frac{(25\text{ ft})^{0.3}}{(14\text{ ft})^{0.59}} = 5092\text{ psi}$$



ASSUMING DOME-OUTS STOP AT THE WINTERSET PILLAR HEIGHTS BECOME 23 TO 25 FEET TALL WHEN SURROUNDED BY FAILURES.

$$S(h=23) = 0.92\left(\frac{10000\text{ lb}}{\text{in}^2}\right)\frac{(25\text{ ft})^{0.3}}{(23\text{ ft})^{0.59}} = 3799\text{ psi}$$

$$S(h=25) = 0.92\left(\frac{10000\text{ lb}}{\text{in}^2}\right)\frac{(25\text{ ft})^{0.3}}{(25\text{ ft})^{0.59}} = 3617\text{ psi}$$

ROOF BEAM

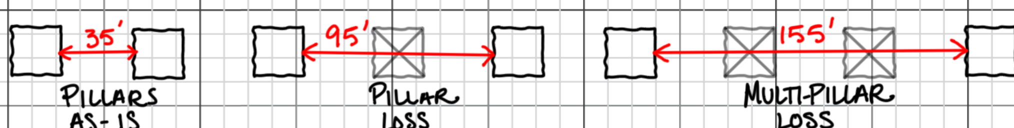
TENSILE STRESS IN ROOF BEAM FROM GOODMAN INTRO TO ROCK MECHANICS 2ND ED. EQ 7.5

$$\sigma_{t\max} = \frac{\gamma L^2}{2t}$$ WHERE: γ = UNIT WEIGHT 163 pcf

L = BEAM SPAN

t = BEAM THICKNESS 4.5 ft

ROOM DIMENSIONS



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$$\sigma_{tmax}(L=35) = \frac{\frac{163\ lb}{ft^3}(35\ ft)^2}{2(4.5\ ft)} = 22186.11\ psf \quad 154.07\ psi$$

$$\sigma_{tmax}(L=95) = \frac{\frac{163\ lb}{ft^3}(95\ ft)^2}{2(4.5\ ft)} = 163452.78\ psf \quad 1135.09\ psi$$

$$\sigma_{tmax}(L=155) = \frac{\frac{163\ lb}{ft^3}(155\ ft)^2}{2(4.5\ ft)} = 435119.44\ psf \quad 3021.66\ psi$$

SHEAR STRESS FROM GOODMAN INTRO TO ROCK MECHANICS 2ND ED. EQ 7.9

$$\tau = \frac{3\gamma}{2}\left(\frac{L}{2} - x\right)$$

WHERE: γ = UNIT WEIGHT

L = BEAM SPAN

X = DISTANCE FROM PILLAR

↑ HIGHEST STRESS @ $X=0$ & $X=L$

$$\tau(L=35, X=0) = \frac{3}{2}\left(\frac{163\ lb}{ft^3}\right)\left(\frac{35\ ft}{2} - 0\right) = 4278.75\ psf \quad 29.71\ psi$$

$$\tau(L=35, X=35) = \frac{3}{2}\left(\frac{163\ lb}{ft^3}\right)\left(\frac{35\ ft}{2} - 35\right) = -4278.75\ psf \quad -29.71\ psi$$

$$\tau(L=95, X=0) = \frac{3}{2}\left(\frac{163\ lb}{ft^3}\right)\left(\frac{95\ ft}{2} - 0\right) = 11613.75\ psf \quad 80.65\ psi$$

$$\tau(L=95, X=95) = \frac{3}{2}\left(\frac{163\ lb}{ft^3}\right)\left(\frac{95\ ft}{2} - 95\right) = -11613.75\ psf \quad -80.65\ psi$$

$$\tau(L=155, X=0) = \frac{3}{2}\left(\frac{163\ lb}{ft^3}\right)\left(\frac{155\ ft}{2} - 0\right) = 18948.75\ psf \quad 131.59\ psi$$

$$\tau(L=155, X=155) = \frac{3}{2}\left(\frac{163\ lb}{ft^3}\right)\left(\frac{155\ ft}{2} - 155\right) = -18948.75\ psf \quad -131.59\ psi$$

UNCONFINED COMPRESSIVE STRENGTH (UCS) = 10000 psi

ESTIMATED TENSILE STRENGTH = 6.5% UCS = 650 psi

ESTIMATED SHEAR STRENGTH = 10% UCS = 1000 psi

CRITICAL BEAM LENGTH

FOR FS = 2

TENSION $\sigma_{tmax} = \frac{650\ psi}{2} = 325\ psi$

$$L_{CRITICAL} = \sqrt{\frac{2\sigma_{tmax}t}{\gamma}} = \sqrt{\frac{2\left(\frac{325\ lb}{in^2}\right)\left(\frac{12\ in}{ft}\right)^2 4.5\ ft}{\frac{163\ lb}{ft^3}}} = 50.8\ ft$$

SHEAR $\tau = \frac{1000\ psi}{2} = 500\ psi$

$$L_{CRITICAL} = \frac{4\tau}{3\gamma} = 588\ ft$$

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MINE MITIGATION STUDY | J035637.02

MINE STABILITY CALCULATIONS

FACTORS OF SAFETY		PILLAR HEIGHT	PILLARS AS-IS		PILLAR LOSS		MULTI-PILLAR LOSS	
PILLAR CRUSHING	55' OVERBURDEN	12'	$\frac{5577}{331.14}$	16.84	$\frac{5577}{496.71}$	11.23	$\frac{5577}{1324.56}$	4.21
		14'	$\frac{5092}{331.14}$	15.38	$\frac{5092}{496.71}$	10.25	$\frac{5092}{1324.56}$	3.84
		23'	$\frac{3799}{331.14}$	11.47	$\frac{3799}{496.71}$	7.65	$\frac{3799}{1324.56}$	2.87
		25'	$\frac{3617}{331.14}$	10.92	$\frac{3617}{496.71}$	7.28	$\frac{3617}{1324.56}$	2.73
	65' OVERBURDEN	12'	$\frac{5577}{397.94}$	14.01	$\frac{5577}{596.91}$	9.34	$\frac{5577}{1591.76}$	3.50
		14'	$\frac{5092}{397.94}$	12.80	$\frac{5092}{596.91}$	8.53	$\frac{5092}{1591.76}$	3.20
		23'	$\frac{3799}{397.94}$	9.55	$\frac{3799}{596.91}$	6.36	$\frac{3799}{1591.76}$	2.39
		25'	$\frac{3617}{397.94}$	9.09	$\frac{3617}{596.91}$	6.06	$\frac{3617}{1591.76}$	2.27
	75' OVERBURDEN	12'	$\frac{5577}{419.42}$	13.30	$\frac{5577}{692.13}$	8.86	$\frac{5577}{1677.68}$	3.32
		14'	$\frac{5092}{419.42}$	12.14	$\frac{5092}{692.13}$	8.09	$\frac{5092}{1677.68}$	3.04
		23'	$\frac{3799}{419.42}$	9.06	$\frac{3799}{692.13}$	5.49	$\frac{3799}{1677.68}$	2.26
		25'	$\frac{3617}{419.42}$	8.62	$\frac{3617}{692.13}$	5.23	$\frac{3617}{1677.68}$	2.16
ROOF BEAM	TENSION		$\frac{650}{154.07}$	4.22	$\frac{650}{1135.09}$	0.57	$\frac{650}{3021.66}$	0.22
	SHEAR		$\frac{1000}{29.71}$	33.66	$\frac{1000}{80.65}$	12.40	$\frac{1000}{131.59}$	7.60

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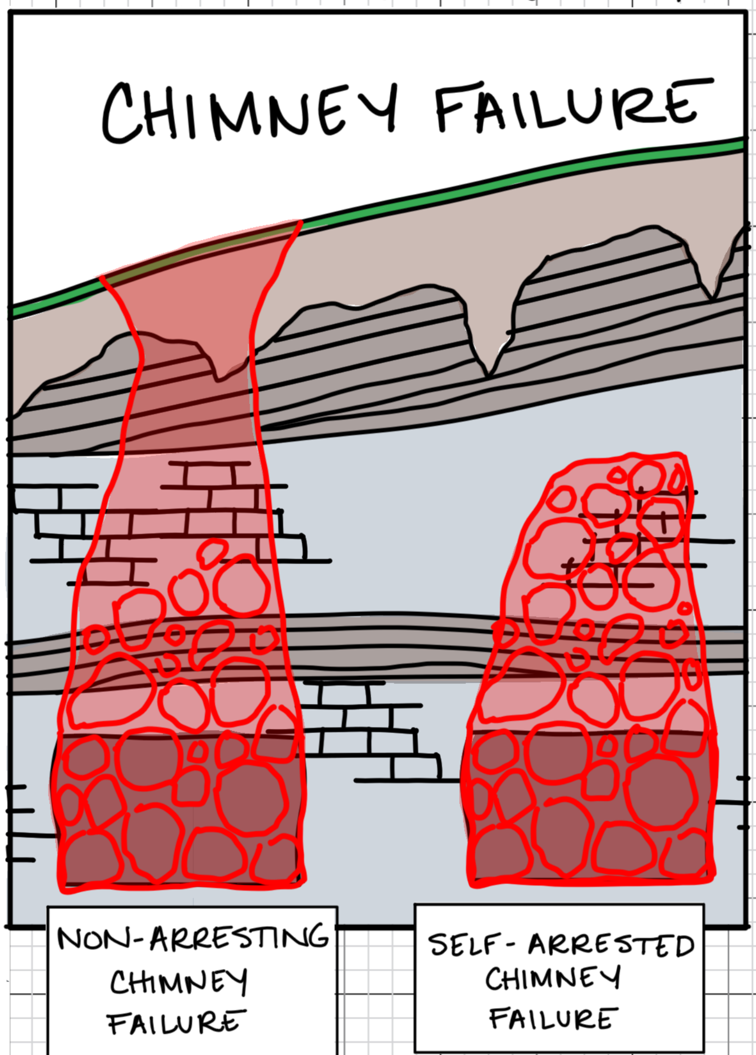
FOR ILLUSTRATIVE PURPOSES ONLY



CHIMNEY FAILURE IS LIMITED WHEN DEBRIS "CHOKES" OFF THE VOID SPACE

FROM PARISEAU "DESIGN ANALYSIS IN ROCK MECHANICS" CHAPTER 8

$$H = \frac{4h}{\pi}\left(\frac{1}{B} - 1\right)$$

WHERE: H = CHIMNEY HEIGHT
h = VOID/MINE HEIGHT
B = BULKING POROSITY
FOR A BULKING FACTOR = 1.25
B = 0.25

$$H(h=12) = \frac{4(12\text{ ft})}{\pi}\left(\frac{1}{0.25} - 1\right) = 45.8\text{ ft}$$

$$H(h=14) = \frac{4(14\text{ ft})}{\pi}\left(\frac{1}{0.25} - 1\right) = 53.5\text{ ft}$$

FOR LOW OVERBURDEN AREAS MINE SHOULD BE FILLED TO A DEPTH OF ATLEAST 2 FEET TO REDUCE CHIMNEY FAILURE RISK

CHIMNEY STRESSES : VERTICAL

$$\sigma_v = \left(\frac{\gamma A}{\mu k C}\right)\left[1 - e^{\wedge}\left(\frac{-\mu k C}{A} z\right)\right]$$

WHERE: γ = UNIT WEIGHT COMPOSITE $\gamma = 151$ pcf
A = CROSS SECTIONAL AREA $35' \times 35' = 1225\text{ ft}^2$
C = CIRCUMFERENCE $35' \times 4 = 140$ ft
$\mu = \tan\phi$ $\phi = 30°$
$k = \frac{1-\sin\phi}{1+\sin\phi}$ $\frac{1-\sin 30°}{1+\sin 30°} = 0.33$
Z = H FROM PREVIOUS CALCULATIONS $H = 46$ ft

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MINE MITIGATION STUDY J035637.02

MINE STABILITY CALCULATIONS

$$\sigma_v = \frac{151\ \text{pcf}\ (1225\ \text{ft}^2)}{\tan 30^\circ (0.33) 140\ \text{ft}} \left[1 - e^{\wedge}\left(\frac{-\tan 30^\circ (0.33) 140\ \text{ft}}{1225\ \text{ft}^2} (46\ \text{ft}) \right) \right]$$

$$= 6934.76\ \text{psf} \left(1 - e^{\wedge}(-1.002) \right)$$

$$= 4390\ \text{psf}$$

HORIZONTAL STRESS

$$\sigma_h = k\,\sigma_v$$

$$= 0.33\,(4390\ \text{psf})$$

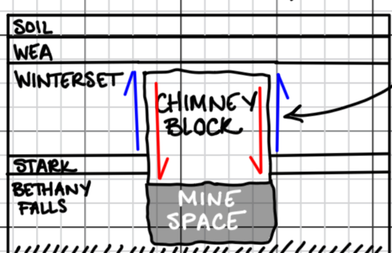
$$= 1448\ \text{psf}$$

SHEAR STRESS

$$\tau = \mu\,\sigma_h$$

$$= \tan(30^\circ)\,1448\ \text{psf}$$

$$= \frac{836\ \text{lb}}{\text{ft}^2} \left(\frac{\text{ft}}{12\ \text{in}} \right)^2 = 5.8\ \text{psi}$$



NOT TO SCALE

WHAT IS THE SHEAR STRENGTH OF THE WINTERSET?

TYPICAL VALUE: ~10,000 psi

SHEAR STRENGTH IS ESTIMATED AS ~10% OF UNCONFINED COMPRESSIVE STRENGTH

SHEAR STRENGTH OF WINTERSET = τ_W = 1,000 psi

FACTOR OF SAFETY AGAINST CHIMNEY FAILURE

$$FS = \frac{\tau_W}{\tau} = \frac{1{,}000}{5.8} = 172.4$$

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APPENDIX D

Mine Filling Methodology

C1

IC 9359

BUREAU OF MINES
INFORMATION CIRCULAR/1993

State-of-the-Art Techniques for Backfilling Abandoned Mine Voids

By Jeffrey S. Walker

UNITED STATES DEPARTMENT OF THE INTERIOR

U.S. Department of the Interior Mission Statement

As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally-owned public lands and natural resources. This includes fostering sound use of our land and water resources; protecting our fish, wildlife, and biological diversity; preserving the environmental and cultural values of our national parks and historical places; and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to ensure that their development is in the best interests of all our people by encouraging stewardship and citizen participation in their care. The Department also has a major responsibility for American Indian reservation communities and for people who live in island territories under U.S. administration.

Information Circular 9359

State-of-the-Art Techniques for Backfilling Abandoned Mine Voids

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UNITED STATES DEPARTMENT OF THE INTERIOR
Bruce Babbitt, Secretary

BUREAU OF MINES

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CONTENTS

ILLUSTRATIONS

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

cfm	cubic foot per minute	in	inch
fps	foot per second	pct	percent
ft	foot	psi	pound per square inch
gpm	gallon per minute		

STATE-OF-THE-ART TECHNIQUES FOR BACKFILLING ABANDONED MINE VOIDS

By Jeffrey S. Walker[1]

ABSTRACT

Abandoned underground mine openings are susceptible to collapse because of the mining methods used, the character of the overburden, and the typically large, wide entries with minimal roof support. The final effect of the collapse of the underground workings is surface subsidence. To reduce the probability of subsidence, methods to backfill the mine void with various types of materials have been developed. This U.S. Bureau of Mines report describes the available technologies for subsidence abatement and discusses their operation and application. The basis of these abatement methods is the replacement of the mined material with mine waste. Backfilling of mine voids is the most common method of stabilization used to abate subsidence and protect surface structures. Hydraulic flushing and grouting, using remote methods from single or multiple boreholes, are the most often-used methods for the placement of backfill material. Other subsidence abatement techniques are available and may be more appropriate under different conditions. These other techniques include pneumatic stowing, either by in-mine or remote methods, and various point support methods that do not completely fill the mine void and are used for the protection of small areas of the land surface and surface structures.

[1]Mining engineer, Pittsburgh Research Center, U.S. Bureau of Mines, Pittsburgh, PA (now with U.S. Department of Energy, Germantown, MD).

INTRODUCTION

Subsidence from abandoned underground coal mines has become an everyday concern of many citizens living in coal-producing regions of the United States. Because subsidence causes direct damage to property, with significant financial loss, and more importantly presents an extreme danger to public health, safety, and general welfare, subsidence and subsidence-related problems are considered by the Office of Surface Mining Reclamation and Enforcement (OSMRE) and affected States to be the highest priority issues relating to reclamation of abandoned mine lands (AML).

The development and improvement of methods to abate subsidence from the collapse of abandoned underground mines has been more or less neglected for the last 10 to 15 years. The same methods and procedures used in the late seventies are still used today. The objectives of the U.S. Bureau of Mines Abandoned Mined Land Subsidence Abatement Research Program are to improve subsidence abatement practices through the evaluation of current backfilling methods, to increase the use of sound geotechnical engineering principles in site investigation and remedial design, and to develop state-of-the-art tools and procedures for subsidence abatement. This work will provide a source of documented information describing the mechanism of subsidence development, applicable abatement methods, and remediation procedures.

BACKGROUND

The principal application of mine backfilling in the United States today is different from that of 100 years ago. Backfilling was first used in the anthracite coalfields of northeastern Pennsylvania to support the surface in active mining operations, to arrest the development of progressive pillar failure, to aid in the recovery of pillars, and to dispose of mine waste (*1*).[2] Today, backfilling is primarily used as a method to abate subsidence from the collapse of abandoned underground coal mines. There have been a few instances in which this technique was applied in active mines. However, with the increased use of high-extraction mining methods, backfilling during the mining operation may be a future option for subsidence control.

There are a number of differences between backfilling methods used in active mines to prevent surface subsidence and those used in abandoned mines. The greatest difference is the lack of access to abandoned mine voids. In active mines it is possible to directly observe the backfilling operation and control the location and amount of the fill material deposited. Whereas, in abandoned mines, often there is no access to the mine void. All work must be done from the surface in a remote fashion. It is this difference that separates the techniques used for in-mine backfilling and remote backfilling of abandoned mine voids. The primary concerns of in-mine backfilling programs are the logistics of material transportation through the mine and the capacity of the system. Remote backfilling operations require consideration of these factors, and also installation of equipment through boreholes, contending with unknown mine configurations and collapsed zones and blind monitoring of the fill.

There has been little change in the backfilling technology used during the past 100 years, and technical information to support the design of artificial pillars is almost wholly lacking in the literature. The most recent discussion of state-of-the-art methods for subsidence control was written in 1974 by the Appalachian Regional Commission (*2*). Since that time, progress has been made in the design of point support methods and delivery equipment, but the basic concepts and limitations remain the same.

There are two options available for the abatement of subsidence: point support and area-wide techniques. Point support techniques are used for situations in which subsidence is occurring in a relatively small area as a result of individual pillar failure. These methods provide additional support at a single location, such as at the center of a large mine opening or beneath a single surface structure of interest (fig. 1). Area-wide techniques are used where a large portion of the mine is at risk of collapse. The area-wide techniques fill large portions of the mine with fill material from multiple injection boreholes.

In general, there are four methods for the construction of artificial supports used in subsidence abatement: hydraulic, pneumatic, mechanical, and hand. Only hydraulic and pneumatic methods have widespread application to subsidence control in AML, because these methods can be implemented from the surface without the need for personnel to enter the mine. This type of backfilling operation is termed "blind" or "remote." It differs from in-mine backfilling techniques in that the remote method is typically performed through a borehole without placing any equipment into the mine except for the material injection pipe and nozzles. In-mine techniques utilize operators inside the mine to direct and control the flow of material into the void. Hand and mechanical methods, such as belt

[2]Italic numbers in parentheses refer to items in the list of references at the end of this report.

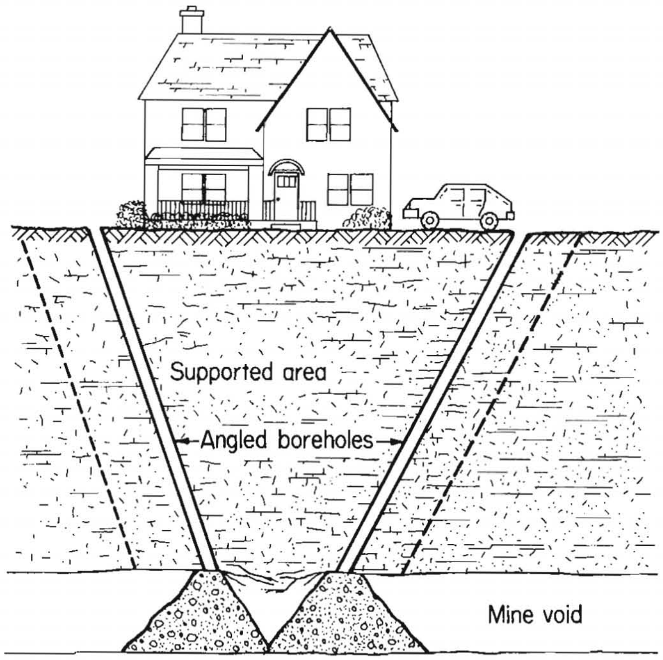

Figure 1.—Idealized example of point support method.

or sling packing machines, are restricted to construction of the support from within the mine. These methods have become obsolete or uneconomical for large-scale subsidence control and have been replaced with the use of pumped concrete for in-mine construction. Cribbing made from timber or concrete blocks remains the only type of hand-constructed support used today.

Inherent to remote backfill systems is the problem of tracking the placement of the backfill. It is difficult to monitor the size of the fill area and to determine the direction in which the fill area is growing (*3*). This problem is relevant to both pneumatic and hydraulic methods. However, because larger areas can be backfilled using the hydraulic method, the need to determine the growth of the fill becomes much more important.

Simply stated, hydraulic flushing consists of washing backfill material into a mine void using large quantities of water. The hydraulic flushing method is believed to have originated in the anthracite coalfields of Pennsylvania in 1864 in an effort to prevent the destruction of a church from mine subsidence (*1*). The practice of hydraulic flushing was further developed during the late 1800's and early 1900's and was used on a regular basis in the anthracite coalfields to extinguish fires and support areas of extremely high extraction. In 1901 hydraulic flushing was introduced into Europe, where it has been used extensively to dispose of mine waste into the abandoned sections of deep room-and-pillar mines. The practice of hydraulic flushing in the United States decreased with the decline of the anthracite industry in the 1940's. Currently, use of hydraulic flushing in the coal industry is limited to subsidence abatement in abandoned underground mines. However, in the metal mines of this country, hydraulic methods are commonly used to remedy ground control problems

Pneumatic stowing was first applied in Germany in 1924, and it has become the favored backfilling method, replacing hand packing and hydraulic methods. Pneumatic stowing consists of the transportation of backfill material into the mine through a pipeline by flowing air. In Europe the general use of pneumatic stowing is to place mine refuse in the gob areas behind advancing longwall faces. This practice not only provides roof support, but reduces or eliminates the disposal of mine waste on the surface. Pneumatic stowing is not used widely in the United States, and is typically limited to small-scale use for building mine seals or filling behind tunnel liners. In recent years, however, pneumatic stowing has been used in ground control applications in the Western United States where there is a lack of abundant water supplies or the mine workings cannot tolerate excessive moisture conditions (*4*).

"Hand packing" and "mechanical packing" typically refer to backfilling of very small mine areas for specific purposes, such as permanent support of unstable roof areas or replacement of barrier pillars during second and third mining operations. However, the term "packing" has been used as general term to describe any type of mine backfilling operation.

The type of abatement method to be used for each situation is not easily determined. Each method has its pros and cons. It is safe to say, however, that all of the subsidence abatement methods need to be further investigated to fully document their capabilities and to improve their application.

AREA-WIDE TECHNIQUES

HYDRAULIC FLUSHING

Hydraulic flushing is the practice of filling the mine void with various types of material (i.e., crushed stone, mine refuse, sand, or fly ash) by washing or pumping the material into the mine with water (fig. 2). The fill material is transported into the mine as a slurry and deposited in the void until the void is completely filled or there is a blockage of the injection pipe. Bulkheads and mine pillars are often used to limit the flow of the material to

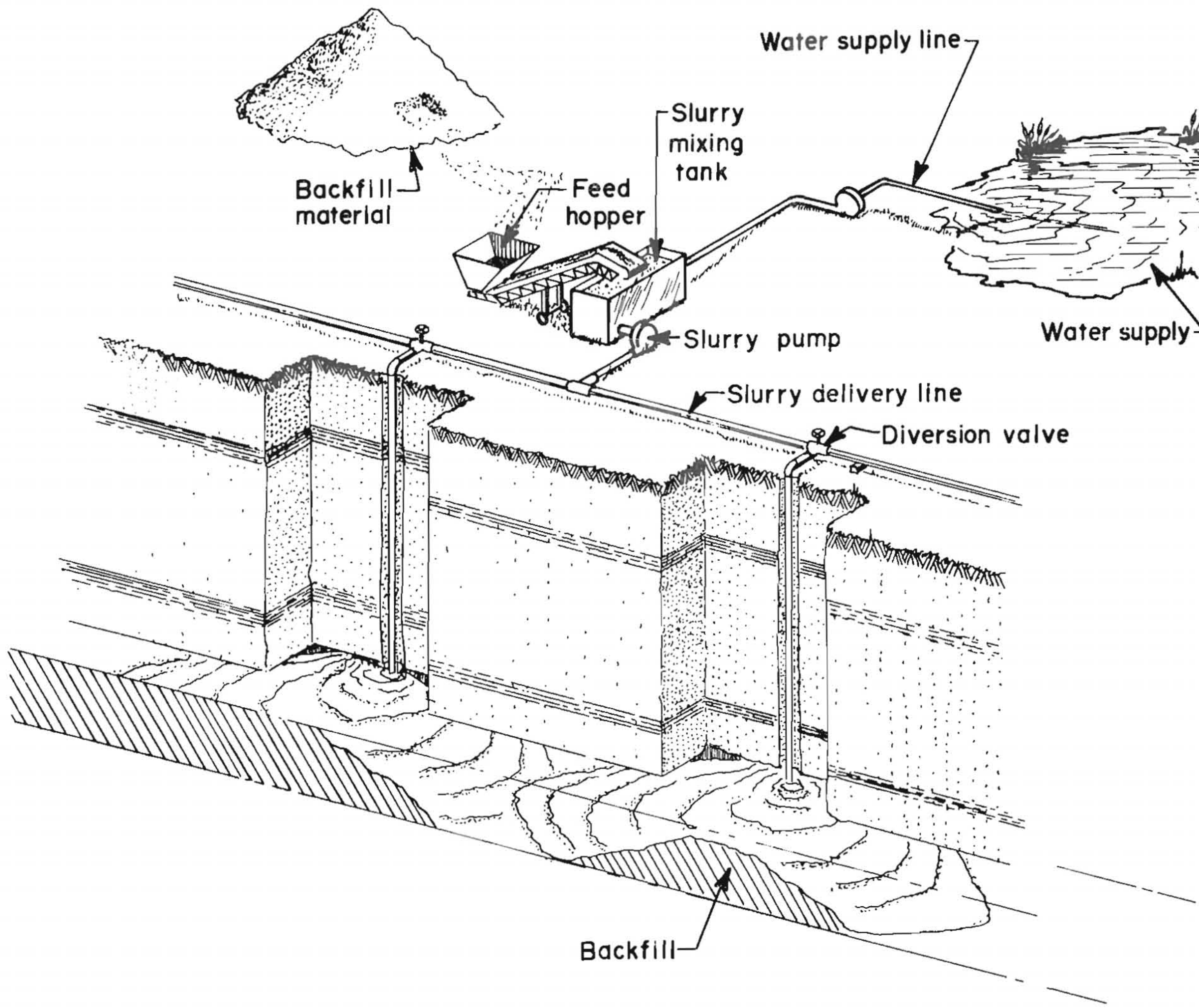

Figure 2.—Hydraulic backfilling.

a specific section of the mine. Current flushing methods require pumping of large amounts of water and material with little control over the direction and compaction of the resultant backfill. The available hydraulic flushing techniques include high-volume pumping, low-volume pumping, and gravity feed (*5*).

In the high- and low-volume pumping methods, granular material is blended with water in a large mixing tank at a processing plant (typically located some distance from the injection borehole) and is then pumped to a series of injection boreholes through pipelines (*6*). The energy provided from the pump and the static head of the borehole gives the backfill material the velocity required to keep the fill material in suspension and determines the size of the area that can be flushed from a single borehole. Normally, many boreholes spaced close to each other are required to backfill a large area.

The efficiency of the backfilling process within the mine is a function of the slurry velocity. When the slurry first enters the open mine void from the borehole, its velocity drops rapidly, and the solid particles drop out to form a donut-shaped mound on the mine floor (*7*). As the height of the mound approaches the mine roof, the velocity of the slurry is maintained by the narrowing of the opening between the roof and the fill material (fig. 3). When the slurry reaches the outer limit of the mound, the velocity decreases abruptly and solids are deposited in a process that is similar to the development of an alluvial fan. In an open void without obstacles, such as pillars or stoppings, the filling occurs in a circular fashion (*7*). However, in actual conditions, large mine openings are not usually encountered; the honeycomb pattern of room-and-pillar sections is more often typical. In a table-sized mine model simulating a typical arrangement of rooms, it was shown

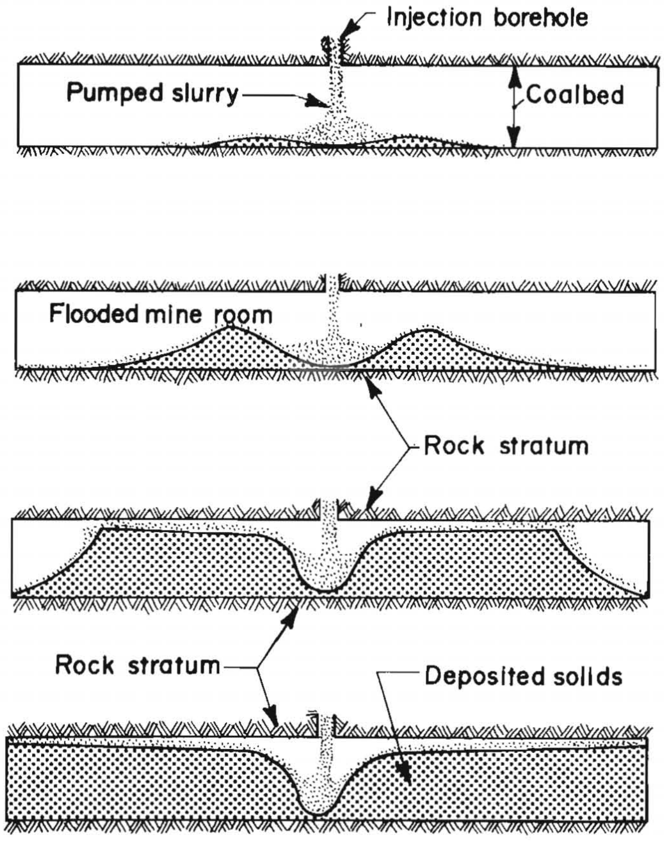

Figure 3.—Idealized hydraulic backfilling process (7).

that the flow of the slurry usually develops in one direction until the resistance to the flow becomes so great that another flow path is developed (*8*). This process is repeated until nearly all of the mine openings are filled or the slurry cannot be pumped any farther. Figure 4 represents the deposition pattern of hydraulic backfill in a model mine (*8*).

The lateral extent of the fill is determined by the energy of the pumped slurry and the condition of the mine. As the mound of fill builds outward into the mine, the flow channels between the mound and the mine roof become longer and more serpentine; therefore, the resistance to flow is increased. When this resistance becomes so great that the slurry can no longer transport the material beyond the fill, "refusal" or the limit of the backfilling is reached. In practice, it is extremely difficult to estimate the volume of fill needed for a specific area because of the uncertainty concerning the configuration of the mine and the tendency of the slurry to cut a channel through the existing fill and flow unrestricted into other parts of the mine (*9*).

The particle size distribution is variable throughout the fill; this is due to the changing velocity of the slurry. As the velocity of the slurry is decreased, the heavier particles fall out of suspension. The smaller and lighter pieces of fill material are then carried farther from the injection point and deposited. The resultant fill is therefore stratified based on the velocity of the slurry.

Where the dip of the coalbed exceeds 40°, a barrier or bulkhead may be needed to prevent large-scale movement of the slurry downdip away from the planned flushing area (*5*). This technique is often called controlled flushing. The barriers, bulkheads, or cutoff walls constructed across the mine openings are permanent obstructions designed to limit the flow of backfill material or to contain the material within a specific area. These obstructions are made from grout or gravel and can be installed remotely or from within the mine if access is available. The construction of these items is similar to the construction of several of the point supports, which will be discussed later in this report.

High-volume pumping is normally used on very large backfilling projects to place mine refuse or other available material into the mine void and spread it over a large area. It is typically used in mine areas that are known to be open and have few obstructions. The high volume of water and its velocity at the injection point ensure a wider spread of material than in any of the other methods since the energy of the slurry is much greater. High-volume pumping requires a large quantity of water and fill material. A minimum of 500 gpm is necessary, and as much as 8,000 gpm may be required depending on the size of the boreholes and the type of material (*5*). Fill materials with higher densities require a greater amount of flow to keep the solids in suspension in the slurry mixture.

The advantages of high-volume pumping are that a large amount of material can be placed in a short period of time and that low-cost, locally available backfill material can be used. The disadvantages of the system are the costs of establishing a central slurry mixing plant, a pipeline network, and a high-volume water supply. Furthermore, the addition of water to a formerly dry mine may cause sloughing and weakening of the pillars, floor, and roof.

In low-volume pumping, a combination of slurry and low-volume pumps is used to carry the material to the injection point. The water requirements are 100 to 500 gpm, about 10 times lower than for high-volume pumping (*5*). The advantage of the system is the typically small, portable, construction-type pumping equipment, which causes little congestion within the project area. The disadvantages are longer material placement times; the need for additional water, leading to adverse effects in the mine; and the need for additional boreholes, because the material spreads a limited distance from the injection point.

Gravity feed systems do not use mechanical pumps to transport material. The solids are dumped into a hopper, which feeds to a stream of water entering the mine. The

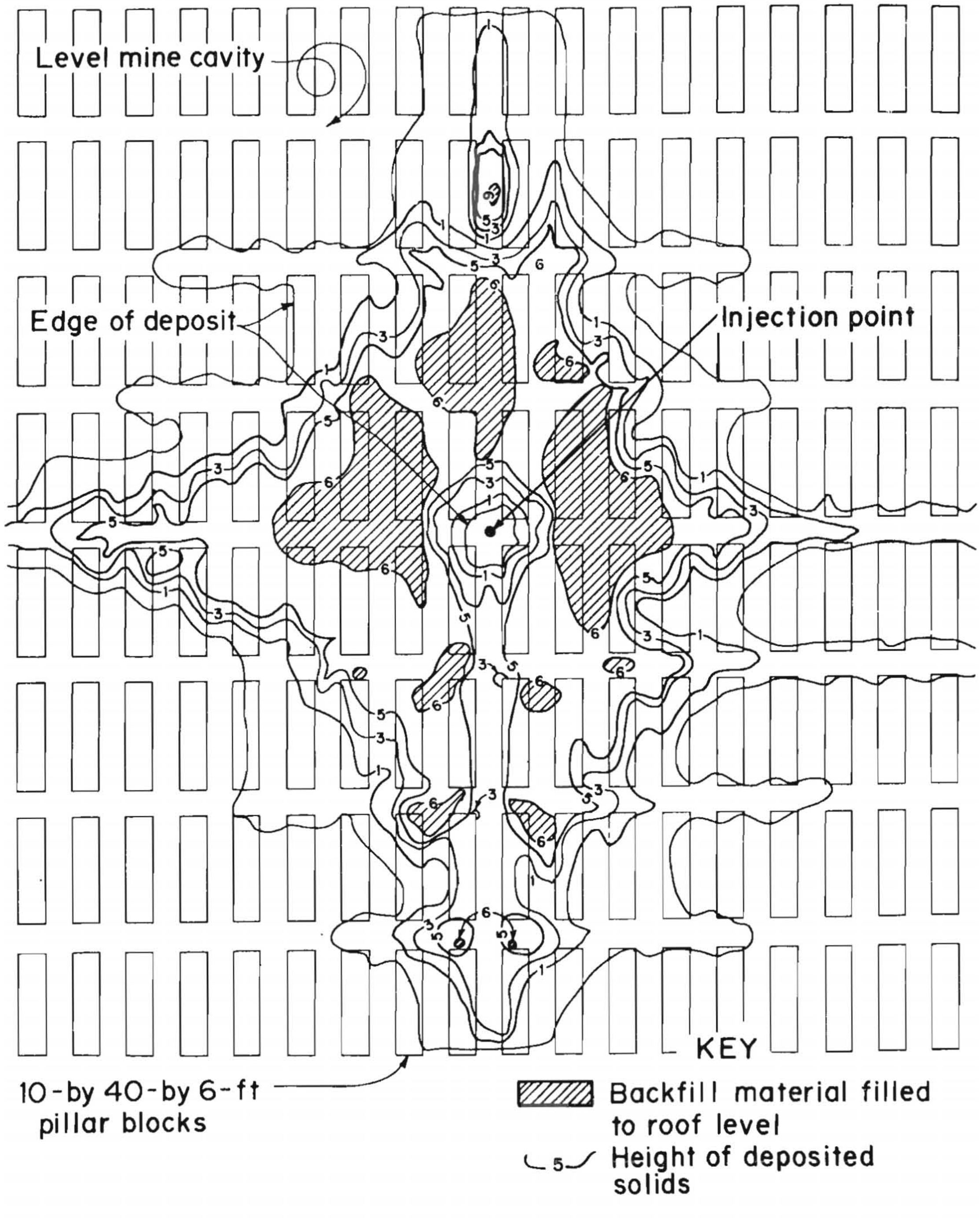

Figure 4.—Hydraulic backfill placement in model mine (8).

capacity of this type of system is very low and is limited by the type of material and the size of the borehole. The advantages of this system are the low operating cost and the small amount of required equipment. The disadvantages of this system include the following: material is not transported laterally beyond the injection point, injection rates are very slow, and filling the void without blocking the injection pipe is difficult.

An extensive amount of equipment is needed when pumping methods are used. In addition to a large supply of water and fill material, a screening and mixing plant and a network of pumps and pipelines to transport the slurry to the injection borehole or supply water to the mixing plant are required (fig. 5). All of this equipment should be sized in accordance with the rate of backfilling desired.

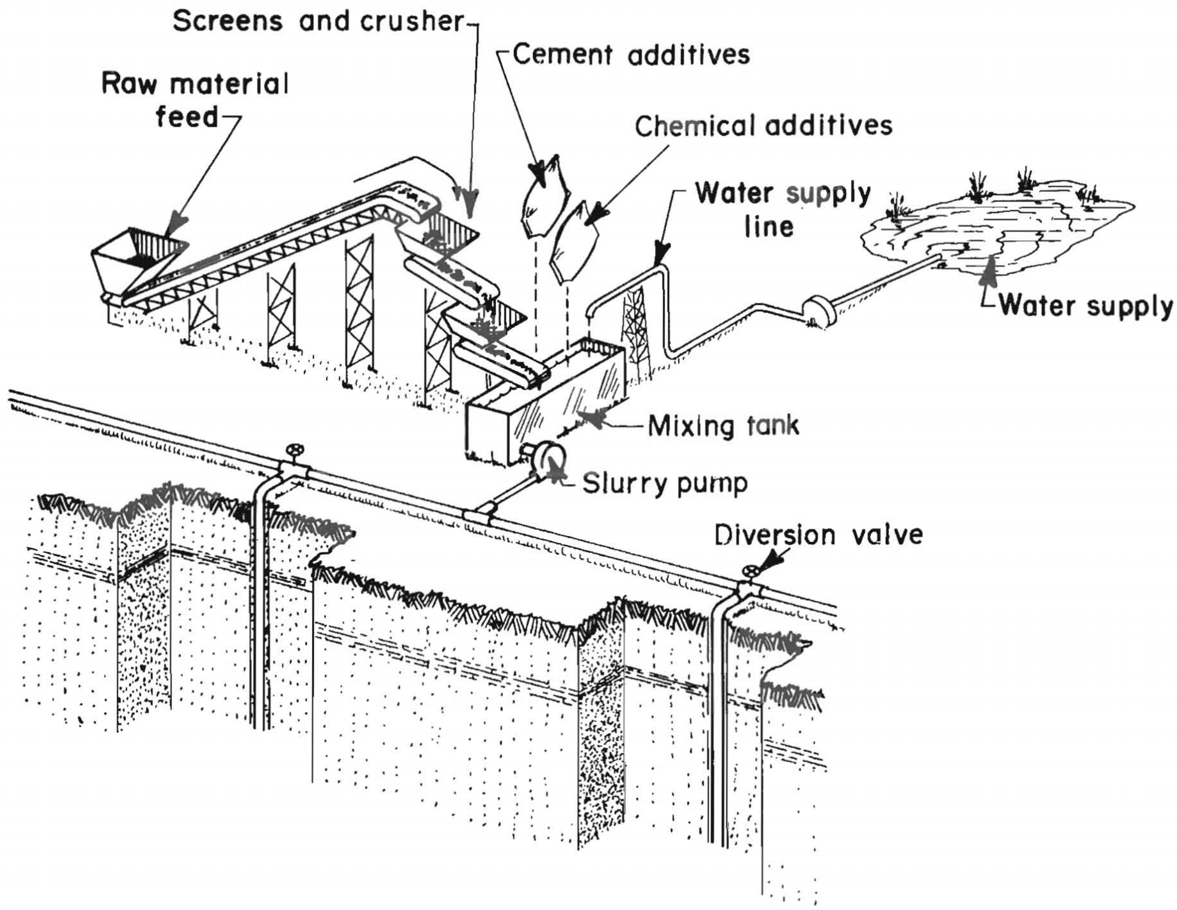

Figure 5.—Hydraulic backfilling equipment.

The greatest difficulty with hydraulic flushing is the control of the material in the mine. Boreholes are typically used to monitor the flow of the backfill during the flushing process. Downhole video cameras and/or observation wells are used to determine the extent of the backfill and the head pressure of the backfill at various locations during backfilling operations. However, because the configuration of the mine void is typically unknown and the opening is often blocked by fallen debris or abandoned equipment, the backfill material frequently flows in unexpected directions. Without knowing where the material has been transported in the mine it is impossible to determine the support capacity of the backfill.

In several cases, an amount of fill material that exceeds the estimated volume of the mine void has been pumped into a borehole without a pressure increase that would indicate filling of the void (*5*, *9-11*). A recent study found that at some sites where the flushing operation was considered a success the backfill material did not completely fill the void (*12*).

PNEUMATIC STOWING

"Pneumatic stowing" refers to a specialized form of pneumatic conveying in which mine backfill material is transported into a mine and "stowed" or placed into the void. At the present time, the technology has not been developed to the point where pneumatic stowing is generally accepted as a viable method of subsidence control. However, the problems are being overcome as its use becomes more common. The advantage of pneumatic stowing over hydraulic techniques is the elimination of water-related control problems. However, the fill material, with current technology, cannot be carried a great distance from the nozzle. This limitation prevents pneumatic stowing from being used as a truly area-wide technique. Furthermore, this method can only be used in dry mines.

The pneumatic stowing system is a more recent development than hydraulic flushing. Recent research indicates that it may be possible to pneumatically project material up to 80 ft beyond the injection site (*4*). Material placed using pneumatic methods can achieve a fairly high degree of compaction and good contact with the mine roof. The drawback to the system is the potential for excessive abrasion of the injection nozzle and elbows, which causes failure of the equipment after only a relatively small amount of material has been stowed.

There are two methods for pneumatic conveying (*3*). One is dense phase, in which the pipeline is nearly filled with material that is moved as a fluid with low-velocity air pressure in slugs. The other method is dilute phase, in which there is less than 5 pct fill material in the pipeline and it is moved at relatively high velocity as a fluid.

In general, the mining industry uses dilute phase conveying. However, some mines use rock dust systems, which utilize dense phase transport. The two major applications of pneumatics in the U.S. mining industry have historically been hoisting cuttings from shaft and tunnel boring operations and stowing.

There are few published cases of remote pneumatic stowing operations. No personnel are present at the discharge end of the pipeline during remote stowing operations, and all equipment used in the mine must be installed through a borehole (fig. 6). The available information indicates that the majority of remote pneumatic stowing operations were unsuccessful either because of a lack of site-specific information or from the inability to direct the trajectory of fill material (*3, 13*).

Research is progressing at the Bureau to improve the capabilities of pneumatic stowing systems. The problems with the systems include excessive wear on the injection pipeline, especially at the elbows and transition points, and inability to project the material into the mine void once it is injected down the borehole. Various configurations of collapsible elbows with nozzles are under development to redirect the material from the vertical injection pipe out into the mine void in a nearly horizontal fashion (fig. 7). The function of these elbow-nozzle assemblies is not only to redirect the material into the mine but to reentrain the material into the airstream. Reentrainment ensures maximum trajectory by placing the fill material above the airstream exiting the nozzle rather than below the airstream. It is calculated that if the research is successful

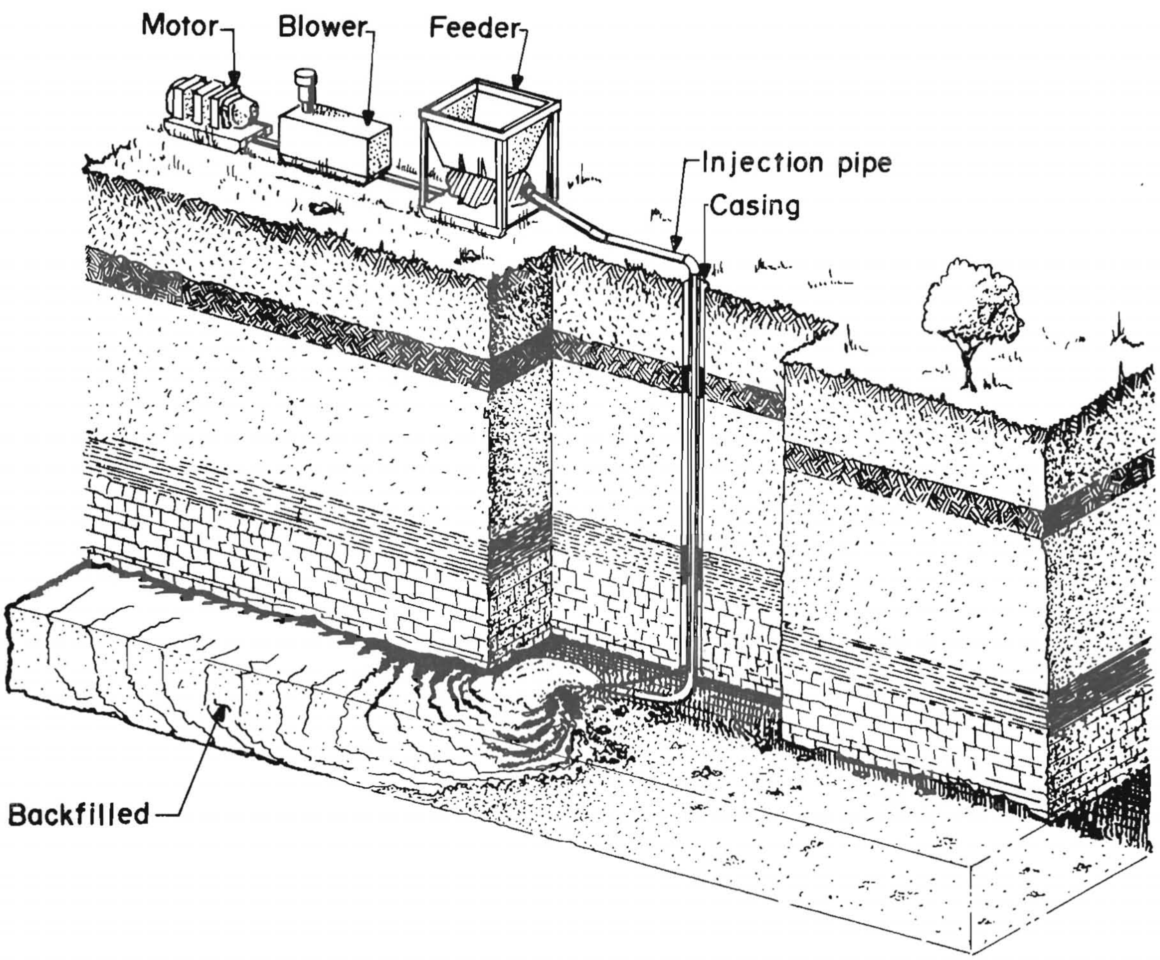

Figure 6.—Idealized pneumatic stowing process.

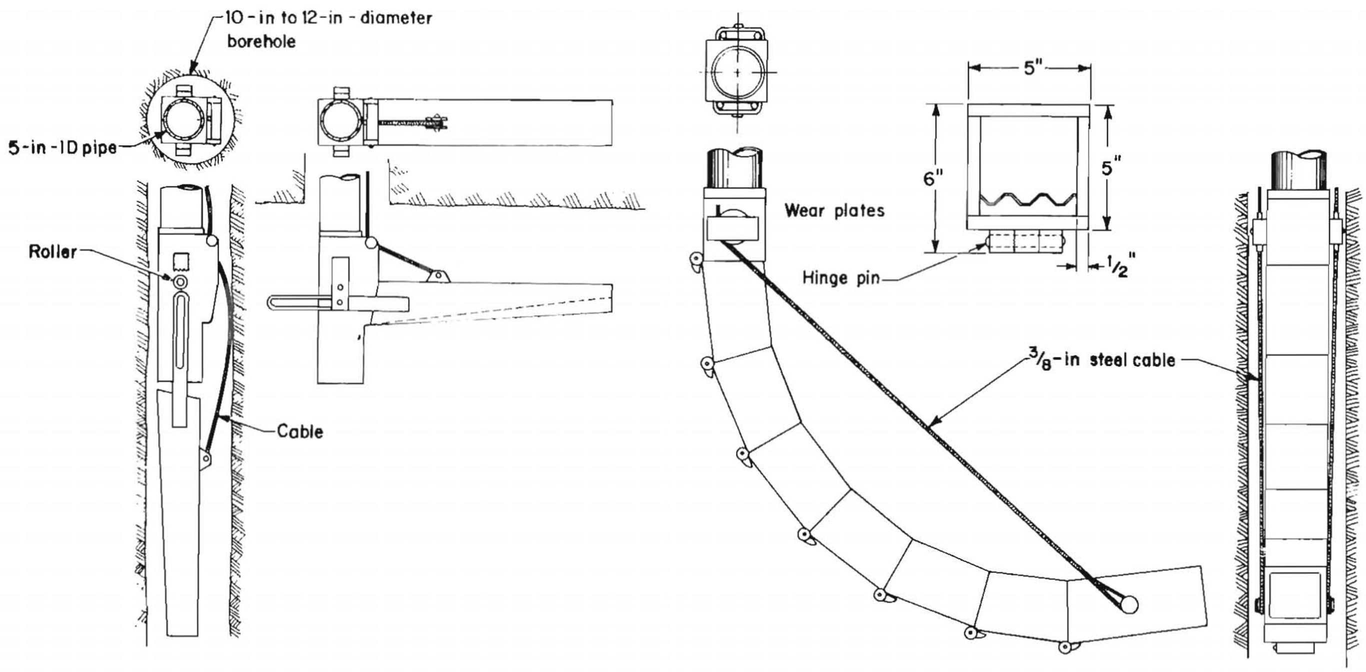

Figure 7.—Remote stowing elbows. Left, Short-radius elbow with blind T; right, segmented wide-radius elbow (*4*). (ID = inside diameter.)

elbow-nozzle assemblies of this type may be capable of projecting the fill material as far as 100 ft in a 6-ft-high mine entry (*4*).

A different type of elbow-nozzle assembly is also being researched. This type is called the pneumatic ejector (*14*). The pneumatic ejector consists of a high-pressure air nozzle fitted to an elbow at the end of the injection pipe (fig. 8). The high-pressure nozzle in the elbow creates a supersonic airstream (in excess of 1,600 fps) across the bottom of the injection pipe. As the backfill material falls through the injection pipe into the mine, a transfer of momentum occurs. The result is a combined stream of fill material and air moving horizontally at a velocity of 100 to 500 fps. This system was tested by the OSMRE and was found to be capable of projecting the material as far as 50 ft horizontally depending on the height of the mine opening (*15*). Theoretically, a 2,000-cfm airstream at 100 psi will project sand material as far as 90 ft in a mine void 7.5 ft high. The advantage of this system is that air velocity turns the fill material, thereby eliminating the abrasion on the injection equipment. Research is continuing, and it is expected that several different types of elbow-nozzle combinations will be developed.

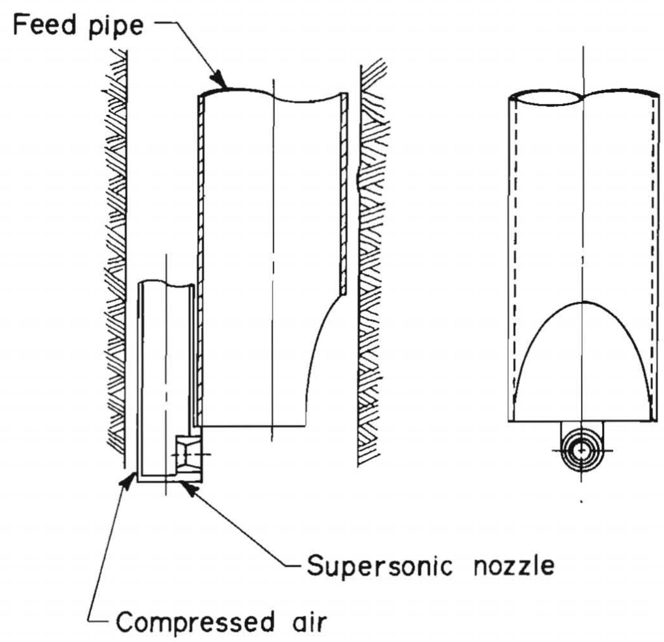

Figure 8.—Pneumatic ejector.

The principal equipment needed to perform pneumatic stowing is the same as for any other pneumatic conveying activity: a power pack, a compressor or blower, and an air-lock feeder (fig. 9). The differences between various applications are the quantity of air required, the pipe size, and the air pressure. These are all functions of the weight of the material to be conveyed and the length and elevation of the delivery pipeline.

The air supply for pneumatic stowing operations is provided by either a positive displacement blower or compressor. The majority of the pneumatic backfill equipment in the United States use rotary lobe positive displacement blowers, usually capable of supplying 3,000 to 5,000 cfm at 14 to 18 psi. Pneumatic stowing machines that are powered by compressed air are available and have been used extensively in European mines where up to 100 psi of air at large volumes is readily available.

The chamber air lock, screw feeders, and rotary air lock are three types of air-lock feeders that can be used to introduce the backfill material into the airstream without excessive loss of air. The rotary air-lock feeder is by far the most popular feeder. The size of the chamber air lock and the lack of positive sealing capability of the screw feeder make them less desirable. The rotary air-lock feeder is essentially a drum with screw-shaped pockets around the outside. Material is fed into one pocket at a time. As the drum rotates, the pocket is sealed in the housing and dumps its load into the airstream while another pocket is being filled. The rotary air lock is a high-maintenance item because of the abrasiveness of the backfill material.

The pipelines, elbows, and nozzles for pneumatic stowing are also high-wear items. In horizontal pipelines, pipe wear is fairly rapid and is most often found on the bottom of the pipe. Because of the excessive wear, the pipelines are made of hardened steel pipe. The pipe is rotated reg ularly to achieve longer life. Ramps are sometimes built into the pipeline to lift the material from the bottom of the pipe and reentrain it into the airstream rather than letting it slide along the bottom of the pipe. The wear on vertical pipelines is much less, usually only 10 pct of the wear found on horizontal piping.

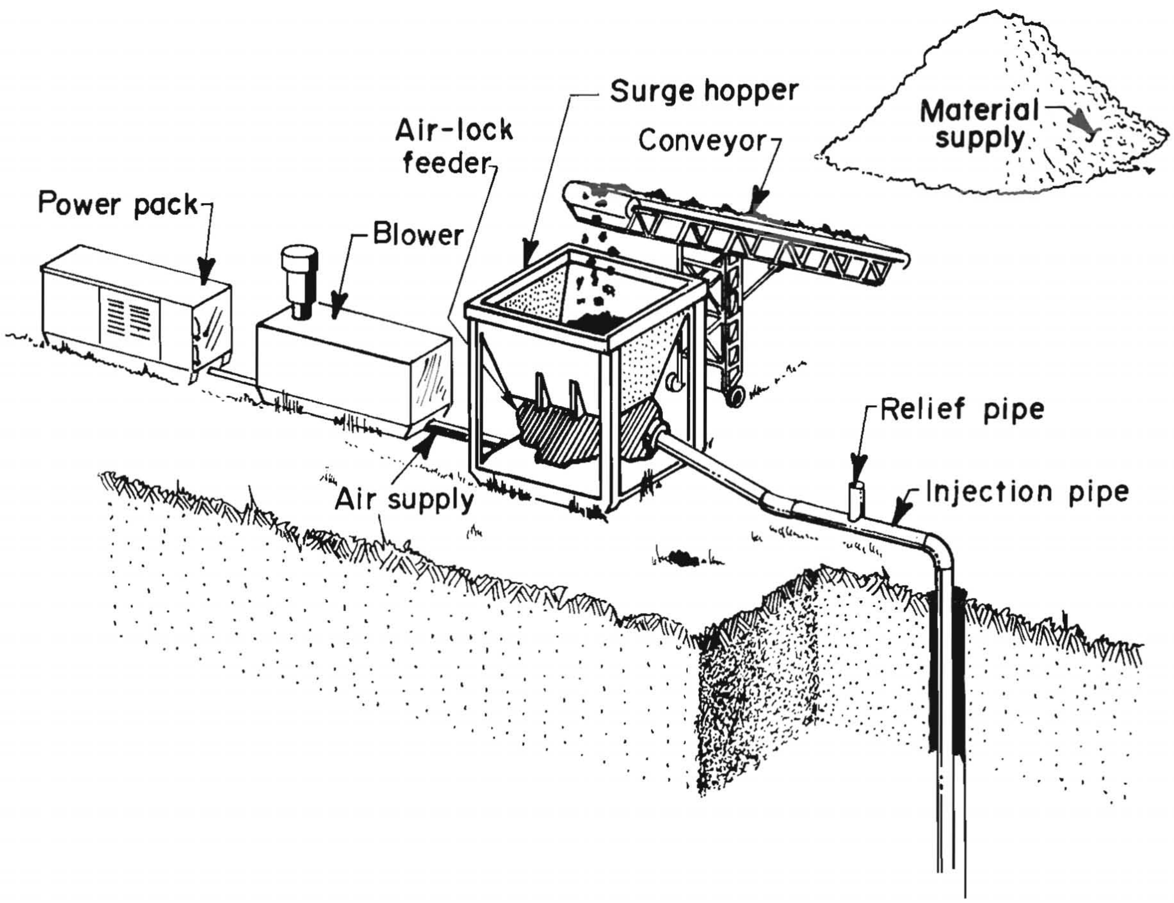

Figure 9.—Support equipment for conventional pneumatic stowing.

Elbows are required at places where the pipeline changes direction. The elbows must be specially designed to maintain the material entrained in the airstream and must include replaceable wearing portions. In a well-designed elbow, there is an impact section at the entry and a transition piece to accelerate the flow at the discharge. The liners of the elbows are typically manufactured from tool steel of at least 650 Brinell hardness and sometimes have a W-shaped cross section to maintain airflow beneath the entrained material and to reduce the wearing surface (*4*).

At the discharge end of the pipeline, a nozzle or deflector plate is usually used to direct the fill material. The deflector plate is usually a steel plate or a section of a pipe cut lengthwise. Nozzles have been used to concentrate the stream of material coming from the pipeline and direct the trajectory without moving the pipeline. It has been suggested that the nozzles would increase the degree of compaction achieved by directing all of the force to a smaller target area (*16*).

EXCAVATION

Excavation or daylighting of the mine void is a technique that can be used for shallow mines. This type of remedial action is similar to a surface mining operation. The area affected by subsidence is excavated to mine level, the remaining coal is removed, and the area is then backfilled to the original grade. This technique is suitable only for areas where little surface activity is present and where the surface topography would require extensive regrading because of the effects of subsidence. The cost of this method is extremely high, especially in areas that are not accessible to heavy surface equipment and are limited by environmental constraints.

BLASTING

Blasting to reduce the overburden to rubble and to collapse the mine roof in a controlled manner has been used as an area-wide subsidence abatement technique in one demonstration project. This project was located at the Urlacker Mine site near Dickerson, ND. This demonstration project was performed at a shallow mine site (30 to 50 ft of depth) and was considered a success. The blast did induce caving that extended to the surface, and the area has had no reported additional settlement or subsidence since 1983, when the project was concluded (*17*).

The concept of blasting to prevent subsidence is based on the fact that as a rock mass is reduced to rubble, it swells to fill a larger volume. Theoretically, the magnitude of the swelled volume is calculated to be equal to or slightly larger than the volume of the mine void. In deeper mines the rubbled zone is formed just above the coal seam, similar to a longwall operation; however, in a shallow mine the rubbled zone extends to the surface. The relationships among the open mine voids; the depth to the coalbed; and blast design considerations, such as charge weight, spacing, and delay times, are not well understood. In addition, the swell factors and the crater formation process are not well known for coal measure rocks.

The concerns with this type of reclamation include problems with blast vibrations in urban areas and environmental concerns such as damage to ground or surface water sources. Additional research is underway to address these concerns and to provide additional technical information.

POINT SUPPORT TECHNIQUES

Point support methods are used in areas where an analysis of the subsurface conditions indicates that the in situ support would not be sufficient for long-term stability. These methods are typically used with the intent of supporting only a small area such as a structure or a specific surface feature. These methods are extremely cost effective when applied correctly. However, the ability to accurately locate the artificial supports is limited by a lack of knowledge of the mine void and overburden characteristics.

GROUTING

"Grouting" is a general term that typically refers to the use of a fly ash-cement mixture as the backfill material. Generally grout is placed as grout columns, which consist of small cones of cemented backfill material extending from the mine floor to the roof directly beneath the injection borehole. The addition of cement to the backfill increases the strength of the material. The support capacity of grout-stabilized backfill is considered to be very good, and grouting is used regularly as a backfill method in small areas.

Historically, grout columns have been constructed from 6-in-diameter boreholes filled with crushed rock or gravel from the mine floor to the surface, which has been pressure grouted to form a permanent support. Newer methods utilize quick-setting, low-slump grout pumped through 2- to 6-in-diameter boreholes to completely fill small areas of open mine voids or to form self-supporting grout columns (fig. 10).

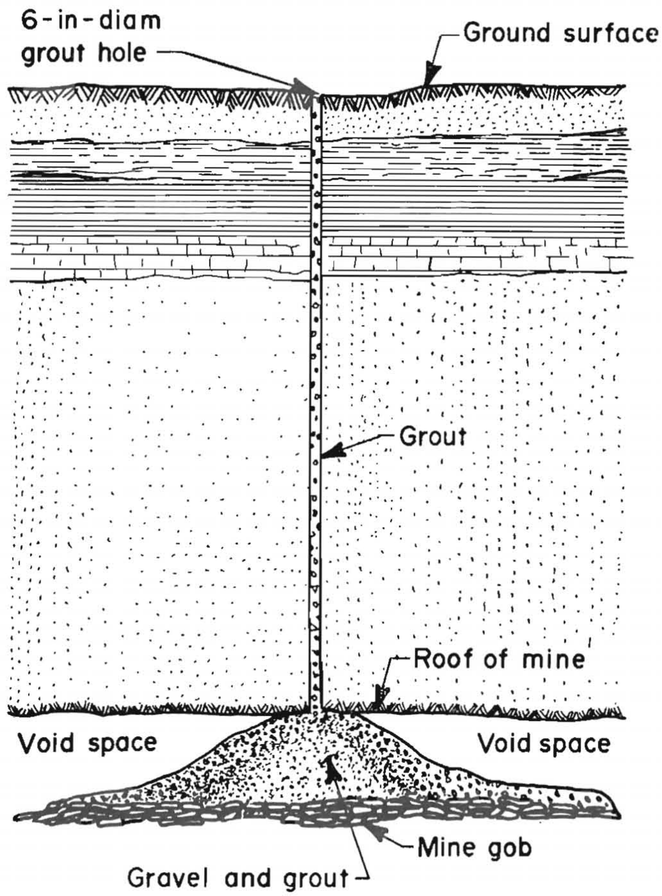

Figure 10.—Typical grout column.

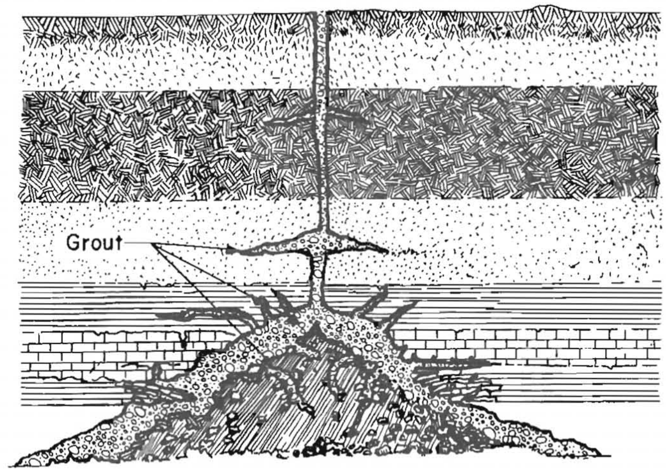

Figure 11.—Pressure grouting to stabilize collapsed overburden.

Grout can also be used in areas where caving has already occurred. In pressure grouting, grout is forced into the voids at a pressure on the order of 1/2 psi per foot of depth. This method not only fills the mine void but penetrates and stabilizes the fractures in the overburden that occurred as a result of subsidence (*5*). Figure 11 shows an idealized use of pressure grouting to stabilize the rubbled material resulting from a roof collapse and the resulting fractures in the overlying rock mass.

Grout mixtures vary according to the requirements of an individual project. For most applications, mixtures of portland cement and fly ash are sufficient, and these mixtures provide inexpensive and reliable fill material. Mixtures of 1 part cement to 3 to 9 parts sand or fly ash are typical (*18*). Chemical additives are available to modify the penetration and setting abilities of the grout mixtures. Chemical grouts are typically several times more expensive than cement grouts.

As with hydraulic flushing, the drawback to grouting is an inability to monitor placement and to determine areal distribution and percentage of mine roof contact.

A new technique for the construction of grout columns is the use of sodium silicate additive to the grout mixture (*19*). In this technique a chemical additive is applied to the outside of the grout as it is discharged. The sodium silicate reacts with the surface of the grout stream, forming a skin of calcium silicate that acts as a barrier. This allows the grout to form a self-supporting column. Figure 12 illustrates the device for applying the sodium silicate to grout as it is entering the mine. The sodium silicate technology can be used in both dry and flooded conditions. In dry conditions the calcium silicate skin contains the high-slump concrete, and in wet conditions the skin prevents water from penetrating the grout mass and diluting the mix. An example of the shape of a grout column using sodium silicate technology compared with an average grout column is shown in figure 13.

The strength and capability of grout columns made by this method have been tested by the OSMRE, and the columns have been proven to be effective (*19*). This technology should increase the use of grout pillars as a means of support, because it is now possible to control the placement of the grout.

Grout bags are another way to control the spread of grout during the formation of grout columns. The bags are placed into the mine void through a 6- to 8-in-diameter borehole. The grout bag, once installed in the mine, contains the grout pumped from the surface so that an

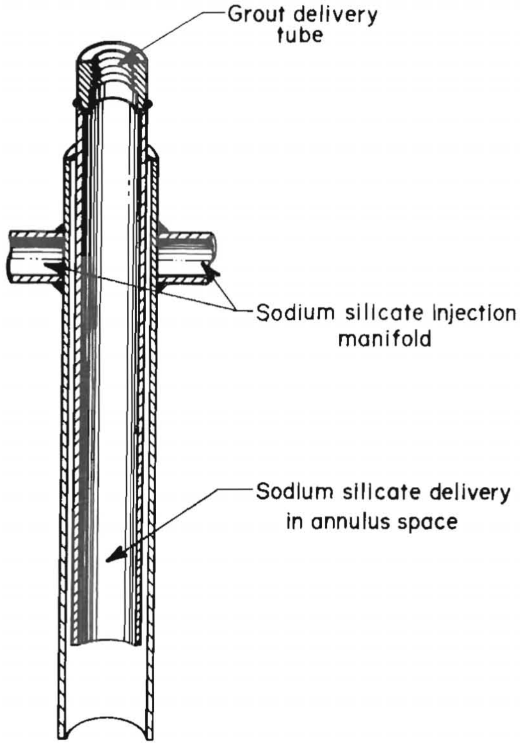

Figure 12.—Sodium silicate injector.

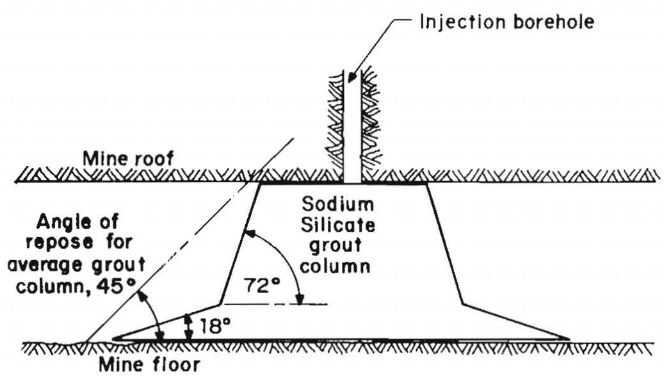

Figure 13.—Idealized grout column using sodium silicate technology.

artificial pillar of a specific size and shape can be remotely constructed. These bags have had limited application and have shown great promise (*20*). Further analysis and testing of the concept is required before the bags can be used on a regular basis.

PIERS

In-mine piers are simply in-mine supports to augment existing coal pillars. They are made from concrete, masonry, and in some cases timber cribbing. Although this method has been used for support in active mines throughout modern mining history, in-mine piers are rarely used in abandoned mines, since it is necessary to physically enter the mine void to install them. The primary application of support piers is to support a specific surface structure. Their use is generally limited to room-and-pillar mines, and the piers are usually constructed immediately after mining.

DEEP FOUNDATIONS

Deep foundations or pile foundations are limited to the protection of individual structures. These foundations are large-diameter drilled piers (usually 8- to 24-in diameter), which are drilled through the mine and bear on the firm strata beneath the mine. Such methods are used to found a structure on the competent rock beneath the mine opening (fig. 14). In general, deep foundations are not used at sites where the overburden thickness exceeds 100 ft. This method is limited almost exclusively to new construction.

The piers are steel casings driven from the surface into the rock beneath the mine and then filled with concrete. The surface structure is then constructed on beams supported by the piers.

BULKHEADS

Bulkheads are not a means of support, but they are used as barriers to control the flow of the backfill material in the mine. They are built between existing pillars so that the area to be backfilled is completely surrounded. In general, bulkheads are made from grout or gravel. They are constructed by drilling injection boreholes closely spaced across the known mine opening. The spacing of the boreholes must be close enough that the cone-shaped deposition spreads into a solid barrier rather than leaving void spaces near the mine roof.

Gravel bulkheads are preferred when used in conjunction with hydraulic flushing operations. The porous nature of the gravel allows the water in the slurry to pass but retains the solids (*21*).

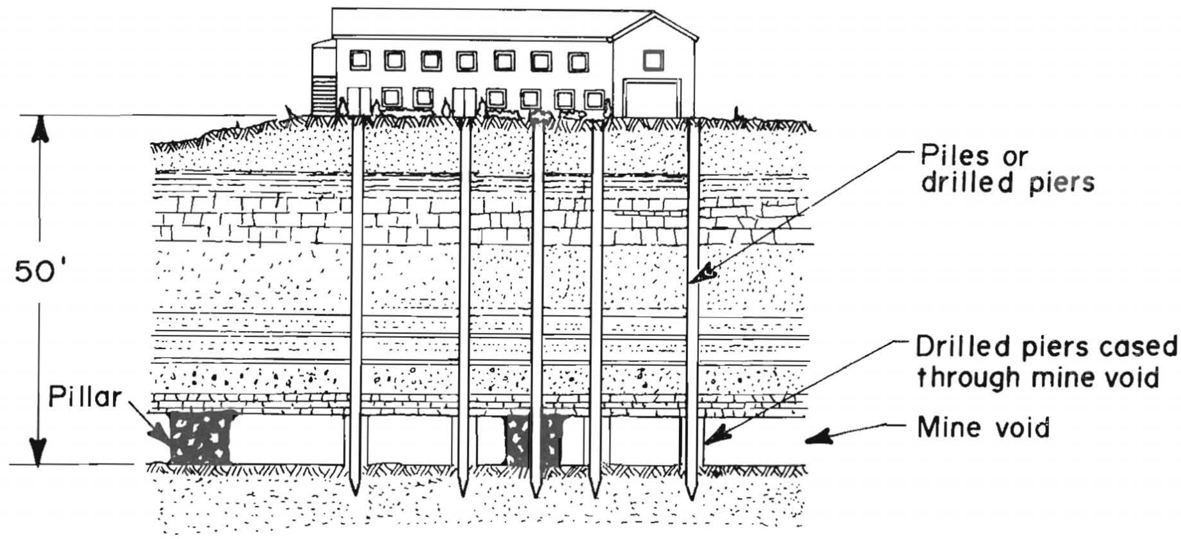

Figure 14.—Deep foundations for subsidence control.

TYPES OF BACKFILLING MATERIAL

The material used in a mine stabilization program is determined by local availability, transportation costs, and desired engineering properties. The most common materials used include waste rock; coal processing waste, fly ash; and prepared aggregates such as sand, gravel, or crushed rock. Almost any type of material can be used, as long as it is environmentally safe and its consistency is such that it can be easily injected into the mine and yet provide a positive support.

The engineering characteristics of a backfill material are very important to the long-term success of a subsidence abatement project. The material must be durable, in the sense that it will not degrade during handling or pumping; nonabrasive, because the wear on the system must be limited to the extent economically possible; strong, for long-term support; and stable, so that it will not become weak when exposed to the mine conditions. Many different types of material have been tried with varying success. However, the following remain the most common backfill materials.

PROCESSING WASTES

Processing wastes from local plants where rock is being crushed for aggregate can provide inexpensive fill material in the form of "off-spec" rock. This material often has sufficient engineering properties but does not meet highway sizing requirements. Limestone dust from such plants is a highly desired mine fill additive because of its pozzolanic (cementing) properties.

MINE REFUSE

Waste product created by coal preparation plants consists of coal, clay, shale, and other rock. The bearing strength is low if the waste product contains a high percentage of clay particles. Therefore, this material is mostly used for stabilization of dry, shallow mines where the weight of the overburden is relatively low and roof falls are the major failure mode. "Red dog" or burned mine refuse has a greater bearing capacity than unburned refuse and can also be used as backfill. However, red dog is very abrasive and does not move freely once it is released from the end of the injection pipe.

FLY ASH

Fly ash and bottom ash are noncombustible residues that result when coal is burned. Fly ash is a fine-grained, lightweight aggregate composed of small globules of glass-like material. Fly ash is easy to transport pneumatically and in slurry form can be transported long distances in mine voids. Though fly ash often has natural pozzolanic properties, it is necessary to mix it with cement to develop significant bearing capacity (*17*). Bottom ash is coarse and heavier than fly ash. Bottom ash is better than fly ash for mine stabilization but is normally more expensive.

PREPARED AGGREGATES

Prepared aggregates are the preferred types of backfill material to use in a flushing or pneumatic stowing operation because the engineering properties of sand, gravel, and crushed stone are well documented by the roadway and heavy construction industries. The drawback to these materials is their cost. Depending on the location, prepared aggregates can cost up to four times as much as other backfill materials.

The environmental impact of backfilling abandoned mines with waste materials is not well understood. The most obvious danger is the contamination of ground water supplies that come in contact with backfill material. For example, coal refuse and fly ash may contain high levels of contaminants such as aluminum, arsenic, boron, cadmium, chromium, copper, iron, lead, manganese, mercury, nickel, selenium, vanadium, and zinc. Research indicates that when this material comes in contact with water these elements go into solution, thus contaminating the water. There is the possibility that other backfill materials may also include one or more of these contaminants. The extent of the assimilation of these pollutants into the natural environment is a function of many variables that are unique to each site. To date, the documented research supports the belief that backfilling is not hazardous to the environment. However, the Environmental Protection Agency (EPA) regulates and classifies backfilling material using criteria stated in the Resource Recovery and Conservation Act and other health-related criteria. Research is currently under way at the EPA to identify the environmental impacts of backfilling.

EVALUATION OF PAST SUBSIDENCE ABATEMENT PROJECTS

Investigations have been performed to measure the results of a number of area-wide mine backfill projects. The majority of these projects, however, were conducted as a method to confirm the location of the fill material, and few or no tests were performed to evaluate the supporting capability of the backfill. One recent report by OSMRE does in fact evaluate the performance of four past subsidence abatement projects that used hydraulic backfilling in terms of lateral and vertical extent of the fill and the engineering parameters of the fill, i.e., grain size distribution, strength, and in situ bearing capacity (*12*). The sites evaluated were Green Ridge Demonstration Project, Scranton, PA; Farmington Subsidence Abatement Project, Farmington, WV; Watson Hill Subsidence Abatement Project, Fairmont, WV; and 70th Street Subsidence Project, Belleville, IL. All of the projects were demonstration projects sponsored by the Bureau in the 1970's. The methods and results of the first two were documented in Bureau reports.

The testing conducted at these sites was standard engineering practice for geotechnical investigations of subsidence-prone land, with the exception of the use of a specialized tool for the determination of the in situ bearing capacity of the backfilled material. Core samples were taken at various locations above the study sites, and split-spoon samples of the fill material were recovered. The general findings of the OSMRE report are as follows:

Backfilling of mine voids was very dependent on the configuration of the mine. Some portions of the mine void received very little fill material. This was presumed to be due to roof falls, stoppings, or other debris that may have blocked the flow path of the backfill material into the void.

In mines that were considered to be successfully backfilled because of the significant amounts of material that were deposited, a void space of more than 1.0 ft was left between the roof of the mine and the top of the fill.

The flow pattern was not the radial distribution that Whaite and Allen (*6*) and Carlson (*7-8*) had described in reports on modeling hydraulic backfilling. Sieve analysis and observations of the bedded character of the backfill indicated that the slurry flowing from the injection point settled in the same manner as sediment in stream channels, forming bars and graded deposits. The fine material in the slurry was segregated from the coarse material and was transported farther from the injection point.

Large concentrations of water-saturated fine material showed very low compressive strengths and the material was determined to be plastic enough to yield when subjected to lateral or horizontal pressures.

In mines where coarse coal refuse was used as the backfilling material, degradation of the clays and shales was apparent. This suggests that backfill that contains large portions of these materials will randomly break down into finer fragments in a saturated environment and will be insufficient to support the overburden.

These findings cannot be applied universally to all remote hydraulic backfilling projects, but the problems found at these sites are consistent with the concerns of those working in the field. Discussions with individuals who have drilled into backfilled underground mines for reasons other than evaluation of the fill indicate that voids and very soft fill material are not uncommon.

SUMMARY AND DISCUSSION

Backfilling of mine voids is the most common method of stabilization used to abate subsidence and protect surface structures. The state of the art of mine backfilling is much the same as it was 10 to 15 years ago. In-mine methods are the most well developed and are the most effective means of constructing support. However, in abandoned mine areas, access to the mine void is rarely possible. Therefore, it is necessary to use remote or blind backfilling methods.

The most common method for the placement of backfill material has historically been hydraulic flushing, either by in-mine or remote methods. These have been area-wide projects in which several acres or more have been stabilized. Pneumatic stowing appears to be increasing in popularity as the technology advances. Other subsidence abatement techniques are available and may be more appropriate under different conditions. These techniques include the use of point support methods, which do not completely fill the mine void, but offer protection to small surface areas and surface structures.

At any location where a potential subsidence problem may exist, the primary task is a thorough investigation to determine the nature and extent of the subsidence problem. It is necessary to gather information on the condition of the overburden, the mine roof, the mine floor, and the mine pillars. Knowledge of the size and condition of the mine void is necessary to effectively design remedial measures to protect the surface. Historically, large area-wide hydraulic flushing methods have been considered the only way to fill the mine void with material to arrest the collapse of the mine opening. However, the development of improved methods for site investigations, the improvement in pneumatic and grouting technologies, the increase in the use of point support methods in emergency subsidence actions, and the increase in the cost of subsidence abatement actions has forced the industry to consider new or improved support methods.

Hydraulic flushing remains the only cost-effective method for backfilling a large area of unstable underground mine void. The reasons for using the area-wide approach are related to the type of subsidence, the size of the area affected, and more importantly, the knowledge of the site conditions. In areas where much is known about the geology, the mine configuration, and the condition of the mine, a specific remedial design can be made. In areas where little is known, area-wide methods are the only choice.

In areas where the cause of the subsidence can be pinpointed, a point source method can be applied. In some cases, a number of point support structures may be necessary to strengthen a large area. Using point source methods is thought to be more cost effective than using area-wide methods if there is sufficient information to provide a sound remedial design.

The point source method most often used is the grouting technique. The grout mix used varies depending on the intent of the project. Where very good subsurface information is available, pinpoint drilling can be performed and low- or no-slump grout can be pumped into the void to provide the support needed in a specific area. As the uncertainty of the subsurface information increases, the size of the area into which the grout is pumped is increased. Grout additives such as foaming agents, plasticizers, or surfactants are used to meet a specific need, such as a slow setup time or increased strength. Using these additives significantly increases the cost of the project.

Pneumatic stowing may be an inexpensive alternative to grout in a dry mine. Pinpoint drilling as used for grouting could be utilized to locate the injection pipe near the collapsing section of the void. Ground support could then be built by directing the trajectory of the fill material.

Other methods are inappropriate for the majority of AML situations. Methods such as deep foundations or drilled piers are more suited to new construction than the mitigation of damage in a subsidence-prone area. Similarly, the blasting method may not be acceptable in urban areas. However, explosives are used regularly in urban areas for other purposes, and if the blasting method is proven to be a viable method for the abatement of subsidence, blasting could possibly be used in urban areas with proper care.

The development of methods for the abatement of AML subsidence has progressed only as fast as the need for new methods. Until recently the need was not a priority of many city planners. As the frequency of subsidence events has increased, so has the awareness of the lack of effective methods to abate subsidence. Parallel to this has been the development of techniques to evaluate the subsurface condition at an AML site. As the capabilities to determine the location and the condition of the mine site become more precise, remedial designs will require a variety of backfilling methods. It is clearly demonstrated from past work that it is possible to backfill a mine to an extent that will lessen or prevent subsidence-related damages. Much more research is needed, however, to provide simple, cost-effective solutions.

REFERENCES

1. Ashmead, D. C. Mining of Thin Coal Beds in the Anthracite Region of Pennsylvania. BuMines B 245, 1927, 133 pp.

2. Gray, R. E., J. C. Gamble, R. J. McLaren, and D. J. Rogers. State of the Art of Subsidence Control. Appalachian Reg. Comm., ARC-73-111-2550, Dec. 1974, p. 264.

3. Thrill, R. E., P. J. Huck, and B. G. Stegman. Monitoring Blind Backfilling in Abandoned Mines. Min. Eng., v. 35, No. 12, Dec. 1983, pp. 1625-1630.

4. Roberts, J. M., F. W. Tobias, A. L. Masullo, and J. E. Powell. Report on the State-of-the-Art of Pneumatic Stowing Techniques. Off. Surf. Min. Reclam. and Enforcement unpublished report, 105 pp.; available upon request from J. L. Craft, Off. Surf. Min. Reclam. and Enforcement, Greentree, PA.

5. Staff, Office of Surface Mining Reclamation and Enforcement. Abandoned Mined Lands Reclamation Control Technology Handbook. 1981, 270 pp.

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

7. Carlson, E. J. Hydraulic Model Studies for Backfilling Mine Cavities (First in a Series of Tests). Bur. Reclam., ERC-73-19, 1973, 32 pp.

8. ____. Hydraulic Model Studies for Backfilling Mine Cavities (Second in a Series of Tests). Bur. Reclam., ERC-75-3, 1975, 38 pp.

9. Faddick, R. R., and W. R. Eby. Slurry Backfilling of Mine Voids in Hanna, Wyoming. Proceedings of the 1986 Symposium on Surface Mining, Hydrology, Sedimentology, and Reclamation (Lexington, KY, Dec. 8-11, 1986). Univ. KY, Lexington, KY, 1986, pp. 209-214.

10. Craft, J. L. Office of Surface Mining Reclamation and Enforcement. Private communication, 1991; available upon request from R. C. Dyni, BuMines, Pittsburgh, PA.

11. Candeub, Flesissing, and Assoc. (Newark, NJ). Report on Demonstration of a Technique for Limiting Subsidence of Land Over Abandoned Mines. June 1971, 65 pp.

12. Elder, C. H. Evaluation of Procedures Used in Four Mine Subsidence Control Projects. Off. Surf. Min. Reclam. and Enforcement unpublished report, 59 pp.; available upon request from J. L. Craft, Off. Surf. Min. Reclam. and Enforcement, Greentree, PA.

13. Juntunen, R., M. Heil, and B. Mundie. Reclaiming Orphaned Lands Using a Pneumatic Backfill Process. Am. Soc. Agric. Eng., Pap. ASAE 83-2035, 1983, 11 pp.

14. Burnett, M., and J. L. Craft. A Pneumatic Ejector for Backfilling Underground Mines Through Boreholes. Proceedings of the 10th Annual AML Conference, Association of Abandoned Mine Land Programs (Wilkes-Barre, PA, Oct. 10-14, 1988). PA Dep. Environ. Resour., 1988, 153 pp.

15. Burnett, M. Final Contract Report, Development and Testing of a Dry Feed Ejector for Backfilling Mine Voids (contract HQ51-GR87-10008, Off. Surf. Min. Reclam. and Enforcement); for inf., contact J. L. Craft, Off. Surf. Min., Reclam. and Enforcement, Greentree, PA.

16. Soderberg, R. L., and D. R. Corson. Support Capabilities of Pneumatically Stowed Materials. BuMines RI 8202, 1976, 48 pp.

17. Head, W. J., and E. E. Saadeh. Assessments of Fly-Ash Cement Grouts for Use in Abandoned Mines. Proceedings of the International Symposium on Environmental Geotechnology, v. 1 (Allentown, PA, Apr. 21-23, 1986). ENVO Publ. Co., 1986, pp. 576-585.

18. Royse, K. W. Underground Mine Reclamation Via Blasting. Proceedings of the 1983 Symposium on Mining, Hydrology, Sedimentology and Reclamation (Lexington, KY, Nov. 27-Dec. 2, 1983). Univ. KY, Lexington, KY, 1983, pp. 247-254.

19. Reifsnyder, R. H., R. A. Brennan, and J. F. Peters. Sodium Silicate Grout Technology for Effective Stabilization of Abandoned Flooded Mines. Paper in Mine Drainage and Surface Mine Reclamation. BuMines IC 9184, v. II, 1988, pp. 390-398.

20. Michael, P. R., A. S. Lees, T. M. Crandall, and J. L. Craft. A Point-Support Technique for Mine Subsidence Abatement. Paper in Mine Drainage and Surface Mine Reclamation. BuMines IC 9184, v. II, 1988, 401 pp.

21. Vandyke, M. W. Gravel Bulkheads for Confining Hydraulic Backfilling of Abandoned Underground Coal Mines. Soc. Min. Eng. AIME preprint 85-363, 1985, 2 pp.

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Practical Design Considerations when Planning a Backfill Operation

A.J.S. (Sam) Spearing and D.G. Millette

ABSTRACT

The design of a cost effective backfill system involves many aspects and various inputs from different engineering disciplines. Backfilling is a very significant mining cost and efforts need to be made in the design stage to maximise efficiency and flexibility whilst minimising costs. The aspects that need to be considered include the following:

- Required fill performance underground
- Peak and average fill tonnage requirements
- Backfill scheduling in the mining cycle
- Material selection, testing and preliminary costing
- Plant design and storage needs
- Infrastructure requirements
- Implications on surface waste/tailings disposal and costs
- Total backfill costs

In order to produce a cost effective and efficient backfill operation, a multi-disciplinary project team is required that should include the following skills and functions:

- Mine Planning
- Rock Engineering
- Metallurgy
- Mechanical Engineering
- Environmental Health and Safety
- Tailings Disposal/Waste Management

This paper discusses the above and indicates important aspects that should be considered.

INTRODUCTION

Backfilling is the process whereby waste material (generally metallurgical tailings) is placed back underground. This has many potential advantages for typical underground mining applications, such as:

- Reduced tailings on surface
- Reduced surface subsidence
- Increased underground extraction
- Reduced rockburst (seismic) damage
- Mine excavation support and/or working surface

In addition, backfilling is an integral part of many mining methods, such as:

- Cut and fill
- Vertical crater retreat
- Drift and fill
- Room and pillar, with pillar recovery
- Long hole sloping with fill

There are three main transportation methods used to move the fill from surface to the underground workings:

- Hydraulic via pipelines, with the fill as a slurry or paste
- Mechanical via conveyor belts or trucks
- Pneumatic via pipelines as a dry (or damped) material

The current most common transportation method by far, is hydraulic, because it is energy efficient, cost effective and able to transport large volumes. Within hydraulic filling, paste filling is gaining popularity, mainly because of the reduced water handling facilities needed underground, and surface disposal considerations for tailings.

The application of backfilling internationally is growing significantly, mainly due to:

- An increased environmental lobby
- Several high profile and damage causing surface tailings dam failures
- The cost of final surface tailings dam rehabilitation
- The financial benefits derived from increased orebody extraction
- The rehabilitation cost of land damaged by mining induced surface subsidence
- More reliable and cost effective methods for placing fill underground

BACKFILL MATERIAL SELECTION

The choice of the correct and stable backfill material for mining is critical to control mining costs in the longer term. Backfill contributes a major cost element in mining operations and must be therefore cost effectively selected at the initial planning stage. It is essential that a correctly designed plant infrastructure be made to ensure the reliable and consistent supply of fill at all times.

The criteria and requirements for backfills vary, depending on the site-specific requirements. The following are however some of the more common and important criteria and requirements:

- The fill should be placed at the lowest possible cost.
- The risk of fill failure (e.g., liquefaction) must be minimised.
- Early strength development needs to be adequate.
- Long term strength should be sustainable.
- Delivery volumes must be adequate.
- Reliable delivery must be achieved.
- After placement, dimensional stability must be achieved.

Figure 1 is an example of a design chart that can be used where the backfill is needed for maximising extraction and reducing the rockburst potential, in a deep narrow tabular hard rock mine. Such a chart can be modified for other applications and is useful in optimising a conceptual backfill design.

The criteria and requirements for the cementitious slurry component of cemented rock fill are also similar to those for hydraulic fill.

The range of performance (stress versus strain) that can be achieved using different backfill types is extensive Figure 2 gives the in situ performance for three fill types under large strain conditions.

THE POTENTIAL FOR ADMIXTURES

Cement is basically an adhesive substance that is capable of uniting fragments or masses of solid material into a compact whole.

Concrete is a composite material that consists mainly of a binding medium (adhesive usually cement), within which are embedded particles or fragments (aggregates). Admixtures are now frequently added, to enhance the performance of the concrete, mortar and grouts before or after hydration of the mix.

An admixture is defined in ASTM C125 (2) as: "A material other than water, aggregates, hydraulic cement and fibre reinforcement, used as an ingredient of concrete or mortar, and added to the material immediately before or during its mixing."

There are several types of admixtures, as defined by ASTM C494 (2):

- Water reducers (type A)
- Retarders (type B)
- Accelerators (type C)
- Various combinations (e.g., water reducers and accelerators, high range water reducers)

In addition, hydration control admixtures and internal curing agents are also frequently added to concrete, as well as foaming agents

Traditional concrete admixtures therefore affect the (physical, chemical and mechanical) properties of concrete. However, admixtures should never be used, without first testing and establishing the benefits in terms of cost and performance. Admixtures do however offer very real benefits, but must be dosed correctly.

BACKFILL MATERIAL SELECTION DESIGN FLOWCHART

FIGURE 1

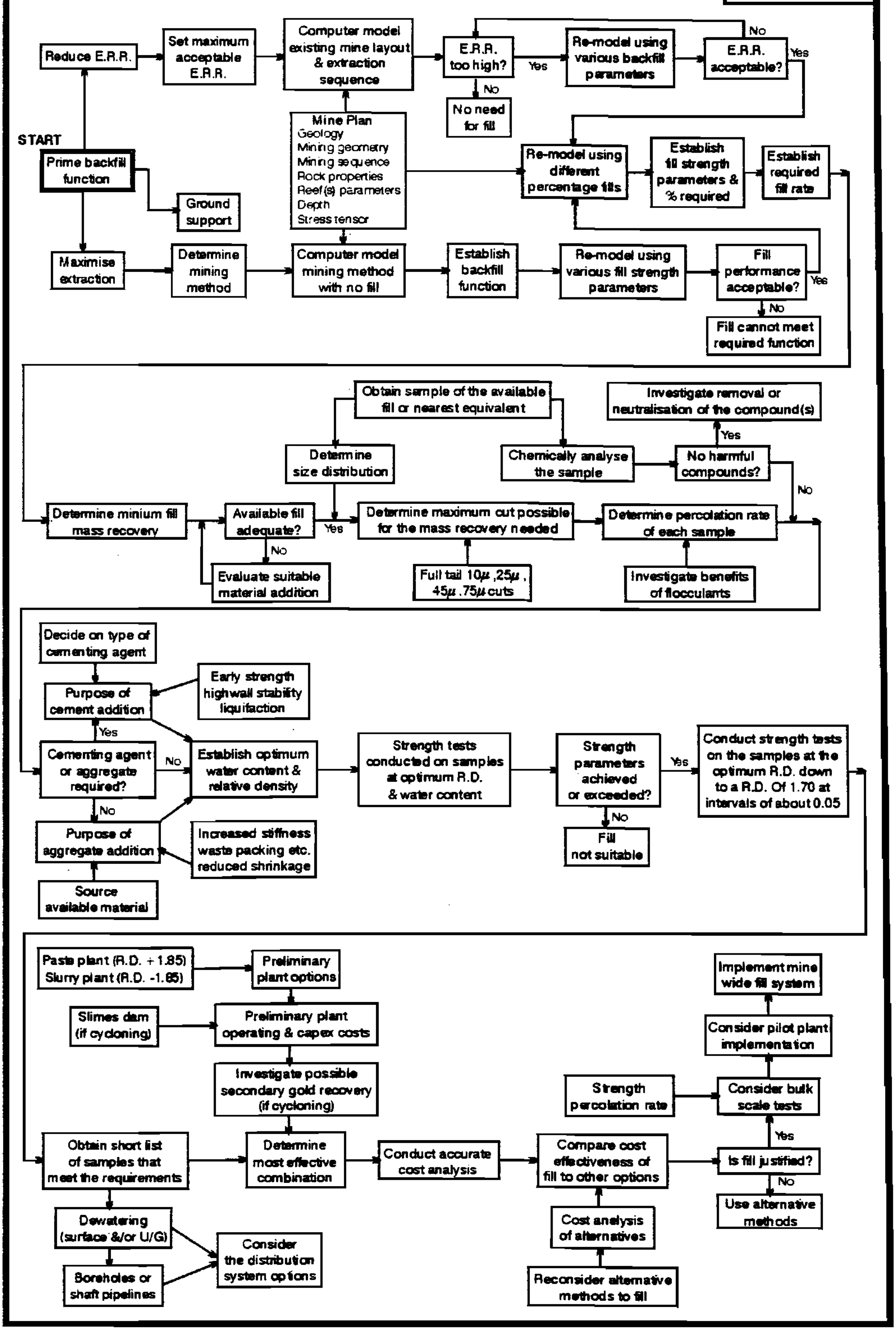
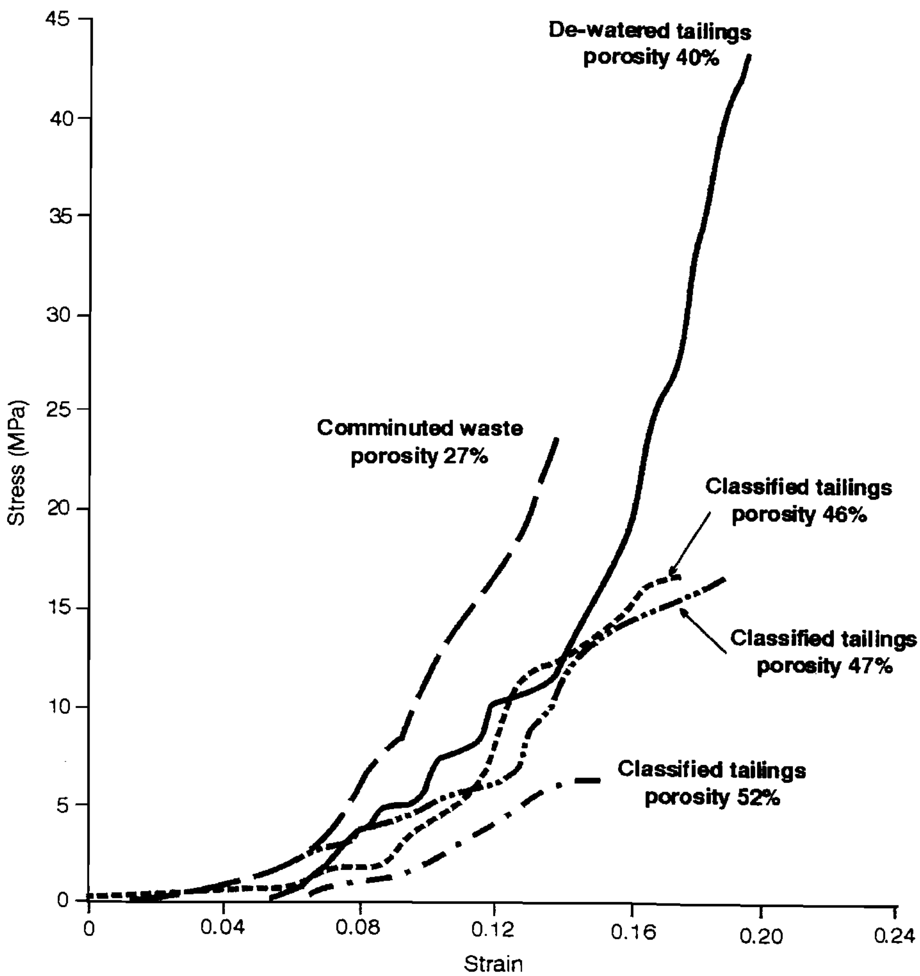


FIGURE 2

THE IN-SITU PERFORMANCE OF VARIOUS FILL TYPES AT LARGE STRAIN



(After Gurtunca)

The admixture product groups found most effective in certain backfill operations can be divided into three distinct categories:

- Rheology modifiers

  Rheology modifiers (admixtures) basically alter the (slump) flowability of a paste (high density) fill at a constant solids density. Rheology modifiers consist of water reducers and superplasticisers, which are disperse fine particles by altering the surface charge of particles or internal lubricants, where the long polymer chains reduce turbulence This can result in:

  - — Reduced pressure losses (longer transportation distance and/or improved flow).
  - — Reduced pipeline wear rate.
  - — Improved fill placement (reduced beach angle).
  - — Reduced binder usage for any given strength requirement
  - — Improved the consistency of the fill mix (reduced segregation).

- Hydration modifiers

  The use of selected hydration modifiers (admixtures) in cemented hydraulic fills can:

  - — Accelerate or retard the strength gain in the placed fill. This can be an important issue depending on the mining method and the cycle.
  - — Stop cement hydration prior to fill placement for up to 72 hours. This can be very useful, from an operational point of view, because wastage is reduced, and the occurrence of costly pipeline blockages significantly reduced.

  By retarding the hydration of cement in backfill mixes, higher compressive strengths are realised at later curing periods. Some set stabilisers also offer water- reducing properties.

- Durability enhancers

  Improving the durability of fill is very site specific and not frequently an issue, but where it is needed, it becomes vital. An application is when backfill needs to be placed in live (running) water, and an anti-washout additive is beneficial.

  As an example where admixtures can be used, paste backfill mixtures exhibit higher water demands due mainly to the high surface area as a result of using total tailings. Reducing the water content is desirable due to a more favourable water to cement ratio, but it results in a loss of flow unless a plasticizing admixture is used to compensate In tests (Gay and Constantiner, 1997), the backfill binder was composed of 90% slag and 10% Portland cement. The results showed that specific admixtures could reduce the water requirements (by between 2% and 28%) without sacrificing the original flow properties Reducing the water content in the paste can significantly increase its compressive strength (5% to as much as 100%), as long as the specific admixture and dosage does not cause excessive retardation.

BACKFILL PLANT DESIGN AND INFRASTRUCTURE

The design of a backfill plant and infrastructure must be carefully considered and implemented to ensure cost effective filling. This includes the delivery of sufficient quality controlled fill on demand, so that mining can progress uninterrupted.

Backfill Storage Design

Backfill storage is often needed, mainly on surface, because:

- The metallurgical plant operates for hours a day, whilst backfill is generally placed for 8 to 1 hours per day, and at an hourly rate of more than the fill plant process rate.
- It provides the pressure head to maintain an adequate feed to the pipelines (usually vertical initially, down boreholes or a shaft). Pumps can also be used to achieve this.

Backfill storage facilities are relatively costly to build, and hence must be sized correctly, and include some overcapacity.

Storage tanks can have a circular or rectangular cross section, but generally the circular section is preferred because it is easier to agitate, and keep the fill homogeneous.

Agitation is required in the storage tank, particularly if slurries are stored. Agitation is achieved mechanically or pneumatically, and it is not uncommon to use mechanical agitation with a pneumatic back-up (in case of mechanical failure). Using both systems simultaneously is counter-productive because mechanical agitation mixes by pushing the material downwards, and pneumatic agitation works by lifting the solids (as in an airlift pump system).

The design of the storage vessel and agitation should be professionally undertaken, but the following guidelines can be considered:

- Wall baffles are generally needed to avoid the fill from vortexing, and improve mixing.
- The main mechanical impeller is generally placed about a third of the slurry level in the tank, above the base of the tank.
- The rotational speed is usually determined by testing.

Distribution Design Steps

Where piping is used, the correct diameter and pressure rating are key to the site specific conditions. The following aspects must be considered when designing a backfill distribution system:

- The transport velocity must be above the depositional velocity, to stop the solids from settling out The depositional velocity is mainly dependant on the pipe diameter, the particle size distribution and the concentration of the solids.
- The transportation velocity must however be minimised to reduce the wear since:

  $$\text{Wear} \propto \text{Velocity}^k$$

  The exponent k lies typically between 1.5 and 30, about 1.8 typically (Steward and Spearing).

- Standard pipe lengths and diameters should be used to reduce the pipe and fitting costs.
- Provision must be made to flush the pipes when necessary, with water at an adequate pressure and flow rate.
- Rupture discs should be considered at locations of peak pressure, to safeguard the pipelines in the case of blockages.

The following are the design steps that should be followed:

- Calculate the monthly volume of fill required, at the planned mining rate.
- Increase this calculated amount to allow for additional mining production (say 10%) and backfill losses due to wastage and drainage (this depends on the fill system to be used).
- Calculate the flow rate required, taking account of the actual filling time per day (typically only hours per day if filling on a single shift).
- Calculate the static pressure head.

  $$\text{Static head (kPa)} = \text{Vertical height (m)} \times \text{gravity } (9.81\,\text{m/s}^2) \times \text{slurry relative density}$$

- Determine the total pipeline length (from surface to the slope for a dedicated pipeline).
- The piping selection is based mainly on the pressure and the velocity needed to achieve the flow rate.
- The frictional losses must be determined (from lab testing or estimated from previous similar experience). For a system to be balanced, the pipeline frictional losses must equal the static pressure head. If this is not the case, the following can be considered:
  - Decrease the pipe diameter selected.
  - Increase the slurry relative density.
  - Choke the system.

Backfill Placement

The placed backfill strength is dependent on the following:

- The delivered density
- The final density (after any water and solids loss and gravity settlement)
- The particle size distribution
- The binder type and content (if any), and the water content (as above)

Backfill is required in situ mainly under two basic placement requirements:

- Bulk placement where the fill is a working floor (platform) or roof. This placement is relatively easy and the only major potential problems are segregation of the fill and binder loss in the drainage water.
- Tight placement where the fill is typically used as an artificial sidewall and needs to have reliable contact with the roof of the slope. The potential problems are to control/eliminate post filling shrinkage and avoid costly containment structures and/or mining at unfavourable gradients from a production point of view.

Quality Management

The need for effective backfill quality management is frequently overlooked, often resulting in a waste of money or even a potentially serious safety risk. Not only should a workable quality system be in place, but actions must be taken should non-compliance to pre-specified criteria be found.

As a minimum, the following should be regularly and routinely monitored:

- The particle size distribution particularly if classified tailings are used (the –10μm size fraction is a useful indication of the percolation rate, and can be quickly obtained if a laser diffraction instrument is available). This should ideally be measured in the surface storage, and material discarded if it fails the criterion.
- The relative density prior to underground discharge (i.e., in storage) is very important. Care must be exercised that any recirculation system does not significantly dilute the backfill with time, due to the introduction of gland service water for example.
- The water content after storage which can frequently be measured using a (slump) density cup measurement.
- Where a cementitious binder is used, care must be taken to ensure that an adequate quantity is actually added. Depending on the plant design, this can be routinely checked, especially if the fill mix is batched. An additional check can be readily and quickly undertaken by measuring the pH of the mix. The cementitious addition should make the final product alkali (Universal Indicator Paper can even be used). Adding too much can also cause problems of wastage (cost) and even pipeline blockages.
- The level of any contaminants (cyanide for example) should also be monitored in the fill prior to placement and from any underground run-off.

IMPLICATIONS OF UNDERCROUND BACKFILLING ON SURFACE TAILINGS DISPOSAL

The metallurgical waste materials can be disposed of in the following ways:

- Into the sea or a lake (this method is not now generally favoured due to environmental pressures)
- Into a river behind a suitable dam wall (this can be a costly method and also has adverse environmental considerations)
- Onto a specifically constructed tailings dam on surface.
- Placing the waste back underground as a backfill material (as mentioned this method cannot be used to remove all the tailings produced, and some surface disposal is still required).

The problems with conventional tailings dams include the following:

- The inherent safety risk of liquefaction such as recently occurred in Spain and South Africa, causing severe property damage (and loss of life in the South African accident at Merriespruit Mine).
- Serious water losses on the dams in regions where water is a scarce commodity (such as in Australia and South Africa).
- The large area needed for the dam and the cost of eventual reclamation.

The above are significantly influenced by the tailings particle size distribution and mineralogy. Underground backfilling can adversely affect this if some form of classification process is used to produce the backfill. An assessment of the impact of the selected backfill type on the surface tailings disposal needs to be undertaken and considered before making a final decision.

TOTAL BACKFILL COSTS

Any backfill system selected must be as cost effective as possible. In most mines utilising backfill, the fill cost is one of the highest and efforts must be made to minimise it (safely). The total (placed) backfill cost is the important cost and often increasing the fill preparation and/or material cost can actually reduce the placed cost.

Backfill operating costs typically reported by mines vary from $2.00/t placed to $6.00/t placed depending on the fill type used and the cost elements actually considered.

Elements that need to be included in any backfill costing exercise include:

- Plant amortisation
- Plant operating cost (including labor, power, consumables, etc.)
- Raw material costs (e.g., cement, sand, admixtures)
- Maintenance (surface and underground)
- Fill placement preparation (e.g., bulkheads, new piping, etc.)
- The cost and frequency of breakdowns (e.g., pipe blockages)

- The influence of the fill on the mining cycle (e.g., would a faster setting fill material improve overall mining efficiencies by permitting secondary extraction sooner)
- The additional cost of handling any spillage and any water and solids run-off after placement

Pre-Design Testing

Much of the performance and cost issues of backfill mix design can be accurately estimated through laboratory testing. This testing can be divided into three stages:

- Physical and chemical properties of the aggregate(s)

  Basic physical testing of the aggregates should include determinations of particle size analyses, specific gravity, bulk density and permeability. Chemical properties and mineral composition can be determined by some combination of the following: pH, zeta-potential measurement, optical microscopy, XRD (x-ray diffraction) which shows the crystalline structure, XRF (x-ray florescence) for specific mineral composition and/or ICP (inductively coupled plasma) for determination of all metals simultaneously.

- Mechanical properties of the backfill mix

  Compressive strength testing is necessary to determine binder contents and blend ratios of binders in a mix design. Two parameters that are very important in mixing compressive strength samples are to use of the same water that is intended for use in actual production, and to size the samples appropriately (if using minus 12"/300mm rockfill aggregate, cast a minimum sample diameter of 18"/450mm).

  Monitoring of set times of the mixes using thermocouples is very valuable. It is important to cure the samples in a similar environment to the slopes where the material is to be poured. If the slope environment is 30°C, then curing test samples at 20°C will not yield results that can be applied.

- Rheological properties of the backfill mix

  For binder slurries used in rockfill applications, flow cone measurements and set time determination are normally adequate.

  There are many different methods of measuring rheology in paste fill applications, but the accuracy of any of them is often questioned. Viscometers and rheometers are commonly used, as are pipe loop testing and slump cone. With standard viscometers and rheometers, the paste plug will often rotate within the container, giving false readings. Pipe loops are valuable if properly set up, but they should be adequately instrumented, be of the same diameter throughout and not a closed loop. Slump testing, although often used is highly inaccurate when a flocculent is present in the tailings or when the slump is greater than 230mm (9"). Slump also varies considerably with temperature changes.

CONCLUSIONS

Before finalising a backfill plant design, various considerations should be made to ensure that the overall cost is minimised and that backfilling does not have an adverse influence on the mining cycle.

The design can be optimised if various key elements are examined during the design phase. These include:

- Considering the overall placed cost.
- Selecting the correct diameter and pressure rating if pipe ranges are be used.
- Consider using admixtures after appropriate testing and costing.
- Ensure that adequate storage is available (if needed) and that the design maintains the correct fill properties during storage.

REFERENCES

Gay, F., Constantiner, D. Additives for improving paste backfill mixes. Tailings and Mine Waste '98. Fort Collins, Colorado, USA. January 1998.

Steward, N.R., Spearing, A.J.S. Combating pipeline wear—An advancing technology. J.S.Afr.Inst.Min.Metal. Vol. 92, No. 6. June 1992.

APPENDIX E

Effectiveness of Mine Filling on Subsidence Control

CONTROLLING SUBSIDENCE EFFECTS USING PARTIAL BACKFILLING

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ABSTRACT: Partial backfilling can be used to reduce and control surface subsidence damage caused by longwall mining or pillaring operations in room-and-pillar mines. The use of partial backfill as opposed to total backfill minimizes the cost associated with such efforts. By comparing surface subsidence, strains, and slopes, the effectiveness of partial backfilling including the amount, geometry, and location of backfill are demonstrated. Results show that it is possible to arrange a given volume of backfill in its geometry and location so that optimum effects are achieved in controlling vertical movement, surface slope, and curvature.

INTRODUCTION

Longwall mining continues to gain popularity for use in U.S. coal mining due to its high productivity, efficiency, and recovery. Unfortunately, both longwall mining and pillaring operations in room-and-pillar mines almost invariably create surface subsidence. Effects from this subsidence frequently include damage to buildings, roads and other structures, and serious disruption of the ground water regime. This damage is so severe on occasions that has resulted in cessation of the mining activity. If total extraction systems with their inherent benefits are to operate successfully, a solution must be found to the problem of surface damage.

Backfilling of underground openings has been used, predominantly in Europe, to control surface disturbances. Such techniques are often expensive, however, because they usually require an extensive backfilling system and a large amount of backfill material.

Field studies show that the most detrimental effects of subsidence on surface structures occur in isolated areas such as those over rib lines. The negative abrupt effect of the rib edge could be reduced by selective backfilling to diminish or control the various elements of surface movement. In order to maximize surface control of subsidence while minimizing associated costs, it is necessary to select an optimum system of partial backfilling for each mining and geologic environment.

SUBSIDENCE AND BACKFILL

The idea of backfill as a means to control surface subsidence is not new. It was used in the early 1930's to enable extraction of coal from beneath European cities[1]. Later studies on this subject in the United States were primarily concerned with abandoned room-and-pillar mines[2-4]. Depending on the scale of the problems, subsidence control techniques generally fall into two categories: point support methods and area backfilling[5]. Point support methods are used to protect individual structures, or even individual foundation elements of structures; whereas area backfilling techniques may be applied to protect much larger surface areas such as entire neighborhoods or urban districts hundreds of acres in extent (Figure 1).

Transportation of backfill material can be done by gravity, mechanical transport , pneumatic or hydraulic stowing[6]. Mechanical stowing transports fill using a combination of conveyors and gravity systems (vertical or inclined chutes, boreholes, pipes). This type of backfilling is rarely used for underground coal mines. Pneumatic stowing utilizes compressed air as the transporting medium and is used almost exclusively with longwall mining, though it may be adapted to any type of mine. Hydraulic stowing, more frequently called hydraulic injection, was first used in underground waste disposal in active mining operations[2]. This method is generally more easily adapted to room-and-pillar mining than to longwall mining, and has been used almost exclusively in the U. S. to backfill abandoned coal mines using either pumped slurry injection or flyash slurry injection.[3,5]

The use of backfilling to control subsidence is based on three basic mechanisms. First, backfilling an opening reduces the volume of void, thus reducing the maximum amount of subsidence. Second, compacted backfill

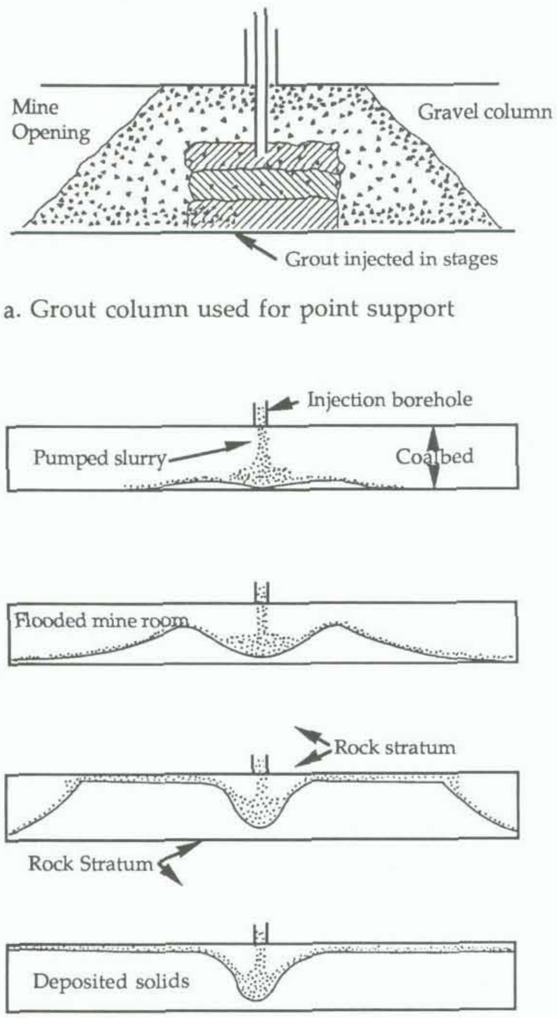


Figure 1 Use of backfill for ground support and subsidence control (after Huck et al. 1982) [5]

material may provide lateral support to the mine pillars and protection against pillar deterioration, which is often the cause of sinkhole subsidence. Finally, some direct support is provided to the roof strata, thus limiting roof collapse and enhancing overall stability. The objective, therefore, is to place as much backfill material as possible in the designated area and to achieve firm contact with the mine roof. Although efforts have been made to monitor the effectiveness of backfilling, most studies have concentrated on preparation and transportation of backfill material rather than on achievement of best results with minimum cost. Certainly where backfilling is done in conjunction with waste disposal, there is incentive to fill up the underground void as much as possible. However, the benefits of increased fill volume must be weighed against increased such costs, as even with complete filling some surface subsidence will normally occur. If acceptable surface subsidence effects can be determined, selective partial backfilling of underground openings can be more cost-effective than total backfilling.

A MATHEMATICAL ANALYSIS

Subsidence in coal mining has been studied extensively and various theories are available for calculating subsidence movements. Figure 2 shows the various elements of ground movements associated with subsidence and their distribution. The following influence function method has been chosen for this analysis because it has been successfully used for surface prediction in the Appalachian coalfields and has the flexibility necessary to allow for irregular geometries and superposition[7].

Using theory based on Knothe's influence function, the subsidence at point x is given by the following equation:

$$s(x) = \frac{S_{max}}{r}\int_{x_1}^{\infty} e^{-\pi\left(\frac{x}{r}\right)^2}\,dx \tag{1}$$

where S_{max} is the maximum subsidence; $r = H/\tan\beta$ is the radius of influence; H is the depth of mining; and β is the angle of influence.

In previous subsidence analysis, the maximum subsidence S_{max} is considered to be constant since seam height is generally considered constant. With the introduction of partial backfilling, S_{max} will be a function of x. For purposes of simplicity, the new subsidence and maximum subsidence are redefined as s(x) and S(x), respectively.

Taking the first derivative of s(x), we have the surface tilt or slope, T(x):

$$\begin{aligned} T(x) &= \frac{\partial s(x)}{\partial x} \\ &= \frac{S(x)}{r} e^{-\pi\left(\frac{x}{r}\right)^2} + \frac{\partial S(x)}{\partial x}\frac{1}{r}\int_{x_1}^{\infty} e^{-\pi\left(\frac{x}{r}\right)^2}\,dx \end{aligned} \tag{2}$$

The second derivative of s(x) gives the curvature:

$$\begin{aligned} K(x) &= \frac{\partial^2 s(x)}{\partial x^2} \\ &= \frac{\partial^2 S(x)}{\partial x^2}\frac{1}{r}\int_{x_1}^{\infty} e^{-\pi\left(\frac{x}{r}\right)^2}\,dx \\ &\quad + \frac{2}{r}\frac{\partial S(x)}{\partial x} e^{-\pi\left(\frac{x}{r}\right)^2} + 2\pi\frac{S(x)}{r}\left(-\frac{x}{r}\right) e^{-\pi\left(\frac{x}{r}\right)^2} \end{aligned} \tag{3}$$

From subsidence studies, surface strain is equal to the curvature multiplied by a constant called the strain factor.

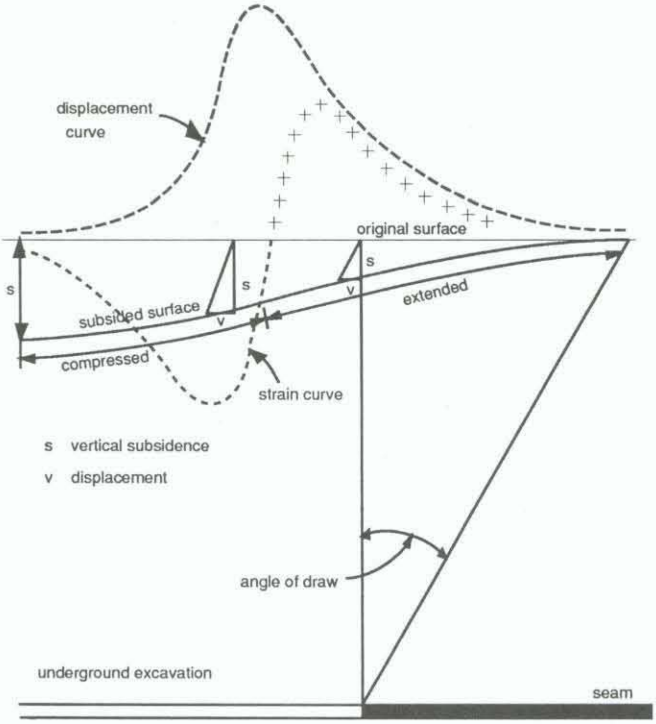


Figure 2 Elements of Ground Movements and Their Distribution [7]

For Appalachian coalfields, the strain factor has been determined to be 0.35 (Karmis at el.[7]). Since the maximum subsidence S(x) is a function of x, the second term on the right-hand side of Equation (3) is always present. However, for most practical purposes, the second derivative of S(x) can be considered to be zero since S(x) at most will be a linear function of x. Therefore, Equation (3) can be simplified into the following:

$$K(x) = 2\pi \frac{S(x)}{r} \left(-\frac{x}{r}\right) e^{-\pi\left(\frac{x}{r}\right)^2} + \frac{2}{r} \frac{\partial S(x)}{\partial x} e^{-\pi\left(\frac{x}{r}\right)^2} \qquad (4)$$

Combining Equations (2) through (4), the following conclusions can be drawn concerning partial backfill in underground coal mines:

1) All subsidence parameters, i.e. vertical movement, surface slope, curvature and horizontal strain, are affected by the placement of partial backfill;

2) Since S(x) is a function of x, and is therefore a function of backfill geometry and location, it should be possible to achieve optimum effects of reducing surface subsidence through proper combination of fill volume, geometry, and location.

These conclusions are important because by varying the location and geometry of backfill, optimum effect can be achieved towards reducing vertical movement, curvature, or surface strain, depending on the engineering requirements.

SIMULATION ANALYSIS OF FILL PATTERNS

Design of Fill Patterns

In the following analyses, it is assumed that the final shape of the backfill pile can be predicted and that the backfill material has the same stiffness as the surrounding collapsed rock material. Based on the preceding analyses and assumptions described in the Introduction, nine basic patterns of partial backfilling have been identified, as shown in Figure 3. These shapes can be formed during backfilling using current geotextile techniques, or they may form as a result of material flowing due to high ground pressure and lack of lateral confinement.

Pattern (a) has the effect of reducing mining height, thus reducing maximum subsidence and maximum tensile strain. The subsidence factor and edge effect are not affected by this pattern. Pattern (b) serves to reduce the panel width. Depending on panel geometry, this can result in changing the subsidence profile from super-critical to sub-critical, reducing the subsidence factor, and changing the edge effect. The pattern shown in (c) yields two separate panels of much smaller width. However, the superposition of tensile strain over the backfill may result in unwanted effects in areas over the backfill. This makes the pattern in (d) more desirable as it has the effect of a yield pillar, which has been proven to mitigate surface tensile strain[7]. This pattern also is a more realistic representation of deformed backfill material which has relatively lower stiffness than coal. Other patterns of non-rectangular shape are possible if the backfill material has low stiffness and is allowed to deform relatively freely under pressure without flowing, as shown by patterns (f) through (i).

Unknown Fill Geometry

With high ground pressure and lack of lateral confinement of the backfill, the material may be compressed so that the final geometry becomes unknown. When this happens, the effect of partial backfilling can be incorporated by adjusting the width-to-depth ratio:

$$\left(\frac{W}{H}\right)' = \frac{TW - V_f}{TW} \left(\frac{W}{H}\right) = \left(1 - \frac{V_f}{TW}\right) \left(\frac{W}{H}\right) \qquad (5)$$

where W is the panel width; V_f is the backfill volume; and T is seam thickness.

The correction on width-to-depth ratio is made only to adjust the subsidence factor and the inflection point (edge effect). In subsequent subsidence calculations, the cor-

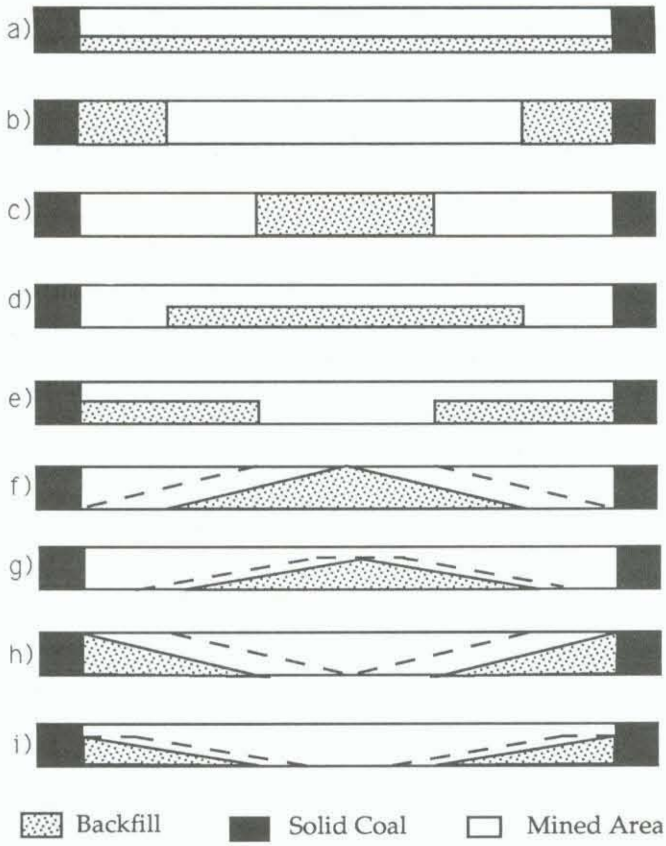


Figure 3 Basic Patterns Of Partial Backfill

rected width-to-depth ratio was used while the panel width remained. Note that if the height of the backfill pile is fairly uniform, only the extraction height needs to be adjusted accordingly.

Analysis of Simulation Results

To understand exactly how these patterns affect the various elements of ground movement, the subsidence package developed at Virginia Tech was used to estimate subsidence, slope, curvature, and strain distributions. Using a constant volume of backfill material equal to one-third the volume of the mined space, the nine basic patterns of backfill have been analyzed and their effects compared. Values for the various parameters affecting subsidence are listed in Table 1. Table 2 shows a comparison of their effects on reducing the four elements of ground movement. While pattern (c) produces the smallest surface subsidence, pattern (d) is the most effective in reducing tensile strain. As can be seen in Table 2, pattern (c) actually produces a higher tensile strain than the situation without partial backfilling, clearly due to the effect of strain superposition on both sides of the backfill pile. Pattern (d) also yields the lowest surface slope among all the patterns. The differing effects on subsidence obtained by varying the backfill pattern is defined as the pattern effect.

Table 1 Parameters used for subsidence analysis

Extraction thickness	6 feet
Total panel width	600 feet
Total panel length	2000 feet
Depth of mining	500 feet
Subsidence factor	0.7
Tanβ	2.31 (Appalachia)
Strain coefficient	0.35 (Appalachia)

A comparison of the distribution of the four elements of ground movement studied is shown in Figures 4 through 7. Generally speaking, pattern (d) is the most effective in controlling surface subsidence for the given volume. The distributions shown in Figure 5 illustrate the effects of strain superposition by fill pattern (c). If strain control is the main objective, pattern (c) should be avoided. In this instance, if the width of the backfill pile is fixed, a reduction in fill volume may be more effective in controlling surface strain because of the yielding pillar effect. This may be designated as the volume effect, by which different results can be achieved through varying the volume of backfill for a given shape, location, and material property. If such a pattern must be used, it is more beneficial to place it near the abutment. This effect is known as the location effect.

It is interesting to note that in Figure 6 patterns (c), (f), and (g) all reduced surface slope by a magnitude of 0.5, but they all carry a significant side effect. Namely, the number of changes in slope direction is increased. While the original surface has only two changes of slope direction, the surface with any one of the three partial backfill patterns would have four changes in direction. This side effect may be extremely undesirable for some surface structures. A comparison is also made in Figure 7. Even though patterns (c) and (d) produce about the same magnitude of surface slope, pattern (d) is more desirable than pattern (c) because it produces fewer changes in slope direction.

Finally, it should be pointed out that backfill patterns can be achieved by design or deformation of the fill material. Different material types may also have differing effects on ground support underground. Such effects due to material deformability and support capability are defined as the material effects.

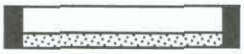
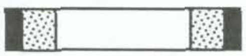
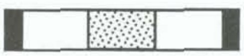
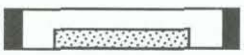
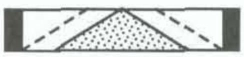
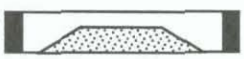
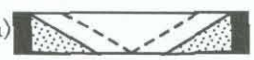
CONCLUSIONS

Based on analyses presented in this paper, the following conclusions can be made:

1) Surface subsidence effects can be controlled by selectively backfilling mined areas of underground openings.

2) The effects of partial backfilling can be classified

Table 2 Comparison of Effects of Fill Patterns on Ground Movements

Fill Pattern	Max Sub. (ft)	Max Slope (1/1000)	Max Curvature (1/1000)		Max Tensile Strain(1/1000)	Max Comp. Strain (1/1000)
	-4.15	1.96	0.137	-0.133	10.11	-10.40
a) 	-2.77	1.31	0.092	-0.090	6.75	-6.95
b) 	-3.83	1.90	0.160	-0.133	10.05	-12.09
c) 	-1.52	1.00	0.174	-0.161	12.20	-13.17
d) 	-2.10	0.82	0.050	-0.047	3.59	-3.78
e) 	-3.38	1.51	0.147	-0.093	7.08	-11.13
f) 	-2.63	1.47	0.176	-0.138	10.43	-13.30
g) 	-2.38	1.32	0.154	-0.126	9.54	-11.64
h) 	-3.84	1.78	0.136	-0.123	9.35	-10.34
i) 	-3.75	1.74	0.137	-0.118	8.96	-10.40

into four categories: volume, pattern, location, and material effects.

3) The volume, location, and geometry of the backfill pile all have differing effects on surface subsidence, slope, and strain, depending on the engineering requirements.

4) Depending on volume and characteristics of the backfill, the best fill location and geometry can be determined to produce optimum surface control.

REFERENCES

1. Rice, G. S. Coal mining in Europe. U.S. Bureau of Mines Bulletin No. 414. 1939, 369 pp.

2. Falkie, T., J. E. Gilley, and A S. Allen. Overview of underground refuse disposal. 1974, First Symposium on Mine and Preparation Plant Refuse Disposal. Louisville, KY. pp. 129-138.

3. Paone, J., R. H. Cox, and A. S. Allen. Subsidence-control project in the Belleville-Maryville area, Illinois. Mining Engineering. 1978, Vol. 30, pp. 183.

4. Brown, A., J. Murphy, and H. Bartell.. Slurry backfilling of an underground coal mine. Geotechnical Special Publication No. 21, ASCE, 1988, pp. 77-94.

5. Huck, P. J., Y. P. Chugh, and M. Jenning. Subsidence control in abandoned coal mines: U.S. practices. Proc. Conference on Ground Control in Room-and-Pillar Mining. 1982. pp.151-154.

6. Munjeri, D. Prevention of subsidence using stowing methods. Colliery Guardian. 1987. pp. 241-244.

7. Karmis, M., Jarosz, A., and Schilizzi, P. Monitoring and prediction of ground movvements above underground mines in the eastern United States coalfields. Proc. 6th Int. Conf. on Ground Control in Mining. 1987. pp. 184-194.

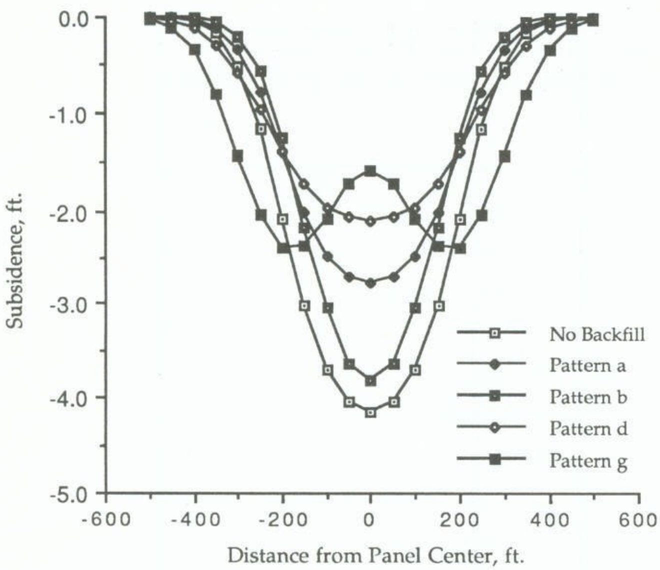


Figure 4 Effects of Fill Patterns on Surface Subsidence

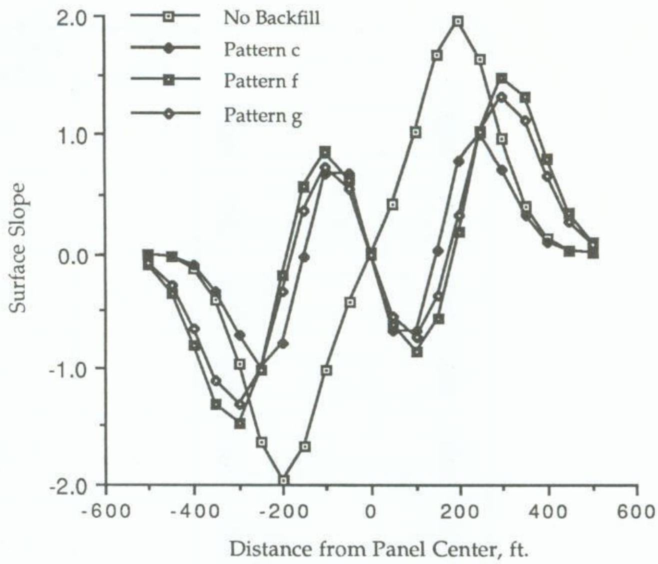


Figure 6 Effects of Fill Patterns on Surface Slope

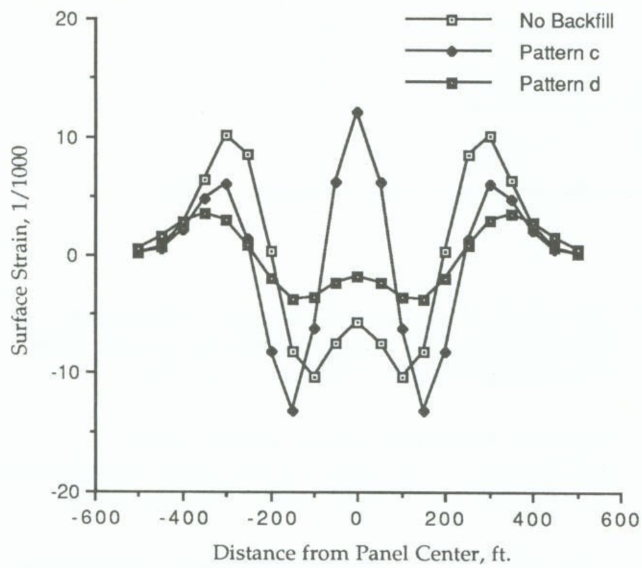


Figure 5 Effects of Fill Pattern on Surface Strain

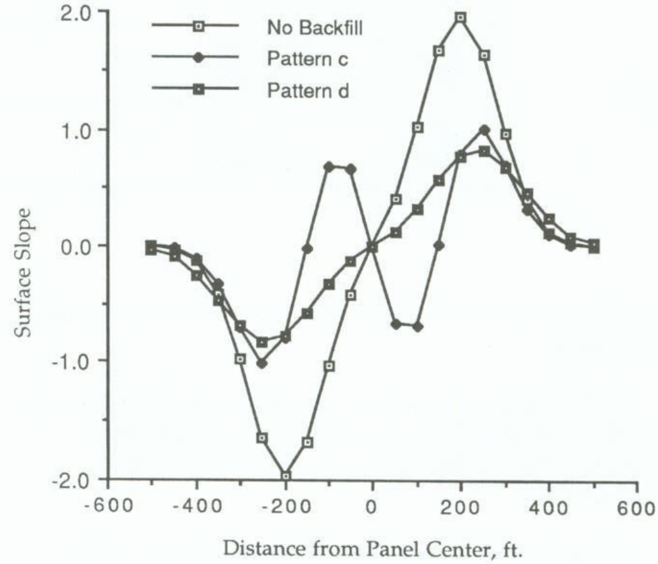


Figure 7 Effects of Fill Patterns on Surface Slope

ACKNOWLEDGEMENTS

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Chapter 4. Subsidence Control by Backfilling

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SURFACE SUBSIDENCE IN MINING

The consequences of subsidence are becoming progressively more serious as the consumption of an ever-increasing quantity of minerals conflicts with the needs of an expanding population for surface land area. The inevitable result will be an increase in the amount of land area on which undermining is restricted because surface subsidence must be prevented. Designation of land areas that may be undermined, provided that subsidence is controlled and kept within specified limits, also will increase. The decisions as to which land is to be used for urban, agricultural, or other purposes clearly involve considerations that go far beyond the scope of mining technology.

Under the authority assigned to the US Bureau of Mines (USBM) by the Organic Act (May 16, 1910) and its succeeding amendments and pursuant regulations (30 U.S.C. 1–11), the Bureau conducts scientific and technologic investigations concerning mining and its related problems. Subsidence-control demonstrations that have been conducted under this authority include projects in Wyoming, West Virginia, Illinois, and the Pennsylvania anthracite region.

The problems for technology are to devise methods for extracting minerals with controlled subsidence in varying geologic settings and for calculating or predicting the amount, extent, and characteristics of the subsidence that accompanies the extraction of our principal minerals mined by high-tonnage methods so that logical choices can be made from the possible alternative approaches.

Where underground minerals and fuels are mined and removed, the voids that are created underground generate strong imbalanced stresses in the surrounding and overlying rock strata. The resulting readjustments in the rock masses may cause subsidence of the ground surface. Subsidence implies vertical collapse, because the most conspicuous component of movement is downward. However, the downward component is accompanied by differential horizontal strains that may be more damaging to man-made surface structures than the more apparent vertical displacements.

SUBSIDENCE CONTROL

The most widespread method of alleviating potential subsidence problems in undermined areas has been to backfill mine voids with mine refuse or some other inexpensive material that provides lateral support to the remaining mine pillars and vertical support to the mine roof and overburden. Most USBM backfilling work has been conducted jointly with the Pennsylvania Department of Environmental Resources in the anthracite region of northeastern Pennsylvania. These joint projects were conducted under the authority of the Anthracite Mine Drainage Control Act of July 15, 1955 (Public Law 84–162, as amended), and the Appalachian Regional Development Act of 1965 (Public Law 89–4, as amended).

In addition to the Appalachian and mine-drainage projects, the Bureau has either conducted or participated in demonstration projects to develop the "pumped-slurry" method of backfilling mine voids. Three of these projects were conducted in Rock Springs, WY. The first was a field test of the pumped-slurry technique, conducted in 1970 under the combined participation of the City of Rock Springs, the US Department of Housing and Urban Development, the Dowell Div. of the Dow Chemical Co., and USBM (Candeub, Fleissig, 1971). The objective was to demonstrate that a large quantity of sand could be hydraulically injected under pressure through a single borehole, and that filling of the mine voids would be essentially complete. In the earlier "blind-flushing" methods, involving sluicing material through boreholes by gravity, quantities per injection hole ranged from 15 to 765 m^3 (20 to 1000 cu yd), and the mine voids were only partially filled. In the first test of the pumped-slurry method, approximately 14 900 m^3 (19,500 cu yd) of sand were injected successfully through a single borehole. Subsequent information obtained through 43 monitoring boreholes indicated that mine voids below 11 330 m^2 (2.8 acres) had been filled.

As a result of the successful initial test of the pumped-slurry process, USBM conducted additional backfilling demonstration projects using the new technique. The first full-scale demonstration was carried out in the Green Ridge section of Scranton, PA, between 1972 and 1974. This project proved the feasibility of using the new hydraulic-injection technique to backfill dry mine voids, as well as flooded voids, and it demonstrated that crushed anthracite refuse could be used as easily as sand in the process. Stabilization was provided for about 202 300 m^2 (50 acres) of Scranton, having a population of approximately 1000.

Additional demonstration projects at Rock Springs, WY, between 1973 and 1975, resulted in the stabilization of about 364 200 m^2 (90 acres). The work was necessary to preserve the physical and economic well-being of the city, because there have been numerous occurrences of subsidence during recent years. A demonstration project completed in 1976 in Rock Springs brought the total stabilized area in that city to 647 500 m^2 (160 acres). At a cost of about \$3,000,000, a population estimated at 7000 persons and property values exceeding \$18,000,000 were protected.

Presently, USBM is participating in subsidence-control projects involving backfilling in Pennsylvania, Illinois, and West Virginia.

Eleven demonstration projects were in progress by USBM in 1977—eight in the Pennsylvania anthracite region and three in bituminous areas of West Virginia (one) and Illinois (two). The estimated total property value is over \$100,500,000, and the total cost of the projects is expected to be nearly \$20,000,000. The demonstration projects are designed to adapt the pumped-slurry technique to a variety of subsurface conditions, to increase efficiency, and to reduce costs.

In several areas, some subsidence already has occurred, causing hazards to the local communities and decreasing property values.

SUBSIDENCE-CONTROL TECHNOLOGY

The methods used in the subsidence-control program —controlled backfilling, conventional blind backfilling, and blind backfilling by pumped-slurry injection—are forms of hydraulic backfilling. The controlled-backfilling method is a long-established technique that provides excellent support, but its use is restricted to accessible mines. For inaccessible mines, the conventional blind-backfilling method only fills about a third of the void space in the mine workings, whereas the newly developed pumped-slurry method of blind backfilling provides essentially complete filling, both laterally and vertically.

An historical review of hydraulic and other methods of backfilling mine workings was published by Pennsylvania State University (Luckie and Spicer, 1966). Some of the general practices and requirements described in the ensuing paragraphs are based on USBM experience, and others are based on state and federal statutory requirements.

Hydraulic backfilling involves transporting or flushing the fill material with water through pipelines or boreholes (or a combination of both) to the point of stowage in the mine. When sufficiently dewatered, hydraulic fills are more resistant to vertical and lateral pressure than pneumatically placed or hand-packed fills because water-deposited particles tend to compact uniformly, producing a degree of confined compressive strength and forming a level bearing surface. In laboratory tests, the bulk density of minus 6.4 mm (0.25 in.) crushed mine waste in the dry condition ranged from 1220 to 1280 kg/m^3 (76 to 80 lb per cu ft), samples hydraulically placed in water-filled receptacles ranged from 1470 to 1550 kg/m^3 (92 to 97 lb per cu ft), and other samples similarly placed and allowed to drain ranged from 1540 to 1700 kg/m^3 (96 to 106 lb per cu ft). Loading tests, simulating 61 m (200 ft) of overburden, indicated that the compressibility of the material was about 5% (Whaite and Allen, 1975).

Normally, water for flushing backfill material into boreholes is obtained from the mine-water pools found in most abandoned mine workings. During active mining, the mine workings are kept dry by pumping. Once mining ceases, the pumping is stopped, and mine water pools grow in size, inundating the lower workings, often over areas of many square miles. The resulting reservoirs of mine water are an ideal source of water for hydraulically backfilling abandoned mine workings. Water is raised to the surface by deep-well pumps that are installed after boreholes to a mine-water pool have been drilled and cased. The water is carried by pipelines either to flushing hoppers or to slurry blenders, where it is mixed with crushed or screened material and flushed into the mine workings to be supported. The mine workings may be flooded, in which case the water simply has returned to its source. However, if the workings are dry, the water normally drains off the deposited fill material and flows downward along the mine floor until it reaches the mine-water pool. Where dry workings are isolated from the mine-water pool, it sometimes is necessary to drill drainage boreholes to lower inundated areas, allowing dewatering and compacting of the fill material.

Generally, USBM has used crushed coal-mine refuse to backfill subsidence-prone voids in Appalachian mining regions. In these areas, preparation-plant refuse is obtained from nearby waste banks, which are numerous and readily accessible as sources of fill material. However, in other areas where coal refuse has been either unavailable or more valuable for other uses, USBM projects have used screened sand for the backfill material.

Normally, preparation of the backfill material is carried out at fill-excavation sites, whether coal refuse or sand. If material is obtained from a coal-refuse bank, it is prepared by a raw-feed crushing plant that is operated in conjunction with equipment for stockpiling the material, as shown in Fig. 1. The material generally is crushed to the desired size and stockpiled at the rate of about 190 m^3 (250 cu yd) per hour. Crushers are equipped with automatic dust-control systems to prevent the discharge of dust into the atmosphere, in accordance with applicable laws. If sand is used instead of coal refuse, the desired size of the material often can be obtained by means of a conveyor belt that is fitted with a screen to reject all oversize particles and unwanted materials.

Preparation and handling can be simplified when fly ash is used for backfilling. This material can be obtained from various power plants or industrial sites where pulverized coal is burned to provide energy. The fly ash is a product of the burning process, and it is collected in the stacks of these facilities by electrostatic precipitators. Stored in enormous quantities, fly ash poses a serious disposal problem. It may be readily applicable to backfilling since it can be placed directly into enclosed pneumatic transport trucks (of the kind used for transporting dry cement), carried to the backfilling sites, and

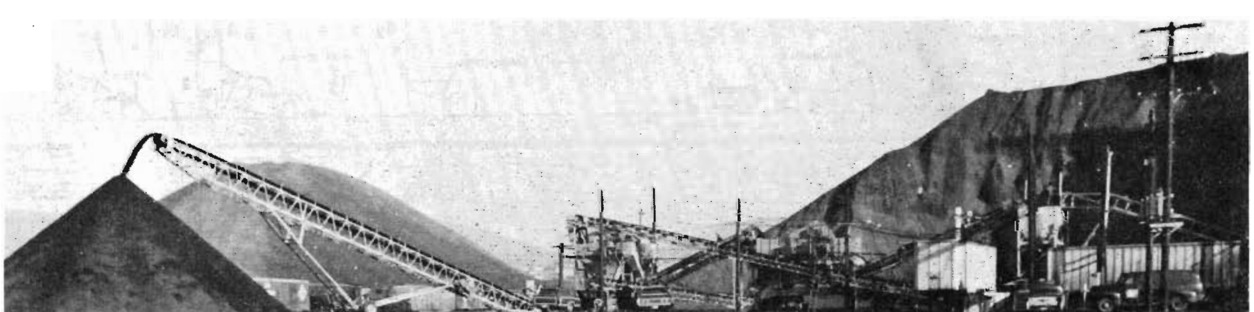


Fig. 1. Crushing plant and stockpiling conveyor at a refuse bank.

Fig. 2. Truck dumping fill at a flushing hopper.

injected into the voids without prior screening or crushing.

The usual method of transporting fill material to backfilling sites utilizes trucks, normally of about 11-mt (12-st) capacity. As shown in Fig. 2, the material may be deposited directly into a portable steel hopper located at the top of a borehole; the material then is flushed into the mine with water. Alternatively, the material may be delivered directly to slurry blenders or stockpiles located nearby. The portable steel hoppers are used in either controlled or conventional blind backfilling, and the slurry blenders are used in pumped-slurry backfilling. In both controlled and conventional blind backfilling, the fill is mixed with water supplied through fire hoses that are mounted on the steel hopper. The mixture of water and fill flushed from the hopper and down the borehole under the force of gravity is over 40% solids by weight. In pumped-slurry backfilling, however, the fill is fed into the blender, mixed with water, and agitated by water jets, sometimes aided by mechanical agitators. The agitated slurry is pumped under pressure through a pipeline to the injection boreholes. The solids content of the slurry ranges between 15% and 30% by weight in this process.

The water for flushing the fill initially is carried from the water wells to the flushing sites (hoppers or slurry blenders) by means of pipelines that range from 152 to 305 mm (6 to 12 in.) diam. The slurry is carried by pipelines with diameters ranging from 152 to 356 mm (6 to 14 in.). Normally, these pipelines are laid on the surface along street curbs; however, at all street intersections, they are laid below the surface and covered by paving material. If the pipelines are unusually large, they are laid entirely below the ground surface. Contractors are required to see that any trench material left over after laying pipelines is properly stored and stabilized until the pipelines are removed and the material is replaced. At entrances to driveways, ramps are constructed over any surface pipelines for the convenience of property owners. The same procedure is followed for all pipelines carrying slurry for the pumped-slurry backfilling process. Contractors are required to obtain any necessary permits or permission to lay pipelines under highway and railroad rights-of-way.

Controlled Backfilling

In mine workings that are safe and accessible to workmen, backfilling is accomplished by flushing the fill through boreholes to the designated mine workings; then, this material is fed directly into pipelines that carry the backfill slurry to a selected portion of the mine, as shown in Fig. 3. This method is called controlled backfilling, due to the accuracy and completeness with which the workmen can fill the voids. Controlled backfilling has been used with considerable success in joint subsidence-control projects conducted by USBM and the Commonwealth of Pennsylvania in the anthracite region of northeastern Pennsylvania. This method was developed by anthracite mining companies to support important surface areas, to extinguish mine fires, and to arrest the development of squeezes in pillars.

It generally is necessary to drill numerous boreholes of various diameters to flush the fill into the mine voids and to provide supplies, access, and ventilation for underground workmen. Also, wells must be drilled to provide flushing water from the mine-water pools, and boreholes must be drilled to drain water from isolated workings to lower areas in the mines. Whenever possible, old mine maps and surface surveys are used to determine the actual locations of voids. This insures accuracy in placing the drill rigs so that boreholes enter voids instead of pillars. Rotary drilling rigs must be equipped with complete dust-control systems to prevent

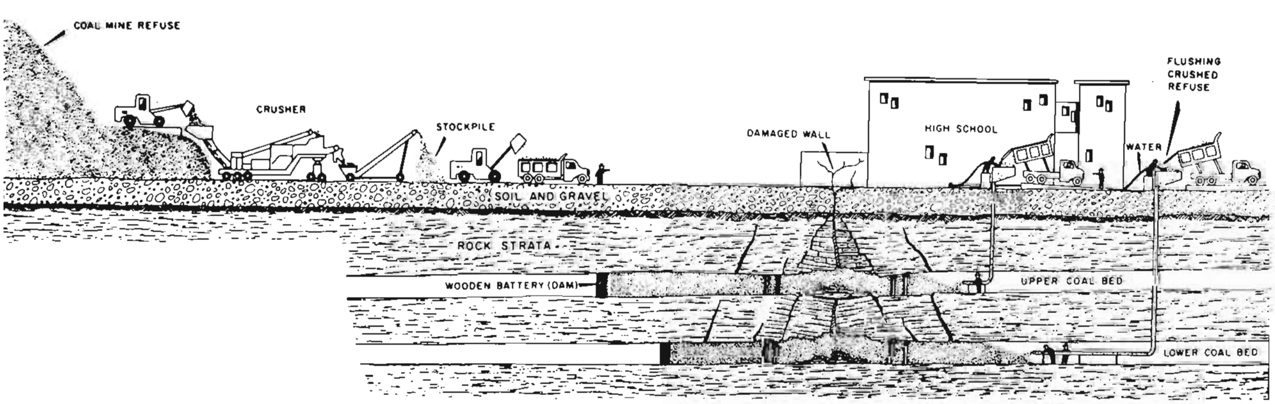


Fig. 3. A typical controlled backfilling project.

the discharge of dust, complying with state and local air-quality standards.

Each borehole is drilled through the alluvium and is cased and cemented to a point 1.9 to 3.0 m (6.5 to 10 ft) below the top of the bedrock before continuing with drilling. The casing prevents disruption of ground water within the alluvium and keeps loose material from blocking the borehole. After the casing is installed, drilling continues at a diameter sufficient to accommodate the required casing to mine openings in the beds to be backfilled. When not in use, the boreholes are capped with steel plates that are flush with the ground surface and spot-welded into place.

Several large-diameter boreholes, for ingress and egress of workmen and for ventilation, are drilled, reamed, and cased to designated voids in each of the beds to be backfilled by controlled methods. Typical specifications require that each hole be reamed to about 1118 mm (44 in.) diam from the surface to a point about 1.2 m (4 ft) below the top of bedrock; new 1118-mm (44-in.) steel casings are sealed into the solid rock with a cement-grout mixture before drilling and reaming continues. From there on, drilling continues at a diameter sufficient to accommodate new 1016-mm (40-in.) steel casings from the surface to the roof of the mine workings. The casing is supported at the ground surface and is welded in several sections to effect a smooth inner surface at all joints. This insures the safety of workmen who travel the access boreholes in hoisting cages (similar to that shown in Fig. 4). Securely locked covers at the tops of the access and ventilation boreholes remain in place at all times, except when men are underground and hoist operators are on duty at the tops of the boreholes.

Underground, the workmen erect props, move loose rock, level roof falls, and move gob (stowed underground waste) to provide sufficient height and width for safe travelways and for installation of flushing pipelines. A 1.6-rad (90°) long-radius, heavy-duty pipe elbow is attached to the foot of each flushing borehole casing; this diverts the backfilling slurry to 152-mm (6-in.) diam underground pipelines that carry it to designated areas for stowage, as shown in Figs. 5 through 7. In controlled backfilling, a single flushing

Fig. 5. Elbow at the foot of a controlled flushing borehole.

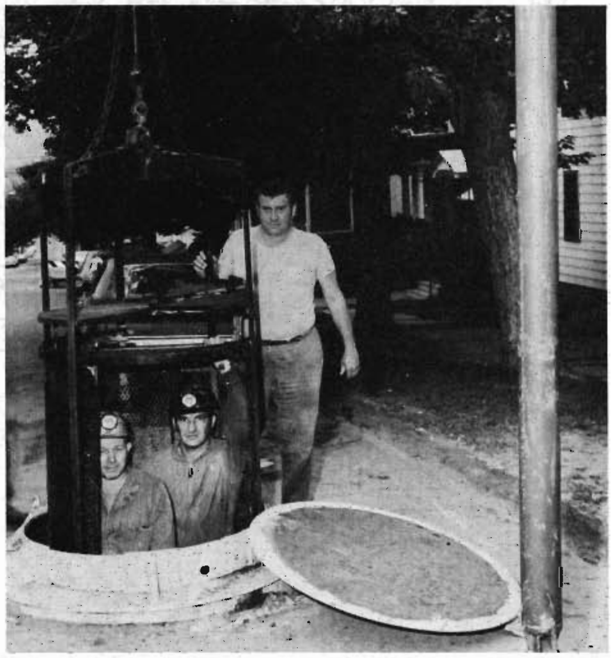


Fig. 4. Hoisting cage for access to the mine workings in a controlled backfilling operation.

Fig. 6. Typical underground flushing pipeline for a controlled backfilling operation.

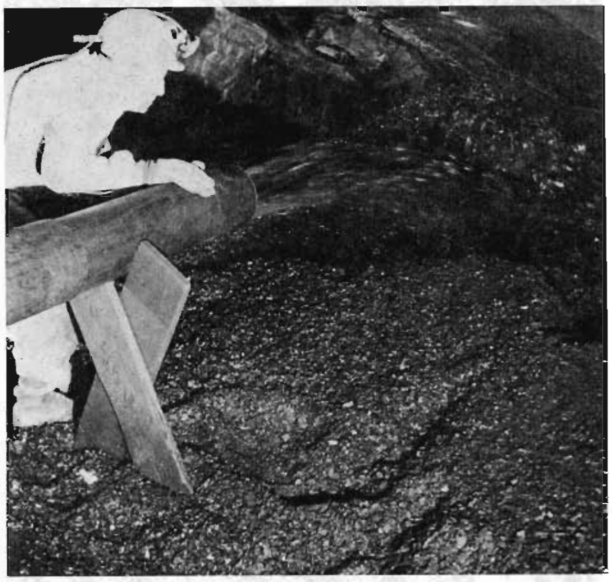


Fig. 7. Slurry flowing from an underground pipeline during controlled backfilling.

borehole may be used to fill voids in areas from 16 200 to 40 500 m² (4 to 10 acres), whereas conventional blind backfilling requires 20 to 25 boreholes per 4000 m² (1.0 acre). Permeable bulkheads are constructed where needed to provide containment of the fill.

Stowage of fill continues until all mine openings are packed solidly to the roof. All cavities above the roof line, resulting from local rock falls, also are backfilled solidly by inserting a flush line into the area and injecting material to the point of refusal. It often is necessary to construct bulkheads on one or more sides of the caved area to contain the material.

Backfilling normally proceeds only during daylight hours, five days per week. Work at a hopper site may block the street partially, but usually only two or three hoppers are used at any one time, so traffic disruption is limited. Flushing at any single borehole may continue for a period of 2 to 12 weeks, depending upon the subsurface conditions. Contractors are responsible for any spillage, and spilled material must be cleaned up immediately.

Conventional Blind Backfilling

Where hazardous or flooded conditions make abandoned mine workings inaccessible to workmen, blind-backfilling techniques must be employed. In past subsidence-control projects in northeastern Pennsylvania, the conventional blind-backfilling technique has been successful, even though the degree of backfilling is much less than that achievable with the controlled method. In conventional blind backfilling, the fill is flushed out of portable steel hoppers in the same manner as in controlled backfilling. Since there is no way to divert or channel the slurry once it enters the mine workings, the material is merely washed into the flushing boreholes by gravity until the boreholes no longer accept the material. In horizontal and gently dipping coal seams, the material forms conical piles in the voids under the boreholes. These piles eventually block the boreholes, preventing much of the mine void between any two boreholes from being filled. Consequently, this method requires closely spaced boreholes with as many as 20 to 25 per 4000 m² (1.0 acre). Only about 61 m³ (80 cu yd) of fill can be sluiced through a typical borehole. In urban areas, most of the drilling must be done in city streeets and alleys; thus, blind flushing under the streets transmits only indirect support to adjacent homes and buildings, as shown in Fig. 8.

Pumped-Slurry Backfilling

The pumped-slurry method has been developed recently by USBM. With this blind method, inaccessible mine workings are filled essentially from floor to roof for considerable lateral distances, using few injection boreholes (Stewart and Heslep, 1974; Whaite and Allen, 1975). In one project, more than 272 000 mt (300,000 st) of fill has been injected through a single borehole, reaching distances of about 250 m (820 ft) from the point of injection.

USBM has been experimenting with the use of coarse fill material to reduce the costs of crushing mine waste. Where sufficient quantities of fill can be prepared by screening alone, both the cost and the environmental impact of a crushing operation are eliminated. Recent demonstration projects have been experimenting with small portable system components to reduce the costs for backfilling mine workings where caved material limits the extent of underground fill distribution.

Solids are placed in suspension in a mixing tank and conveyed in a closed system through a slurry pump, distribution pipeline, and injection borehole into the subsurface voids. During most of the injection process, slurry-pump pressures need be only sufficient to convey the slurry through the distribution pipeline to the top

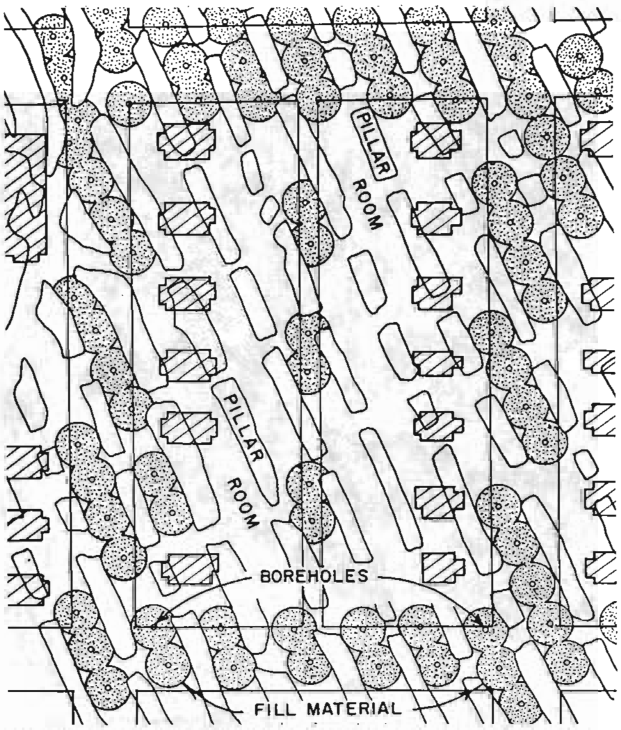


Fig. 8. A typical residential block over a mined area, showing the pattern of conventional blind-flushing holes in streets and alleys (circular areas around hole represent backfill material emplaced in the void).

of the injection hole; during brief periods of resistance to slurry distribution underground, slurry-pump pressures are increased to overcome the resistance.

The relatively few injection boreholes are located carefully according to mine maps and exploratory drilling, obtaining maximum benefits from distributing large quantities of fill material through each injection hole. A sonar caliper may be lowered into probe holes to obtain evidence that adjacent mine openings have not become obstructed by caving of the mine roof. In locations favorable for widespread distribution of fill, large-diameter slurry pipelines and injection boreholes are used. The underground dispersal of backfill material is traced during the injection period by lowering a calibrated line through monitor boreholes placed at frequent intervals. The system of monitor holes usually consists of the exploratory boreholes plus additional holes needed to delineate the boundary of the area that has been filled effectively.

All boreholes are drilled, cased, and cemented to a point 2.0 to 3.0 m (6.5 to 10 ft) below the top of the rock strata before continuing drilling to the designated voids. This practice prevents disruption of the shallow alluvial ground-water table and keeps loose material from blocking the borehole. Subsequently, the boreholes are drilled and cased to the specified mine openings. Prior to use, each borehole is fitted with a removable cap that is recessed several inches below ground level. Provision is made for raising and lowering the casing in monitoring and injection boreholes, providing access to a coalbed or isolating it from underlying or overlying beds.

Sites are selected for stockpiling fill material and installing necessary equipment such as conveyors, slurry blenders, slurry pumps, and deep-well pumps. Blending and pumping equipment is connected to the deep-well pumps by steel pipelines. Fig. 9 shows an operating pumped-slurry plant; the on-site equipment consists of the following:

1) A conveyor is used to feed crushed or screened material into the slurry blender at a steady rate. The conveyor is fitted with a hopper that is filled by a dump truck, a front-end loader, or a belt originating at the stockpile of crushed solids. The conveyor also is equipped with screening equipment that prevents oversize material or rocks from impeding the backfill process.

2) The slurry blender is a mixing tank that receives water from the deep-well pumps and fill material from the conveyor. The tank is equipped with water jets or mechanical devices, and it agitates the mixture to a fluid condition.

3) The slurry is fed through suction lines from the blender to a large slurry pump that forces the slurry into steel pipelines for transport to the injection boreholes.

Initially, water is pumped from the flooded mine workings, through the blending equipment and slurry pipeline, and back into the mine by means of an injection hole; this tests the dependability of the equipment. When it has been determined that the equipment is adequate to inject the fill material safely, the crushed or screened fill material is added to the system gradually until the desired rate of injection is attained. The agitated slurry is impelled with such velocity that the injected material continues to behave in a fluid manner when it reaches the bottom of the borehole. Instead of forming a cone such as in conventional blind backfilling, the deposited material forms a doughnut-shaped mound at the bottom of each injection hole. As the mound builds, slurry flows radially outward over the upper surface of the mound, with deposition continuously occurring on the outer slopes where the flow velocity no longer suffices to carry material in suspension. This is shown in Fig. 10.

Model studies of this process and USBM demonstration projects conducted at Rock Springs, WY, and Scranton, PA, have indicated that the fill material

Fig. 9. A pumped-slurry plant; water enters pipe at left; solids enter tank from conveyor at right; suction line transmits slurry to pump in center; slurry enters distribution pipeline that passes under water pipeline to right foreground.

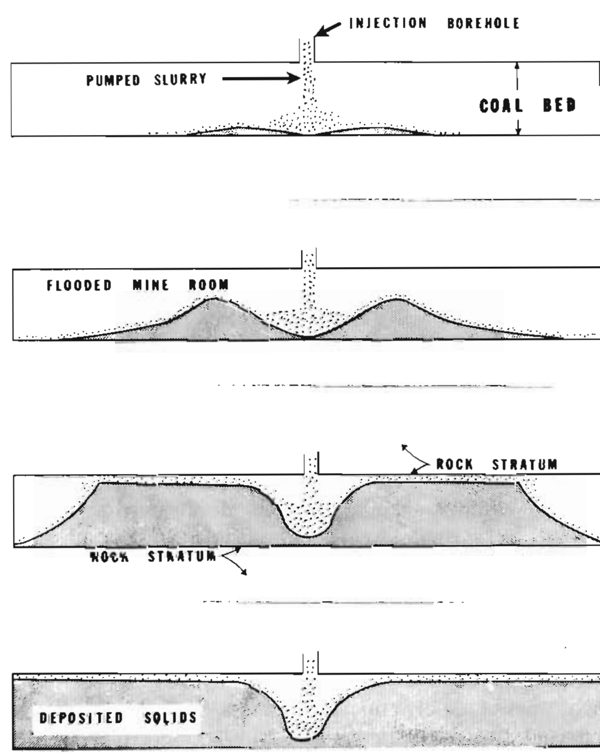


Fig. 10. Diagrammatic sections through a flooded mine room at the point of slurry injection, showing movement of particles and growth of fill deposit.

spreads along the path of least resistance until all interconnected mine voids surrounding the injection holes are filled (Carlson, 1975 a, b; Whaite and Allen, 1975). Throughout the course of backfilling operations, contractors must work closely with USBM and state representatives in monitoring the progress of backfilling and determining when injection should cease at one borehole and begin at another. Contractors also are required to have all functions of the equipment under control at all times, including the slurry-pump pressure, the water volume, and the slurry density, all of which remain variable and dependent upon the conditions encountered at each project. Limitations are delineated in individual project specifications.

Use of Fly Ash: In the bituminous coal region of southwestern Pennsylvania, fly ash has been injected into voids to control fires in abandoned coal mines in federal-state cooperative projects. Fly ash also has been used by the Pennsylvania Department of Environmental Resources to control subsidence (Magnuson and Malenka, 1970; Murphy, et al., 1968). The mined coalbeds in this region are flat, shallow, and above the drainage level in many areas. Fly ash also was used in one locality in the anthracite region. Subsidence protection is provided for individual structures by injecting fly ash to form in-situ piers. Fly ash is injected either in the dry state by pneumatic methods, or as a slurry.

Pneumatic-backfilling systems have been limited almost entirely to active mines, but they are applicable to abandoned mines where controlled hydraulic flushing is used. Pneumatic systems provide a possible alternative to controlled hydraulic-flushing projects, particularly where drainage problems are substantial.

In the pneumatic injection of fly ash for subsidence control, the fill material can be obtained at power plants or industrial facilities that burn pulverized coal. Trucks used for transporting the material are pneumatically sealed bulk-tank trucks of the kind used for carrying dry cement. These trucks have a capacity of approximately 18 t (20 st), and they are equipped with standard pneumatic transportation (injection) facilities and pumps. Thus, fly ash may be driven from a dry-storage facility directly to an injection borehole, where it can be injected into the mine voids with no equipment other than that on the truck.

Fly ash has a low angle of repose of about 0.14 rad (8°) when injected pneumatically and has shown good flow characteristics in pipes and passageways. Thus, it would be relatively easy to fill either open or caved voids with this material. Moisture is present in the voids, and the fly ash tends to absorb the moisture and harden without shrinkage. It is doubtful that fly ash would be suitable for injection into inundated mine voids, because it is a very fine material and probably would not settle out rapidly to form a sufficient depth of compact fill under water.

Dry fly ash normally would be injected at pressures ranging up to 103 kPa (15 psi). The quantity of fly ash pneumatically injected into a void could be increased by intermittently stopping the introduction of fly ash while permitting injected air to open a new channel through the fine material already in place. The same results also could be achieved by maintaining a high rate of airflow while decreasing the rate of material flow.

Hydraulic slurrying of fly ash is effective for injection into caved mine workings. A water content between 25% and 30% by weight produces a slurry that pumps easily, yields very little drainage water while settling, and produces no shrinkage when dry.

The increasing availability of fly ash as a waste product has stimulated considerable interest in its suitability for use in subsidence control. USBM is investigating the chemical and physical characteristics of fly ash admixtures with other waste materials, to explore possible cementitious properties.

Demobilization and Cleanup: Upon completion of backfilling, all boreholes are filled to within 2 to 3 m (6.5 to 10 ft) of the top of the rock strata and then sealed with concrete to the top of rock. The remainder of the boreholes to the surface are filled with tamped soil. Where boreholes are located in paved roadway surfaces, the top 0.3 m (1.0 ft) of each borehole is filled with pavement material. If the borehole casing pipe is not recovered, the top of the casing is cut off at least 0.6 m (2 ft) below ground. As the filling and plugging of boreholes continue, pipelines for supplying water and conveying slurry are removed, except such pipelines as may be needed for completing project work at more distant backfill sites. Streets and work areas are restored to prior or better condition. In the unusual situation where naturally occurring radioactive materials lie between the backfilled bed and an aquifer, boreholes should be completely sealed and plugged throughout their entire length.

At backfill excavation sites, the equipment is removed, all reject material from the screening or crushing process is buried, and the area is regraded. If a coal-refuse bank has been used as the source of fill, the remaining refuse is gently sloped and contoured so that no hanging walls remain. However, if naturally occurring materials have been used, all borrow pits must be reseeded with appropriate vegetation after having been regraded to gentle and stable contours.

POTENTIAL SUBSIDENCE AREAS

It is possible for subsidence to occur in any area where minerals or fuels have been extracted by underground mining.

Most land affected by subsidence is removed from population centers. Adverse effects in these areas normally are decreased land values, altered drainage patterns, flooding of crop or forest land (in areas having high groundwater tables), or near-total loss of stream flow through fractures in the underlying rock. In urban areas, considerable human distress and millions of dollars worth of damage have resulted from the caving of buildings, pavements, and utilities. Not all of the subsidence damage in urban areas is severe, however. Some of the damage is minor and cannot be distinguished from that resulting from improper construction. Some subsided areas may have reached a stable condition.

Table 1 lists urban areas where mining has occurred and indicates the extent of undermining and the possible locales of future projects. No survey has been conducted to identify present subsidence problems or the

Table 1. Urban Areas where Mining Has Occurred

State	Mining Activity and Urban Area
Alabama	Coal and iron mines: adjacent to Birmingham.
Arizona	Copper mines: Bisbee and Jerome.
Arkansas	Coal mines: Hartford, Montana, Paris, and Spadra.
Colorado	Coal mines: Dacona, Firestone, Frederick, Lafayette, and Louisville. Lead-zinc mines: Leadville.
Idaho	Gold, silver, and lead-zinc mines: Burke, Gem, Kellogg, Mullan, Murray, and Smelterville.
Illinois	Portions of cities and towns probably underlain by mines include Belleville, Benton, Breeze, Carbondale, Centralia, Collinsville, Danville, Decatur, Edwardsville, Harrisburg, Herrin, Johnston City, Marion, Maryville, Mount Vernon, Springfield, Streater, West Frankfort, and Ziegler. Lead-zinc mines: underlie Galena.
Indiana	Coal mines: Ashboro, Augusta, Boonville, Brazil, Buckskin, Carbon, Centerpoint, Chandler, Dugger, Evansville, Fort Branch, Francisco, Gibson, Hymera, Kings, Knightsville, Linton, Newburgh, New Goshen, Petersburg, Seelyville, Terre Haute, and Yankeetown.
Iowa	Coal mines: Boone, Centerville, Des Moines, Knoxville, Oskaloosa, and Ottumwa.
Kansas	Zinc-lead mines: Galena and Treece. Limestone mines: Kansas City. Coal mines: Alma, Atchison, Burlingame, Cherokee, Croweburg, Franklin, Frontenac, Lansing, Leavenworth, Mineral, Mulberry, Osage City, Pittsburg, Pleasanton, Scammon, Scranton, Weir, and Williamsburg. Salt mines: Hutchinson, Kanopolis, and Lyons.
Kentucky	Coal mines: Madisonville. Limestone mines: Lexington.
Maryland	Dimension stone mines: Cardiff. Coal mines: Frostburg.
Michigan	Iron mines: Bessemer, Iron River, Ironwood, Ishpeming, Negaunee, and Wakefield. Salt mines: Detroit. Gypsum mines: may be under Grand Rapids. Copper mines: adjacent to and probably underneath Calumet, Hancock, and Houghton.
Minnesota	Iron mines: Aurora, Biwabik, Chisholm, Eveleth, Hibbing, and Keewatin.
Missouri	Zinc-lead mines: Alba, Aurora, Caterville, Duenweg, Neck City, Oronogo, Purcell, Webb City, and Wentworth. Lead mines: Annapolis, Bonne Terre, Desloge, Doe Run, Flat River, Leadington, Leadwood, Valles Mines, and Viburnam. Coal mines: Bevier, Brookfield, Bucklin, Cainsville, Cameron, Carrollton, Clifton Hill, Deepwater, Elmira, Farber, Huntsville, Kansas City, Kingston, Kirksville, Knoxville, Lexington, Macon, Marceline, Melbourne, Milan, Mindenmines, Missouri City, Montgomery City, New Cambria, Richmond, St. Louis, Trenton, Vibbard, Waverly, Wellington, Windsor, and Winston. Clay mines: Deepwater and St. Louis. Limestone mines: Carthage, Kansas City, and Neosho. Sandstone mines: Crystal City.
Montana	Copper mines: Butte, Centerville, and Walkerville.
Nevada	Gold and silver mines: Tonopah and Virginia City.
New Jersey	Iron mines: Dover, Hibernia, Mine Hill, Ringwood, Rockaway, and Wharton.
New Mexico	Potash mines: Carlsbad.
New York	Iron mines: Lyon Mountain, Mineville, and Witherbee.
Ohio	Coal mines: may underlie some urban areas in the southeastern and eastern parts of the State. Salt mines: Cleveland
Oklahoma	Coal mines: Bokoshe, Broken Arrow, Coalgate, Coalton, Cottonwood, Dewar, Haileyville, Hartshorne, Henryetta, Krebs, Lehigh, McAlester, McCurtain, Tulsa, and Wilburton. Zinc-lead mines: Cardin, Commerce, North Miami, Peoria, Picher, and Quapaw.
Oregon	Coal mines: Coos Bay. Iron mines: Oswego.
Pennsylvania	Anthracite mines: The anthracite region and particularly the Northern anthracite field including Scranton and Wilkes-Barre. Bituminous mines: portions of the following urban areas are undermined: Brownsville, Canonsburg, Charleroi, Donora, Johnstown, metropolitan Pittsburgh, Monongahela, and Uniontown.
South Dakota	Gold mines: Lead.
Virginia	Gypsum mines: Plasterco. Coal mines: Norton
Washington	Coal mines: Bellingham, Black Diamond, Carbonado, Centralia, Chehalis, Cle Elum, Issaquah, Newcastle, Ravensdale, Renton, Ronald, Roslyn, and Wildeson. Iron mines: Hamilton. Gold mines: Chewelah, Republic, and Wenatchee. Lead-zinc-silver mines: Leadpoint and Metaline.
West Virginia	Coal mines: Barrackville, Bartley, Bradshaw, Fairmont, Fairview, Farmington, Grant Town, Monongah, Rivesville, and Welch.
Wisconsin	Lead-zinc mines: Benton, Hazel Green, Mineral Point, New Diggins, Platteville, Shullsburg, and Tennyson. Iron mines: Hurley and Montreal.
Wyoming	Coal mines: Reliance and Rock Springs.

proximity of the mine workings to the existing urban development.

Interest in subsidence control has been expanding since passage of the Surface Mining Control and Reclamation Act of 1977, and the inclusion in that legislation of surface effects of underground mining. The resulting federal and state programs will be increasingly concerned with subsidence effects of both active and abandoned mining. To obtain a permit for underground mining, the operator is required to mine in such a way as to prevent subsidence that will cause material damage, to the extent technologically and economically feasible, except where the mining method requires planned subsidence in a predictable and controlled manner (Sec. 516). Among the uses set forth for the Abandoned Mine Reclamation Fund are sealing and filling deep mine entries and voids and the prevention, abatement, and control of coal mine subsidence (Sec. 401). Subsidence control activities in the USBM Mined Land Demonstrations program will be supplemented by investigations and demonstrations cooperative with the Office of Surface Mining.

ENVIRONMENTAL CONSIDERATIONS

USBM subsidence control projects generally have been conducted in heavily built-up urban and suburban areas that historically have been centers of coal-mining activity. Normally, these areas have been undermined by the room-and-pillar system of mining, and two or more coal seams usually have been mined. Individual project areas have ranged in size from 8100 to over 810 000 m^2 (2 to 200 acres). Much of the USBM subsidence-control work has been conducted jointly with the Commonwealth of Pennsylvania in the Northern anthracite field of eastern Pennsylvania. Primarily, this work has been centered in the Scranton and Wilkes-Barre area. However, initial experimentation with the pumped-slurry backfilling process was conducted at Rock Springs, WY. The subsidence-control technology developed in these areas is considered equally applicable to all mining regions of the country where tabular deposits in stratified rock have been mined. In areas selected for hydraulic backfilling, subsidence may or may not already have affected the surface, but, in all cases, there is strong evidence that remaining mine pillars and structures are no longer adequate to support the mine roof and overburden. The potential for subsidence is greatest where the extraction ratio for coal has been high (often with less than 40% of the coal remaining as pillars), where the patterns of mining have been irregular, or where pillars in closely spaced mined beds are not columnar. Faults in the rock strata may be particularly significant in determining the location and severity of subsidence.

Occupants of surface areas already affected by subsidence have experienced buckled streets and sidewalks, heavily damaged homes and other structures, and broken or ruptured utility lines. Broken gas lines have proven extremely dangerous, increasing the possibility of gas explosions. In many cases, residents have been forced to either abandon their homes or shore up foundations and make other structural improvisations to compensate.

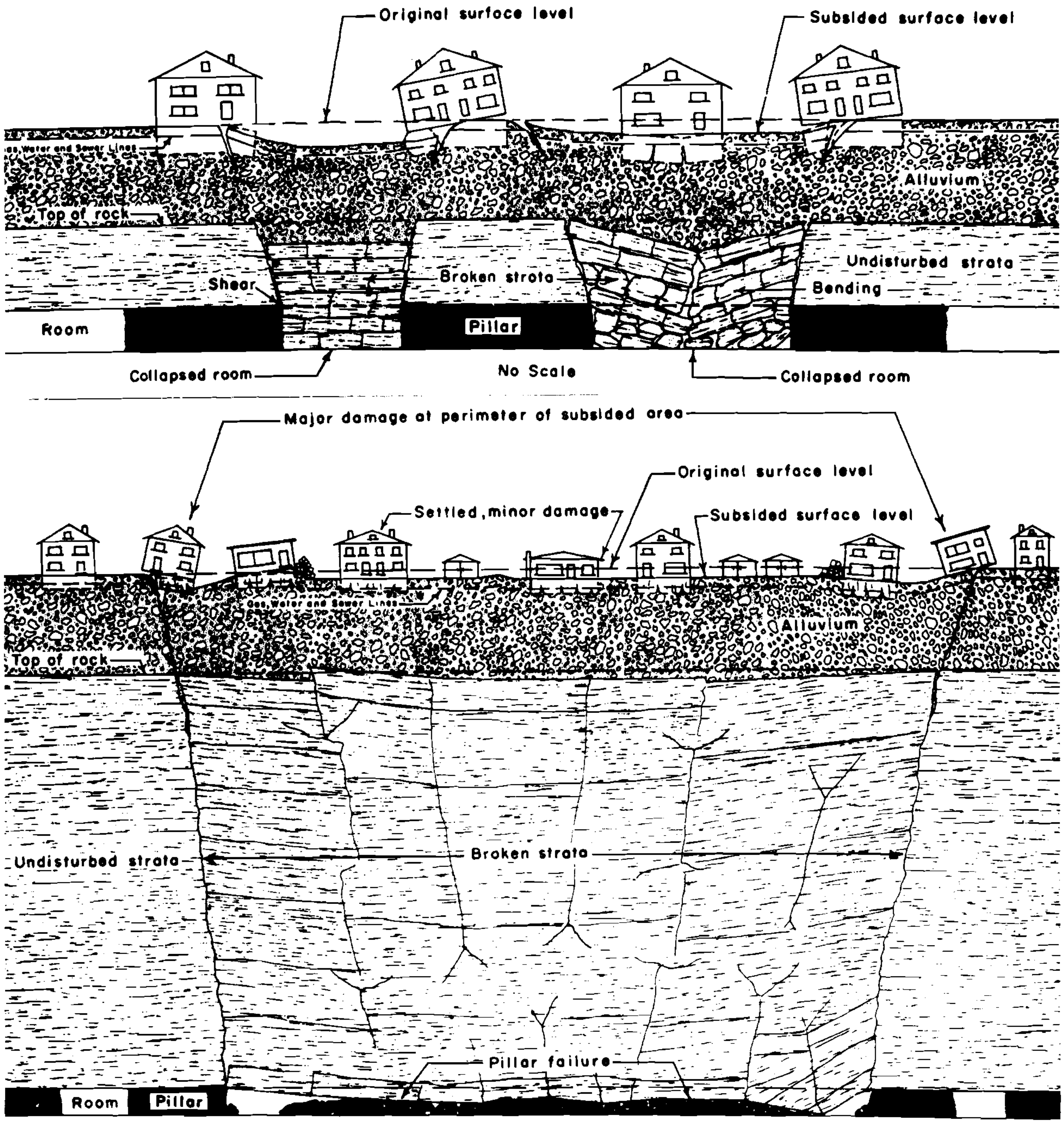
Without backfilling operations, subsidence-prone areas can be affected either by localized and random "pothole" occurrences or by general surface-subsidence movements that can attain a magnitude of up to 1.0 m (3.0 ft) or more of vertical displacement, as shown in Fig. 11. In either case, there would be a high horizontal component of strain that could heavily damage buildings, streets, and utilities. In all such cases, the economic impact upon residential and business areas would be severe, and both on-site and adjoining property values would drop considerably. The distress experienced by residents of these areas, sometimes for long periods, cannot be described adequately.

Using coal-mine refuse to backfill voids provides an opportunity to rid mining areas of unsightly surface waste piles occupying space that could be used for more productive purposes. In residential areas, these mine-refuse banks have an adverse effect on property values; many of them have no vegetative cover, so fine material is carried away by wind and creates a nuisance to nearby residents.

In addition to the unattractive appearance of these banks, there are disadvantages associated with pollution and public health and safety. Coal-refuse banks generally consist of rock, slate, "bone" (a high-ash medium-carbon-content material), and a small amount of coal. The size of the refuse material in the anthracite region generally ranges from more than 0.3 m (1 ft) long to silt size; much of the material is in a size range of 4.8 to 82.6 mm (0.1875 to 3.25 in.). Water from rain and snow percolates through these materials and becomes acidic in the presence of oxidizing pyritic materials. This results in acid drainage, which may appear as seepage that reaches storm drains as well as nearby ponds, lakes, and streams.

Another environmental aspect is the possibility of fire in a coalbed or in a deposit of coal-mine refuse on the surface. Once started, a fire emits noxious fumes and particulate matter, and it often develops burning pockets and cavities into which people and animals can fall. Surface coal-waste fires can ignite fires in coal seams where burning material lies against an exposed coalbed or above a mined bed that is thinly covered. Underground, the limited air supply in abandoned mines considerably reduces the possibilities for combustion of coal-refuse material. Spontaneous ignition of coalbeds is unlikely in such mines; spontaneous ignition in coal mines is more likely to occur during active mining when the mines are ventilated. Oxidation of coal faces is most rapid when the coal is first exposed by mining; after a period of exposure, a surface film of coal becomes oxidized, protecting the coal from further heating. Another factor reducing the likelihood of combustion is the limited circulation of air in abandoned mines; circulation is inhibited even further when voids are backfilled over a wide area. For spontaneous combustion to take place, the supply of air must be sufficient for oxidation to occur but not sufficient to dissipate the heat generated by the oxidation.

In some localities, it may be desirable to use crushed and burned coal refuse (red dog) for mine backfilling, provided that sufficient quantities are available. Studies of red-dog samples taken from burned refuse banks in the anthracite region of Pennsylvania have been found to contain the following average percentage contents by weight as received in the laboratory: 7.4% moisture;



No Scale

Fig. 11. Examples of subsidence from underground mining; top view shows strata failure at shallow depths causing surface potholes; bottom view shows general subsidence caused by collapsing pillars at greater depths.

1.4% volatile matter; 0% fixed carbon; and 91.2% ash. Because all of the acid-forming pyritic compounds have been burned out of this material, and the remaining minerals have been oxidized, it is believed that this material would help make mine water more alkaline. Advantages of using refuse that has been burned must be weighed against its abrasiveness, which tends to increase the wear on crushing equipment and pipelines and inhibits slurry dispersal in underground workings.

Some of the coarse tailings deposits from various mineral-processing operations also could be used for backfilling, provided that they meet the proper size requirements and are not composed of materials that have an adverse chemical effect on water quality. Whether or not certain materials could be used would depend upon the circumstances of the particular subsidence-control project under consideration.

The use of fly ash for backfilling could prove to be another invaluable way to alleviate a waste-disposal problem, while providing surface support. Fly ash consists of tiny glassy spheres that are composed mainly of silicates of aluminum, magnesium, calcium, and iron. Fly ash injected into a void above the drainage level would absorb water and harden, providing support to mine pillars and roof strata. In addition, the soluble materials in fly ash usually are alkaline, having a neutralizing effect on acidic mine drainage when injected into mine workings.

Where a suitable waste product is not available for use as backfill material, it may be necessary to obtain fill from nearby sand and gravel deposits. These may be found as river-terrace, floodplain, alluvial-fan, talus, windblown, residual, or glacial deposits. They all are rather close to the surface and are extracted easily. Deposits of sand and gravel have proven very valuable when located close to the growing area of cities, whereas deposits located at distances greater than 32 km (20 miles) have not been readily considered for construction purposes due to the high cost of transportation.

Under some circumstances, it may not be desirable to use sand or gravel for subsidence-control projects because these materials have many competing engineering, commercial, and industrial uses, as well as environmental effects related to their extraction. Since subsidence-control projects generally are conducted only in built-up urban or suburban areas, prior use and depletion could make it even more difficult to obtain sand and gravel for mine backfilling. In some areas, these materials are not available for mine backfilling, because they never existed in an adequate supply, or they have become unavailable as a result of building structures over the deposits. Whether or not certain remaining sand and gravel deposits should be used for a specific subsidence-control project would depend upon environmental factors and the social or economic importance of the area to be protected in relation to the competing demands for the sand and gravel in that general locality.

In choosing a source deposit for mine backfilling, it generally is believed that the most desirable materials for this purpose should be well-graded (containing a wide range of sizes) and either noncohesive or only slightly cohesive. In such materials, the particles all tend to merge compactly, with smaller particles filling the space between larger ones. This gives the fill a higher density and greater bearing strength. Cohesive materials with clay content over 12% generally are considered undesirable for backfill, as they tend to be more plastic when moist, and they deform easily under static loads.

Floodplain and river-bottom deposits usually contain high proportions of silt and sand, with many variable and heterogeneous lateral and vertical gradations. Locally, some deposits could contain a higher percentage of clay than is desirable for mine backfill, so care would have to be exercised in selecting the portions of the deposit to be used. Terraced glacial-valley deposits and glacial-outwash plains usually are favorable sites for sand and gravel pits; the overburden is thin, and the deposits are extensive. The height of these deposits above the valley floor determines whether a pit is wet or dry.

Two other types of glacial sand and gravel deposits are kames and eskers. Kames are mounds or hummocks composed of poorly sorted water-laid materials. Because the materials tend to be too large or too small for commercial purposes, kame deposits often are not considered for commercial production. Furthermore, these deposits tend to be very limited in areal extent. Esker deposits generally are considered to be the fillings of old stream channels that ran over, under, or through Pleistocene glaciers. Today, they form sinuous surficial ridges of sorted and stratified sand and gravel. These deposits, though containing good gradations of construction material, may be limited because they are normally segmented rather than continuous. Even where they are continuous, development of a commercial pit is likely to involve either acquisition of rights from several property owners or buying large tracts of land. The limited value of kames and eskers as commercial deposits, makes then good potential sources of mine fill in northern mining areas.

In the 1970 mine backfilling demonstration project at Rock Springs, WY, wind-blown sand was used with considerable success in the first application of the pumped-slurry process. Sand was obtained easily from nearby dunes, and only screening was necessary to attain the desired material size of minus 6.35 mm (0.25 in.).

Only in the absence of suitable waste deposits such as coarse mineral tailings, coal refuse, or fly ash would the naturally occurring deposits be used for mine backfill. Methods of reclaiming such areas are described in the section on Protective Measures herein.

Environmental Impact of Subsidence-Control Projects

Stabilization of Ground Surface: Controlling subsidence by backfilling has had a significant long-term beneficial impact upon the local environments of former mining areas. By lending lateral support to remaining pillars and vertical support to overlying rock strata, the threat of subsidence in these areas has been greatly reduced, if not entirely eliminated. Thus, local land-use patterns have been preserved, and the social and economic well-being of local residents has been enhanced.

Elimination of Mine-Waste Piles: In regions where it is both desirable and necessary to use mine refuse as backfill material, unsightly waste piles are partially or entirely eliminated. Removal of refuse banks from exposure to air necessarily reduces opportunities for bank fires to develop by spontaneous combustion or accidental ignition. Once ignited, waste-bank fires may burn for decades, and extinguishment or control of the fires is costly (Dierks, Whaite, and Harvey, 1971). Formerly the atmosphere in Scranton, PA, and other coal-mining areas near burning refuse piles was affected by yellowish smoke, particulate matter, and poisonous and noxious gases that threatened health, damaged vegetation, and caused deterioration of structures (McNay, 1971). The most toxic gases produced by burning mine refuse are carbon monoxide, carbon dioxide, hydrogen sulfide, sulfur dioxide, and ammonia. Other undesirable products are sulfur trioxide, sulfuric acid, and the oxides of nitrogen. The removal of waste banks also reduces the sources of acid drainage and sediment that may enter nearby streams and ponds.

Restoration of Community Confidence: In addition to guarding against definite future subsidence damage, subsidence control removes the prolonged threat of unpredictable damage that pervades subsidence-prone neighborhoods. Some residents are fearful that their homes may become damaged. Possible disruption of public utilities is a constant source of anxiety, particularly the danger of explosion from broken gas lines. Property values are adversely affected by the uncertainty, and neighborhood appearances decline because maintenance and investment for improvements tend to be postponed. Community confidence is restored when the subsurface mine workings have been backfilled.

Disturbance at Source Areas for Fill Material: Where source areas for fill material are piles of accumulated mine waste, the disturbance caused by excavation of the material for subsidence-control projects is minimal. If only part of the pile is used, the final waste pile would be smaller, and the configuration may be different. If the entire waste pile is removed, a distinct environmental benefit is achieved when the site is cleared and becomes available for suitable surface development.

For most subsidence-control projects, preparation of mine waste has included crushing. Crushing plants, located at waste-bank sites, are sources of noise and dust during their operation. Where crushing plants are located in residential districts, dust and noise control measures are required. Several recent projects are experimenting with the use of material having a wider range of particle sizes, which is obtained from waste banks by screening without crushing. In preparing for these projects, even the temporary inconvenience of a crushing plant is eliminated.

Where borrow pits are developed in naturally occurring fill material, such as the widespread windblown sand deposits around Rock Springs, the disturbance caused by excavation and removal of material is more obvious. The topsoil is stripped and stockpiled; a pit is developed below the natural drainage level; debris and oversize material are left in the pit. Further disturbance may be caused by developing temporary haulage roads. Rehabilitation measures are described in the section on Protective Measures herein.

Potential Impact on Surface Water: Areas to be considered in connection with potential surface-water effects of subsidence-control projects are source areas that are excavated to produce fill materials, the vicinity of the plant where slurry is mixed and injected, and possible surface outlets of waters draining from the backfilled mine workings.

The development of borrow pits in naturally occurring deposits of material may alter local drainage patterns. However, such disturbance affects a very limited area and is of short duration, because the pit is filled in and the area revegetated at the close of the project. Where coal-refuse banks are used, there would be some erosion and runoff from the excavated sites, which could have an adverse effect on nearby stream quality. Siltation would be minimized by the use of intercepting ditches and appropriate grading procedures.

Oxidation of freshly exposed pyrite associated with the coal waste may temporarily increase the rate of acid formation from fill-source areas. Pyrite oxidizes in the presence of air and moisture to form sulfates that may hydrolize in solution to form sulfuric acid and iron hydroxide. During periods of precipitation, acidity from the newly exposed pyrite may affect surface runoff into nearby streams or ponds. However, in coal-mining regions, the effect of any increased acid drainage from a recently disturbed source is relatively small when compared with the acidity already being contributed by that source and when compared to the total acid drainage entering streams from other refuse banks and from mined and unmined coal seams in the area. Furthermore, to the extent that the material from a waste pile can be injected underground, a source of acid drainage can be eliminated. Notwithstanding any temporary increase in the rate of acid formation caused by the initial disturbance of the waste banks, returning material to the mine voids from which it originated reduces the long-term total amount and rate of acid formation from accumulations of the waste. The precautions taken to prevent the escape of water or slurry from the plant site or from pipelines are described in the section on Protective Measures herein.

The possibility that pollutants may be contributed to surface waters by discharges from underground workings that have been backfilled is considered slight. The backfill materials that have been used are indigenous waste rock and inert sand. For past and present projects, the volume of emplaced backfill constitutes a very small part of the total volume of mine voids, and the quantity of water displaced by the placement of backfill would increase the discharge by an amount well within the range of annual fluctuation. During injection there might be a small daily increment, but, after injection has ceased, the discharge would return to normal. Moreover, in the backfilled portions of a mine, contact with air essentially is eliminated. If the impact is only minor or temporary, the overall benefits of the project would outweigh any adverse effects; if a major impact on a surface stream or lake is foreseen, the effluent should be treated as necessary to minimize those effects.

Potential Impact on Ground Water: Two potential impacts on ground-water quality must be considered in relation to backfilling: the effect on mine water, which may be acidic or may contain other dissolved mineral matter; and the effect on developed or potential ground-water resources in the vicinity.

Most pumped-slurry backfilling in which USBM has participated has been in flooded mine workings. When the fill is submerged in the mine-water pool, it is not exposed to the oxygen necessary for acid formation. Even where the fill is placed in voids above the drainage level, it tends to contribute less acid drainage than it does on the surface, because the material is no longer exposed to rainfall. The deposited crushed refuse compacts as the water drains out of the fill, and the resulting random particle size and increased packing density lower the permeability of the material. Thus, areas of backfill reduce the underground circulation of air and water, slowing the rate of acid formation. Capillarity in the refuse maintains a partially saturated condition that limits the amount of pyrite subject to oxidation. Furthermore, ground water entering the mine normally is alkaline, partially neutralizes the acidity, and further reduces pyrite oxidation.

To provide surface support, it is economically feasible to backfill mine voids only under areas of relatively high social importance or economic value. Consequently, backfilling projects usually protect a few city blocks in areas where there are numerous refuse piles and many square miles of underground mine workings that may contain from 30% to 45% of the original coal reserves. Any temporary additional acid drainage resulting from the backfilling would be insignificant when compared to the total acid drainage that already occurs as a result of oxidation and water seepage over many square kilometers of exposed coal faces and mine pillars. This can be verified from samples taken from mine-water pools where extensive subsidence control has been conducted.

For example, the Lackawanna mine-water pool inundates the lower-lying mine workings under Scranton, PA, and all of the general vicinity from Old Forge southwest of the city to Blakely on the northeast; this is a total distance of about 16 km (10 miles). The maximum width of the pool is about 6.4 km (4 miles). The pool discharges to the surface at Old Forge, with a total flow ranging between 2210 and 3470 L/s (35,000 and 55,000 gpm). Over the years the Commonwealth of Pennsylvania has sampled the mine-water discharge at Old Forge to determine its quality. During 1962 the pH values for the water generally remained lower than 4.0. However, the mine-water quality has improved since then, even though extensive backfilling with crushed coal refuse has been carried out under Scranton. From 1965 to 1975, USBM and the Commonwealth of Pennsylvania jointly conducted nine subsidence control projects in Scranton, involving the placement of over 2 300 000 m^3 (3,000,000 cu yd) of crushed coal mine refuse to support 2 400 000 m^2 (600 acres) of surface area, constituting about 3% of the total mine-pool area. According to the records of the Bureau of Water Quality Management, Pennsylvania Department of Environmental Resources, pH readings for the Old Forge mine-water outfall during 1974 fluctuated between 5.3 and 5.7. Although a change in pH is not an indication of a change in total acidity, there is no evidence that the acidity was increased by backfilling.

Pollution of local ground-water resources has not been observed at sites of past or current subsidence-control projects. However, in some areas that may be considered for subsidence control, it is possible that the water draining from the injected slurry will mingle with, and degrade the quality of, potable water sources. Investigations of proposed backfilling sites include the determination of whether or not aquifers are present. Where aquifers are present, geohydrologic conditions at the site should be analyzed to determine the potential impact of slurry water upon the quality of water in the aquifer. For each project, the plans are tailored to fit local site conditions; where the possibility of aquifer pollution is foreseen, plans and specifications should be modified to protect the aquifer. Water quality and water levels in the aquifer should be monitored both during and after slurry injection to predict and assess the long-term effects.

Pollution of an aquifer requires both a physical mingling of backfill water with aquifer water and chemical changes that degrade the water quality in the aquifer. Physical access to the aquifer depends upon changes in the relative levels of head in the aquifer and in the backfilled mine workings over a long period.

Any accurate prediction of changes in water quality requires qualitative and quantitative analyses of the aquifer water, of the slurry water and solids, and of the materials through which the water moves. Thus far, only inert or indigenous materials have been used for backfill.

Projects involving potential pollution problems must consider state programs designed to implement the Safe Drinking Water Act (Public Law 93–523). Such programs include regulations covering underground emplacement of waste materials. For example, Illinois requires a permit from the Pollution Control Board of the Illinois Environmental Protection Agency prior to recovering materials from mine-refuse piles or slurry ponds, giving the state agency the opportunities for case-by-case reviews.

In addition to water quality investigations, possible impacts on storage, rate of flow, and direction of flow of ground water in the vicinity of the backfilling operation should be considered.

Other Minor Impacts: Other potential adverse environmental impacts, such as noise, dust, and spills are minor, temporary, and confined primarily to backfilling sites (flushing hoppers or slurry blending and pumping equipment), excavation sites, and preparation sites. Local residents are inconvenienced temporarily by some dust, noise, and partial blockage of streets by drilling and trenching equipment, as well as by trucks, open trenches, excavated material, and paving equipment. It is unlikely, however, that the occupants of any single home or business establishment would be inconvenienced by the presence of flushing hoppers, trucks, and workers for more than 12 weeks, and the time might be as short as 2 weeks. Access boreholes for controlled backfilling operations would remain active for the duration of the project, but such openings would be located so as to minimize disturbance to the community and to assure safety for the workmen. In pumped-slurry projects, where the slurry plant is adjacent to the community being protected and distribution pipelines are buried under the streets, the filling process is noiseless and may be conducted both day and night.

Normally, there are some accidental material spills; however, contract provisions require that they be cleaned up immediately, and that reasonable preventive measures be taken. At fill sources, some noise, dust, and mud are associated with excavation, preparation, and handling. There is minor noise disturbance to residential areas as trucks haul fill from excavation sites to backfilling areas.

Every precaution is taken to prevent any outflow of water on the surface. However, a water or slurry pipeline could rupture due to a structural weakness in the line or an accident; the contractor's responsibilities in a case of pipeline rupture are described in the section on Protective Measures herein. It is possible that excess water could form puddles and become a nuisance to nearby areas, and contractors are required to drain any resulting impoundments and to clean up any spilled material.

With the pumped-slurry process, pressure in shallow underground voids could become great enough to force slurry to the surface through fissures in the overburden, causing hazardous conditions or damage to property. Thus, the pressure in the injection system should be adjusted to allow for possible weaknesses in the overburden. Also, pressure in a monitoring borehole could become great enough to cause a slurry "geyser," so care must be exercised in removing borehole caps. Employees should be informed of proper procedures for removing borehole caps to avoid accidents or injuries.

During any project involving heavy equipment, conditions exist that are potentially hazardous to public and employee safety, including open excavations and falling equipment. During controlled backfilling other potential hazards are peculiar to underground work, including the presence of gas and the possibility of rock falls.

For most subsidence control projects, the most readily available sources of water have been the mine-water pools that inundate abandoned mine workings. In many regions, the pools are of such magnitude that any changes in mine-water levels caused by pumping water to the surface are not significant. Any water withdrawn is replaced from all directions in the vast mine-water pools (containing billions of gallons of water) and by water returning to the mines in the injected slurry. Therefore, any effect on the mine-water level is temporary and is probably less than that normally occurring during seasonal fluctuations.

Protective Measures for Subsidence-Control Projects

Operating procedures include a variety of measures for protecting the neighborhoods in which the projects are located, as well as the workmen engaged in the operations. These protective measures are based on (1) an analysis of problems specific to the project site, as identified by detailed surface and subsurface site investigations; (2) experience gained by USBM engineers and others in previous projects; and (3) pertinent federal, state, and local statutes and regulations.

Whenever a subsidence-control project is considered initially, existing mine maps are obtained. USBM engineers examine these maps to determine the relationship of mining patterns and structures to each other (as in the case of multiple-seam coal mining) and to surface structures. Information is compiled on the dates of mining and abandonment, the extraction ratios, and the dewatering history. Local mining officials, miners, and others are consulted for additional information on mine conditions that are not shown on the mine maps. Such information may include the strength characteristics of the immediate roof and floor strata, as well as the extent of backfilling performed during active mining. Geologic and hydrologic data are analyzed to determine the cause of actual or potential subsidence and possible impacts on groundwater resources. If safely possible, the voids are entered for an inspection of the mine pillars and structures. A report on mine conditions then is prepared, including recommendations for dealing with the actual or potential subsidence problems. This preliminary examination and analysis of site conditions is requisite for the formulation of protective measures included in USBM's approach to subsidence problems.

Most protective measures included in subsidence control projects are detailed in contract specifications based on USBM experience and on state and federal Dept. of Labors' Safety and Health Regulations for Construction, promulgated under Section 107 of the Contract Work Hours and Safety Standards Act, as amended, and commonly known as the Construction Safety Act. These regulations are published as "Safety and Health Regulations for Construction" by the Dept. of Labor in the Federal Register of April 17, 1971, Vol. 36, No. 75, Part II, page 7340.

All activities must be conducted in a safe and workmanlike manner. Contractors are required to prevent any employee, whether underground or on the surface, from working under conditions that are unsanitary, hazardous, or dangerous, as determined by regulations promulgated by the Secretary of Labor. Contractors must comply with all applicable provisions of state and municipal safety, health, and sanitation codes. Employees must be thoroughly instructed in all aspects of equipment operation and safety procedures, and regular safety meetings must be held. Supervisors must make regular inspections of surface work areas to verify that all equipment is positioned properly and is in good working order, and any underground utilities must be located and protected from drilling and trenching operations.

For projects in which workmen or observers are to have access to underground mine workings, underground operations are carried out in conformance with existing state mining regulations, including supervision by a certified mine foreman assisted by the required number of authorized mining personnel. Arrangements for ingress to and egress from mine workings, under regular and emergency conditions, follow state mine regulations.

Contractors must carry comprehensive public liability insurance, covering potential personal injury, death, or property damage that might occur as a result of project activities. Unauthorized personnel are barred from work areas and backfilling facilities by barricades, which must be locked at all times when work is not in progress. At night, barricades and surface work areas must be illuminated for the safety of vehicular and pedestrian traffic, and warning flashers and safety fences must be used at open excavations and equipment sites. When necessary, traffic in work areas is rerouted with the cooperation of local officials.

During operations, contractors are required to minimize noise and to keep the streets free of debris and dirt. Trucks carrying material to backfilling sites must be loaded in accordance with the weight capacities indicated by their licenses and in a manner that prevents fill material from spilling onto public roadways, streets, or private property. Depending upon weather conditions, dust discharge from moving trucks is suppressed either by use of suitable coverings or by spraying each load with water. To further reduce dust discharge and spillage, the tires and bodies of the trucks are cleared of loose material at the loading area. Drivers are required to follow routes selected to minimize disturbance to residential areas.

Drilling, crushing, and screening equipment must be equipped with dust-control systems, and excavation areas may be wetted as weather conditions necessitate. Any water used in the crushing or screening process is either recirculated or allowed to filter into the ground at the excavation site. Spilled material must be cleaned up and is either injected into the mines or returned to the fill excavation site. Engines on drilling and slurry pumping equipment must be muffled for operation in residential and business areas. Deep-well or submersible pumps for supplying water are usually electrically powered and served by the city electric utility system. Whenever substation transformers are necessary to convert power for these pumps, they must be installed, barricaded, and operated in accordance with local safety ordinances.

Water used for slurrying and injecting backfill, which may be acidic, must be prevented from escaping into surrounding areas or streams. The water is contained within a closed pipeline system throughout the operation, from the well to the mixing tank and from the mixing tank through the slurry pump, distribution pipelines, and cased injection boreholes into the mine workings. Enclosure of the mixing plant area is required to confine water to the plant area. Where advisable, sumps are provided, usually draining through a borehole into the underlying mine pool. Should a water or slurry pipeline rupture, pumping must be stopped immediately and may not be resumed until the break is repaired. If the break is outside of the plant enclosures, most of the spillage is carried away by city storm-sewer systems, and the remaining water filters into the ground.

During most projects noise, dust, and traffic disruptions would be of short duration and would be confined to only two or three locations at any one time during the project. Upon completion of backfilling, all equipment would be removed, and the boreholes would be sealed. Streets and all work areas would be returned to prior or better condition.

Control of erosion and off-site environmental damage due to excavation is required. Where piles of mine waste have been only partly removed and there are no plans for continued excavation of fill material, the remaining portion of the waste pile is graded. Where sources of borrow material have been excavated for fill, rehabilitation is required. The pit floor is leveled to coincide with the existing watershed slope, and screening rejects are scattered over the pit floor. Topsoil, which had been removed and stockpiled, is replaced in such a way that pit slopes do not exceed a 3:1 ratio. Haulage roads are seeded with species and concentrations of vegetation suitable to the climatic and terrain conditions; seeding must continue until a satisfactory stand has been established and approved. It is USBM's intention to prevent any backfill excavation site from becoming a major environmental nuisance or a hazard to the public health and safety.

During all operations, contractors are required to comply with such federal and state provisions for protection of air and water quality as are in effect at the time of the contract award.

Unavoidable Adverse Effects: Based on previous USBM experience in subsidence-control projects, it can be stated with reasonable certainty that projects properly designed to fit the specific sites should not induce either ground movement or subsidence that otherwise would not have occurred had the voids not been backfilled. Noise and physical disruption resulting from drilling and other project operations are unavoidable. However, such disturbances generally are of limited duration at any given location. Crushing and screening operations often result in some noise and dust spillage, and these disturbances are minimized to the greatest extent practicable by measures such as muffling engines and wetting excavation sites and stockpiled material. Crushing operations and their attendant noise and dust are being reduced by the experimental use of waste and material that are processed by screening without crushing. Contractors are required to clean up any accidental spills immediately following their occurrence.

Following the initial disturbance of mine-refuse banks to obtain fill material, the rate of acid formation may increase temporarily from these sources. However, it is likely that the additional acid would be insignificant in relation to the acid already emanating from these banks and nearby coal seams and mines. The long-term effect of using mine-refuse material to backfill voids will reduce or eliminate the banks as eyesores, sources of air and water pollution, and public safety hazards. Moreover, the cleared land is again available to the community for useful purposes.

Temporary loss of vegetation and increased erosion would occur in any areas disturbed to obtain fill material. This impact would be minimized, although not entirely eliminated, through use of intercepting ditches and appropriate grading and revegetation practices.

Any subsidence control project would use equipment that would be an attraction to children. Such equipment, however, would pose no more hazard to children than would any conventional equipment used for light construction.

The most disruptive period would be the earliest weeks, during which boreholes are drilled and cased, pipelines are laid, and equipment is moved into position. At later stages, the work would be less disruptive; there would be less movement of equipment, and local residents would become more accustomed to the operations in their neighborhoods. Work at any one borehole probably would range from 2 to 12 weeks; the injection phase of pumped-slurry projects in urban communities where distribution pipelines are buried proceeds for months, with little sign of activity other than the checking of pressure gages and monitor holes.

Effects on Availability of Resources

In areas where surface deposits of mine refuse are plentiful, backfilling projects can remove some of these deposits and clear the areas for other purposes. The refuse could otherwise be burned to form cinder material for use as low-grade aggregate, fill, secondary road material, or lightweight concrete and cinder block. However, loss of this material would have insignificant effect on these purposes, because there usually are many additional waste banks throughout mining regions. In many areas, sand and gravel deposits are available and may be better suited for construction purposes.

Coal pillars remaining under project areas normally contain from 30% to 45% of the original reserves. As long as the value of the existing land use remains greater than the cost of reclaiming the remaining reserves, the pillars would not be mined. However, it is conceivable that, due to strategic or economic necessity, these reserves could become sufficiently valuable to warrant their subsequent recovery. Under such circumstances, the added ground stability provided by backfilling may permit selective pillar extraction.

ALTERNATIVES TO FEDERAL BACKFILLING FOR SUBSIDENCE CONTROL

One alternative is to take no action. In such a case, subsidence may or may not occur in a given area.

Although the local adverse impacts and short-term disruptions described might not take place, it is very unlikely that conditions in any critical area would remain stable indefinitely. An analysis of mine structures, pillar distributions, depths of mining, and percentages of extracted reserves would indicate whether eventual surface subsidence appears to be inevitable. Subsidence occurring in a critical area could be heavily damaging to property and could create conditions potentially injurious or fatal to local residents.

Another alternative would remove all buildings and public facilities from threatened areas. This would involve the extremely high cost of relocating densely populated neighborhoods. Such an alternative would totally disrupt the long-established land-use and social patterns of such areas, and would cause displaced inhabitants immeasurable grief and inconvenience. Furthermore, such an undertaking would be far more costly to taxpayers than the implementation of one of the mine backfilling techniques described in this chapter.

The foundations of existing buildings could be reconstructed in such a way that subsidence damage would be minimized. This measure also would be more costly than mine backfilling; it would involve direct disturbance of each business establishment or home within a project area, and it would do nothing to support existing streets and public utilities. It would be beneficial, however, to construct foundations of new buildings in such a way as to minimize subsidence damage where construction is necessary in potential problem areas.

It is possible to install artificial supports in underground mines. This has been done where it has been necessary to provide support for large-scale construction. It usually involves injection of a concrete or grout mixture through boreholes drilled from the surface. Performing this operation over a wide area is extremely costly, providing support much stronger than normally would be necessary to support the mine overburden. Constructing various types of supports underground would require employment of large groups of workmen, and it is doubtful that the supports would be as effective a surface protection as mine backfilling. Installation of supports would involve a longer completion time and greater hazards to underground workers than a controlled-backfilling operation.

By delaying action until subsidence actually occurs, thousands of dollars worth of property would be damaged before remedial actions could be implemented, negating many of the benefits that could have been gained by preventive stabilization. Furthermore, once caving in the mines and subsidence at the surface begins, it is too hazardous to send workmen underground to carry out a controlled-backfilling operation. Any measures to prevent further subsidence then would be limited to blind-backfilling techniques, and the probability of success would be diminished by the uncertainty of actual subsurface conditions.

Continued repairs could be made to surface structures as subsidence damage occurs. This practice often has been followed in critical areas overlying mines. With this alternative residents of threatened areas have experienced the effects of buckling sidewalks and driveways; "roller coaster" streets; whole buildings or sections of roads or yards dropping out of sight; disrupted utilities and gas explosions resulting from broken lines; buckling floors, cracked walls, and doors that cannot be closed; and thousands of dollars of damage to real property. Local residents have found this alternative extremely inconvenient and dangerous, if not totally untenable.

As a final alternative, other methods may be developed to control subsidence or to reduce the damaging effects. In areas of future mining, future surface development, or both, programs other than backfilling may be developed to reduce the effect of future subsidence. Such programs include developing improved mining techniques to reduce the damaging effects of subsidence, designing buildings and other structures to withstand subsidence movements with minimal damage, and planning surface or subsurface development (or both) for optimum multiple-land use. Such programs offer promising alternatives to backfilling under appropriate conditions, but they cannot benefit areas of existing mine workings that already are sites of urban development.

RESEARCH

Mining technologists need to develop methods and techniques that will enable the prediction and control of the extent and nature of subsidence consonant with the required surface use.

Investigations and studies will be required to determine quantities and rates of vertical and horizontal displacements, tilts, curvatures, and horizontal strains at the surface of undermined areas. Classes of surface use such as agriculture, recreation, transportation, industrial, commercial, residential, or other uses should be studied to ascertain acceptable limits of movement for each class. The amount of anticipated surface movement then could be correlated with those classes of surface use able to accommodate the amount of disruption expected.

A key element in understanding and controlling subsidence is the ability to take measurements with adequate sensitivity and accuracy. During the past 15 years, USBM has made substantial progress in developing instrumentation and techniques for measuring the pressures and strains in rock and in backfill. New thrusts are directed at lowering data acquisition and processing costs and developing new techniques for detecting mine voids, including seismic, electromagnetic, and acoustic systems.

A precise knowledge of the subsidence mechanisms associated with the complete-extraction method of mining is basic to the technology. Full extraction, with total subsidence occurring as soon as possible after extraction, would minimize the adverse impacts of surface subsidence in the shortest period of time. Research will be required to gain a more complete knowledge of ground control to permit sound engineering design based on the variables inherent in the geologic structures and the stress field of a mining system.

Studies and research of options relating to total extraction will include backfilling techniques that consider problems of backfilling mine workings above the water table, the adaptability of various backfilling techniques, the support capacity after fill emplacement, the long-term stability, and the environmental compatibility of fill emplaced below the water table.

Research on suitable construction techniques will be directed toward minimizing the damaging effects of subsidence movements on surface structures. Efforts and attention to preplanning of surface and subsurface development for multiple land use will continue to attract support because of the associated social, economic, and technical potential and acceptability.

REFERENCES AND BIBLIOGRAPHY

Candeub, Fleissig and Associates, 1971, "Demonstration of a Technique for Limiting the Subsidence of Land Over Abandoned Mines," PB 212 708, Sponsored by Urban Renewal Agency, City of Rock Springs, WY, National Technical Information Service, Springfield, VA.

Carlson, E.J., 1975a, "Hydraulic Model Studies for Backfillng Mine Cavities," REC–ERC–73–19, US Bureau of Reclamation, Information Circular 8667, Appendix, US Bureau of Mines.

Carlson, E.J., 1975b, "Hydraulic Model Studies for Backfilling Mine Cavities—Second Series of Tests," REC–ERC–75–3, US Bureau of Reclamation, prepared for US Bureau of Mines.

Dierks, H.A., Whaite, R.H., and Harvey, A.H., 1971, "Three Mine Fire Control Projects in Northeastern Pennsylvania," Information Circular 8524, US Bureau of Mines.

Luckie, P.T., and Spicer, T.S., 1966, "Methods Employed for Underground Stowing (Resume of a Literature Survey)," Spec. Res. Rept. No. SR–55, Pennsylvania State University, Industrial Coal Research Section.

Magnuson, M.O., and Malenka, W.T., 1970, "Utilization of Fly Ash for Remote Filling of Mine Voids," *Proceedings*, Second Ash Utilization Symposium, Information Circular 8488, US Bureau of Mines, March 10–11, pp. 83–96.

McNay, L.M., 1971, "Coal Refuse Fires, An Environmental Hazard," Information Circular 8515, US Bureau of Mines.

Murphy, E.M., Magnuson, M.O., Pete Suder, Jr., and Nagy, J., 1968, "Use of Fly Ash for Remote Filling of Underground Activities and Passageways," Report of Investigation 7214, US Bureau of Mines.

National Academy of Sciences, 1975, "Underground Disposal of Coal Mine Wastes. Report to the National Science Foundation by the Study Committee to Assess the Feasibility of Returning Underground Coal Mine Wastes to the Mined-Out Areas," available from the Printing and Publishing Office, National Academy of Sciences, Washington, DC.

Stewart, J.D., and Heslep, M.E., 1974, "Method of Filling Subterranean Voids With a Particulate Material," US Patent 3,817,039, June 18.

Whaite, R.H., and Allen, A.S., 1975, "Pumped-Slurry Backfilling of Inaccessible Mine Workings for Subsidence Control," Information Circular 8667, US Bureau of Mines.

APPENDIX F

Drill Tech Drilling & Shoring, Inc. Proposal for Mine Filling

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P**ROJECT:** Streets of West Pryor

Date: 10/26/2020

Drill Tech Drilling & Shoring, Inc. (DTDS) is pleased to submit a proposal for placing material inside the abandoned mine in Lee's Summit, MO. Portions of the mine are partially flooded. The estimated volume is 75,000 cubic yards and 65 drill holes. There are ~10 to 15' feet of overburden soils overlying shale and limestone. The depth to the roof of the mine varies but is approximately 75 feet.

SCOPE OF WORK:

We plan to place material inside the mine by drilling holes from the surface into the mine and placing 2" minus material processed and provided to us in the rooms. The work will be done on a checkerboard pattern filling every other room.

Layout- Holes will be staked by others for the location of the drill holes. No surveying is included in the proposal.

Drilling- the holes will be drilled with a rotary drill with auger and down the hole hammer drilling tools Casing will be placed in the overburden and the rock will be drilled open-hole and left uncased. The equipment for this work is a rock drill, drill rod handling equipment, and a high-pressure air compressor. If water is necessary for dust control it will be provided to us in a tank located near the drilling equipment. A separate daily rate is provided for the drilling activity since it is anticipated it will take significantly fewer days than the material placing.

Placing Material- Material will be provided to us crushed and screened to a 2" minus size. Existing material on-site will be processed and used as minefill. The material will be transported to us and placed in stockpiles adjacent to the drill hole so we can load the material into our hopper with an end loader. The proposed technology is to drop the material down a 12 to 15 inch drill hole and scatter the material with a rotary slinger. The slinger will be attached to a drill with a variable speed rotation to create a cone of material that extends to or nearly to the mine roof. It is understood that the material will be placed loosely with no specific compaction effort. Initial efforts will be in an area that is accessible and can be observed at the mine level to determine the proper rotation speed. When the material approaches the mine roof elevation, the slinger will be removed and the mine backfilled to the surface by dropping material down the shaft.

Verification and quality control- The material being used is random and this is not an engineered fill. Nothing is included in the proposal for the support capacity of the material placed. Also, nothing is included to assure complete filling of the void or capping the fill with grout to assure roof contact.

Production- The process proposed is unproven. We have confidence in the performance and have estimated a production rate of 540 cy per day. It is likely we can place more than this per day and there are no performance guarantees.

Test program- We propose using the existing vent hole on the north end of the property as a test hole. If desired additional holes can be drilled to expand the test area by utilizing a local subcontract driller to reduce mobilization costs and provide a quick response time. The bid schedule below is based on the test being performed. The construction of special tooling is included in the test program.

Bid Schedule

Bid price

TASK NO.	DESCRIPTION	QTY	UNITS	UNIT PRICE	AMOUNT
1	Test Hole Equipment and Mob	1	LS	$40,000.00	$ 40,000.00
2	Placing Material	6	days	$ 6,800.00	$ 40,800.00
3	Mobilization	1	Ea	$ 48,000.00	$ 48,000.00
4	Drilling	33	Days	$ 10,225.00	$ 337,425.00
5	Placing Material	134	Days	$ 6,800.00	$ 911,200.00
				BID SUB TOTAL:	**$1,377,425.00**

<u>SCHEDULE:</u>
We propose to progress the work in 15 hole increments or roughly ¼ of the project. The work start and progress in any direction, as long as the work is continuous. 15 holes would include placing ~19,000 cy and take about 35 working days.

<u>EXCLUSIONS:</u>

- Plans, permits, property surveys, inspections, inspectors, monitoring, noise monitoring, and all environmental monitoring, 3rd-party testing, and reporting to authorities. DTDS will be responsible for its business licenses, contractor's license, and qualifications to do business or similar approvals that are not specific to the project.
- All minefill materials.
- Survey and layout.
- Site clearing, grubbing, grading, and compaction.
- Potholing, location, relocation, support, or removal of utilities, underground & overhead as required by DTDS.
- Continuous removal of drill spoils/grout and shotcrete rebound as required by DTDS.
- Removal of or drilling through existing man-made objects.
- Hoisting or lifting of materials or equipment other than in this scope of work.
- Repair and/or replacement of pavements and existing site improvements.
- Development, restoration of laydown areas, fences, gates, ramps.
- Dewatering, waterproofing, site water handling, collection, pumping whether contaminated or clean, if required.
- Excavation and site prep for drilling and placing.
- Erosion control, dust control, and SWPPP establishment and maintenance.
- Removal, handling, covering, storage, and disposal of hazardous materials.
- The cost for additional insurance requirements and changes to wording, over & above our industry standards listed.
- Any work not specifically included in the "Scope of Work" above.
- All liquidated and consequential damages.
- The provision of a performance and payment bond is available for an additional price.

PROJECT SPECIAL CONDITIONS:

Contractor shall provide at no cost to DRILL TECH DRILLING & SHORING, INC:

- Safe work area as determined by DTDS and firm, level all-weather access for material deliveries, storage, drilling equipment, and mixer trucks moving under its own power.
- Provide 2500 sf min. level working working area at each drill hole location
- Traffic control and lane closures required to complete our work and mobilize equipment onsite.
- A continuous supply of water is placed in holding tanks by others at each injection if needed for dust control.
- Secure and adequate staging and storage areas for equipment and materials.
- Cost of permits and bonds if required.
- *DTDS is a Specialty Contractor.* DTDS must be notified in advance, before the scheduling of any its work, of any project requirements regarding labor compliance and/or local hiring goals. When properly notified in advance of such goals and requirements, DTDS shall make every good faith effort to comply with such goals. Notwithstanding, participation by DTDS in such labor compliance goals shall be limited to DTDS's good faith efforts in selecting its own specialty team.
- Upon issuance of subcontract a schedule of work shall be provided to DTDS and mutually agreed upon.
- This proposal to be part of the subcontract and this proposal's conditions preside over conflicting subcontract conditions.
- Winter work- nothing has been included for winter work or associated delays.

ABOVE PRICES ARE BASED ON:

- A mutually agreed to schedule, scheduling our work so it can be completed, working 10-hour daytime shifts, 5 days per week excluding Saturdays, Sundays, and Holidays. If weekend or night work required, O.T., premium, plant openings, lighting costs to be paid by the general contractor.

PAYMENT TERMS:

- Progress Payment: Payments in accordance with a mutually agreed schedule of values due and payable thirty (30) days after receipt of our monthly billing or ten (10) days after receipt of payment from the owner, whichever occurs first.
- Mobilization: Mobilization is due and payable ten (10) days after mobilization of the drill rig on site.
- Retention: Retention of a maximum five percent rate is due and payable in full within forty-five (45) days after substantial completion of our work. Retention to be no greater than a specified amount between the owner and the general contractor.
- Materials on Hand: All on-site production materials delivered to the site to be paid in full thirty (30) days after receipt of our invoice, or ten (10) days after receipt of payment from the owner, whichever comes first.
- DTDS does not and will not waive or release any rights or remedies provided under any applicable prompt payment statute and nothing herein shall be construed as a waiver or release by DTDS of any such rights or remedies.
- DTDS shall be promptly reimbursed and paid by the Contractor for any cost which DTDS may be required to incur for its participation in, subscription to, and/or otherwise mandated use of any administrative programs, systems, or software for any aspect of project management which are or might be mandated for use by DTDS for this project, plus 15% of such costs.
- DTDS expressly does not waive any statutory right to collect prompt payment penalties and nothing herein shall constitute or be construed to affect a waiver by DTDS of any statutory right to collect prompt payment penalties.

- *Other Payment Conditions:* DTDS will submit a conditional waiver and release executed by it as a condition to each payment and it will submit an unconditional waiver and release promptly after payment funds have been posted, without restriction, to its account. It shall not be a condition of any payment to DTDS that a waiver and release from any third party in a contract with DTDS (as a sub-tier subcontractor or as a material supplier) be submitted. In the event of any third party claim against the owner or its surety, the general contractor or its surety, or the property to which the work of improvement relates, for alleged nonpayment by DTDS concerning the subject work of improvement, DTDS, at its sole option, shall indemnify the claim recipient(s) concerning the claim or bond around the claim or do both. DTDS's duty herein shall only arise provided DTDS has been promptly notified of the claim when it initially arose. "Claim" means a mechanic's lien as evidenced by a recorded mechanic's lien, a stop payment notice served as required by law, a bond claim received by a surety, a cause of action or a claim for relief in a filed lawsuit, and a claim made in papers which initiate mediation or arbitration.

DAMAGES, DELAYS, BACK-CHARGES, and EXTRA WORK:

- DTDS shall not be held responsible for any direct, indirect, and consequential damages.
- DTDS must be notified in writing within 24 hours of the cause of any potential claim for damages and be given the opportunity and reasonable time, to resolve the damages with our forces.
- No damages may be assessed without DTDS’s specific agreement in writing, signed by our authorized company representative. Any damages shall be proportional to DTDS’s share in the total fault and the total assessed will be limited, not to exceed our subcontract value less the materials purchased.
- No back-charges of any kind to DTDS will be accepted unless received daily and agreed to and signed for by our authorized representative.
- Unless specified otherwise, extra items of work not included in our scope of work, but performed by DTDS, or delays resulting from interference or non-performance of others, shall be invoiced at cost plus 20%.
- If the owner/contractor asks DTDS to prepare submissions, secure materials, purchase or modify equipment and accessories or perform contract-related work before the contract document being finalized, all costs incurred will be paid at cost plus 20%.

INSURANCE, INDEMNIFICATION:

A) DTDS shall maintain in place during the performance of its work hereunder, worker's compensation coverage as required by law, bodily injury and property damage liability insurance (XCU exclusion deleted), with a combined single limit of $1 million. "Additional Insured" coverage thereunder is limited to DTDS's proportionate share of the total fault causing the loss or damage on which such claim is based. Coverage afforded shall be Type III. For additional insurance requirements beyond that mentioned above, and total additional cost and premiums required in the DRAFT subcontract provided at bid time have been included in the bid price. Any costs due to changes in insurance requirements after bid time shall be solely born by the Prime Contractor and/or Owner.

B) DTDS shall indemnify Prime Contractor (if any) and Owner concerning any claim which arises directly from the negligence, if any, of DTDS in performance of its work hereunder, but the extent of such indemnity shall be only in proportion to DTDS's share of the total fault causing the loss or damage on which such claim is based. DTDS shall have no other responsibility to defend, indemnify, or provide insurance for Prime Contractor (if any), Owner, or others concerning claims arising from or related to DTDS’s work hereunder. Variations to wording, additional requirements and waiver of subrogation can be requested at the owner’s additional cost and subject to our carrier’s availability.

C) **ACCEPTANCE:**

- This bid proposal subject to acceptance within ninety (180) days from the bid date.
- Acceptance of a bid proposal shall:
  - Constitute a contract between both parties and this bid proposal shall be an integral part of any contract agreement and shall supersede and control any conflicting or ambiguous language in the contract documents.
  - Acknowledge acceptance by the General Contractor that DTDS will be held harmless for all costs, direct and indirect damages caused by or resulting from drilling within the contract's specified drilling tolerances at the locations specified in the contract plans and approved engineered drawings.
  - DTDS shall not be required to mobilize, order materials, or work before receiving a fully executed contract with terms agreeable to DTDS.

CONTRACTOR: DRILL TECH DRILLING & SHORING, INC.

Drill Tech Drilling & Shoring, Inc. is an Equal Opportunity Employer

BY: ___________________________

BY: Patrick Carr, P.E.
Patrick.Carr@DrillTechDrilling.com
Phone 913-378-2580
Estimate number E - 65820

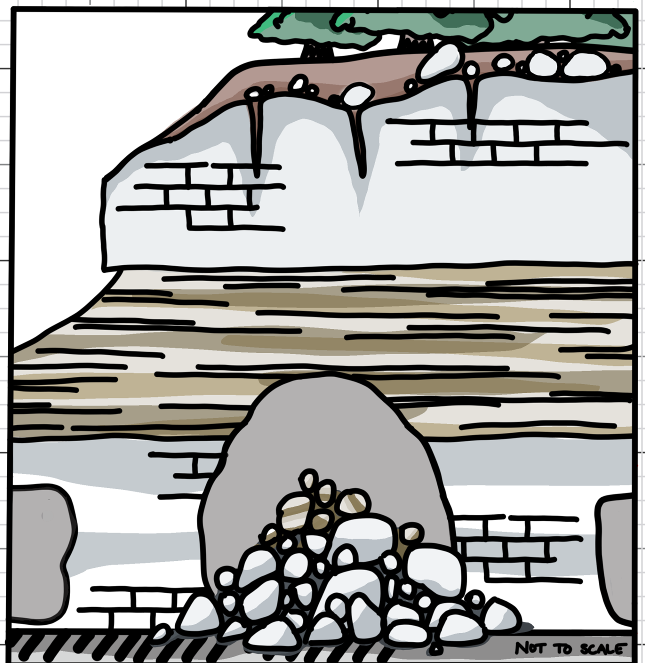
DATE: __________________________

DATE:

APPENDIX G

Calculations for Minimum Fill Height Requirements

MINE MITIGATION STUDY	J035637.02
MINE FILLING CALCULATIONS	

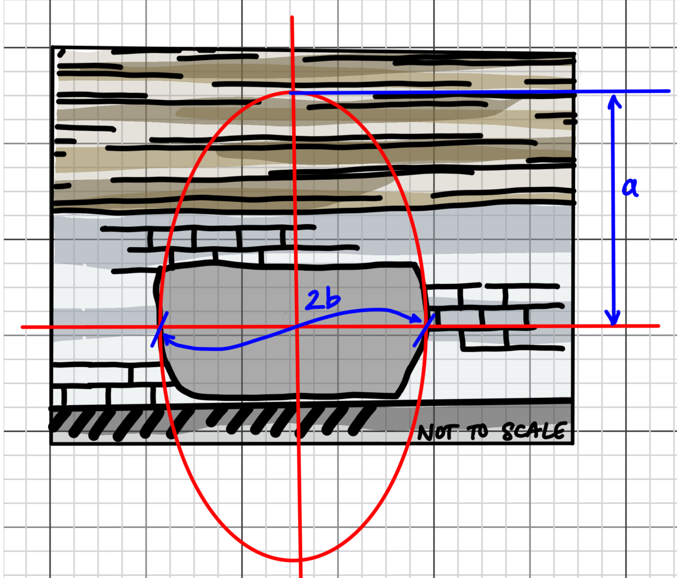


IN GENERAL, ROOF FAILURES CONTINUE TO PROPOGATE LATERALLY AND HORIZONTALLY UNTIL A MORE STABLE CONFIGURATION IS REACHED, GENERALLY AN ARCH.

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 $(0.7P_v)$
σ_t = TENSILE STRESS

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

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

$\gamma = 163\ pcf$
$L = 35\ ft$
$t = 4.5\ ft$

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

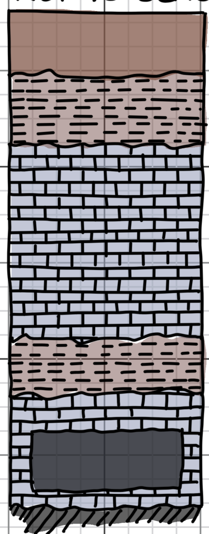
Calc. By: aly	Checked By:	Appvd By:
Date: 12-22-2020	Date:	Date:

MINE MITIGATION STUDY | J035637.02

MINE FILLING CALCULATIONS

P_V = VERTICAL STRESS

NOT TO SCALE



		ASSUMED
OVERBURDEN	10.1'	γ = 120 pcf
WEA SHALE	8.7'	γ = 140 pcf
WINTERSET LIMESTONE	36.1'	γ = 163 pcf
STARK SHALE	6.2'	γ = 140 pcf
BETHANY FALLS LIMESTONE	ASSUMED BEAM THICKNESS 4.5'	γ = 163 pcf

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

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

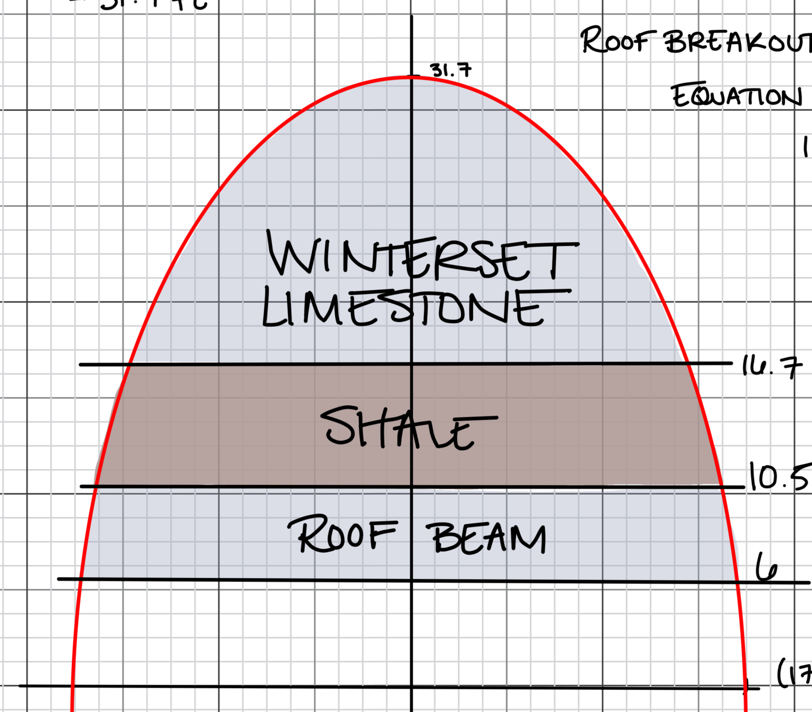
$$P_h = 0.7 P_V = 6941.1\,psf$$

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

$$= 31.7\,ft$$

ROOM HEIGHT 12 ft

ROOF BREAKOUT DEPTH 31.7 − 12 = 19.7 ft



EQUATION OF AN ELLIPSE

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

WHERE: a = 31.7 ft
b = 17.5 ft

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

IN TERMS OF Y

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

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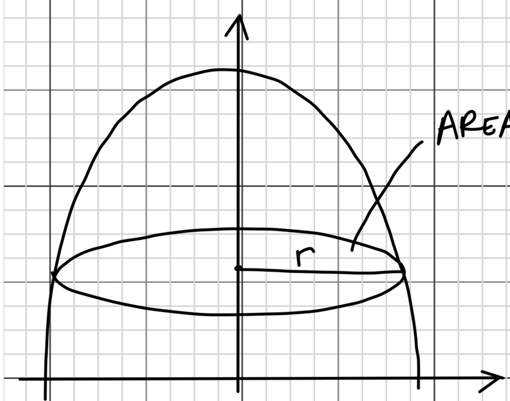
Date: 12-22-2020	Date:	Date:

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

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

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

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

FOR THE ROOF BEAM:

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

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

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

$$= 17.5^2 \pi \left(4.5 - 1.025\right)$$

$$= 4{,}025 \text{ ft}^3$$

VOLUME OF SHALE

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

$$= 4{,}835 \text{ ft}^3$$

TOTAL ROCK VOLUME $= 8{,}860 \text{ ft}^3$

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Date: 12-22-2020	Date:	Date:

MINE MITIGATION STUDY	J035637.02
MINE FILLING CALCULATIONS	

EXPANDED VOLUME

ASSUMED BULKING FACTORS

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

	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 $4{,}025\ ft^3 (1.25) = 5{,}030\ ft^3$

EXPANDED SHALE VOLUME $4{,}835\ ft^3 (1.12) = 5{,}415\ ft^3$

$10{,}445\ ft^3$

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

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

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

$$= 20{,}265\ ft^3$$

$$V_{FILL} = V_{ROOM} - V_{DEBRIS} = 20{,}265\ ft^3 - 10{,}445\ ft^3 = 9{,}820\ ft^3$$

FILL HEIGHT		
	11 ft	$10{,}473\ ft^3$
	10.5 ft	$10{,}002\ ft^3$
	10 ft	$9{,}530\ ft^3$

RECOMMENDED MINIMUM WOULD BE WITHIN 12 INCHES OF MINE ROOF

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Date: 12-22-2020	Date:	Date:

<u>APPENDIX H</u>

Industry Standards for Quality Control in Blind Backfills

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.

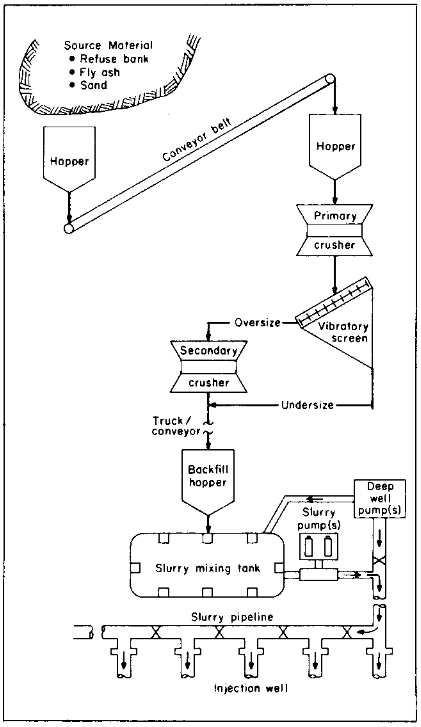

Fig. 1—Pumped-slurry backfilling operation.

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

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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

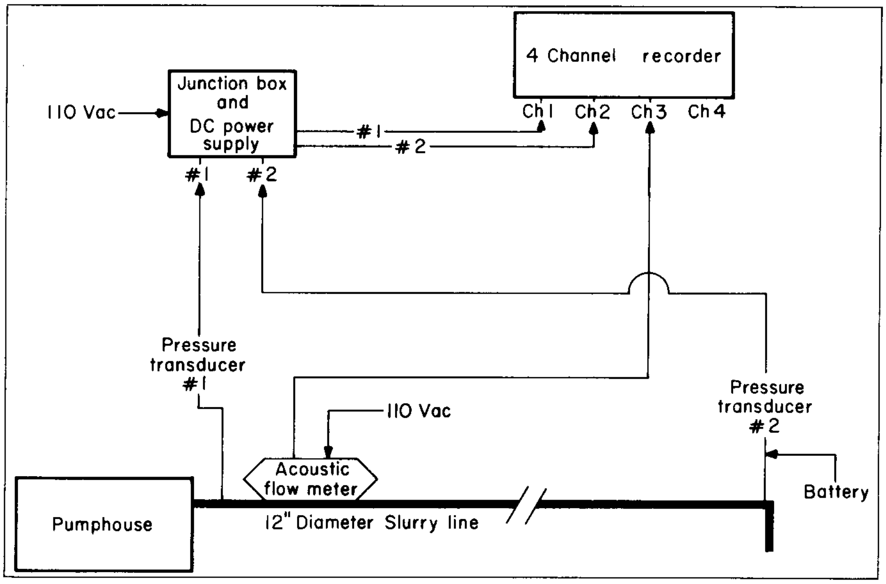

Fig. 3—Process monitoring system.

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

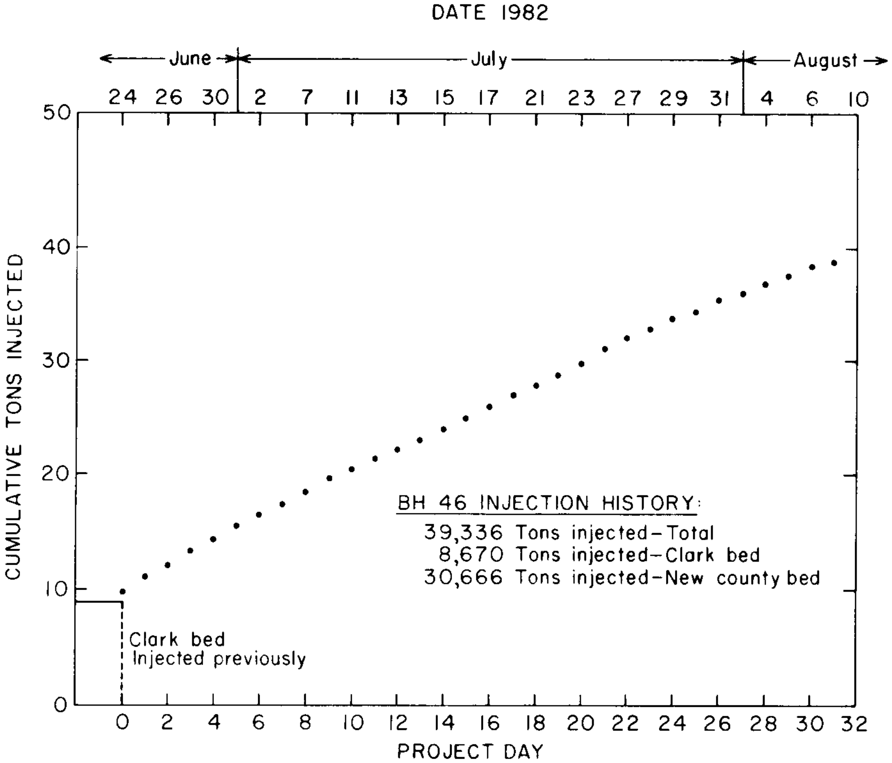

Fig. 4—Injection record for borehole No. 46.

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. ■

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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.

UNDERGROUND MINING

Mine backfill design and testing

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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:

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

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:

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

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:

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

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

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

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:

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

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%

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

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:

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

where γ_w = unit weight of water = 9.81 kN/m³.

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):

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

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

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

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

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

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

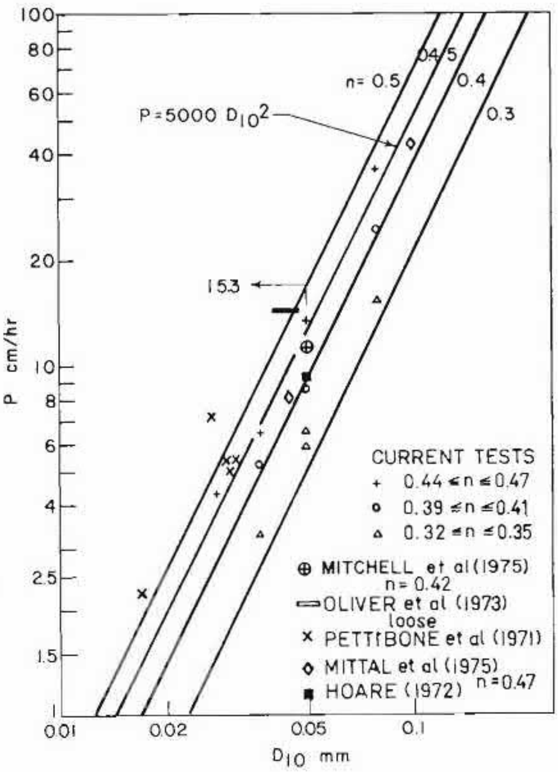


FIGURE 1. Percolation test data.

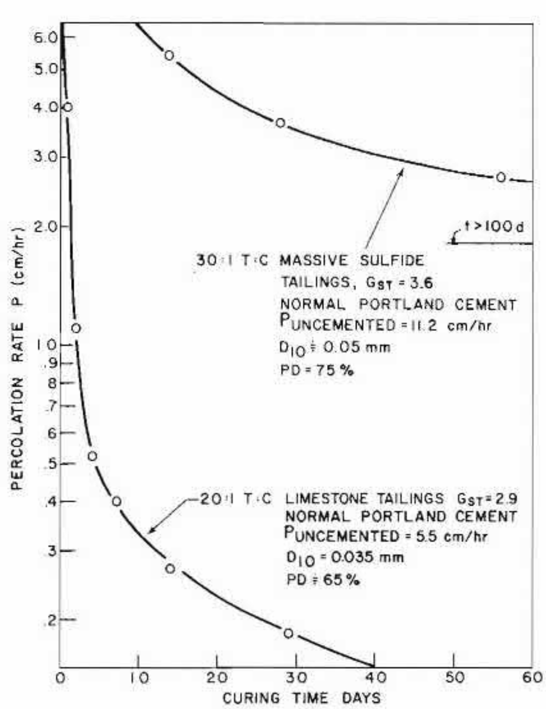


FIGURE 2. Percolation in cemented tailings.

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:

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

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.

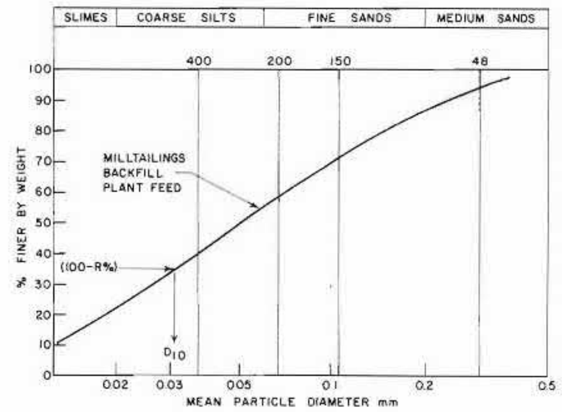


FIGURE 3. Estimation of D_{10} from plant feed recovery.

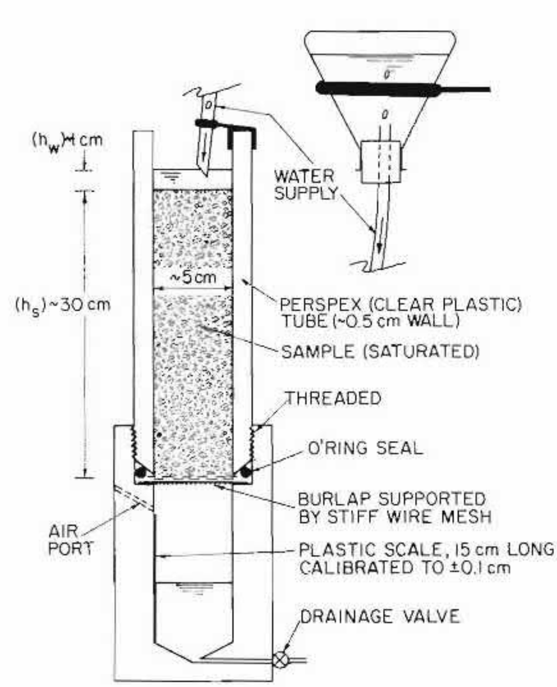


FIGURE 4. Percolation test arrangement.

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 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:

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

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

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

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

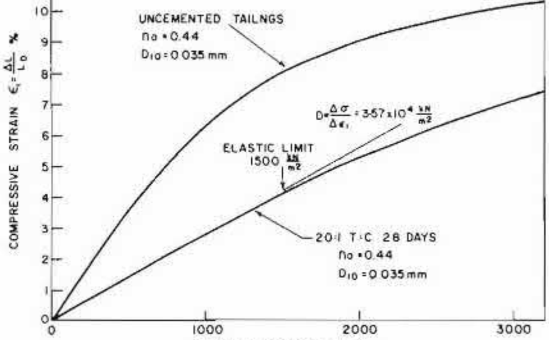


FIGURE 5. Compressibility test results.

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 $\gamma = 20$ kN/m^3, $\sigma_v' = 400$ 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), $\Delta\epsilon_1 = 11.6\%$ ($\Delta H = 2.30$ 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.

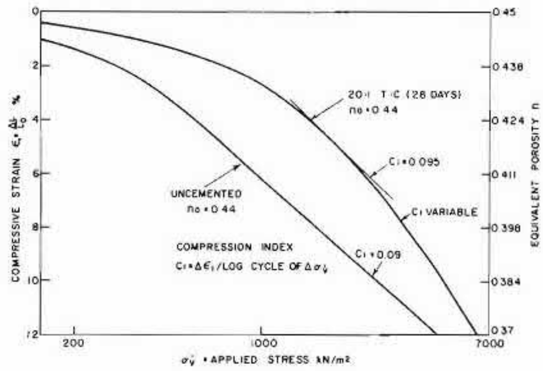
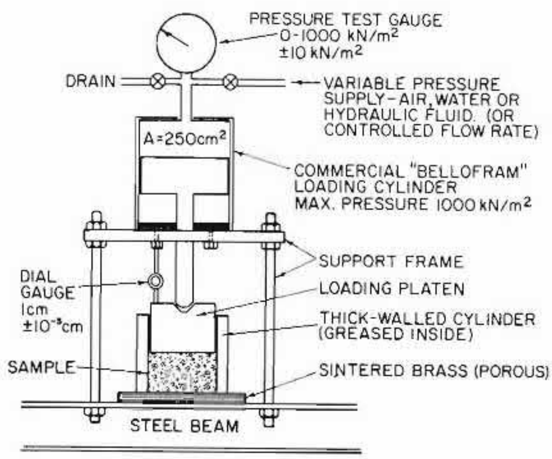


FIGURE 6. Compressibility vs logarithm of pressure.



SAMPLE PREPARATION: POUR AND CURE (AS REQUIRED) IN MOLD.
SAMPLE SIZE: 8 cm I.D. BY 4 cm HIGH (A = 50 cm^2)
MAXIMUM σ_v: 1000 kN/m^2 $\times \frac{250 \text{ cm}^2}{50 \text{ cm}^2}$ = 5000 kN/m^2

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.

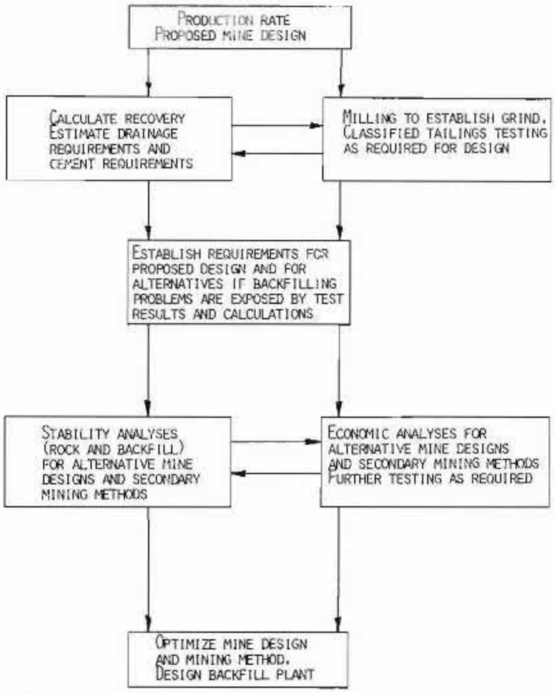


FIGURE 8. Proposed design flow chart for mine backfill design.

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.

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.*

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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.

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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.

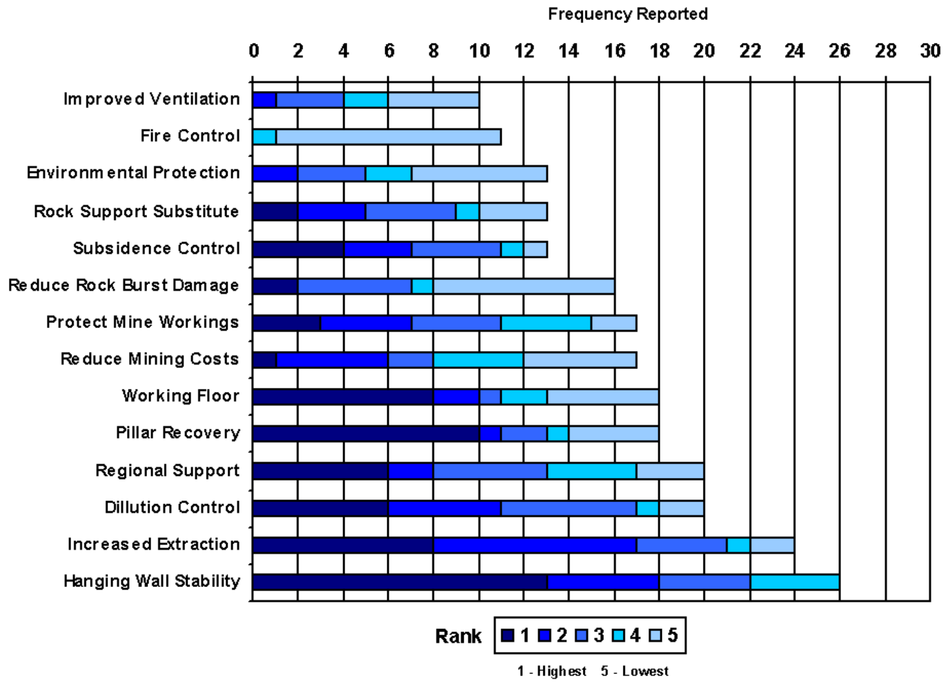

Figure 1. Backfill functions in underground mining.

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.

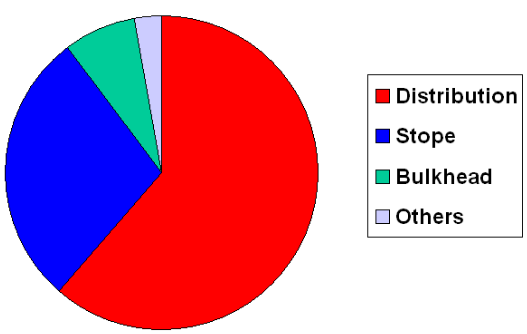

Figure 2. Reported failures in backfill systems.

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.

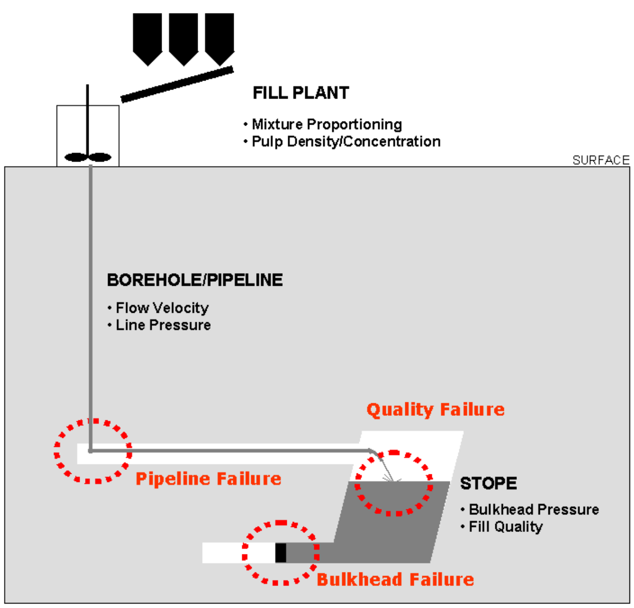

Figure 4. Simplified mine backfill system components.

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).

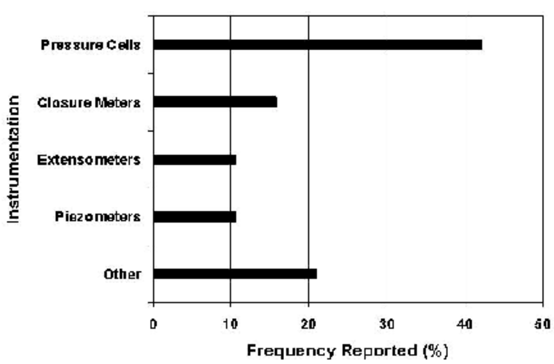

Figure 5. Reported backfill instrumentation.

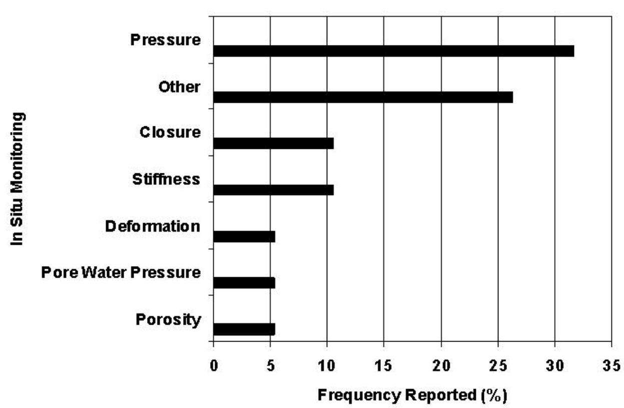

Figure 6. Reported backfill monitoring.

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.

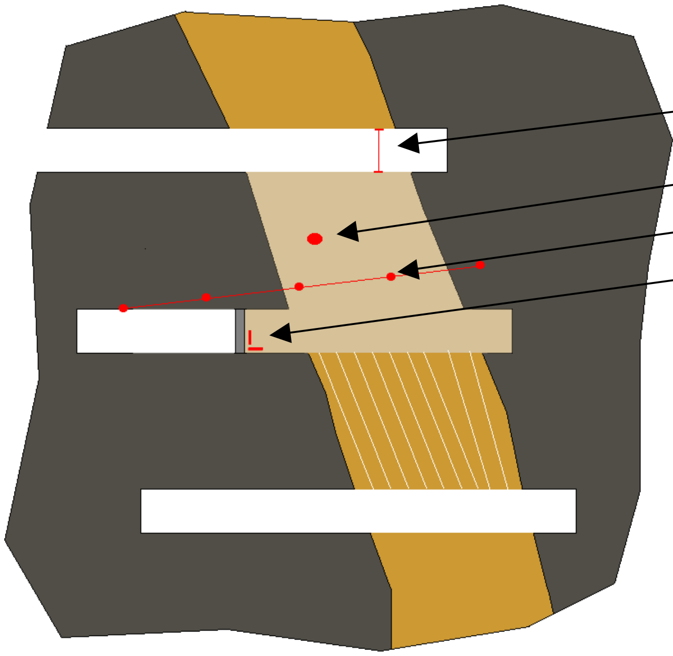

Figure 7. Basic instrumentation of a backfilled stope.

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.

APPENDIX I

Project Manual for Mine Filling at Streets of West Pryor Development

PROJECT MANUAL
FOR
MINE FILLING
AT
PRYOR CROSSING

NORTHWEST QUADRANT OF NW PRYOR ROAD AND NW LOWENSTEIN DRIVE
LEE'S SUMMIT, MISSOURI

JANUARY 15, 2021

GEOTECHNOLOGY, INC.
5055 ANTIOCH ROAD
OVERLAND PARK, KANSAS 66203

SECTION 00 01 01
PROJECT TITLE PAGE

PROJECT MANUAL
FOR
MINE FILLING AT PRYOR CROSSING

GEOTECHNOLOGY PROJECT NUMBER J035637.02

NORTHWEST QUADRANT OF NW PRYOR ROAD AND NW LOWENSTEIN DRIVE
LEE'S SUMMIT, MISSOURI

DATE: JANUARY 15, 2021

PREPARED BY:
GEOTECHNOLOGY, INC.

END OF SECTION 00 01 01

SECTION 00 01 02
PROJECT INFORMATION

1.01 PROJECT IDENTIFICATION

A. Project Name: Mine Filling Pryor Crossing
Northwest Quadrant of NW Pryor Road and NW Lowenstein Drive
Lee's Summit, Missouri

B. The Owner, hereinafter referred to as Owner: Streets of West Pryor, LLC

C. Owner's Project Manager: David Olson
1. Address: 7200 W 132nd Street, Suite 150
2. City, State, Zip: Overland Park, Kansas 66213
3. Phone: 314-413-3598
4. E-mail: daveolson@monarchprojectllc.com

1.02 PROJECT DESCRIPTION

A. Summary Project Description: The mine space, formerly known as Union Quarries Mine, will be filled prior to the construction of single-family homes and a multi-family apartment complex on the Streets of West Pryor property. The mine will be filled from the surface by drilling holes into the mine space and placing a 2-inch minus prepared aggregate to within 12 inches or less of the mine roof. The mine will be filled in a checkerboard pattern, filling every other room, and will extend two rooms beyond the footprint of the planned development.

B. Contract Scope: Mine Filling

1.03 PROJECT CONSULTANTS

A. The Engineering Geologist, hereinafter referred to as Engineering Geologist: Geotechnology, Inc.
1. Address: 5055 Antioch Road
2. City, State, Zip: Overland Park, Kansas 66203
3. Phone: 913-438-1900
4. E-mail: aprince@geotechnology.com

B. The Mine Filling Contractor, hereinafter referred to as Mine Contractor: Drill Tech
1. Address: 8334 Ruby Avenue
2. City, State, Zip: Kansas City Kansas, 66111
3. Phone: 913-422-5088
4. Email: Patrick.Carr@drilltechdrilling.com

1.04 PROCUREMENT TIMETABLE

A. The Owner reserves the right to change the schedule or terminate the entire procurement process at any time.

1.05 PROCUREMENT DOCUMENTS

A. Availability of Documents: Complete sets of procurement documents may be obtained:
1. From Owner at the Project Manager's address listed above.

END OF SECTION 00 01 02

SECTION 00 01 03
PROJECT DIRECTORY

1.01 SECTION INCLUDES

A. Identification of project team members and their contact information.

1.02 OWNER

A. Name: Streets of West Pryor, LLC
   1. Address Line 1: 7200 W 132nd Street, Suite 150
   2. City: Overland Park
   3. State: KS
   4. Zip Code: 66213
   5. Telephone: 314-413-3598

B. Primary Contact: All correspondence from the Mine Contractor to the Engineering Geologist will be direct, with copies to this party, unless alternate arrangements are mutually agreed upon at the preconstruction meeting.
   1. Title: Owner
   2. Name: David Olson
   3. E-mail: daveolson@monarchprojectllc.com
   4. Telephone: 314-413-3598

1.03 PROJECT CONSULTANTS

A. Engineering Geologist: Design Professional of Record. All correspondence from the Mine Contractor regarding construction documents authored by the Engineering Geologist will be direct, with copies to this party, unless alternate arrangements are mutually agreed upon at the preconstruction meeting.
   1. Company Name: Geotechnology, Inc.
      a. Address Line 1: 5055 Antioch Road
      b. City: Overland Park
      c. State: KS
      d. Zip Code: 66203
      e. Telephone: 913-438-1900
   2. Primary Contact
      a. Title: Engineering Geologist, Senior Project Manager
      b. Name: Andrea Prince, R.G.
      c. E-mail: aprince@geotechnology.com
      d. Cell Phone: (913) 998-0527

B. Project Surveyor
   1. Company Name: BHC Rhodes
      a. Address Line 1: 712 State Avenue
      b. City: Kansas City
      c. State: KS
      d. Zip Code: 66101
      e. Telephone: 913-371-5300
   2. Primary Contact

a. Title: Surveyor
b. Name: Matthew Schepmann
c. E-mail: matthew.schepmann@ibhc.com
d. Cell Phone: 816-898-2832

C. The Mine Filling Contractor

1. Drill Tech
    a. Address Line 1: 8334 Ruby Avenue
    b. City: Kansas City Kansas
    c. State: Kansas
    d. Zip Code: 66111
    e. Phone: 913-422-5088
2. Primary Contact
    a. Title: Manager
    b. Name: Patrick Carr
    c. Email: Patrick.Carr@drilltechdrilling.com
    d. Cell Phone: 913-378-2580

END OF SECTION 00 01 03

SECTION 00 01 10
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END OF SECTION 00 01 10

SECTION 00 31 00
AVAILABLE PROJECT INFORMATION

00 31 13 – PRELIMINARY SCHEDULES

00 31 13.13 – PRELIMINARY PROJECT SCHEDULE

A. A total of 173 days is estimated for completion of the project, utilizing 10-hour daytime shifts, 5 days a week.
   1. This time estimate excludes survey and layout of the drill locations, clearing, grubbing, or preparation of a working surface, removal of excess materials or spoils, and repair/restoration of site.
   2. This project timeframe assumes the availability of at least 540 cubic yards of fill material per day.

00 31 13.26 – PRELIMINARY CONSTRUCTION SEQUENCING

A. The project Surveyor shall tie the existing surveys for the surface and mine space together. If additional work is required to do so, it shall be at the Owner's expense.

B. Boreholes, approximately 20-inch diameter, shall be drilled from the surface penetrating the mine space on a checkboard pattern within the footprint of the planned development. The drill holes shall be cased from the surface to the top of rock and capped to prevent the leakage of water into the mine space.

C. The mine shall be filled at each drill hole location by dropping material down the hole until the resulting pile reaches the mine ceiling. A device capable of spreading the mine material such that the pile extends radially from the drill hole should be employed.

D. The mine space should be filled to within 12 inches or less of the mine roof.

E. At the completion of filling at each drill hole, the hole shall be promptly backfilled.

00 31 19 – EXISTING CONDITION INFORMATION

A. Based on the previous work performed by Engineering Geologist. for the subject mine space, a majority of the mine area is underwater with depths of up to 8 feet; however, water could be deeper in unexplored areas of the mine. A number of dome outs are located along the major joint directions. These dome outs extend into multiple rooms and evidence of propagation can be seen in the surrounding rooms.

00 31 21 – SURVEY INFORMATION

A. Survey of the surface and mine space has been performed by BHC Rhodes. Updated survey information is required in order to begin laying out the borehole locations required for filling.

00 31 32 – GEOTECHNICAL DATA

A. Observation Letter: Summary of Preliminary Mine Evaluation, Streets of West Pryor Development over Former Union Quarry Mine, Lee's Summit, Missouri, dated May 15, 2020.
   1. Prepared by Geotechnology, Inc., Overland Park, Kansas
   2. The letter identifies potential sources for instability after the construction of the Street of West Pryor development.

3. The recommendations described shall not be construed as a requirement of this Contract, unless specifically referenced in the Contract Documents.
4. This letter, by its nature, cannot reveal all conditions that exist on the site. Should subsurface conditions be found to vary substantially from this report, changes in the design of the mine filling will be made, with resulting credits or expenditures accruing to the Owner.

END OF SECTION 00 31 00

SECTION 01 31 00
PROJECT MANAGEMENT AND COORDINATION

01 31 13 – PROJECT COORDINATION

A. Project coordination will be in collaboration with the Engineering Geologist and Mine Contractor as to mine filling activities and source materials.

01 31 19 – PROJECT MEETINGS

01 31 19.13 – PRECONSTRUCTION MEETING

A. A preconstruction meeting will be scheduled by the Engineering Geologist and held after the completion of survey tying the subsurface and surface together and at least 7 days prior to the anticipated start of any construction activities. Attendance by the Mine Contractor and any subcontractors is mandatory. The preconstruction meeting will be conducted to clarify the construction requirements, discuss submittals for the work, to coordinate the construction schedule and activities, and to identify contractional relationships and delineation of responsibilities among the Mine Contractor and any approved subcontractors.

01 31 19.16 – SITE MOBILIZATION MEETING

A. A site mobilization meeting will be scheduled by the Engineering Geologist and held prior to site mobilization. The site mobilization meeting will be conducted to clarify remaining questions about the site mobilization and discuss staging areas and sequencing of room filling.

01 31 19.23 – PROGRESS MEETING

A. At least one progress meeting will be scheduled by the Engineering Geologist during construction. The progress meeting will be conducted to update the Owner on the construction progress and to discuss changes in scope, if required.

END OF SECTION 01 31 00

SECTION 01 32 00
CONSTRUCTION PROGRESS DOCUMENTATION

01 32 29 –WORK OBSERVATION

A. The Owner will retain Engineering Geologist full-time to monitor borehole drilling and mine filling and to collect a backfill sample from each borehole location.
   1. The Mine Contractor shall coordinate with Engineering Geologist access to material samples and work areas for personnel and equipment.

B. Periodic observation of the mine space by the Engineering Geologist, or their representative, is required to confirm placement of the fill material in accordance with specifications. Observation should be completed such that fill piles do not obscure previous work prior to observation.
   1. Coordination of access will be the responsibility of the Owner.
   2. Additional personnel may be required to accompany the Engineering Geologist through the Star Excavation property and through the mine space to the project site. Additional personnel, where required, will be at the Owner's expense.

C. Employment of testing and observation services in no way relieves Mine Contractor of obligation to perform Work in accordance with requirements of Contract Documents.

END OF SECTION 01 32 00

SECTION 01 33 00
SUBMITTAL PROCEDURES

01 33 19 – FIELD TEST REPORTING

A. Engineering Geologist will provide the Owner with field reports detailing observations regarding the depth of the borehole, the height of the mine at the borehole location, the volume of backfill materials placed, confirmation of the required fill height via borehole camera, the thicknesses and materials used to backfill the borehole, and samples collected for testing.
   1. Field reports will be sent to the Mine Contractor with Owner permission.

01 33 26 – SOURCE QUALITY CONTROL REPORTING

A. Engineering Geologist will provide the Owner with reports detailing the origin of test samples, the drill hole(s) filled with the associated material, and the results of laboratory testing and observation.
   1. Testing reports will be sent to the Mine Contractor with Owner permission.

END OF SECTION 01 33 00

SECTION 01 35 00
SPECIAL PROCEDURES

01 35 26 – GOVERNMENTAL SAFETY REQUIREMENTS

A. Applicable OSHA Requirements for Health and Safety Plan
    1. The Mine Contractor shall be responsible for developing a site-specific Health and Safety Plan, as required by 29 CFR 1926.65(b), based on site characterization, monitoring, and exposure assessments as stated in 29 CFR 1926.65 and 29 CFR 1926.62.

B. The Mine Contractor shall comply with all applicable provisions of 29 CFR 1926.65 and 29 CFR 1926.62.

C. All Mine Contractor personnel involved in construction shall receive training meeting the requirements on 29 CFR 1926.65(e)(3)(iii) before construction begins.

D. The Mine Contractor shall submit the Health and Safety Plan to the Engineering Geologist one week prior to beginning construction. Along with the plan, the Mine Contractor shall submit the monitoring and exposure assessment data used to determine the requirements of the plan. This plan will be reviewed and filed by the Engineering Geologist for availability to enforcement agencies. The Mine Contractor shall also follow all other safety requirements incorporated elsewhere in these provisions and include them in this Health and Safety Plan.

01 35 29 – HEALTH, SAFETY, AND EMERGENCY RESPONSE PROCEDURES

A. The Mine Contractor shall exercise extreme caution when performing any activities on the site and especially around a known dome-out. The Mine Contractor shall take appropriate precautions to ensure all personnel are aware of the hazards and provide adequate safety equipment.
    1. The construction site shall be a hard-hat and steel-toe shoe area.
    2. The Mine Contractor shall continually monitor the area for instability and notify the Engineering Geologist immediately if any signs of potential instability are noted. The Engineering Geologist, in conjunction with the Mine Contractor, shall then determine a course of action, including if and when the work can continue.
    3. Dust control will be provided by others.

B. The Mine Contractor shall have communication equipment on the construction site or immediate access to other communication systems to request assistance from the police or other emergency agencies for incident management. The Owner and Engineering Geologist shall be notified when the Mine Contractor requests emergency assistance.

C. In addition to the 911 emergency telephone number for ambulance, fire, or police services, the following agencies may also be notified for an accident or emergency situation within the project limits.
    1. Missouri Highway Patrol: (816) 622-0800
    2. Jasper County Sheriff: (816) 524-4302
    3. City of Lee’s Summit Police: (816) 969-7390
    4. City of Lee’s Summit Fire: (816) 969-7407

D. The Mine Contractor shall notify enforcement and emergency agencies before the start of construction to request their cooperation and to provide coordination of services when emergencies arise during the construction at the project site. When the Mine Contractor completes this notification with enforcement and emergency agencies, a report shall be furnished to the Engineering Geologist on the status of incident management.
E. No direct pay will be made to the Mine Contractor to recover the cost of the communication equipment, labor, materials, or time required to fulfill the above provisions.
F. Parties entering the mine space for observation of the fill piles will review Geotechnology's Mine Entry Safety Plan prior to entry and adhere to the procedures therein.

END OF SECTION 01 35 00

SECTION 01 43 00
QUALITY ASSURANCE

01 43 26 – TESTING AND INSPECTING AGENCY QUALIFICATIONS

A. Prior to start of work, submit agency name, address, telephone, and names of responsible officer.

01 43 36 – FIELD SAMPLES

A. The fill material should be sampled at each borehole location and at intervals when the source material changes. The samples should be taken from the stockpile in accordance with ASTM standards and (1) marked with their source/origin and the room(s) the material was placed and (2) transported to the laboratory for testing.

END OF SECTION 01 43 00

SECTION 01 45 00
QUALITY CONTROL

01 45 13 – SOURCE QUALITY CONTROL PROCEDURES

A. Mine filling material should consist of 2-inch minus aggregate with less than 20% fines. Organic materials should be excluded from the stockpile.

01 45 16 – FIELD QUALITY CONTROL PROCEDURES

A. Material should be visually inspected for aggregate large enough to block the drill hole.
B. Saturated material should be excluded from the stockpile.
C. Samples of the backfill should be taken in accordance with the procedure indicated in Section 01 45 23
D. Following placement of the fill, the borehole camera should be placed downhole to verify the 12-inch or less clearance between the top of the fill and the mine roof.
E. Drill holes should be backfilled after filling of the mine space to prevent surface water from entering the mine space which may cause washout of the placed materials.
   a. The temporary casing should be used as a tremie pipe for backfilling.
   b. The abandoned borehole should be plugged with bentonite chips, cement bentonite or bentonite grout, or concrete.

01 45 16.13 – CONTRACTOR QUALITY CONTROL

A. All Mine Contractor plans for the completing the work of mine filling shall be submitted to the Engineering Geologist for approval prior to execution in the field. Changes made to the work plan during the course of construction shall likewise be approved by the Engineering Geologist.

01 45 23 – TESTING AND INSPECTION SERVICES

A. A representative sample shall be taken from the stockpile or conveyor for each borehole location and at intervals when the source material changes.
   1. Sample will be marked with the following:
      a. Date
      b. Mine Room filled
      c. Aggregate Source
   2. Testing to Include:
      a. Gradation
      b. Slake Durability (if determined necessary by the Engineering Geologist)
B. Observation should include, but is not limited to:
   1. Depth to Mine Floor
   2. General Description of Fill Materials
   3. Depth to Top of Fill as verified by borehole camera
   4. Placement of Backfill Materials in the Borehole
C. Inspection of the mine space is required to verify fill height and breadth of each fill pile. Additional borehole and fill piles may be required if large portions (more than two

connecting rooms) are not filled to a minimum depth to be determined during the rock slinger test holes.

01 45 29 – TESTING LABORATORY SERVICES

A. Prior to Mine Filling, testing shall be performed on an aggregate sample prepared from the source material and processed in the project crusher:
    1. Gradation
    2. Slake Durability (if determined necessary by the Engineering Geologist)
B. During construction, the following tests will be performed on collected samples:
    1. Gradation
C. Significant variation in the source rock may warrant additional gradation and slake durability testing. The Engineering Geologist should be notified of changes in the source aggregate.

END OF SECTION 01 45 00

SECTION 31 23 00
EXCAVATION AND FILL

31 23 23 – FILL

A. Mine fill material shall be:
   1. 2-inch minus aggregate with less than 20% fines
   2. Organic materials should be excluded from the stockpile.
   3. Free of compressible and other deleterious materials

END OF SECTION 31 23 00