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# INTERNATIONAL LAND RECLAMATION AND MINE DRAINAGE CONFERENCE

# AND

# THIRD INTERNATIONAL CONFERENCE ON THE ABATEMENT OF ACIDIC DRAINAGE

## Volume 4 of 4: Abandoned Mine Lands and Topical Issues

Proceedings of a conference held in Pittsburgh, PA on April 24-29, 1994.
This conference served as the annual meeting for both the American Society for Surface Mining and Reclamation and the Canadian Land Reclamation Association.

Copies of this and the three companion volumes can be obtained by through NTIS.

Volume 1: Mine Drainage - SP 06A-94
Volume 2: Mine Drainage - SP 06B-94
Volume 3: Reclamation and Revegetation - SP 06C-94
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## UNIT OF MEASURE ABBREVIATIONS USED IN THESE PROCEEDINGS

| | |
|---|---|
| Å | angstrom |
| ac | acre |
| Bq | becquerel |
| bu | bushel |
| cal | calorie |
| cm | centimeter |
| cmol | centimole |
| cps | count per second |
| °C | degree Celsius |
| d | day |
| dS | decisiemen |
| ft | foot |
| g | gram |
| gal | gallon |
| gpd | gallon per day |
| gpm | gallon per minute |
| h | hour |
| ha | hectare |
| hL | hectoliter |
| hp | horse power |
| Hz | hertz |
| in | inch |
| J | joule |
| K | kelvin |
| kcal | kilocalorie |
| kg | kilogram |
| kHz | kilohertz |
| km | kilometer |
| kmt | kilo metric ton |
| kN | kilonewton |
| kPa | kilopascal |
| kV | kilovolt |
| L | liter |
| lb | pound |
| m | meter |
| M | molar |
| mA | milliampere |
| meq | milliequivalent |
| mg | milligram |
| Mg | megagram |
| min | minute |
| mL | milliliter |
| mm | millimeter |
| mmho | millimho |
| mol | mole |
| mmol | millimole |
| MPa | megapascal |
| mS | millisiemen |
| mt | metric ton |
| mV | millivolt |
| MW | megawatt |
| µg | microgram |
| µL | microliter |
| µm | micrometer |
| µmho | micromho |
| µmol | micromole |
| N | normal |
| N | newton |
| nm | nanometer |
| Pa | pascal |
| pCi | picocurie |
| ppb | part per billion |
| ppm | part per million |
| ppt | part per thousand |
| psi | pound per square inch |
| psig | pound per square inch gauge |
| r | revolution |
| rad | radian |
| rpm | revolutions per minute |
| s | second |
| st | short ton |
| vol % | volume percent |
| wt % | weight percent |
| yd | yard |
| yr | year |

# INTERNATIONAL LAND RECLAMATION AND MINE DRAINAGE CONFERENCE AND THIRD INTERNATIONAL CONFERENCE ON THE ABATEMENT OF ACIDIC DRAINAGE

Proceedings of a conference serving as the annual meetings of the American Society for Surface Mining and Reclamation, the Canadian Land Reclamation Association, and the West Virginia Surface Mine Drainage Task Force, in conjunction with the Third International Conference on the Abatement of Acidic Drainage.

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## ABSTRACT

Mine drainage and mine reclamation are topics of major interest to the mining industry, federal and local governments, and the general public. This publication and its companion three volumes are the proceedings of a conference held in Pittsburgh, Pennsylvania, April 24-29, 1994. There were twelve sessions (69 papers) that dealt with mine drainage, including modeling, geochemistry, prediction, treatment, control strategies, characterization, hydrology, and case studies. These sessions comprise volumes 1 and 2. The six sessions (34 papers) that dealt with reclamation and revegetation of disturbed lands are included in volume 3. Volume 4 includes the six sessions (34 papers) that dealt with such topical issues as fires at abandoned mine sites, subsidence, hydrology, mine wastes, and policy. Poster session presentations are represented by 49 papers and 92 abstracts that have been placed in the back of volumes 2, 3, and 4, consistent with the subject of that volume.



Reference to companies and specific products in these papers does not imply endorsement by the Bureau of Mines.

# INVESTIGATION OF A SUBSIDENCE EVENT NEAR FLUSHING OHIO[1]

Charles M. Ledney and Joan L. Hawk[2]

**Abstract**: An investigation was undertaken to determine the cause and extent of events which caused problems to a number of residences along State Route 149 near Flushing, Belmont County, Ohio. The events began in 1988 and continued through 1991 and affected nine homes. The type of problems occurring, as well as surface effects, compared to available mine maps of the area, indicated the problems were caused by subsidence from coal mining. The mining occurred in the Pittsburgh seam at a depth of between 180 and 220 feet. The mining beneath the site took place between 1975 and 1977 and was of the room and pillar type. A subsurface investigation was performed, along with "down the hole" video camera inspections to provide necessary subsurface information for analysis of the subsidence event. Factors of safety were calculated for pillars throughout the mine. Based on this analysis, it was determined that pillar failure caused the subsidence event. Once a determination was made as to the likely cause of the subsidence, the data was re-examined to determine the possible location of pillar failure, as well as the type and extent of subsidence. This analysis involved the use of RQD versus depth plots and the compilation of isopach maps of the mine overburden and the Sewickley Sandstone. The trend of the two maps suggested that a relationship existed between the sandstone thickness, the overburden and the surface expression of the subsidence. In order to determine this relationship, the two maps were combined into a second order map showing the mine overburden - Sewickley Sandstone thickness ratios. The combination was accomplished by computer matrix operations using the grid values of the two previous maps that were generated by kriging. It was concluded that the ratio of the Sewickley Sandstone thickness to the mine overburden had a tremendous effect on the amount of damage that occurred to specific residences.

## Introduction

Underground mining for the removal of coal has taken place in the eastern portion of the United States since the early part of this century. Millions of acres of abandoned room and pillar mines exist, many of these below existing structures. As the coal pillars, mine floor and roof strata of these mines degrade with age, subsidence of these abandoned mines continues to occur.

Many factors contribute to the cause, extent and effects of the subsidence. These factors include overburden thickness and composition, seam thickness, pillar configuration, condition and strength, groundwater levels, mine floor conditions, location and composition of structures above the mine, etc. The number and variability of these factors has precluded the development of methods to predict the likelihood and extent of room and pillar mine subsidence with any degree of accuracy. Because of this, it is necessary to examine each subsidence event with a broad based approach, paying particular attention to trends in the data that may give an indication of cause and extent.

This paper presents a case history of a subsidence event near Flushing, Ohio. In cooperation with the Office of Surface Mining (OSM), this investigation was undertaken to determine the cause and extent of subsidence occurring at this site. Using standard investigation techniques in conjunction with some unique analysis of the data, a relationship between the surface expression of the subsidence and the thickness of a sandstone bed and the overburden stress was developed. While this relationship is site specific, it provides insight into the means and methods necessary to investigate room and pillar subsidence.

## Site Description and Mining History

The study area is situated along State Route 149 beginning approximately 518 m. north of its intersection with S.R. 331 and extending north approximately 549 m. in Belmont County, Ohio. The site encompasses eleven (11) single family dwellings as well as three (3) mobile homes.

The area is underlain by room and pillar mine workings in the Pittsburgh or No. 8 seam with the depth to the top of the seam varying between 54.9 m. and 67.1 m. The mine began operation in the late 1960's and according to MSHA records closed in November, 1990. Dates on the mine map indicated that mining under the study area took place between 1975 and 1977.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

[2]Charles M. Ledney, Senior Engineering Geologist and Joan L. Hawk, Senior Geologist, L. Robert Kimball & Associates, Inc.

According to former employees of the mine, this particular section was mined with conventional mining techniques, with retreat mining performed by a continuous miner. The shape of many of the pillars on the mine map indicate that slabbing and/or splitting of pillars was conducted. The extraction rate was 61%. The mine map indicates that the overlying No. 9 (Meigs Creek Coal) was mined approximately 1.2 km. west of the site. However, the map does not show any deep mining in this seam within the study area, although it has been surfaced mined adjacent to the site.

## Geology

Strata underlying the study area belong to the Pennsylvania Age Monongahela Group. This group averages 76 m. thick, and is composed of alternating beds of limestone, claystone, shale, sandstone, and coal (see Figure 1).

Known as the Upper "Productive Coal Measures", the Monongahela Group contains three regionally economic coal seams, as well as several non-persistent, locally minable coal seams. Within the study area, the top 9 m. of Monongahela strata has been eroded. The remaining strata of this group contain the other two economic coal seams - the No. 9 and the stratigraphically lower No. 8. The No. 9 coal lies from 34 m. to 37 m. below ground surface and averages 1.2 m. thick. The No. 8 coal lies from 55 m. to 67 m. below ground surface and averages 2 m. thick and has been extensively mined beneath the study area.

Limestones comprise nearly 50% of the overburden. There are five limestone units within the Monongahela Group in the study area. They are in ascending order: the Redstone, the Fishpot, the Benwood, the Arnoldsburg and the Uniontown.

The Redstone Limestone lies from 1 m. to 2.7 m. above the No. 8 coal. The limestone consists of light gray limestone at the bottom grading into a shaley limestone at the top. The interval between the coal and the limestone consists of an incompetent slickensided claystone.

Approximately 1.5 m. above the Redstone Limestone is the Fishpot Limestone. The interval between the limestones is composed of claystone, and the thin discontinuous Redstone coal horizon which denotes the division between the two limestone units. The Fishpot is composed predominantly of light gray fine-grained limestone. There are also several beds of claystone 0.6 m. to 2.1 m. thick within this unit.

Another limestone lies from 0.3 m. to 3.7 m. below the No. 9 coal. This limestone, although persistent within

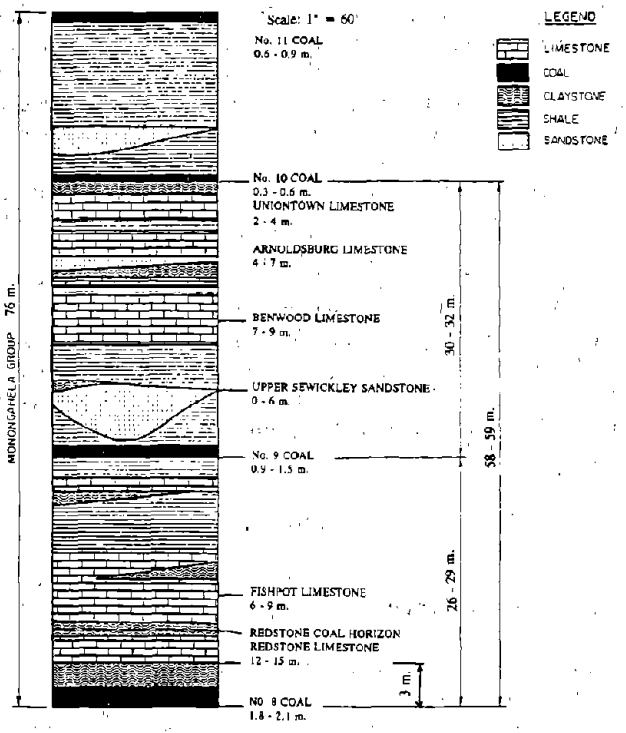


Figure 1. Generalized stratigraphic column of project area.

the study area, may not be aerially extensive. It is very thin, averaging only 1.4 m. thick.

The Benwood Limestone lies approximately 10.7 m. above the base of the No. 9 coal and like the Fishpot is composed predominantly of light gray fine-grained limestone. It ranges in thickness from 7.0 m. to 8.5 m. thick and contains beds of shale and clayshale.

The Arnoldsburg Limestone lies from 0.6 m. to 2.7 m. above the Benwood Limestone, separated from it, by a thin sandstone. The Arnoldsburg is shalier than the Benwood or Fishpot, although some fine-grained limestone beds are present along with the beds of shale and claystone.

The Uniontown Limestone occurs within a meter of the base of the Uniontown coal. It is separated from the underlying Arnoldsburg by 1.5 m. to 3.0 m. of shale and

claystone. Several on-site drill holes show a thin sandstone layer at the base of this limestone. In the study area, the Uniontown is predominately a medium gray shaley limestone.

The Upper Sewickley Sandstone is the only prominent sandstone occurring within the study area. It is a cross-bedded channel sand ranging from 3.0 m. to 7.0 m. thick. It lies from 2.7 m. to 6.7 m. above the top of the No. 9 coal.

## Site History and Structural Damage

The event or events causing the problems to the homes began in 1988. The Office of Surface Mining conducted an investigation at an adjacent property in early 1988, which appears to coincide with the beginning of these events. Subsidence monitoring indicated that movement was still taking place as of 1990. Also, sudden and severe structural failures at an adjacent property during September 1990 are further evidence that the movement is ongoing.

The subsidence has caused damage to most of the residences in this area. The following is a summary of problems which were found in the Marinucci, Doty, Taylor and Robert Hyest residences:

1. Tension cracking in floor slab.
2. Floor slabs separating from basement walls.
3. Cracks in the masonry block basement walls.
4. Cracking and/or separation of concrete sidewalks.
5. Staircase style cracks between and through exterior brickwork.

In addition, all homes other than the Robert Hyest residence experienced waterline breaks.

The Britton residence experienced the following unique problems which were:

1. Septic system failure.
2. Apparent uplift of the furnace into the first floor joists.
3. Main floor support beam lifted off pilasters.
4. Basement walls bulging inward.

The tension cracking in the three perimeter homes examined indicate an oblong type subsidence feature bounded by the Marinucci, Doty and Robert Hyest residences. It should be noted that these are only arbitrary boundaries, due to a lack of surface expression of subsidence beyond these homes, (i.e. except for a ground crack that developed behind the Doty residence in September 1990.

The problems at the Britton residence is indicative of compressional forces. The damage at the remaining residences is indicative of tensional forces.

## Methods of Investigation

### Field Reconnaissance

The field reconnaissance for the project consisted of three site visits. The initial site visit was performed on May 1, 1990 to examine conditions at the site. A second site visit was conducted prior to the drilling investigation, to more thoroughly examine the structures involved and the proposed boring locations. A third site visit was made upon completion of drilling to re-examine the structures and any new subsidence features as well as to examine a new and recent event that had damaged an adjacent residence.

### Subsurface Investigations

A drilling program was begun on July 18, 1990. A total of 12 drill holes were completed to aid in determining subsurface conditions. Split spoon samples of soils were taken and drill holes were advanced through bedrock with an HQ core barrel. The HQ core size was chosen to allow the use of OSM's borehole camera and distant viewing light source. Due to fracturing in the overburden and numerous clay seams, drilling was difficult and extended through September 10, 1990.

Upon completion of the drilling, all holes were examined and videotaped with OSM's borehole camera. Each hole was first videotaped using the axial lens, followed by the close-viewing right angle lens. It was anticipated that any mine voids encountered would then be viewed using the distant-viewing right angle lens and light source; however, due to the collapsed condition of the mine, this was only possible in one borehole.

## Subsurface Conditions

Based on the drilling and examination of rock core as well as examination of the borehole video the following subsurface conditions were found to exist.

### Overburden

A thin to moderately thick mantle of soil was encountered above the rock overburden at the site. The soil thickness ranges from approximately 2.1 m. to 5.5 m. The soils were generally fine to medium grained residual soils,

consisting of silty to sandy clays with varying amounts of rock fragments. The soils were soft to stiff in consistency and moist.

The rock overburden encountered during drilling generally consisted of alternating layers of limestone, shale, claystone, sandstone and coal. The rock quality designation or RQD of the rock core varied extensively ranging from very poor to excellent. Core recovery during drilling was generally excellent. However, some areas of poor recovery and poor RQD were encountered within the first 12.2 or 15.2 m. above the mine. This is attributed to small voids and broken rock strata.

**Fractures**

The rock core shows that the overburden of the No. 8 coal is fractured. High angled fractures are common. The fracture surfaces are rough to jagged, with some forming a "criss-cross" pattern while others consist of 2 high angle parallel fractures, generally less than 3 cm. apart. These irregular fractures are indicative of subsidence of the overburden and are most commonly found in the zone of caving and/or zone of breaking as shown in Figure 2.

Both naturally occurring and irregular fractures were observed in the core; however, the irregular fractures are more common. Both fracture types are iron stained, at depths less than 15.2 m. At greater depths, generally only the naturally occurring joints are iron stained, indicating that the irregular fractures are relatively recent. Iron-stain development (iron hydroxide) is dependent on the presence of oxygen. Fractures at greater depths, where oxygen is not as readily available as near the surface, would need a longer exposure time to develop a coating of iron hydroxide. Subsidence fractures, being a recent development, may be slightly iron-stained at the surface where oxygen is abundant, but, not at greater depths.

The irregular fractures occur primarily in the light gray fine-grained limestone of the Fishpot, Benwood and Arnoldsburg limestones. These limestones are extremely hard and brittle. In DDH-7, the top part of the Fishpot limestone core was shattered. The fragments consist of small angular, sharp-edged pieces along with some "cork screw" shaped pieces.

The borehole camera showed, in addition to numerous high angle fractures, low angle and horizontal fractures. The horizontal fractures generally occur along bedding planes in shaley layers or at changes in rock type. Low angle fractures occurring within limestone, sandstone, shale, and claystone beds are thought to be subsidence related. Most horizontal and low angle fractures occur in the rock strata lying above the Upper Sewickley Sandstone, while high angle fractures predominate below the sandstone. The horizontal separation ranges from 2.5 cm. to 5.1 cm. in width. This separation is indicative of the zone of downwarping of strata as shown on Figure 2.

**Analysis**

To analyze the mode of subsidence we looked at three possible causes:

1. Roof failure.
2. Pillar punching.
3. Pillar crushing or failure.

Any individual or combination of the above causes could have been responsible for the mine subsidence. The following is a description of each.

Pillar Punching

The potential for a bearing capacity or pillar punching failure depends on the engineering properties of the mine floor. Pillar punching occurs when weak floor strata exist in the mine and bearing pressures of the pillar exceed bearing capacity of the floor. When this occurs the pillar either settles or, in the worst case, the pillar punches through the floor strata. This generally occurs when the mine floor consists of underclay. However, the floor in this mine is a fairly competent limestone and/or siltstone unit. One boring, encountered a 1.2 m. thick seam of underclay. However, due to the low recovery in this particular section of the boring and the nature of the core, it is unclear whether this is truly an underclay or possibly mine gob. The RQD of core samples in most borings that penetrated the floor are good to excellent indicating that the floor material is competent. This indicates that pillar punching most likely did not cause or contribute to the subsidence.

**Roof Failure**

Roof failure occurs when the strata comprising the roof of the mine are not strong enough to span the distance between pillars. Degradation of roof strata can be accelerated by weathering and/or of groundwater levels in the mine.

Plots of elevation vs. RQD show that the rock quality of the roof strata is poorest within the first 12.2 m. above the mine. This is particularly evident in borings that were drilled through the rooms of the mine. However, even drill holes, which penetrated pillars, show this trend, though to a lesser degree than the borings into mine voids. It should be noted that this is likely due to both poor roof

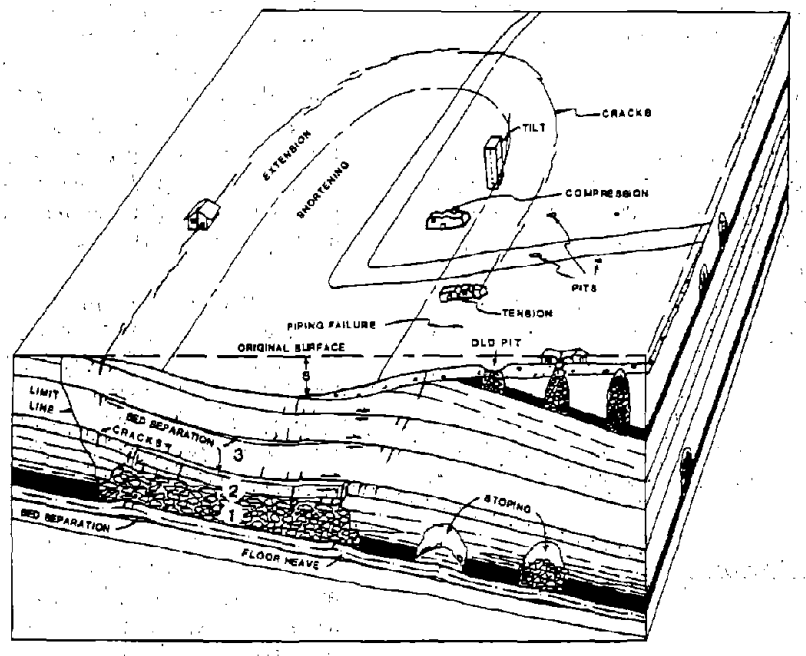


Figure 2. Conceptual diagram showing coal mine subsidence processes and deformation above extraction areas and mine openings. In downwarped zone: 1) zone of caving, rotation of caved fragments, and permanent bulking; 2) zone of breaking with little rotation of caved fragments, much cracking and bed separation, some permanent bulking; 3) zone of downwarping of strata as laterally constrained plates with local separation along planes of stratification, little or no permanent bulking. (AEG, 1987)

and pillar failure. This concentration of fracturing indicates that roof collapse has taken place, and that quite possibly this has contributed to subsidence at the site.

In the portion of the study area north of State Road 149, 10 holes were drilled through the No. 8 mine; 3 encountered coal pillars and 7 encountered mine passageways. The 7 holes that encountered passageways all show that the mine roof has fallen and that the mine is inundated. The height of the failure ranges from 0.37 to 0.82 m. above the top of the coal. The 2 holes drilled on the Watson property both encountered pillars at mine level.

The cause of the immediate roof failure is attributed to the nature of the claystone that forms the roof. This claystone is soft and slickensided, which makes it difficult to support. The height of the roof collapse observed during drilling, however, exceeds the claystone thickness; this

indicates that either the strata overlying the claystone are also incompetent and cannot span the support pillars or that the pillar spacing is inadequate to support the resultant span. As noted in the geology section, the Redstone Limestone (the first limestone above the mine) is split by shale and claystone partings. These abrupt lithologic changes form natural planes of weakness in the strata. The thinner beds of limestones in the base of the Redstone are probably not strong enough to support the overburden and the roof spalls upward to the more competent limestone beds above.

## Pillar Crushing or Failure

The strength of the coal pillars depends on the size, shape and compressive strength of the coal. Pillar failure occurs when the overburden stress on the pillar exceeds the strength of the pillar. This can occur when pillars are too small or when the strength of the pillar is reduced by weathering effects or other factors (additional surface loading, redistribution of stresses due to other pillar failures, etc.)

Based on the mine map, the topographic mapping, and the drilling results, factors of safety were calculated for the pillars. Average stress was calculated based on the following formula:

$$S_{AVG} = \frac{D \times 25.9\ kg/cm^3/m.}{1\text{-}\%\ Extraction}$$ (Given, 1973)

Where:

$S_{AVG}$ = Average stress
D = Depth to top of seam = approx. 64 m.
% extraction = 61%
25.9 $kg/cm^3/m$ = Overburden stress/m.

Based on this, the average overburden stress is approximately 41.6 $kg/cm^2$

To determine pillar strength the Holland-Gaddy formula was used:

$$S = \frac{K\sqrt{L}}{T}$$

Where:

S = Pillar strength $kg/cm^2$
L = Least lateral dimension of pillar (cm)
T = Thickness of seam or pillar height (cm)
$K = Sp\sqrt{D}$

Where:

Sp = compressive strength of cubical sample of coal in question.
D = Edge dimension of cube

Based on testing of samples obtained during drilling, as well as data from previous testing of samples from the Pittsburgh seam, a value of 5400 was calculated for K. The smallest pillars under the site had a least lateral dimension of approximately three meters. A value of 53.3 $kg/cm^2$ was then calculated for the pillar strength based on a least lateral dimension of three meters. Because of degradation of the pillars with age, as well as pillar fracturing induced by conventional mining techniques, 0.6 m. were subtracted from the least lateral dimension when calculating allowable pillar strength. This resulted in a pillar strength value of 47.7 $kg/cm^2$ for 3.0 m. wide pillars.

The resultant factor of safety is 1.15, a figure considerably less than the recommended factor of 2.0 using this formula [SME Handbook (1973)]. Because of this, we felt that any pillars equal to or less than 3.0 m. wide were of very questionable stability. We then back calculated and found that pillars equal to or less than 8.2 m. would be of questionable stability, and pillars greater than 8.2 m. should be stable. (It should be noted that because of pillar failure and redistribution of stresses to other pillars, pillars greater than 8.2 m. wide may also fail.) These pillars were highlighted on the mine map showing their distribution across the site. From this map it can be seen that many of the pillars with the lowest factor of safety lie on the west side of the main heading below S.R. 149 and parallel it throughout the site. The pillars with factors of safety between 1 and 2 make up the majority of the rooms below the site.

The drilling revealed evidence that some of the support pillars have been "robbed" or split, and are not as shown on the mine map. Two boreholes that should have bottomed in pillars, however, both encountered passageways. Another borehole, which encountered a pillar, shows only 1.25 m. of coal while adjacent drilling shows the coal to be approximately 1.83 m. thick. This may be due to a local roll in the coal seam, but the fractured nature of the coal more likely indicates that the pillar has failed and "crushed out".

## Discussion

Once a determination was made as to the likely cause of the subsidence, the data was re-examined to determine the possible location of initial pillar failure, as well as type and extent of the subsidence.

## Main Area

As mentioned previously, structural damage at the site varied. The Britton residence experienced compressional damage while the remaining structural damage in nearby residences was tensional in nature. This suggests that the Britton residence is near the center of the event within the compressional zone (see figure 2).

The possibility of the subsidence event beginning near the Britton residence was then investigated. An examination of the geologic sections reveals that the Sewickley Sandstone is the prominent competent unit underlying the study area. Numerous limestone units exist, but are interbedded with shales and claystones. Examination of the rock core and the borehole videos show the limestones to be fractured and brittle in nature. Graphic plots of RQD vs. depth show a consistent spike (increase in rock quality) in the Sewickley Sandstone with RQDs generally above 80%. This indicates that the sandstone is competent and has largely withstood the stresses produced by the mine subsidence. The same plots show varying RQDs in the limestones and shales, indicating that these units are not as consistently competent as the sandstone and that they have not withstood the subsidence-induced stresses as well.

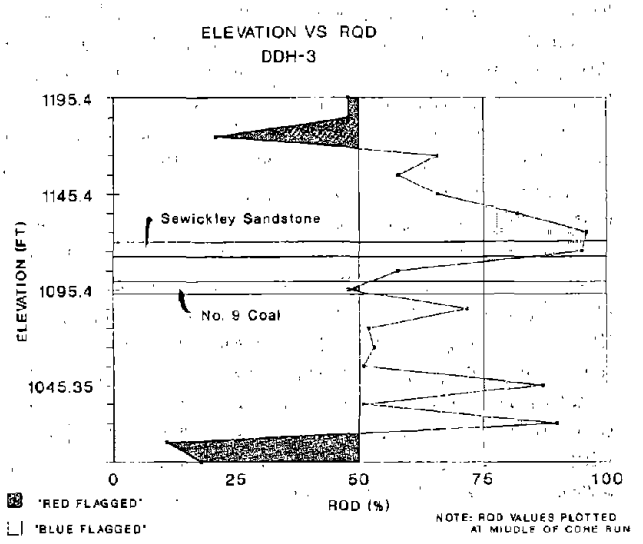


Figure 4. Example of graph depicting RQD values obtained from drill core versus elevation.

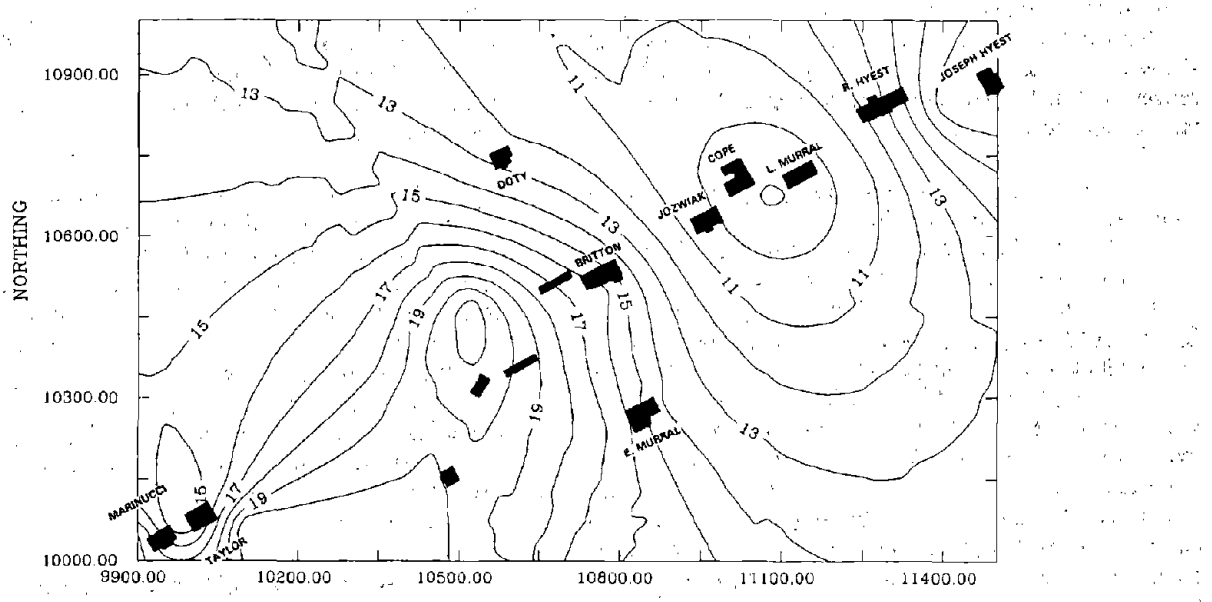


Figure 3. Map showing ratio of mine overburden thickness to Sewickley Sandstone thickness.

The ability of the Sewickley Sandstone to withstand stress is primarily dependent on its thickness as well as its location within the overburden. The Sewickley Sandstone is located within the zone of breaking or zone of downwarping, as shown in Figure 2. Because of this location, the sandstone is not subject to the higher degree of stresses that many of the stratigraphically lower limestones are. To determine the variations in thickness of the Sewickley Sandstone, we produced an isopach map of the unit. Values were obtained from the drill holes based upon sandstone thicknesses and the laterally equivalent siltstone. These values were input into the Surfer computer program and grid values were calculated using a kriging algorithm. Based on these grid values, the computer produced contour lines representing areas of equal sandstone thickness. The resulting map shows a thickening of the unit in the area surrounding the Jozwiak, Cope and Murral residences, while showing the sandstone to be thinnest in the region between the Britton and Taylor residences.

Because many of the pillars had factors of safety near unity it was determined that slight increases in overburden stress may cause pillar failure. Therefore, a second map was produced using the Surfer program. This map represents the thickness of the overburden between the top of the mine and the ground surface. Values for this map were obtained from drill holes as well as from the topographic map and elevations on the mine map. Again a kriging algorithm was used to calculate grid points; subsequently, contour lines representing areas of equal overburden thickness were derived. As expected, the map showed that the overburden was a somewhat subdued reflection of surface topography with greatest thickness near the Everett Murral residence.

The trends of the two maps suggested that a relationship exists between this sandstone thickness, the overburden and the surface expression of the subsidence. In order to determine this relationship, the two maps were combined into a second order map called the Overburden/Sewickley Thickness Ratio Map (see Figure 3). To combine the two maps, we again used the Surfer computer program. In this procedure, a matrix operation was performed in which the grid values from the overburden grid were divided by the grid values from the sandstone thickness grid. The resulting grid was then contoured. Contour lines on this map represent lines of equal overburden/sandstone thickness ratios. The areas with the larger values represent areas where the sandstone is the thinnest, and overburden is the greatest. Areas with the lower values represent areas where the sandstone is thicker and overburden is less. The mapping shows that the area slightly west of the Britton residence has a ratio of 21:1; that is 21 m. of overburden for every meter of sandstone. This indicates that while the overburden stress on the pillars is greater in this region, the competent sandstone unit is thinner and less able to provide ground support. The area with the largest ratios (18 to 21) corresponds with the area in which we believe the initial subsidence event occurred.

To further investigate the location of the initial event, plots of RQD vs. elevation for DDH-1 through DDH-10 were examined. The RQD values are indicative of fracturing in the strata, with low RQD values being representative of areas with the greatest fracturing. A "flagging" technique was used on the individual RQD vs. elevation plots. The flagging technique involved drawing vertical lines at 50% and 75% RQD on the individual plots. Areas less than 50% on the plots were "red flagged" and areas above 75% were "blue flagged" (see Figure 4). The 50% value was chosen as it represents the cut-off below which rock is considered poor. The 75% was chosen as it represents values above which rock is considered good. All pre-mining factors being equal at the site, the RQD values at the site should be fairly consistent and furthermore, below the zone of weathering, the rock quality should generally be good at this site. The latter is true due to the generally competent lithologies present and general lack of premining structural deformation in the units above the mine (both of which could affect the RQD).

RQD values in the uppermost portions were low, representing the weathered zone; most borings demonstrated higher values in the Sewickley Sandstone and markedly lower values in the No. 9 coal. The trends in RQD below the No. 9 coal then began to vary until most borings disclosed a dramatic drop above and continuing to the No. 8 coal horizon.

In addition, the flagging revealed some anomalous trends within individual drillholes. As expected, the borings above pillars exhibited the least "red flagging" and the most "blue flagging". The inference here is that the strata in these areas withstood the subsidence best and/or experienced the least stress. DDH-5 and DDH-10 have the most "red flagging", while DDH-3 has the least "blue flagging". This is indicative of areas where strata have endured the most fracturing. A closer examination of DDH-3 also reveals that the RQDs in this boring are considerably lower than any other boring at the site. An examination of the borehole videotapes confirms the RQD data.

Based on the RQD plots, borehole videotapes, Sandstone - Overburden Thickness Ratio, mine map analysis and surface damage, it is evident that the subsidence event at the site began in the room just south of the Britton residence and propagated radially outward.

## Conclusions

Based on our analysis, the following conclusions are presented:

1. Structural distress at the site is related to subsidence from the collapse of mine workings in the No. 8 coal seam.

2. Poor roof conditions exist in the mine. The shales, claystones and thin limestones are incompetent and spall with time. This collapse may be a contributing factor to the subsidence; however, due to the presence of the competent Upper Sewickley Sandstone member and the overburden thickness, roof collapse does not appear to have been the primary cause of subsidence.

3. The floor of the No. 8 seam is a fairly competent limestone unit and therefore pillar punching most likely is not a contributing factor to the subsidence.

4. Based on our calculations, the majority of the pillars at the site have a factor of safety of 2.0 or less. In addition, degradation of pillars with time likely further decreases pillar strength.

   Therefore it appears that pillar crushing has occurred and is the major cause of the mine subsidence.

5. Based on our superposition of the topographic map on the mine map (using state plane coordinates as the common control), the location of the structures on the mine map are not accurate. Structures as shown on the mine map may be mislocated by as much as 12.2 m. In addition, drill holes DDH-3 and DDH-7 should have penetrated mine pillars but encountered passageways. This indicates that actual pillar configuration appears to be different from that shown on the mine map.

6. Initial subsidence most likely began in the section just south of the Britton residence and propagated elliptically with a northeast-southwest trend.

7. Subsidence is generally of the "sag" type (Craft, 1990) with some individual room subsidence contributing to the overall damage.

8. Subsidence propagation at the site is likely related to the overburden - Sewickley Sandstone thickness. Areas where the overburden sandstone ratios are below 12 seem to have the least, if any, damage, while areas with ratios above 12 have the most.

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# SHAWNEE NATIONAL FOREST SUBSIDENCE STUDY[1]

Uldis Jansons and Diann D. Gese[2]

**Abstract:** The U.S. Bureau of Mines initiated a mine subsidence study in the Shawnee National Forest in southern Illinois in 1990. Abandoned mine lands may constitute a hazard and the U.S. Forest Service did not have sufficient records of the abandoned tripoli mines to ensure public safety. Identifying areas of potential subsidence is vital for land-use planning, land acquisition, and ensuring the public's safety. During the Bureau's study, approximately 200 inactive tripoli mine and prospect sites were located and 120 of these underground mines and subsidence holes were mapped. Some immediate results from the Bureau's study follow: (1) Undermined land was withdrawn from the timber base, (2) a road-closure point was moved and a land acquisition was reconsidered because in both instances the land was undermined, (3) an all-terrain-vehicle road site was relocated, and (4) an access road was closed because it was undermined and caved to within 4.5 m of the surface. Because mapping underground mines in active collapse is hazardous and because many mines were inaccessible, the use of reconnaissance geophysical methods is being considered to outline areas of potential surface collapse. Geophysicists are currently attempting to determine possible geophysical methods to identify undermined areas and areas of potential collapse.

**Additional Key Words:** tripoli, subsidence, Shawnee, Illinois.

## Introduction

In 1990, the Intermountain Field Operations Center of the U.S. Bureau of Mines initiated a mineral appraisal of the Shawnee National Forest in southernmost Illinois (fig. 1), as requested by the U.S. Forest Service. Early in this study, the Forest Service decided that a mine subsidence study of the Jonesboro district should be given the highest priority and requested the Bureau to determine the location and extent of the undermined areas. This district has been mined extensively for tripoli, and many of the inactive underground mines are caving, in some cases leading to surface collapse.

The Jonesboro district is in the southwest part of the Shawnee National Forest in Union and Alexander Counties. Tripoli has been mined here since the early 1900's, and much of the area is honeycombed with shallow, inactive mines. Tripoli is composed of microcrystalline quartz crystals or silica. It is fine grained (particle sizes usually range from 1 to 10 m), soft, porous, and friable. The tripoli deposits in the Jonesboro district occur as flat-lying beds, mostly in the Devonian Clear Creek Chert Formation (fig. 2). This formation consists of interbedded chert, limestone, and dolomite. Deposits are discontinuous and there is no specific stratigraphic horizon within the Clear Creek Formation in which the tripoli deposits occur. These deposits appear to be confined to areas where silica has been introduced and the carbonates leached. (Berg and Masters 1993.)

Historically, tripoli deposits were interpreted to be a weathering product of chert and limestone. Berg and Masters (1993) indicate that the introduction of hydrothermal silica to this sequence of sedimentary rocks was accompanied by leaching of all carbonates forming a sequence ranging from hard chert to friable

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

[2]Uldis Jansons, Chief, Economic and Engineering Branch, Diann D. Gese, Geologist, U.S. Department of the Interior, Bureau of Mines, Denver, CO, USA.

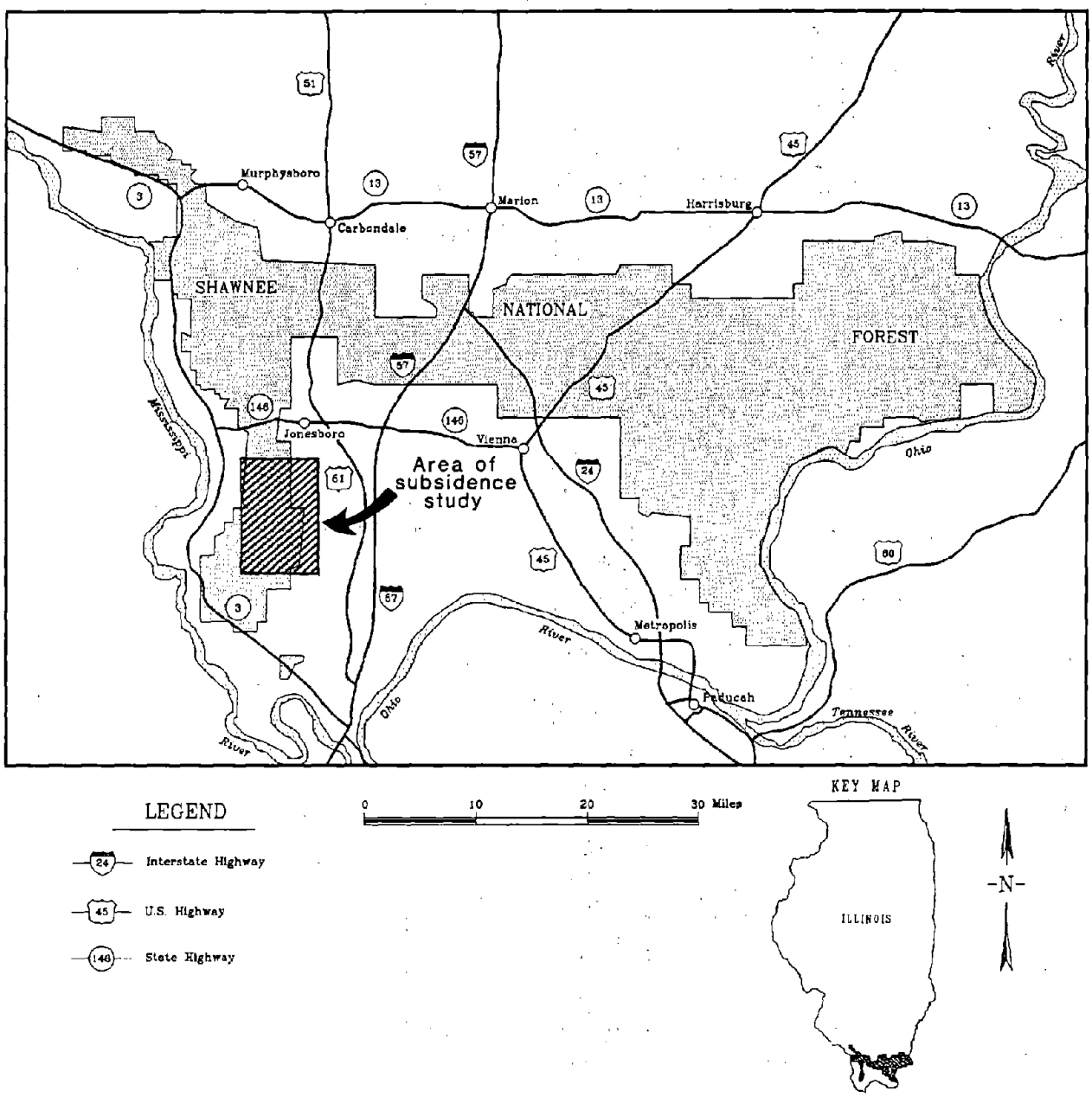


Figure 1. Index map of the Shawnee National Forest, IL.

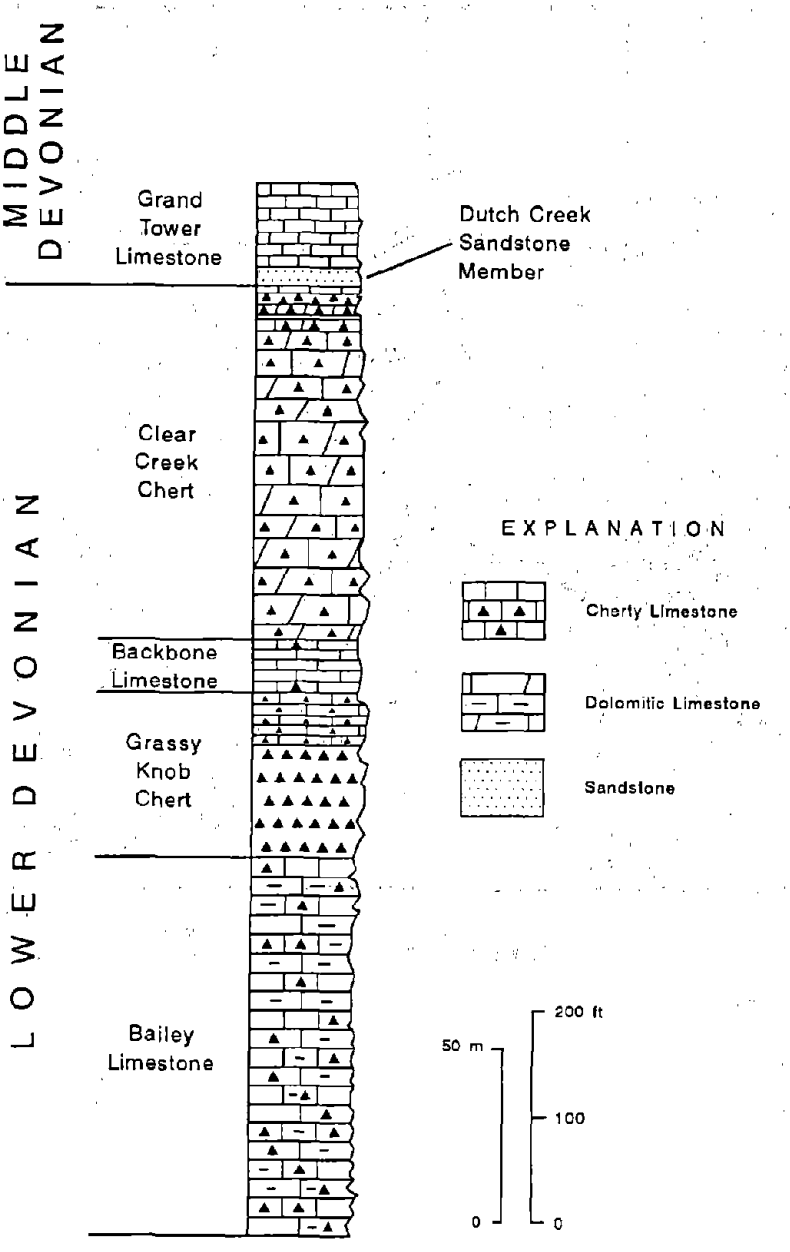


Figure 2. Generalized stratigraphic section for part of the Devonian in southwestern Illinois (Berg and Masters, 1993).

microcrystalline silica with thin clay beds. The nature and origin of the tripoli deposits is discussed in detail by Berg and Masters (1993).

Pulverized tripoli is used primarily as functional fillers and extenders in paint, plastic, and rubber and as mild abrasives. Southwest Illinois has been and continues to be a major domestic source of tripoli (U.S. Bureau of Mines).

This report is based on field work carried out by Brian J. Hannigan, Steven E. Tuftin, and Mark L. Chatman.

## Subsidence Problem

The abandoned shallow tripoli mines are caving, resulting in surface collapse of land to which the Forest Service holds the surface rights. In April 1986, about 60 m (200 ft) of a major Forest road was down-dropped 15 m (50 ft) in one catastrophic event. Numerous mines in other areas also had collapsed, resulting in deep, steep-walled holes and sheer highwalls which were unfenced and unposted. The number of additional potential collapse sites was not known. Surface subsidence could result in damage to personal property, personal injury, or loss of life.

The Forest Service was aware that it had a subsidence problem in the Jonesboro district. However, it did not have sufficient records or information concerning the number and location of old mine workings to ensure public safety or manage land use adequately in this area. It had no way of determining the number of abandoned mines, how extensive the undermined areas are, or which undermined areas were the most likely to subside.

At the early tripoli operations, many companies and individuals did not make mine maps because the shallow deposits were prospected by driving exploratory adits into steep slopes on the sides of flat-topped ridges; if a tripoli deposit of desired quality was found, the operations were expanded into a room-and-pillar underground mining system. Haulage ways were typically 4.5 to 6 m (15 to 20 ft) high and 6 m (20 ft) wide (Berg and Masters, 1993). The locations and configurations of the mine workings were, for the most part, unknown, few mine maps were available, and the mining records are vague or incomplete.

This was the situation in 1990, when the Bureau was requested to supply the Forest Service with subsidence data to enable it to make informed land-use and land-acquisition decisions. In addition to timber use, this area also has a high recreation use (hunters, hikers, all-terrain-vehicle users, etc.). Abandoned mines tend to attract the curious.

## Methods Used

To begin the inventory of the tripoli workings, the Bureau identified probable sites of past mining activity by reviewing published literature, examining aerial photographs, interviewing Forest Service and mining company employees, carrying out an aerial reconnaissance, and conducting surface traverses. Because virtually all the mined material was hauled away to a mill, there is little or no dump material outside any of these workings. The extent of underground workings was determined by inspection on a "one mine at a time" basis. Underground workings were mapped using the tape and Brunton compass method and electronic distance meters (fig. 3). Only those workings where the Forest Service holds the surface rights were mapped. A total of 50 field days involving two to three people was required to carry out this project.

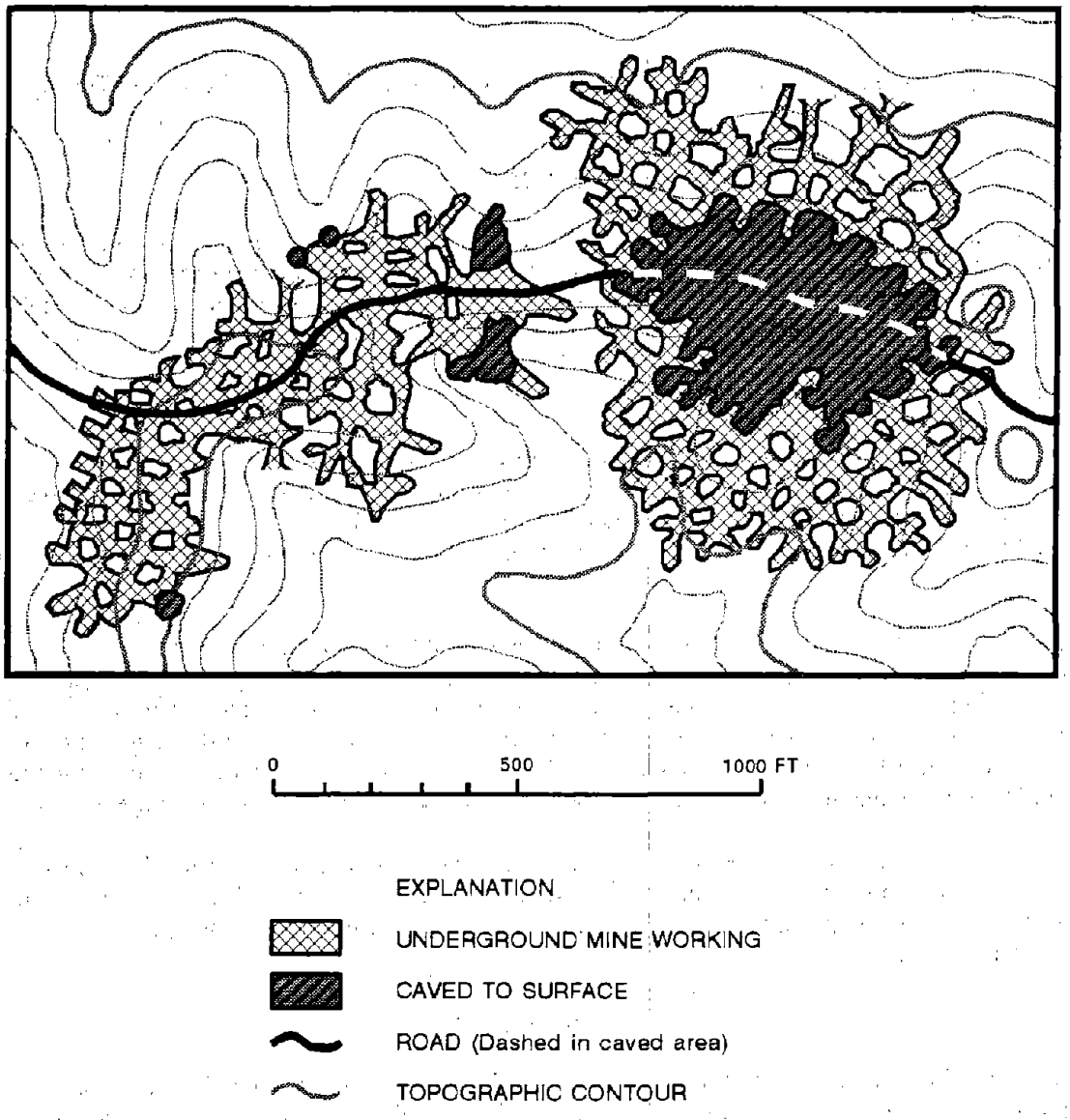


Figure 3. Map of abandoned tripoli mines showing surface collapse, Shawnee National Forest, IL.

## Study Results

Starting in the fall of 1990 and continuing through the spring of 1991, all potential mines sites were field-checked. About 200 inactive mines and prospects were identified and located on land of interest to the Forest Service (fig. 4). Approximately 120 underground mines and subsidence holes were mapped (over 30.5 km [19 miles] of workings)(fig. 3). The use of electronic distance meters expedited the work and allowed measurements to be taken without compromising personnel safety. Potentially hazardous conditions, such as winzes and areas with active roof fall, were identified and avoided; a premium was placed on personal safety. The mine maps were digitized using ARC/INFO geographic information system (GIS) software, plotted, and released to the Forest Service. For planning and presentation purposes, the mine configurations were superimposed on an enlarged topographic base (fig. 3). This presentation method is ideal for land-use planning purposes. Use of the GIS system enables the user to present the information at any scale and make revisions quickly and easily.

The Forest Service's immediate benefits from the Bureau's study were that: (1) previously unknown subsidence holes were identified, posted, and fenced, (2) undermined land was withdrawn from the timber base for future timber sales, (3) the use of an undermined service road was stopped, (4) the site of a proposed all-terrain-vehicle trail was relocated, (5) gates were erected at mine entrances to prevent ATV entry and to avoid disturbing bat populations, and (6) a land acquisition was reconsidered.

The Forest Service initiated a subsidence-remediation project which included fencing and posting of areas of immediate danger to the public. The Forest Service is also using the mine maps as a base for a 5-yr study to monitor additional subsidence. Data from this study are also available for future land-use decisions. To date, the Forest Service does not plan to fill any of the collapse holes because of the expense involved and because such action would disturb threatened and endangered populations of the Indiana bat, which inhabit some of the caves associated with these openings.

## Future Studies

Because mapping abandoned mines in active collapse is hazardous, and many mines in the Jonesboro district are inaccessible due to caving, flooding, and bad air, the use of reconnaissance geophysical methods is being considered to outline general areas of potential collapse. The Intermountain Field Operations Center is currently working with various U.S. Bureau of Mines Research Centers to determine possible geophysical methods to identify undermined areas and/or to assist in determining which undermined areas are more likely to collapse.

Two aspects of a proposed geophysical investigation are to: (1) determine a workable, cheap reconnaissance technique for outlining the general area of shallow underground workings that could lead to surface collapse and (2) where shallow workings are known, determine if surface subsidence is imminent, which would be important in areas accessible to the public.

Geophysical systems that are being considered include seismic, electrical resistivity, electromagnetic, ground penetrating radar, and gravity surveys. Seismic surveys have been used by the Bureau to detect, locate, and characterize a subsiding mine. Ground penetrating radar may work for outlining shallow openings (less than 30 m [100 ft]) in areas of no or very thin overburden. Gravity surveys may be used to detail gravity lows in areas of possible workings. Field methods would be tried to determine if they can detect underground openings and, if so, can they be adapted to airborne reconnaissance methods.

Testing for the immanency of surface subsidence would make use of a Bureau sponsored and tested seismic system (Munson, 1992). This system detected and measured characteristic seismic signals prior to surface collapse in tests conducted at both a hard-rock and a coal mine. Such seismic monitoring can accurately

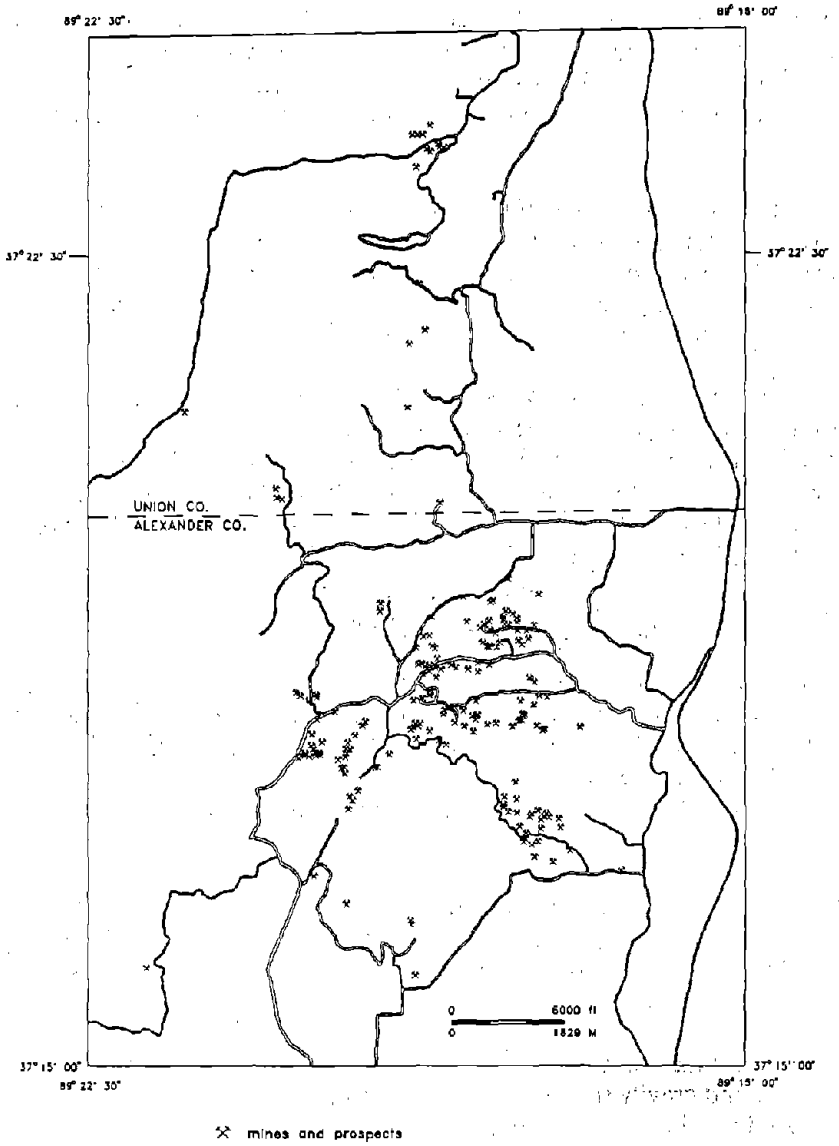


Figure 4. Area of subsidence study showing abandoned mine sites, Shawnee National Forest, IL.

locate the site and define the stage of subsidence and its areal extent. This system may be applicable in identifying areas of potential collapse in the Shawnee National Forest, which would allow the Forest Service to withdraw these areas from public access. An added feature of this system is that detection equipment can be placed to encircle areas of possible undermined areas, detect seismic activity, and prioritize areas for more focused work.

## Conclusions

A subsidence study of the Jonesboro district was carried out by the Bureau in order to provide a Federal land management agency with detailed information regarding the locations and extent of the undermined areas. This information is enabling the Forest Service to ensure public safety and manage the resources in this area much more easily, efficiently, and accurately. Geophysical tests could lead to the development of quick and relatively safe methods to evaluate other undermined areas and determine if they are susceptible to surface subsidence. These areas could then be blocked off from public access.

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# RESOLVING ABANDONED MINE CONDITIONS SUSCEPTIBLE TO SURFACE SUBSIDENCE[1]

K. K. Cohen and M. A. Trevits[2]

**Abstract:** Surface subsidence attributable to abandoned underground mines ranks high among abandoned mine land issues. Resolving the conditions that lead to unplanned mine subsidence remains a problem to geotechnical personnel charged with such tasks. A poor understanding of the conditions that lead to subsidence and geotechnical tools with limited subsurface resolving power compound the problem of determining abandoned mine land susceptible to subsidence. Drilling on closely spaced centers is a standard approach to characterizing an area of investigation. Alternative technologies that augment standard procedures have been tested to a limited extent, and are investigated here. Two case studies that represent different stages of subsidence assessment are presented, including an abandoned iron ore mine site in New Jersey investigated with magnetometry and an abandoned anthracite coal mine site in Pennsylvania investigated with seismic techniques. Results from the two case studies demonstrated that the alternative technologies of magnetometry and seismic techniques were valuable in assessing the extent of mining and in identifying different physical conditions within the mine level and overburden. A review of these and other subsidence investigations identified the following tasks as useful in assessing abandoned mine properties: (1) delineating the areal extent of mining, (2) developing models for mine and overburden conditions, (3) conducting field studies with appropriate alternative technologies to target instability zones, and (4) if warranted, implementing a program to monitor mine overburden stability through time. Alternative technologies identified to augment standard drilling practices include geophysical techniques, remote sensing, geochemical surveys, and displacement sensors.

## Introduction

Subsidence, mine fires, slope instability, and acid mine drainage are several issues that are now being addressed in mine land reclamation. Subsidence ranks high on this list of issues and affects abandoned coal, metal, and nonmetal mine lands (National Research Council 1987). Surface subsidence attributed to underground mines results from failure and collapse of mine overburden. When this failure propagates to the surface, it can be manifest as surface disturbances ranging from small potholes or sinkholes (a few meters to tens of meters in diameter, fig. 1) to complex and extensive collapse features such as subsidence troughs. Surface subsidence poses safety risks to residents of communities underlain by abandoned mines, and can damage cultural infrastructure including roads, utilities, and structures.

A wide spectrum of abandoned mine land subsidence research has been conducted to date, and has been directed by Federal, State, and other organizations. This research has focused on such topics as developing subsidence prediction technology (ICF Technology Inc./SRW 1992, Bauer et al. 1991, Walker and Trevits 1990, Munson 1987), underground mine delineation and void detection (Cohen et al. 1992, Friedel et al. 1990, Branham and Steeples 1988), and remediation (Dyni 1993, Burnett et al. 1993). These research topics address issues of subsidence risk assessment, prediction, prevention, and remediation. Few would disagree that, although much progress has been made by this and other research, we are far from answering all the questions that surround these issues. Conditions of the overburden rockmass and at the mine level are ultimately responsible for subsidence; thus, knowledge of these conditions are central to subsidence risk assessment, prediction, prevention, and remediation. Determining mine level and overburden rockmass conditions for

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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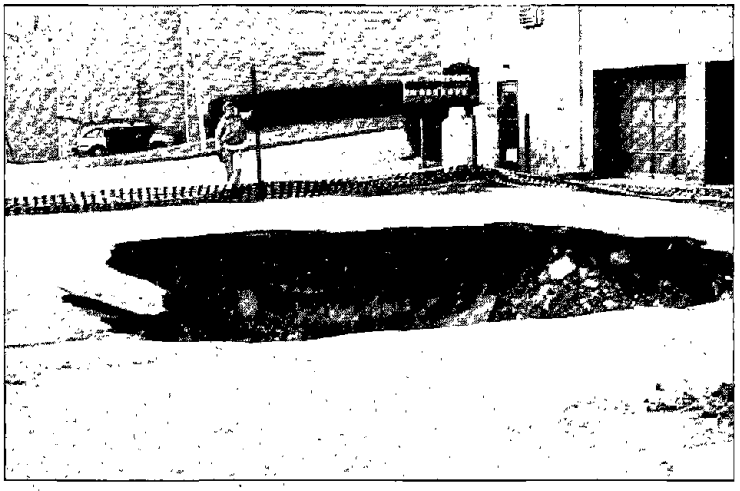


Figure 1. Surface subsidence attributed to collapse above an abandoned underground iron ore mine in northern New Jersey.

abandoned mine land susceptible to subsidence is the focus of this paper.

A review of past subsidence studies can provide geotechnical investigators with insight into activities useful for identifying subsidence-prone conditions; technology successful in completing these tasks can also be assessed. In the following, two recent subsidence studies conducted by U.S. Bureau of Mines researchers are presented. These studies used technologies not conventionally utilized to study subsidence-prone sites, and included an abandoned iron ore mine site in New Jersey and an abandoned anthracite coal mine site in Pennsylvania. From these and other similar studies, tasks that are helpful and often necessary to assess overburden rockmass and mine level conditions prone to subsidence are identified and discussed. In addition, alternative technologies such as geophysical investigations that can augment standard drilling practices (commonly used to assess overburden and mine conditions) are also identified.

## Case Studies

In the following case histories, abandoned mine conditions susceptible to subsidence were investigated through use of field studies. These field studies were selected because they integrated mining and geotechnical information with ancillary geophysical technology.

### Abandoned Iron Ore Mine

For several decades, a community located in north-central New Jersey has been plagued by subsidence attributable to more than 20 abandoned iron ore mines located under the town. Some of these mining operations predated the Revolutionary War. The mining operations used both open pit and underground mining practices for extraction of the high grade magnetite ore. Mine maps and other geotechnical information are limited and are often over a century old.

Mined property on which a school was built in the 1960's has generated considerable concern. Two subsidence collapse features developed on the property behind the school building in the 1960's. An available mine map for the area was combined with a recent map of the property (Shea and Pustay 1989); the resulting map suggested the subsidence was attributed to slopes used to access the underground workings (fig. 2). Prior to the building of the school, the surface had been graded, and any surface evidence of mining had been destroyed. The two subsidence sites were eventually remediated by excavating down to bedrock, emplacing a 30.5 m (100 ft) long concrete cap over the area, and backfilling to the surface.

This site warrants additional subsidence assessment due to its past history. Numerous boreholes have been drilled, and other geotechnical investigations have been conducted. The boundary of the mining is poorly known and, in part, determined by an historic mine map that is over a century old. Because the limits of the

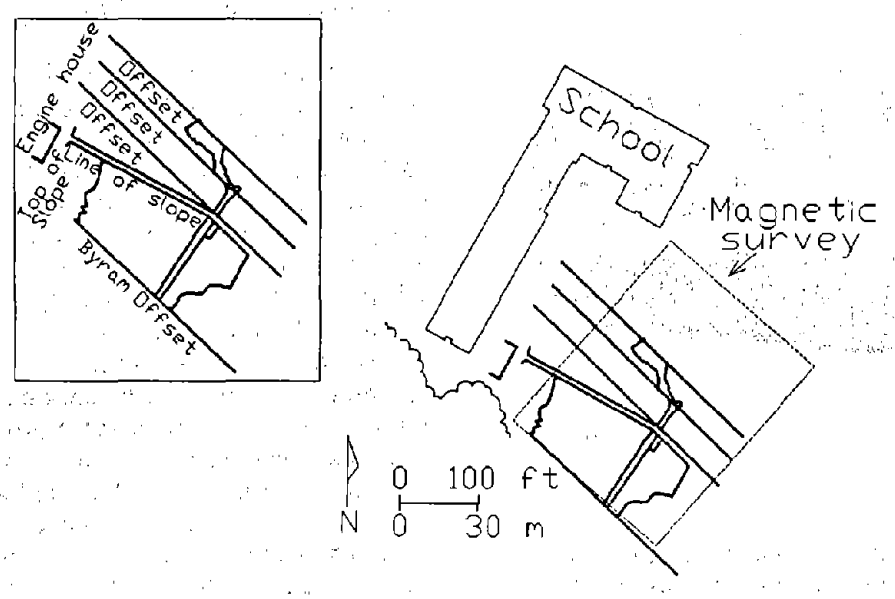



Figure 2. Property and mine map of abandoned iron ore mining site in northern New Jersey. Location of magnetic survey indicated on map. Inset shows historic mine map.

underground mining operations are poorly defined, delineating the areal extent of mining could aid in later targeting specific areas for detailed instability studies.

U.S. Bureau of Mines researchers used magnetometry as a tool to aid in delineating the areal extent of the underground workings (Cohen et al. 1992). This tool was chosen because the ore has an extremely high magnetic susceptibility, a physical property that determines a material's ability to become magnetized. It is proportional to ore grade in this case. The magnetic signature of a magnetic body of known dimensions and magnetic susceptibility can be computed, and extraction of ore from the body should distort this modeled signature. The magnetic signature (prior to mining) for the ore body at the site was computed, with the necessary model parameters obtained from historic mine records.

A magnetic survey was conducted over the area with a magnetometer using a gridded station array, and these data were contoured (fig. 3*A*). An anomalous magnetic field pattern was observed, and a profile through the map was compared with a profile for the expected, modeled magnetic field (fig. 3*B*). The field data confirmed that the site had been substantially undermined. Conclusions drawn from this study were that magnetic surveying of an area confirmed the suspected location of an abandoned iron ore mine; anomalously low magnetic field intensities identified in the survey were attributed to extraction of iron ore. Thus, the extent of mining and areas susceptible to subsidence were delineated. If necessary, more quantitative modeling of these data could be completed to determine the volume of ore removed from the ore body.

**Abandoned Anthracite Coal Mine Site**

A town located in the Northern Anthracite Field of eastern Pennsylvania was the site of several subsidence collapse features that developed in 1992. The affected area was an athletic field adjacent to an elementary school (fig. 4*A*). An old mine map indicated that mining occurred in the earlier part of this century by the room-and-pillar method (fig. 4*A*). The map suggested that all but four coal pillars had been extracted. The U.S. Office of Surface Mining Reclamation and Enforcement (OSM), Wilkes-Barre, PA, implemented a remediation program for the site. Eleven boreholes were drilled to determine subsurface conditions. These data suggested the mine was in various stages of degradation. In cooperation with OSM, the U.S. Bureau of Mines conducted a geophysical investigation at the site to locate large open voids. The planned remediation program for ground stabilization was to include grout injection through the boreholes. Large, open underground voids were required to effectively inject the grout into the subsurface.

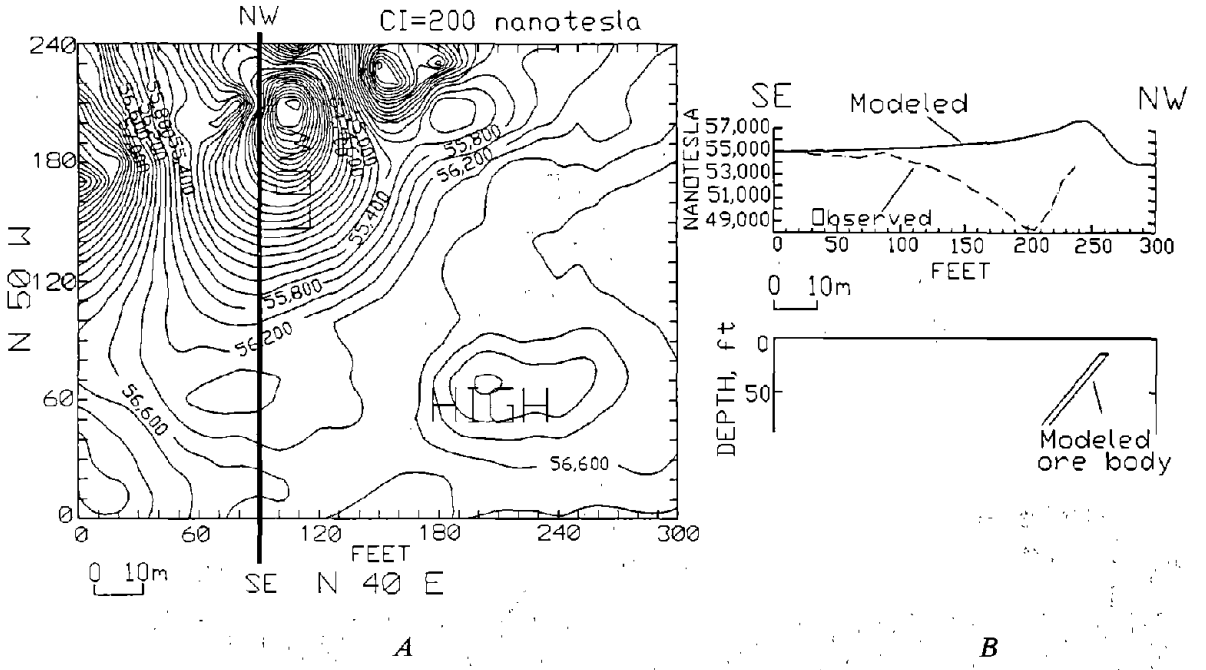
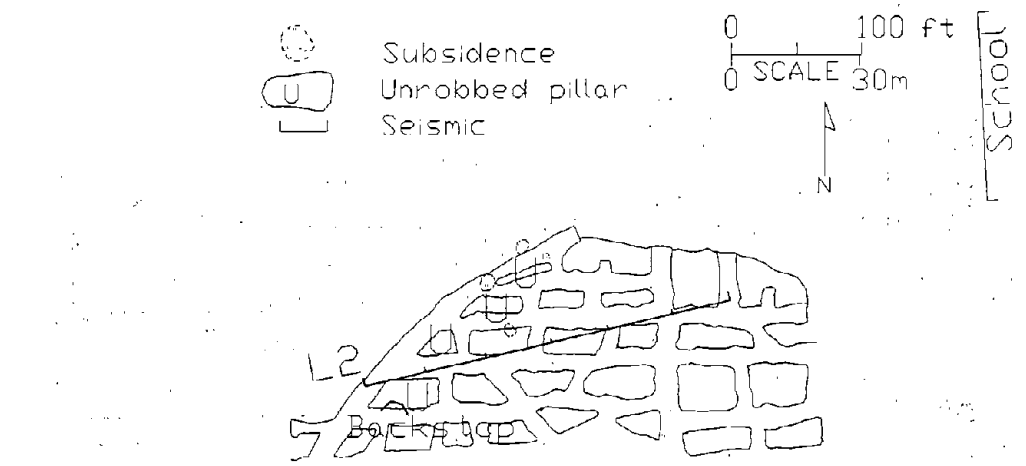



Figure 3. Magnetic field map (*A*) and observed and modeled profile through map (*B*).

A seismic method, the optimum common offset technique, was selected for investigation of the site. An engineering seismograph with signal enhancement, 100-Hz geophones, and a sledge hammer energy source were used for data acquisition. A walkaway test was conducted and extended from the unmined zone (as determined from the mine map) into the mined zone; recording parameters as well as shot-receiver offsets varied for this test. Shots were stacked to increase signal-to-noise ratio. Seismic noise, significant arrivals, optimum offset, and optimum recording parameters were identified from the walkaway test. An early coherent, high-amplitude arrival on the seismic record was observed over both the unmined and the mined zone. This was thought to represent the overburden-coal and overburden-open mine void interfaces from the unmined and mined zones, respectively. It was recorded between 40- and 50-ms traveltime at offset of 36.6 m (120 ft).

Several common offset lines were conducted (using 36.6-m (120 ft) offset) over the area of interest; one of these lines, L2, is provided for the following discussion. In line L2 (figs. 4*A* and *B*), only the westernmost portion of the line shows the 40- to 50-ms arrival thought to represent the overburden-coal and overburden-large open void interfaces. This arrival is not observed on the remainder of the record. It is thought that, for the remainder of the record, the mine and overburden conditions are chaotic; along this part of the seismic line it is likely that voids are small and are partly or totally rubble filled. Thus, they do not provide sufficient contrast in physical properties across this boundary and do not effectively return seismic energy to the surface. On this basis, it was decided that two zones existed over the area of investigation (fig. 5): (1) a zone of coal left in place or of large open voids and, (2) a collapsed zone of small voids that are partly or entirely rubble filled. Subsequent to this study, extensive drilling by OSM resulted in more than 20 boreholes and accompanying logs which confirmed these interpretations. Conclusions drawn from this study were that seismic surveying techniques proved effective in characterizing mined property prone to subsidence. Based on the seismic data, two distinct zones were identified at the site, a zone characterized by rubble filled voids and a zone characterized by coal or open voids. Subsequently, borehole data supported these interpretations and provided physical evidence for the presence of the two zones. Subsidence activity at the site indicated that the



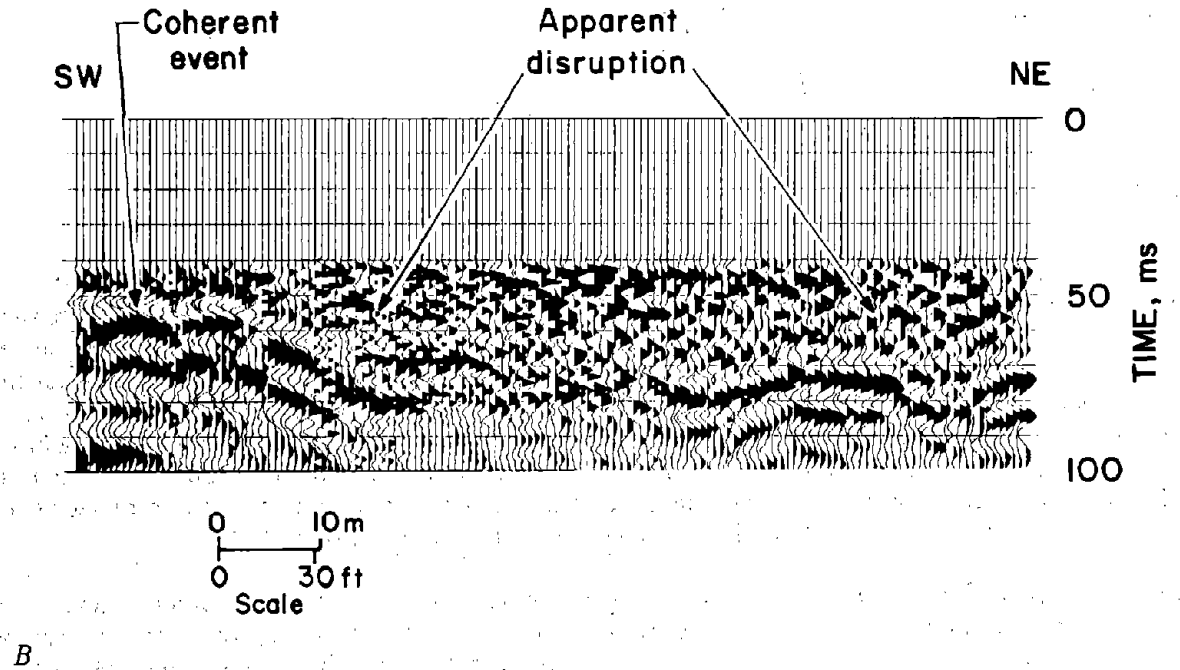




Figure 4. Property and mine map of abandoned anthracite mine site in eastern Pennsylvania and seismic results. *A*, Property and mine map with location of seismic line; *B*, seismic record from L2.

zone characterized by rubble filled voids was susceptible to ground instability.

## Assessment of Abandoned Mine Conditions

The cases studies discussed represent two different stages in what could be a sequence of steps designed to resolve mine conditions susceptible to subsidence. From these and other similar studies, steps toward identifying subsidence-prone mine conditions should target the problem area. Tasks that may be necessary or

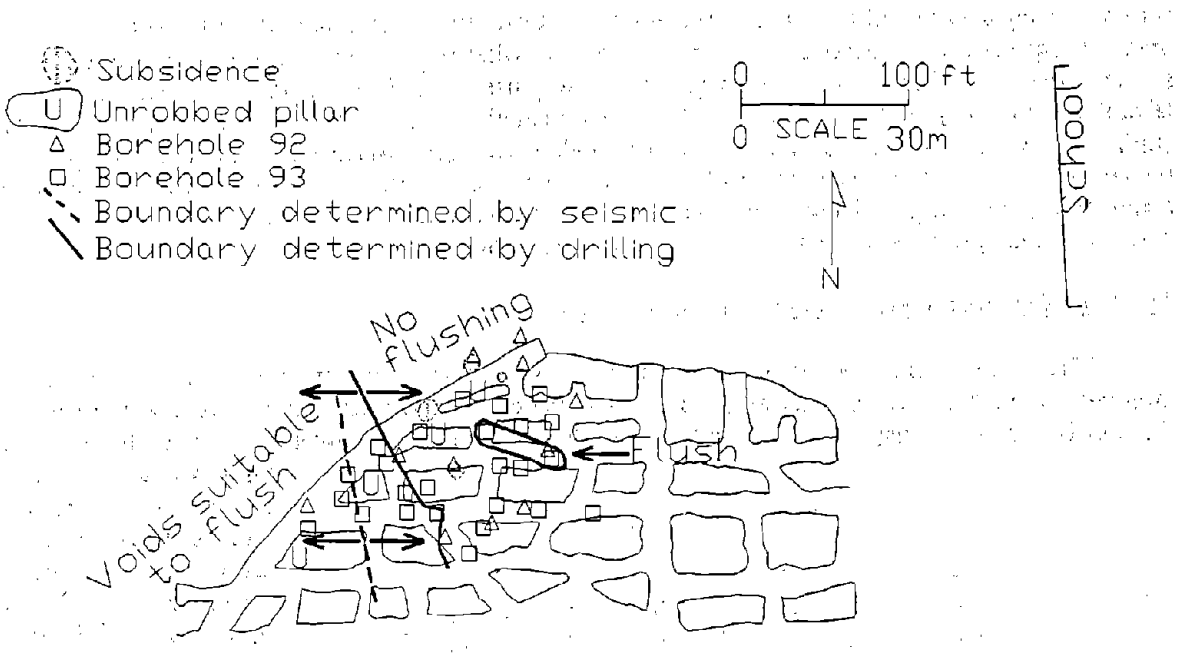



Figure 5. Area shown in figure 4 with the addition of the two zones delineated by seismic and drilling (postseismic) results.

that could prove helpful to accomplish this goal include the following: (1) delineate mined versus unmined areal extent, (2) develop reasonable models and mechanisms for subsidence in the study area, assign physical properties to the mine level and overburden rockmass conditions, and predict physical property changes that occur in these models as instability develops, (3) select appropriate tools to conduct field studies that will resolve these changing properties for the targeted area of interest, and (4) when prediction is required, implement a program to monitor mine overburden stability based on measuring precursory signals of subsidence through time. The goal of these activities is to identify the site prone to subsidence with respect to spatial coordinates and time. Several of these tasks are discussed in more detail in the following.

**Delineating the Extent of Mining**

Delineating mined from unmined areas is the first task that must be completed to target a subsidence-prone area if that information is not readily available. Historical mining records in the form of mine and other maps, as well as written and verbal accounts, are of great value and must be obtained when possible. If the area has not been developed, surface remnants of mining may still be observed. Field inspection of these sites can provide valuable information to reconstruct mining conditions. If the area has been developed, such field inspection is most likely of little value.

Even with mining records in hand, it may still be necessary to further delineate the extent of mining from other than historical information. Excavating and drilling are standard procedures used. Valuable information can be provided from these activities because they physically sample the subsurface; however, the information is spatially limited to the excavation or drill hole site. Additionally, these activities can be time consuming and expensive, and cause significant disruption to the ground surface.

Surface geophysical techniques (such as electromagnetics, resistivity, seismic, gravimetry, and magnetometry) and other remote sensing methods provide tools by which large regions can be imaged rapidly and without intrusion (Ackman and Cohen 1994). At sites of investigation, points, lines, and areas can be

surveyed using a variety of geophysical technologies. From these data, models for the subsurface can be derived and, subsequently, a determination can be made as to whether the subsurface is undermined. Numerous case studies exist in the literature in which geophysics was used to aid in delineating mined versus unmined areas (Branham and Steeples 1988, Dobecki 1988, McCann et al. 1987); these are often referred to as "void detection" studies. However, solution sets and subsurface models from geophysical data can be nonunique and often must be verified by some invasive method. Studies should be carried out by personnel experienced in data acquisition, processing, and interpretation. Additionally, cultural development or unique geologic or engineering conditions may preclude the use of some tools listed above.

## The Need for Subsidence Models and Mechanisms

The areal extent of mining may already be known, or a smaller area within the mined zone may be targeted for further investigation based on past subsidence history or signs of imminent subsidence. What if the task is to resolve unstable mine conditions in the study area? As suggested earlier, the next recommended tasks are to construct reasonable models of failure for the study area and assign specific physical properties to the mine and overburden conditions. Appropriate field techniques can then be selected for use in identifying failing mine and overburden conditions. These need not be labor-intensive activities, and a specific example follows.

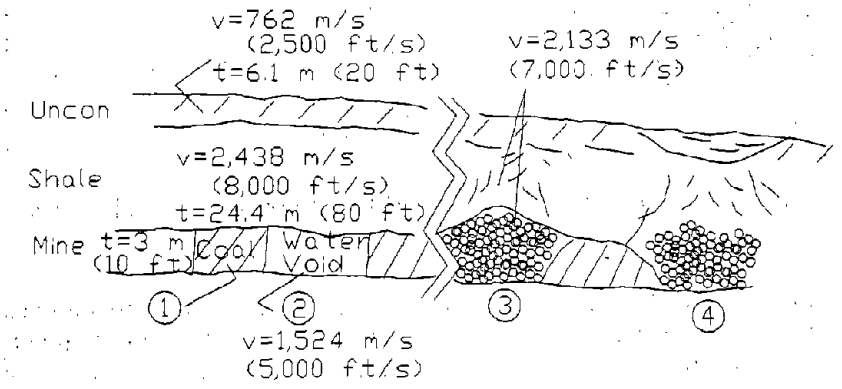



Figure 6. Model for abandoned mine in various stages of degradation.

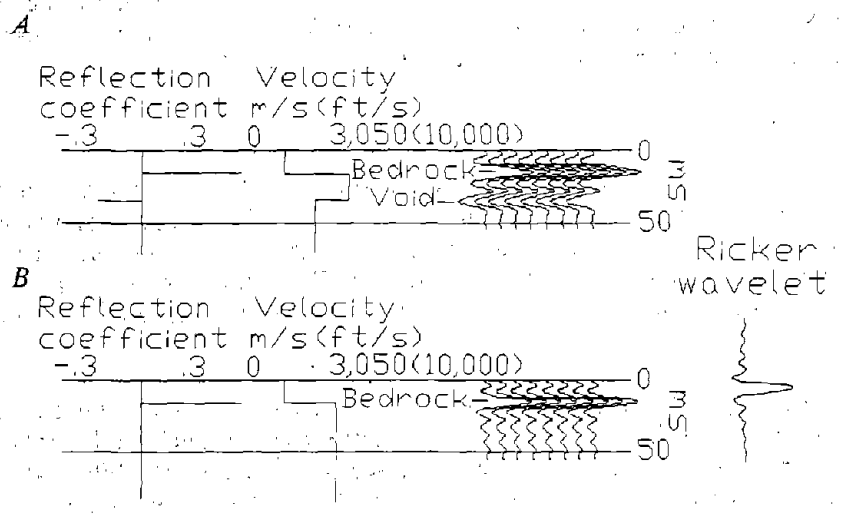



Figure 7. Synthetic seismograms with seismic response for zone 2 (*A*) and zone 3 (*B*) in figure 6. Densities not accounted for in these models.

In this hypothetical situation, an abandoned room-and-pillar coal mine is located at depth of 30.5 m (100 ft) (fig. 6). The seam mined is 3 m (10 ft) thick, and the mine is in variable stages of degradation in the model. The mine overburden is composed of a shale interval 24.4 m (80 ft) thick and unconsolidated material 6.1 m (20 ft). Four distinct zones of the mine (fig. 6; 1, 2, 3, and 4) can be characterized with different physical properties, and are discussed here. Seismic velocity was selected as a property to illustrate the utility of modeling. Resultant seismic responses for zones 2 and 3 are presented in figure 7; when compared, the seismic signals are clearly different for each zone. This model suggests that seismic data could provide valuable information

concerning variable subsurface overburden rockmass and mine level conditions. Reports in the literature indicate previous field studies have attempted to determine mine conditions using models and rationale similar to that described above; some have met with limited success (Schwarz 1992, Dobecki 1988).

## Alternative Technologies For Field Investigations

Alternative technologies to drilling which can be used to investigate subsidence-prone mine conditions are provided in table 1 and include such technology as geophysical techniques and satellite imagery. The suggestion here is not to replace drilling or excavating, but to augment these conventional practices. Identifying appropriate models of failure and consideration of the physical properties of the mine and overburden are tasks critical to resolving subsidence-prone conditions using alternative technologies. With this information, field studies of the area can be completed with the appropriate technology. These tools could have a twofold purpose: (1) to delineate the areal extent of mining and (2) to detect physical property changes in the overburden rockmass that are indicative of instability. As discussed earlier, site-dependent variables will control the selection of tools. Such variables include but are not limited to the following: depth of mining, resource mined, geometry and configuration of mine, condition and physical properties of mine, lithology and physical properties of overburden, topography, hydrologic regime, and cultural interference. These variables determine four controlling factors (McCann et al. 1987) for tool selection: depth of penetration, resolution, signal-to-noise ratio, and contrast in physical properties.

The examples provided previously emphasized geophysical tools, specifically seismic and magnetic techniques. Other nongeophysical tools exist and range from use of remote sensing (aerial or satellite imagery) to geochemical surveys such as soil gas sampling (Trevits and Cohen 1991, Sibley et al. 1988). Table 1 summarizes many of these techniques. The methodologies are divided into geophysical, remote sensing, geochemistry, and displacement sensors. Displacement sensors constitute technology to monitor and detect changes in overburden stability over time; many operate in a "real-time" mode. Field observations such as "Near Surface Indicators" (NSI) are not included here.

## Summary

A fundamental goal of abandoned mine subsidence investigations is to better target the zone with high risk for subsidence (spatial attributes); assessing mine overburden and mine level conditions is necessary to

Table 1. Alternative technologies with potential to assess subsurface mine conditions prone to subsidence.

| | |
|---|---|
| GEOPHYSICAL | REMOTE SENSING |
| Electromagnetic | Satellite imagery |
| Electrical resistivity | (e.g. visible and near infrared bands) |
| Magnetometry | Aerial imagery (e.g. black and white |
| Gravimetry | stereo photos, infrared photos) |
| Seismic (active surveying) | |
| Microseismic/acoustic emissions | DISPLACEMENT SENSORS |
| (passive monitoring) | Tiltmeters |
| | Time-domain reflectometry |
| GEOCHEMISTRY | Extensometers |
| Soil gas sampling (e.g. carbon dioxide, | |
| oxygen, methane, helium, radon, others) | |

complete this goal. Two recent subsidence investigations that used alternative technologies in conjunction with conventional practices to study the sites were presented. These and other studies provide insight into identifying tasks helpful in assessing mine level and overburden conditions. Tasks discussed include: (1) delineating mined versus unmined zones, (2) developing realistic models for mine and overburden failure and assigning physical properties to these conditions for use in the models, (3) identifying tools capable of resolving these physical changes and conducting field investigations with these tools, and (4) when required, implementing a program to monitor mine overburden stability based on measuring precursory signals of subsidence through time. A synopsis of these alternative technologies was provided in table 1.

It is emphasized that drilling and excavating to determine subsurface mine conditions will always comprise an important part in subsidence investigations. Physical sampling of the subsurface cannot be replaced. However, alternative technologies can be used in combination with these standard procedures. In many cases, off-the-shelf technology is currently available to help complete tasks such as delineating the mined zone or determining physical conditions of mine and overburden that could be indicative of instability. Technological advances are on the horizon for many of these tools and will, undoubtedly, increase resolution, depth of penetration, and signal-to-noise ratios. These future advances promise an increased toolbox from which to choose for assessing mine conditions susceptible to subsidence.

## Acknowledgments

Appreciation is extended to the community of Mine Hill, NJ for cooperation on the subsidence assessment work completed in that area. Cooperation with the U.S. Office of Surface Mining Reclamation and Enforcement (OSM), Wilkes-Barre, PA, enabled the work in the Northern Anthracite Field to be completed.

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# REMOTE SENSING OF ABANDONED MINE WORKS USING DOWNHOLE RADIO IMAGING TECHNIQUES[1]

David F. List,[2] James G. Schotsch,[3] and Gennaro G. Marino[4]

**Abstract:** Because of the potential for surface subsidence, electromagnetic radio imaging method or RIM was employed to identify old (prior to 1928), mine workings in a 300-foot deep by approximately 5-foot thick coal seam below a proposed construction site near Carlisle, Indiana. Cross-borehole RIM delineated the worked regions in the coal seam between pairs of boreholes using distinct variations in the measured signal strength (i.e., attenuation) of continuous radio waves transmitted at three different frequencies (422.5, 532.5, and 612.5 kHz). At the Carlisle site, 33 holes spaced 113 feet to 293 feet apart were drilled within an approximately 900,000 square-foot area. The holes were specifically positioned to calibrate signal attenuation for worked and unworked sections of the seam and then to verify the original mine workings map. Of the 43 pairs of holes measured, 36 clearly showed the presence or absence of mining using single-frequency data. In the remaining 7 cases, multiple-frequency data provided clear answers. In 3 areas, mine workings were detected that were not indicated on the original map.

## Introduction

Unmapped abandoned mines are hazards for underground mines, surface mines, and surface structures. Mining operations risk employee safety, flooding, decreased reserves and loss of revenue, if old workings are not predetermined. Surface structures can incur damage ranging from cracked foundations to a total loss of the structure. Mine engineers and architects can avoid risks by delineating unmapped mine works and adjusting operations accordingly (Schotsch and Sutton 1991). One method for delineating unmapped mine works is the radio imaging method or RIM.

RIM is an electromagnetic method that generates a subsurface image by mapping variations in wave transmission. To date, we have successfully used RIM in a variety of settings for a variety of reasons. We used in-mine RIM across production panels in coal, trona, and potash mines to image geologic features, including sandstone scours, faults, collapse structures, and lithological variations (Fry, et al. 1985, Hill 1984, Miller and Schreiber 1990, Stolarczyk and Fry 1989). We used cross-borehole RIM in proposed mining zones to provide long-term planning information (Miller, et al. 1986, Miller and Schreiber 1990). In the case study presented here, we used cross-borehole RIM, as an alternative to costly, closely spaced drilling, to locate abandoned mine works.

In this study, the survey site was targeted for building the Wabash Valley Correction Institution (WVCI) near Carlisle, Indiana. Because of the uncertainty of a mine map made in 1928 and the potential for subsidence above worked zones, RIM was employed to delineate worked and unworked areas of the seam. RIM measurements made at the site verified the accuracy of the map in all but three areas. In these areas, RIM detected mine workings not indicated on the original map.

## Radio Imaging Method - Theory

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The cross-borehole RIM, as used at Carlisle, works by transmitting an electromagnetic (EM) or radio wave at a specific (radio) frequency from a transmitting antenna in one drill hole to a receiver antenna in a second hole. The antennae are free hanging and, for the Carlisle survey, were lowered to a depth within the coal seam. Coal is typically much less electrically conductive than the surrounding roof and floor rock. This conductivity contrast or layering confines the wave, preventing most of the wave energy from vertically escaping the coal seam and extending the horizontal range of transmission. In this regard, the coal seam acts as and is called a *waveguide.*

Wave guide transmission offers two unique benefits to RIM and the problem of locating abandoned workings in coal seams. First, as state above, the confining of the signal in the layer allows for longer wave transmission than would occur if there were no a waveguide. And, second, because it is confined, the radio wave is very sensitive to electrical conductivity variations within the wave guide (i.e., the contrast between worked and unworked regions).

**Single Frequency**

Figure 1 shows a schematic of a RIM signal transmitted through a coal seam, the electric property of the coal seam, and how we analyze the recorded signal. Figure 1-A cartoons a continuous, single-frequency EM signal that is emitted from transmitter $T_x$ and travels through the coal seam waveguide to receiver $R_x$, where the amplitude of the signal, A, is recorded. In the figure, the line representing the wave thins from $T_x$ to $R_x$; this is an indication of the loss in signal amplitude.

Although the signal is essentially confined by the coal seam waveguide, transmission is not lossless. Some of the signal is lost or radiated into the surrounding rocks and some of the signal is absorbed by the waveguide media. Radiation loss into the surrounding rock is minor and we neglect it. As a result, the amplitude of the signal at $R_x$ depends on the intrinsic absorption or attenuation rate of the transmission path and the distance between $T_x$ and $R_x$ (i.e., termed "geometric spreading" which we discuss later).

Figure 1-B shows a representative attenuation rate of a coal seam waveguide. The shaded region represents the flooded mine workings and, since water has a relatively high electrical conductivity, it has a much higher attenuation rate than the unworked coal seam.

Attenuation rate in a coal seam waveguide is a function of the electrical conductivity of the coal and the contrast in conductivities between the coal and surrounding rocks (Hill, 1984, Stolarczyk and Fry 1989, Stolarczyk, 1990). The greater the electrical conductivity of the coal, the greater the attenuation rate. The greater the conductivity contrast, the smaller the attenuation rate. For analysis purposes, we calculate attenuation rate in units of decibels or dB per 100 feet.

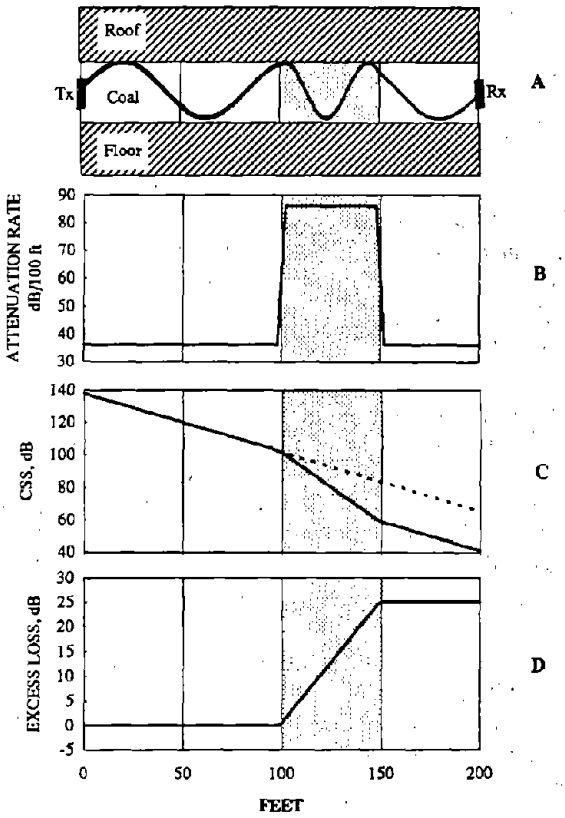


Figure 1. Diagram of the RIM transmission and analysis curves in a coal seam with water filled void (shaded area). A) EM wave in a coal seam, width of line indicates decreasing signal amplitude. Antennae are denoted by $T_x$ and $R_x$. B) Attenuation rates in coal seam and mine void. C) CSS, corrected signal strength of the EM wave as it passes through seam with workings present (solid line) and in absence of workings (dashed line). D) Excess loss of the EM wave in the seam.

[Note: dB = $20\log_{10}(A/A_o)$, where $A_o$ is a reference amplitude, which we define as 1 nano-volt].

Figures 1-C and 1-D show the results of the analysis of the measured signal strength at $R_x$. The signal is processed to provide the corrected signal strength (1-C) and the excess loss (1-D). The corrected signal strength or CSS is the measured signal strength corrected for geometric spreading. Geometric spreading is the reduction of signal amplitude with increased distance from the transmitter and its effect must be removed from the data. It occurs because the wave radiated by the transmitter horizontally spreads over an increasing wave front or area within the waveguide as the wave travels to the receiver. In the coal seam, the spreading is a linear function of the $T_x$-$R_x$ antenna separation when amplitude is cast in dB. The results of the removing geometric spreading is showed by the solid line, the corrected signal strength (CSS) in Figure 1-C.

In addition to the solid line, Figure 1-C shows a dashed line. This line is the projection of corrected signal strength in the absence of the increased attenuation rate due to the flooded, worked coal seam. We term the difference between the solid and dashed lines the excess (signal) loss. Excess loss is an indicator of the presence of the mine workings. If the ray path does not cross mine workings, the excess loss will be negligible; if mine workings are crossed, the excess loss will be high.

**Multiple Frequencies**

The previous example discussed single-frequency RIM. RIM can also be done in a multiple-frequency mode. In practice, a multiple-frequency RIM survey consists of a series of single frequency RIM measurements, each operated at a different frequency. However, analysis of the multiple-frequency RIM data gives more information than simply the direct sum of information from the individual frequencies. The information gain allows for predicting, not only the presence of anomalous features in a coal seam, as with single frequency data, but also the type and location of the features. This is important because for a setting in which mine workings run parallel to, but are offset from, the direct line between the measuring holes may appear in single frequency data to be the same as mine workings that cross the direct line. With multiple-frequency data, these two configurations of mine workings can be differentiated.

As implied above, signals transmitted between $T_x$ and $R_x$ are influenced not only by features along the direct line between the antennae (termed the "ray path"), but also by features near the ray path. The zone surrounding the ray path that influences the signals is called the zone of influence. In planar or map view, this zone is an ellipse with $T_x$ and $R_x$ as foci and the ray path as the major axis. The zone of influence ellipse, also termed the "scattering ellipse," is the outer boundary where signal amplitude is great enough to travel from $T_x$, hit a perfect reflector (i.e. 100% reflection) at this boundary, and still be detectable at $R_x$. The size of the ellipse is a function of geometric spreading, signal strength, but most important for RIM, frequency and attenuation rate. Higher frequency signals attenuate more readily and have smaller ellipses than lower frequencies. Overlaying scattering ellipses from different frequencies and eliminating the common region of both ellipses results in an elliptical annulus. This is the basis of multiple-frequency RIM surveys.

By comparing one frequency signal with another frequency signal, features within the elliptical annulus can be detected and isolated. To make this comparison, we define an attenuation index at two frequencies,

$$\text{Index} = [\alpha(f_2)/\alpha(f_1) - \alpha_s(f_2)/\alpha_s(f_1)]*100, \quad (1)$$

where $\alpha(f_i)$ is attenuation rate at frequency $f_i$, $\alpha_s(f_i)$ is the standard attenuation rate at frequency $f_i$, and $f_1$ and $f_2$ are the two frequencies of the analysis. The standard attenuation in equation (1) is defined at the attenuation under uniform conditions (i.e., no anomalous features within the zone of influence of either frequency $f_1$ or $f_2$).

**Multiple-Frequency - Theoretical Case Studies**

The utility of multiple-frequency data for detecting and distinguishing abandoned mine working configurations is depicted in Figure 2. Figure 2 shows the four possible configurations between the mine workings and the ray paths. To determine which case exists in the survey data, we determine values of excess loss and the attenuation index. The characteristics of ray paths and zone of influence for two frequencies of each case in Figure 2 are discussed below:

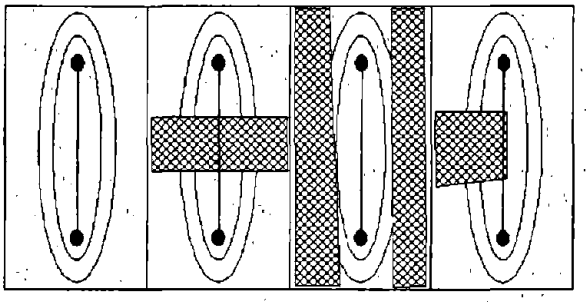


Figure 2. Map view schematic of four possible mine workings (hatched areas) ray path (solid line) configurations. Circles represent antennae locations. The larger ellipse surrounds the zone of influence in the coal seam for the lower frequency signal. The smaller ellipse surrounds the zone of influence for the higher frequency signal. A) Case A - unworked seam. B) Case B - workings crossing ray path. C) Case C - workings parallel and offset to ray path. D) Case D - workings into but not traversing ellipses.

**Case A** Ray paths in a uniform coal seam, characterized by low excess loss on both frequencies. Since neither zone of influence for the two frequencies encounter anomalous areas,the index is approximately equal to 0.

**Case B** Ray paths crossing mine workings, characterized by large excess loss. High frequencies are more affected than low frequencies, leading to a positive attenuation rate index.

**Case C** Ray paths close to but not crossing workings, characterized by elevated excess loss since the workings are in the zone of influence. Low frequencies are more affected by the workings more than the higher frequencies, which leads to a negative index.

**Case D** Mine working crossing the ray path and ending, characterized by slightly higher excess loss than case C, but lower than case B. The larger zone of influence of a low frequency signal can propagate around the disturbance easier than the smaller high frequency zone leading to a positive index.

## Survey Site and Operations

### Mine No 1.

The proposed WVCI site is in the northwest corner of the Illinois Basin just north of Carlisle in southwestern Indiana near the Indiana-Illinois border. Extensive coal mining took place here over the past century. The coal mine beneath the site was called the Mine No. 1 and was operated by the Carlisle Coal Mining Co. (Marino, 1990). The operation appears to have begun in 1908 and was abandoned in 1928 (Marino, 1990).

Although several coal seams exist beneath the site, only the Springfield No. 5 Coal seam was exploited (fig. 3). The seam's depth ranges from about 290 feet to 310 feet below the surface. The water table is 7 feet to 19 feet below the surface. The average thickness of the seam is 4.3 feet and is bound above by a thin black sheety shale overlain by limestone and below by fire clay or black shale. The only available map of the mine (fig. 4) indicates extensive mining in the southern half the site.

### Survey Design and Field Operations

The goal of this RIM survey was to verify the 1928 mine map. The analysis procedure requires distinguishing

ray paths crossing worked versus unworked. To do this we designed 45 ray paths into 3 groups: a calibration group with ray paths that crossed worked areas, a second calibration group away from worked areas, and a third group to verify the original mine map. We positioned 9 ray paths to cross mapped-worked areas and they are referred to as the **worked** ray path group. We positioned 10 more far away from worked areas and they are referred to as the **unworked** ray path group. We positioned the final 26 to verify the map and are referred to as the **verification** ray path group. Figure 4 gives the numbering of the ray paths and their separations.

To measure the 45 ray paths, 33 boreholes were drilled into the Springfield No. 5 Coal Seam. The data collection took 3 days to complete at a rate of 2 ray paths per hour.

Data collection for each ray path follows a prescribed sequence to maintain consistency. The first step is to center the antennae, operating at 422.5 kHz, in the coal seam at locations giving the maximum amplitude. This is done by by lowering the $R_x$ to the center of the coal seam then moving $T_x$ to the depth of maximum signal amplitude. The $T_x$ is then fixed and $R_x$ moved to find the maximum signal amplitude depth. Keeping the antennae at these depths, the amplitude and phase of the signal are recorded at three different frequencies (422.5 kHz, 532.5 kHz, and 612.5 kHz).

## Results

Of the 45 ray paths attempted at the site, two in the **unworked** group could not be recorded because the signal was below the noise level. This occurred because the ray path distances were too long for the frequencies. The two ray paths were 22 and 43 and had distances of 284 feet and 293 feet, which is 50 feet longer than the next longest path and well above the average ray path distance of 170 feet.

The remaining 43 ray paths were processed to obtain CSS for the three frequencies recorded. These values for ray paths in the **unworked** group were plotted versus ray path distance and a least squares regression line fit to them. The least squares line is the line from which excess loss was calculated (i.e. dashed line in Figure 1C). The y-intercept was 145 dB and the slope was 42.0 dB/100 feet for the 612.5 kHz signal. Using the regression lines, excess loss, and attenuation rate,

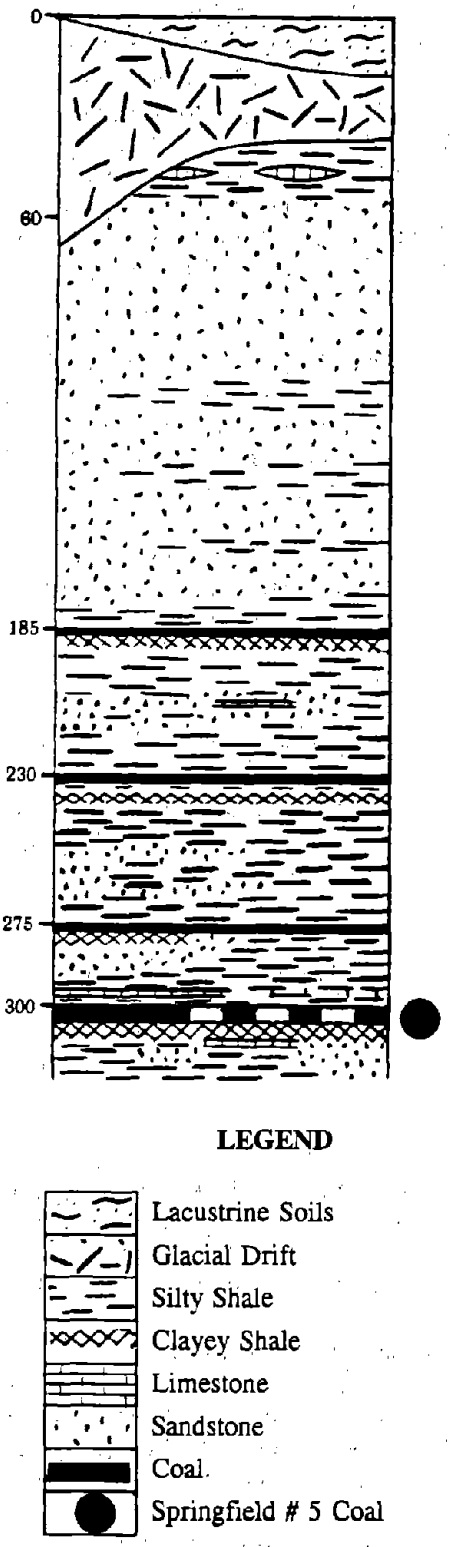


Figure 3. Generalized geologic column beneath the Wabash Valley Correctional Institution site near Carlisle, Indiana.

we calculated attenuation rate indices for all the ray paths.

Our calculations showed ray path excess loss values at 612.5 kHz ranged from -8.6 dB in unworked areas to 42.6 dB on ray paths crossing worked areas (fig. 5). These excess loss values were used in the single-frequency interpretation discussed below. Attenuation rate indices were calculated between the 612.5 kHz and 422.5 kHz data

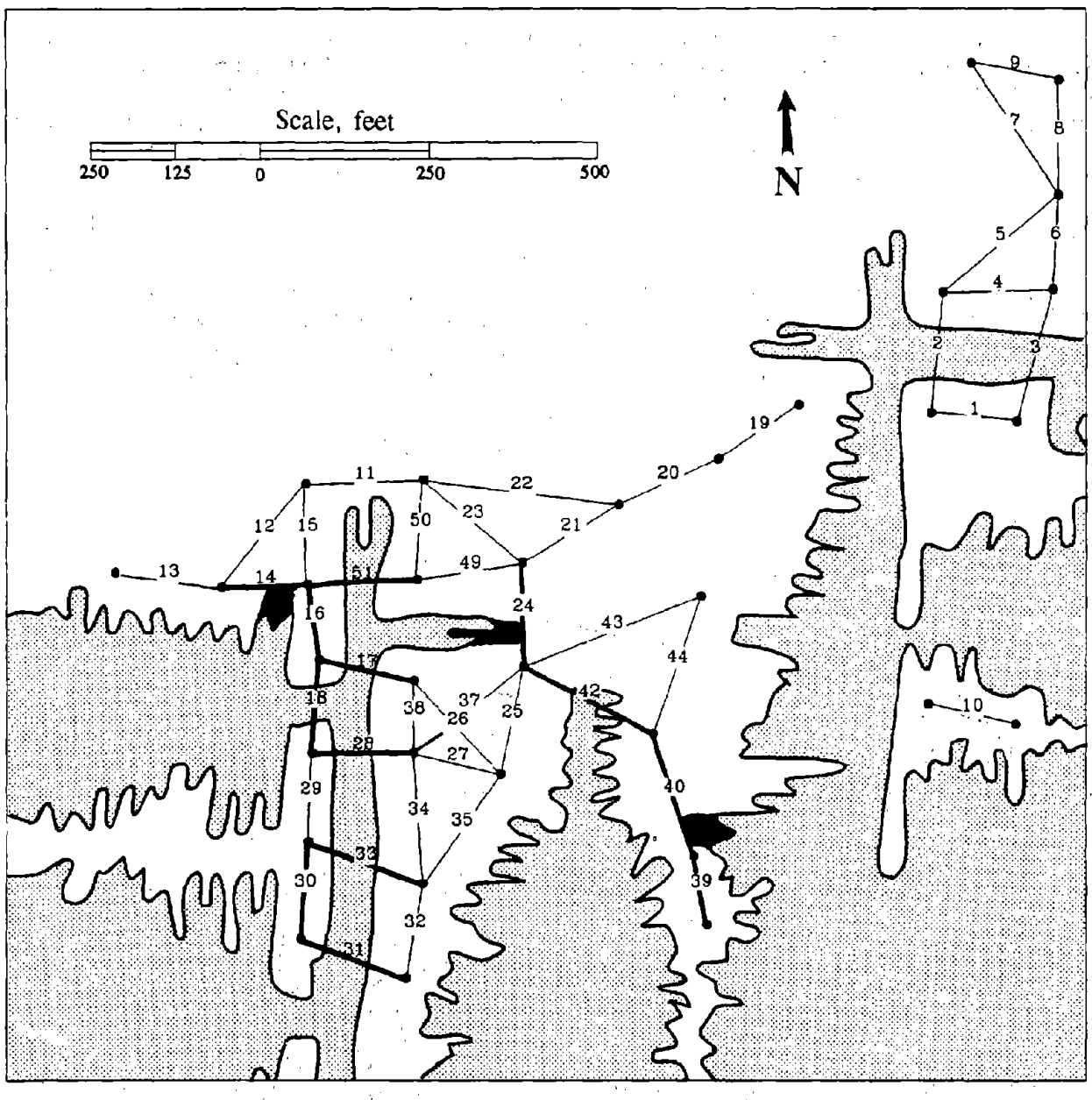


Figure 4. Map showing mine outline of the Carlisle Coal Mining Co. Mine No. 1, interpreted from the original 1928 map, and the RIM survey. Solid lines show ray paths identified by number. Shaded areas represent believed worked areas. Small circles are boreholes drilled through the coal seam. Multiple-frequency analysis ray paths are bold lines. Solid areas are mining identified by RIM and not on original map.

and used for the multiple-frequecy interpretation. The indices varied between +10 and -13. We found erratic values for the index on ray paths with measured signal strengths close to the noise level. [Note: Due to the success in processing the amplitude data, the phase data were not processed at this time.]

## Interpretation

### Scheme

We interpreted the results in a two-fold scheme. The first was a single-frequency approach that focused on the **worked** and **unworked** calibration groups to determine an excess loss value to use to separate ray paths in the **verification** group into those definitely not near mine works (low excess loss) and those that may be (elevated excess loss). The second step of the scheme was to take the ray paths in the **verification** group that may be near mine workings and apply multiple-frequency analysis to determine the position and possible type of anomalous feature causing the elevated excess loss.

### Single-frequency

From the excess losses of **worked** and **unworked** ray path groups we identified an excess loss discrimination level for the **verification** group. We recognized that this value should be less than excess loss in the **worked** group but greater than any in the **unworked** group. We determined a value of 10 dB with one exception. Figure 5 shows all but one ray path in the **worked** group to have excess loss values greater than 14 db and all in the **unworked**

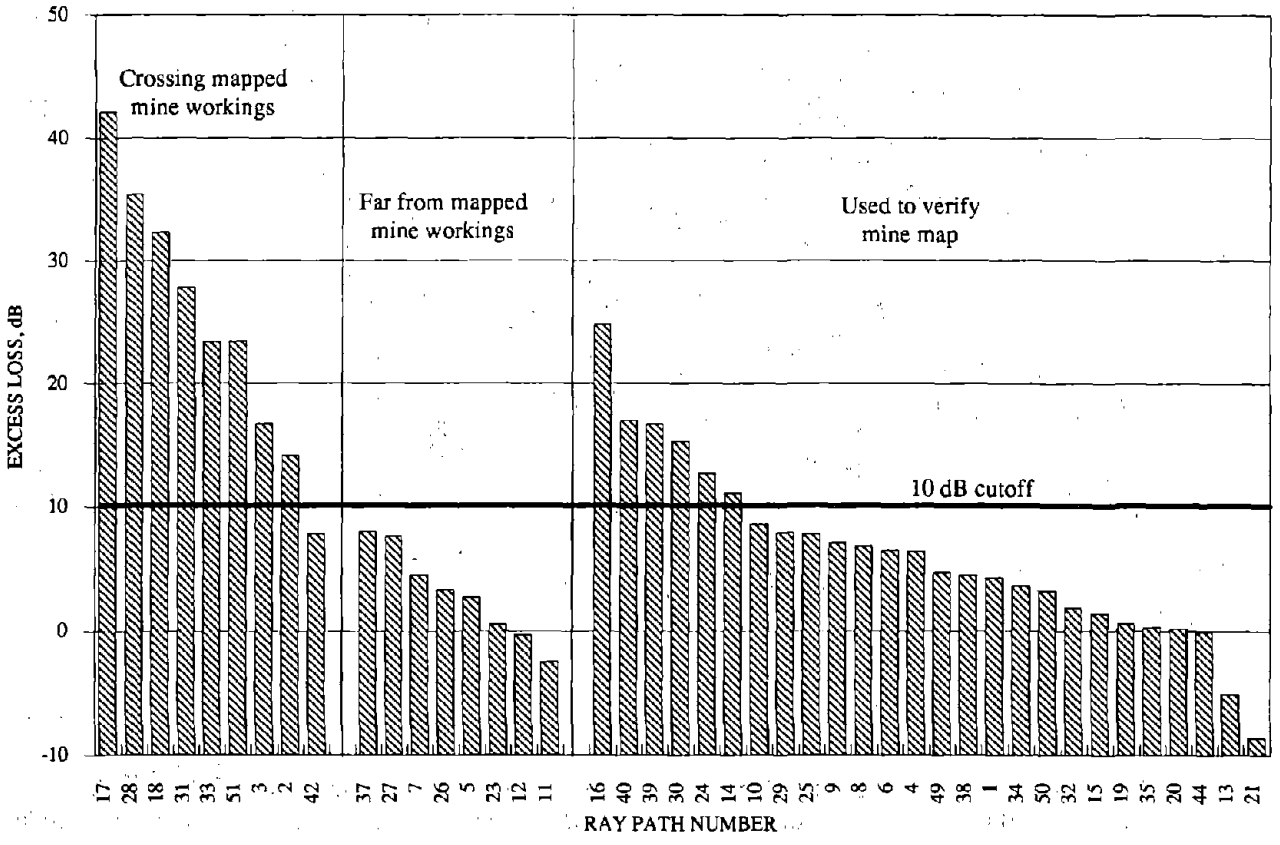


Figure 5. Excess loss of ray paths collected at the proposed Wabash Valley Correctional Institution.

group to be less than 8 dB. Ray path 42 (fig. 5) in the **worked** group is the exception with an 8-dB excess loss. This ray path only crosses the very end of a mapped-mine working (fig. 4), which explains the anomalous low excess loss.

Using the cutoff of 10 dB excess loss, we examined the ray paths in the **verification** group. Of the 26 ray paths, six have excess loss greater than 10 dB (nos. 16, 40, 39, 30, 24, and 14; fig. 4) indicating they are crossing or close to anomalous areas of the seam. We then investigated these six ray paths using the multiple-frequency approach to determine possible causes of the anomalous excess loss. Seven ray paths from the **worked** group were also analyzed in the same manner to provide calibration data for the interpretation. The seven **worked** group ray paths came from the same area as the anomalous **verification** group ray paths. These 13 ray paths are highlighted on Figure 4.

Multiple-frequency

In the multiple-frequency analysis of the six anomalous **verification** ray paths, we separated the ray paths into one of the four theoretical case studies presented earlier. We then adjusted the mine map, if necessary, to be consistent with this interpretation. First ray paths in the **worked** group were examined; then the anomalous ray paths in the **verification** group.

Worked Group. The main purpose for examining the worked group was to provide calibration information to allow us to recognize ray paths that have the potential for crossing worked areas (case B). Ray paths that cross worked areas should have some minimum excess loss. Just as in the previous section where we defined a cutoff value for excess loss to detect ray paths that may be near mining, now one is needed to recognize ray paths that may be crossing workings. Once this cutoff is determined we can confidently put the six anomalous ray paths in the **verification** group into case B, C, or D using excess loss and attenuation rate index in the following manner:

**Case B** Excess loss above the cutoff value and a positive index.

**Case C** Excess loss above or below the cutoff value and a negative index.

**Case D** Excess loss below the cutoff value and a positive index.

The mine map predicts six ray paths (nos. 17, 28, 18, 31, 33, and 51; fig. 4) should be case B ray paths and one (no. 42) are in case D. The six ray paths crossing mapped mine works all have excess loss above 20 dB (Table 1). Thus, the cutoff value will be defined as excess loss larger than 20 dB.

All but ray path 51 has a positive attenuation index as predicted by case B (Table 1). The field notes were re-examined for this path and it was found that $T_x$ calibration data taken from the borehole used for this path showed anomalous phase readings. This indicates the coupling between the $T_x$ and coal seam was different for this path than all others. This may explain the discrepancy in the index from what was expected. Even though we do not understand the exact physics of this situation, we can recognize data exhibiting these $T_x$ characteristics that may be providing misleading results using our current processing scheme.

Ray path 42 showed a much lower excess loss, 7.8 dB, (Table 1) than the six crossing mapped mine workings and a positive index, as expected if the mine map is accurate. This ray path demonstrates the utility of the multiple-frequency approach.

Verification Group. Using the 20 dB cutoff in combination with the attenuation rate index, the six anomalous ray paths in the **verification** group can be analyzed. Only ray path 16 exceeds the cutoff value (Table 1) indicating it could possibly cross worked areas. It, however, has a negative index placing it in case C. The map (fig. 4) is in agreement with this finding, which shows the ray path parallel to mine workings. Since the remaining five do not

have excess loss greater than 20 dB, they cannot cross working areas and, therefore, must fall into either case C or D.

Ray paths 39 and 30, due to a negative index, fit the criteria for case C. These findings are in agreement with the mine map (fig. 4) that shows mine workings paralleling these ray paths on both sides within 50 feet.

All ray paths discussed up to this point have verified the mine map. The last three ray paths discussed (40, 24, and 14) show discrepancies. Each of the ray paths exhibit a positive index and excess loss less than 20 dB (Table 1) placing them in case D, ray paths approaching but not crossing the ray path. However, the map indicates these ray paths should either be in case A or case C. To honor the RIM data, the following modifications to the mine map were suggested:

1) The workings mapped south of ray path 14 may extend approximately another 50 feet to the north beyond the current mapped location.

2) The workings mapped west of ray path 24 may extend approximately another 50 feet to the west beyond the current mapped location.

**Table 1. Interpretation of ray paths that either cross abandoned mapped mine workings or show excess loss above 10 dB in southwestern area of survey.**

| Path | Loss | Index | Case |
|---|---|---|---|
| **Worked** ray path group | | | |
| 17 | 42.1 | 8 | B |
| 28 | 35.4 | 10 | B |
| 18 | 32.2 | 4 | B |
| 31 | 27.8 | 6 | B |
| 33 | 23.4 | 1 | B |
| 51 | 23.4 | -4 | B |
| 42 | 7.8 | 6 | D |
| **Verification** ray path group | | | |
| 16 | 24.8 | -3 | C |
| 40 | 17.0 | 4 | D |
| 39 | 16.7 | -13 | C |
| 30 | 15.3 | -3 | C |
| 24 | 12.7 | 20 | D |
| 14 | 11.1 | 2 | D |

3) The workings mapped east of and at the south end of ray path 40 may extend another 10 or 20 feet.

## Conclusions

Multi-frequency radio imaging (i.e., single and multi-frequency analysis) was used to identify abandoned mine working in a 300-feet deep, 5-foot thick coal seam under the proposed building site of the Wabash Valley Correctional Institution near Carlisle, Indiana. Part of the motivation for the study was to verify an old (1928) map of the workings. The results of the study modified the map in three areas. In these areas, the RIM data indicated the worked area extended further than the map indicated. The RIM cross-hole program resulted in considerable financial savings since closely spaced drill holes would have been needed to verify the existing mine map.

For the study 33 holes with spacings between 113 feet to 293 feet were drilled in an approximately 900,000 square-foot area. The holes were specifically positioned to create calibration data for transmission thorough both worked and unworked sections of the seam that could be used to interpret data across sections of unknown mining history. Of the 43 pairs of holes measured, 36 clearly showed the presence or absence of mining using single-frequency data. In the remaining 6 cases, multiple-frequency RIM data analysis provided clear answers.

Verification of the mine workings, to the level obtained from the exploration program at the site, provided sufficient data to reduce the subsidence risk at individual structure locations. For example, without adequate proof, structures liable to be affected by mine subsidence would require designing for the worst case scenario. However with the subsurface information obtained, many combinations and magnitudes of ground displacement could be eliminated making the subsidence-resistant designs simpler and, in most cases, significantly less stringent. The total expense of this exploration effort was a fraction of the costs saved in foundation design under the worst case

scenario.

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# DEVELOPMENT OF A CONCRETE PLACEMENT DEVICE FOR SUPPORT OF ABANDONED MINES[1]

Mackenzie Burnett[2], Dr. Tahar El-Korchi[3], and James M. Burnett[2]

**Abstract**: Burnett Associates, Inc. (BAI), under contract to the U.S. Bureau of Mines, has developed a reliable and cost effective method of remote placement of point support columns in abandoned mines through boreholes to provide local support, especially under surface structures in subsidence prone areas. The development of the system to remotely build a concrete support cylinder in an abandoned mine required the coordination of mechanical system and concrete design. The mechanical system was designed to remotely place concrete in a cylindrical shape. The concrete was designed to meet the requirements of low slump with high enough strength to resist the forces applied by the ground above the mine. The support cylinder is fabricated through an 8-inch borehole by pumping concrete through a second 4-in pipe inside the borehole. The 4-in pipe has a flexible trunk on the lower end that is bent from the surface when it is inside the mine void. When pumping starts, the 4-in pipe is rotated and a spiral of concrete is placed on the mine floor. Operation continues until the concrete seals at the roof. A normal weight concrete as recommended by ACI 211 having a maximum slump of 1-2 in, a maximum coarse aggregate size of 1/2 in, and a minimum compressive strength of 5,000 psi was used. Cylinders have been fabricated to roof heights of 6 ft. There does not appear to be a technical height limitation. The concrete cylinder can support up to 40 $\times$ $10^6$ lbs when fully cured and filled with gravel, depending on cylinder diameter.

**Additional Key Words:** mine subsidence, cement, grout, roof support.

## Introduction

An improved reliable and cost effective method of remote placement of point support columns is needed to provide local support in subsidence prone areas especially under surface structures. A novel mechanical device to place very low slump concrete in an annular ring so as to build a cylindrical wall from floor to roof remotely through a borehole has been developed. The void in the center of the cylinder is filled with dry gravel. The system design for placing concrete underground in a controlled fashion is presented.

The conventional method of concrete or grout placement through boreholes has been to pump or drop concrete by gravity into the mine and to create a conical plug or seal by controlling slump through the mix design. Typically these methods use a large amount of concrete to obtain a fairly small area of support in the mine. The system presented here constructs a circular wall of concrete 10 ft or more in diameter. After this cylindrical wall has been built and the concrete has cured, the volume in the center of the annular wall is filled with a material such as gravel. This concept offers the advantage of lower cost than an all grout column and has the potential of providing full contact with the roof. The system was tested and developed at the Subsidence Abatement Investigation Laboratory (SAIL) at the U.S. Bureau of Mines.

## Concrete Placement System Description

The system concept is shown in figures 1 - 3. The design allows the remote placement of concrete from the

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surface of a borehole to create an annular wall of concrete in the mine void. Figure 1 shows the system installed at SAIL with placement of concrete at the beginning of the process. A section of hose (trunk) is mounted to the bottom of the delivery pipe. This trunk is fitted with hardware that allows it to be controllably bent from the surface after it has come out of the bottom of the cased borehole as shown in figure 2. The elevation and degree of bending of the hose is controlled using the system support device on the surface. The trunk hardware has been designed so that it will bend through a 120 degree angle. It has hinges to resist the torsion load on the trunk and stops so that it cannot be over bent. The support device also rotates the delivery pipe and trunk so that a continuous spiral of concrete can be laid to produce a concrete cylindrical wall that makes contact with the roof of the mine. The concrete is placed at a rate of 10 $ft^3$/min. The pipe rotates at approximately 3 rpm. Following construction of the cylindrical concrete support, the center of the cylinder is filled with gravel as shown in figure 3. The mechanical design of each system element is discussed in the following section. The concrete design description follows this in the cylinder performance and concrete design consideration section.

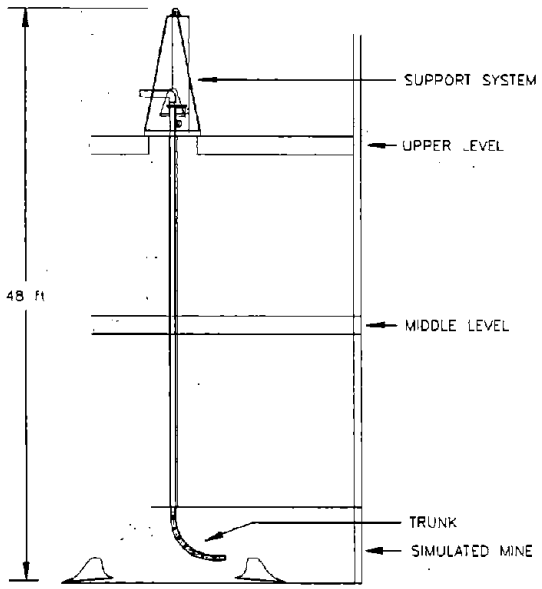


Figure 1. Controlled concrete placement system at SAIL.

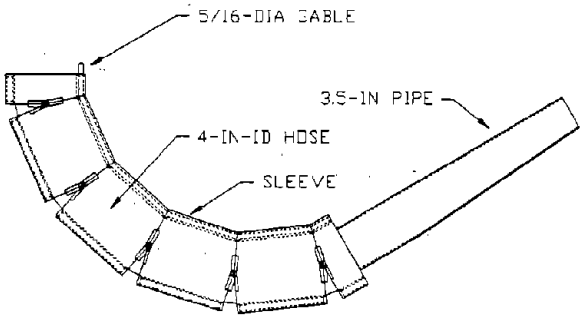


Figure 2. Trunk assembly.

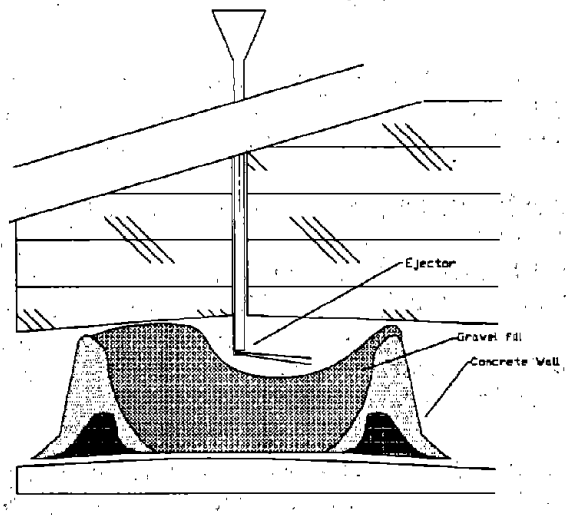


Figure 3. Filling the middle of the concrete annulus with backfill material.

## Mechanical Design of Concrete Placement System

The following sections describe the mechanical design of the concrete placement system. The concrete placement system consists of a series of mechanical elements that allow remote placement of low slump concrete in a mine void through a borehole. The concrete placement system is made up of a support frame, a pipe installation system, a series of modified 4-in pipes and couplings, a rotation device, and a flexible trunk.

**Support Frame.** The support frame serves several functions. The frame system makes it possible to install the delivery pipe in sections as it is lowered into the borehole, and it provides a means to hold the pipe and trunk in the borehole. It contains the mechanism to rotate the pipe and trunk and to bend the trunk into the desired shape for concrete placement.

Figure 4 is a drawing of the complete system installed over a borehole. The frame was designed to be easily assembled on a job site. This allows an efficient means of transporting the system from site to site. The beams are pinned together at the ends and to

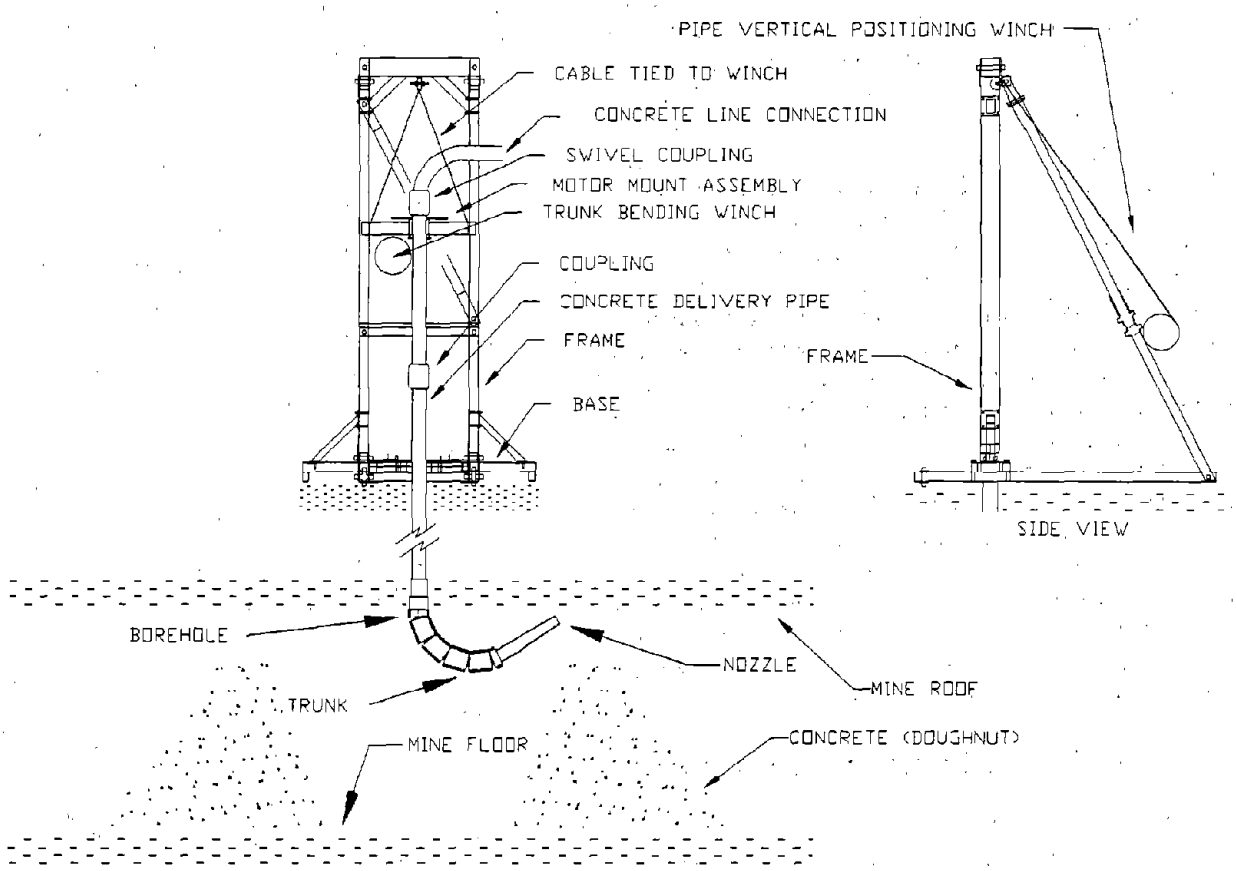


Figure 4. Concrete placement system.

brackets on the base of the frame. The motor mount assembly fits between the vertical frame members and is moved up and down by a winch and cable system. The frame is sized to handle 7-ft long pipe sections, but can be easily modified to accommodate longer sections.

**Pipe Installation System.** The pipe hoist fixture consists of cable mounts and a centered hole just large enough for the 4-in schedule 40 pipe to slip through. This allows the operator to slide the pipe into the hole and place a quick connect grooved coupling over the end of the pipe. After the coupling has been clamped on the pipe end, the winch is used to lift the pipe. In this manner the pipe is safely held in place while it is being connected to the pipe already in the borehole. The pipe hoist fixture is used only during pipe installation and removal and is removed from the headframe during concrete pumping. Sliding plates in the base of the frame close to support the pipe in the borehole while a section of pipe is added or removed.

**Low Profile Pipe Clamps.** Special, low-profile pipe clamps were designed for clearance in an 8-in borehole and to carry two cables alongside the pipe sections. One cable supplies the tension for trunk bending and the other will be used in the future for a borehole camera. The coupling is designed to grip a groove turned in the outside diameter of the pipe.

**Pipe and Trunk Rotation System.** This rotation system holds the motor and the pipe support bushing, and has slide grooves at the ends that ride ont the vertical members of the frame for torque resistance. The rotation system is used during operation only and is out of the way during pipe installation. It consists of a hydraulic motor, a section of 4-in pipe welded to a chain sprocket, a bronze bushing designed to fit over the pipe and slide in a fitting on the motor mount. The pipe winch, not shown, is clamped to the pipe just underneath the motor mount. This allows the cable that supports the bent trunk at the bottom of the borehole to rotate with the pipe assembly. A swivel pipe coupling is mounted to the top of the rotating feed pipe. The swivel coupling allows rotation of the trunk while pumping concrete. A pipe elbow is mounted on top of the coupling and connected to the hose from the concrete pumper.

**Trunk Design.** Figure 2 is a view of the trunk. The flexible hose (trunk) at the bottom of the concrete delivery pipe is the key to the system concept. It is obvious that the mechanism at the bottom of the pipe must fit through the 8-in borehole. The mechanical design of the trunk requires a means of bending the hose. The hose is bent after it is out of the borehole and in the mine void, by pulling on a cable attached to the end of the trunk with a hand winch. The required take-up is less than 5 inches. The trunk is made of several sleeves that are hinged together. These sleeves contain the tensioning cable, and the hinges resist torsion and bending forces applied to the end of the outlet nozzle.

The cable force required to bend the hose assembly for the 4-in hose is less than 2,400 lb. The end sleeves are designed with chamfered leading and trailing edges to insure that the assembly will go in and out of the borehole without hanging up. The borehole end of the hose has a standard high pressure fitting for a standard grooved 4-in pipe coupling. The outlet end of the hose has a specially designed nozzle that is directly connected to the hose. The nozzle tapers from 4-in pipe to a D-shaped outlet with a nominal diameter of 3.5 inches.

## Concrete Design

**Strength Considerations of Concrete Support.** The Strength considerations for the concrete conical barrier will be twofold:

1. How soon after initial placement can the barrier be backfilled with gravel?
2. What is the overburden load that the concrete cylinder can support?

Initial backfilling time. The curing time before initial placement can begin is calculated by determining the active lateral earth pressure on the concrete wall from the gravel alone. The active earth pressure creates a tensile stress on the concrete cylinder. The concrete must develop enough tensile strength to hold the gravel before it can be filled. The cross-section of the conical concrete support is shown in figure 5. The lateral earth pressure induced by the gravel and the overburden surcharge due to subsidence will be modeled using the Rankine theory and calculated as follows.

$$P_a = \frac{1}{2}\gamma_s H\ K_a \frac{1}{144} \qquad (1)$$

Where: $P_a$ = lateral earth pressure lb/in$^2$
$\gamma_s$ = gravel density lb/ft$^3$
H = height of wall
$K_a$ = active lateral earth pressure coefficient $\approx 0.307$ for $\phi = 32°$ (loose gravel)

With a gravel density of 100 lb/ft$^3$, and an internal wall height of 3.5 ft, the active lateral earth pressure from the gravel alone is only 0.4 psi.

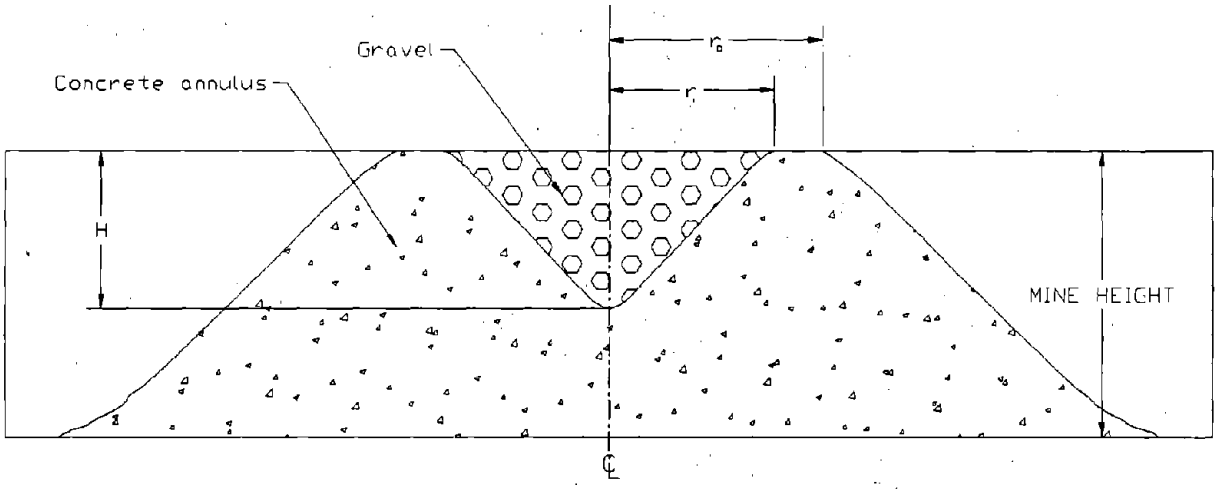


Figure 5. Cross-section of the conical concrete support with gravel.

The tensile strength of the concrete support during backfilling especially at the early ages of concrete curing is only 10% of its compressive strength. The conical concrete support may be modeled as a thick walled cylinder and the radial and tangential stress $\sigma_r$ and $\sigma_t$ respectfully are given as shown by equations 2 and 3. The maximum tensile and compressive stress occurs at $r = r_i$. The compressive stress due to the active lateral earth pressure is less than 1 psi and the tensile stress was calculated to be 1.6 psi. The concrete will easily achieve the required strength after one day of curing. Therefore, the conical concrete support may be backfilled as early as 1 day after the concrete has set.

$$\sigma_r = \frac{p_i r_i^2}{r_0^2 - r_i^2}\left(1 - \frac{r_0^2}{r^2}\right) \tag{2}$$

$$\sigma_t = \frac{P_i r_i^2}{r_0^2 - r_i^2}\left(1 + \frac{r_0^2}{r^2}\right) \tag{3}$$

where

$p_i$ = the internal pressure simulated by the active earth pressure
$r_0$ = outside radius of cylinder top
$r_i$ = inside radius of cylinder top

Overburden load. The total overburden load that the cylinder can support is the sum of load that can be supported by the gravel and the concrete cylinder wall. The concrete cylinder wall support capability is calculated by determining the area of the cylinder top and multiplying by the compressive strength of the cylinder as follows:

$$Load\ (lb) = C_s(\pi r^2_0 - \pi r^2_i) \quad (4)$$

Where: $C_s$ = compressive strength of concrete psi

The compressive strength of this concrete is between 4,000 and 5,000 psi. The inside radius at the top is 42 in, and the outside radius at the top is 54 in for a typical cylinder. Therefore, the load carrying capacity of this cylinder is between 14,500,000 and 18,100,000 lb.

The load carried by the gravel is limited by the tensile strength of the concrete since the gravel exerts pressure on the inside wall of the cylinder. Therefore the maximum load that the gravel can carry is determined by calculating the pressure placed on the inside wall of the cylinder due to the surcharge and active lateral pressures from the gravel. This load is calculated using the following equations:

$$P_{it} = P_a + P_s K_a \quad (5)$$

Where: $P_{it}$ = total lateral internal pressure
$P_s$ = surcharge pressure from overburden

At large values of $P_s$ the active earth pressure $P_a$ becomes insignificant. Equation 3 provides the tensile stress on the concrete as a function of total internal pressure. The maximum tensile strength of the concrete is expected to be 1/10th of its compressive strength. Therefore the maximum overburden pressure that the gravel in the structure will hold is 284 psi or 1,500,000 lb total load. The total load that the filled concrete cylinder will carry is the gravel load plus the cylinder load. The total is between 16,000,000 and 20,000,000 lb depending on the compressive strength of the concrete.

## Concrete Mix Design

Concrete mix designs were conducted during the development at a concrete plant. Several trial mixes were conducted to obtain a low slump pumpable concrete using local materials. The trial mixes are presented in table 1. The final mix was selected based on stiffness (low slump) and potential flowability (sufficient amount of mortar) and pumpability. Several admixtures were used. A water reducer was used to maintain a low water-cement ratio and to allow for water adjustment at the job site. Experience has shown that fly ash added to the mix will increase slump loss due to the absorption of water, especially at high pumping pressures. Indications are that rounded aggregate will pump more easily than crushed stone, with a corresponding small loss in strength. The retarding admixture was used to prevent any premature setting of the concrete in the pipe system and to allow sufficient time for cleanup in the mine roof simulator. The final specification for the mix design is presented in table 2.

## Testing

The point support system was tested at SAIL. The SAIL facility used for this portion of the program consists of a tower containing a simulated borehole that is 31-ft above ground level and connected to the quarry highwall by a catwalk. A 20-ft-wide by 4' to 6-ft-high simulated mine entry constructed of concrete block ribs and a corrugated steel roof is located at the bottom of the borehole. The headframe was placed on the platform at the top of the simulated borehole. A 100-ft concrete pipeline was laid along the catwalk to the pump location on the top of the highwall and two 10-ft-long flexible hoses were used to connect the pipeline to the elbow on the headframe. Ready-mix concrete was brought to the site and poured directly into the hopper on the pump. The concrete point support cylinders were constructed in the simulated mine entry at the base of the borehole.

Table 1. Concrete mix design experiments.

| | Mix 1 | Mix 2 | Mix 3 | Mix 4 |
|---|---|---|---|---|
| Mix type | 26 gal mix | 28 gal mix | Repeat mix 2 | Mix 3 (Air & Ret) |
| Cement lb/yd³ | 611.6 | 611.6 | 611.6 | 611.6 |
| Gravel No 67 lb/yd³ | 1666 | 1666 | 1666 | 1666 |
| Sand lb/yd³ | 1771 | 1725.3 | 1628 | 1361 |
| Water gal/ yd³ | 159.3 | 175.5 | 216 | 235 |
| Water reducer oz/100 lb Cement | 2.8 | 2.8 | 2.8 | 2.8 |
| Air admixture oz/100 lb Cement | 0 | 0 | 0 | 0.8 |
| Retarder admxture oz/100 lb Cement | 0 | 0 | 0 | 1.4 |
| Moisture content sand | 2.7 % | 2.7 % | 2.7 % | 2.7 % |
| Moisture Content Gravel | 0.7% | 0.7% | 0.7% | 0.7% |
| Initial Slump in | 0 | 0 | 1/2 | 3¼ |
| Final Slump in | - | - | - | 2 ½ |
| Air | - | - | - | 6 1/2 % |
| Observations | Dry, crumbles not pumpable | Dry, crumbles not pumpable | - not pumpable | pumpable |
| Water added lb | 6.5 | 3 | - | - |
| New slump | 2 in | 2 in | - | - |
| Compressive Strength (29 day) | | | | Cylinder A 5300 psi<br>Cylinder B 5200 psi<br>Cylinder C 5240 psi<br>Avg Calc tensile Strength =510 psi Tensile |

## Construction of Concrete Mine Point Support Structures at SAIL.

During testing, there was an observer at the bottom of the borehole, an operator at the headframe location, and an operator at the pump location. Communication was achieved with radio headsets. In addition, a radio was located at the video recording location so the conversation could be recorded. The video camera was tripod mounted, and the entire run during pumping of each concrete cylinder was captured on video tape.

The test had three objectives. The first was to operate the concrete placement device and check its performance in terms of rotation control, concrete flow, hose performance and control, ease of assembly, and pipe installation. The second objective was to observe the slump and other properties of the concrete cylinder as a function of concrete mix and determine the compressive strength of the concrete. The third objective was to determine the operating parameters required to form a seal to the roof of the simulated mine. A total of six concrete cylinders have been constructed. The concrete mix design was selected based on the initial mix design experiments conducted at the concrete plant. The results of the initial mix designs were shown in table 1 and the final specifications are listed in table 2.

**Equipment Setup and Operation.** Equipment setup includes headframe setup, pipe installation, equipment operation, pipe removal and headframe takedown. During all phases of the operation, the equipment performed flawlessly. The initial Setup took 4 hours with three persons. With some experience and minor changes to the design the setup time could be reduced to 2-4 hours with two persons. The pipe installation was safe,

straightforward, and easily accomplished with two persons in less than 1 hour. Takedown time was less than the setup. This translates into a turnaround of two days per concrete cylinder in the field with a new setup required for each borehole location. With some experience and close proximity between boreholes, it may be possible to fabricate one or two concrete support structures per day.

Equipment operation includes the pipe rotation system, the pipe raising and lowering mechanism, the pipeline swivel, and the hose bending system. The rotation of the pipeline was easily varied down to 3 rotations per minute. This could have been made lower by simply raising the pressure in the hydraulic line but there was no need for slower operation. The pipeline couplings prevented twisting of the line during rotation and held the cable as designed. The pipe raising and lowering mechanism provided the control of pipe height needed to make the cylinders. The pipeline swivel functioned well and did not become a source of leakage. The hose bending system including the sleeves, and the hose tension cable provided the control over hose bend radius that was required.

Table 2. Specifications for low slump pumpable concrete.

| | |
|---|---|
| Max coarse aggregate in ..... | 0.5 |
| Max slump in ..... | 2.5 |
| Min slump in ..... | 1.75 |
| Min mortar fraction $ft^3/yd^3$ ..... | 17 |
| Min cement factor bags.. | 6.5 |
| Max air content %vol..... | 5 % vol |
| Min 28 day compressive strength psi ..... | 4000 |

Note: Chemical and mineral admixtures may be used depending on specific site and construction conditions.

## Results

The concrete placement system functioned well. The top of the cylinder sealed at the roof and the contact area width at the roof was controllable. After contact was made, continued pumping placed concrete on the outside of the cylinder and the width of the annular roof contact increased. When roof contact is made, the end of the nozzle begins to drag on the inside edge of the cylinder in the wet concrete. This increases the load on the hydraulic motor, and it either stalls or slows dramatically. This is a strong, foolproof indicator of roof contact. When the motor stalls, the operator simply boosts the pressure on the hydraulic power supply, and the hydraulic motor begins turning again. Concrete pumping continues until the desired roof contact as attained. Very accurate predictions of cylinder volume were attained. The concrete angle of repose for both the inner and outer cylinder walls is near 45°, and calculation of the concrete volume is a matter of geometry. Concrete mix design is very critical to the construction and final geometry of the mine support structure. A low slump pumpable concrete is essential. A stiff concrete is essential to minimize slumping of the concrete during the construction process. This reduces the volume of concrete required for a given height and final roof contact area. However, as was observed in the field, slump determines pumpability, with lower slump mixes being much easier to pump. The difficulty is attaining the lowest slump while maintaining a pumpable concrete. Continuous adjustments may be needed at the construction site for desirable concrete properties. Table 2 includes the starting point for mix design recommendations for a low slump pumpable concrete. Adjustment to the mix design may be required depending on changes in material and climate conditions.

## Conclusions

The cylindrical concrete point support system adds a useful tool to the arsenal of the subsidence abatement official. It appears to be particularly well suited to the abatement of beam subsidence as described by Craft[1]. In areas where the floor of the mine is the primary cause of the subsidence, this point support approach offers a useful tool since the compressive stress on the floor will be at least 4 times lower than the compressive stress on the roof

due to the cone shape of the cylinder. The concrete cylinder will provide a support capable of carrying over 22,000 tons using only 28 $yd^3$ of concrete in a 5-ft-high mine. The cylinder design can be adjusted for the conditions at a particular site to account for the load requirements of the site and the load carrying capacity of the roof and floor. This is accomplished be varying the diameter and annular roof contact thickness of the cylinder. The mechanical placement system was designed to be robust, to be easy to assemble, disassemble, and operate, and to accurately place concrete through a borehole. Testing at SAIL shows that the system works as designed. The concrete cylinder point support system testing was a success and should provide a useful subsidence abatement technique to AML programs, mine closure personnel, and locations requiring subsidence abatement.

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# FULL-SCALE EVALUATION OF THE PERFORMANCE OF GROUT COLUMN SUPPORTS FOR MINE SUBSIDENCE ABATEMENT[1]

Thomas M. Barczak,[2] David F. Gearhart,[3] and Robert C. Dyni[2]

**Abstract**: This U.S. Bureau of Mines report evaluates the load performance characteristics of grout columns used as point-supports for mine subsidence abatement. Several full-scale grout columns were constructed at the Subsidence Abatement Investigation Laboratory (SAIL) in water-filled pools to determine the optimum sodium silicate concentration for a flooded mine environment. The scope of work included the testing of seven full-scale grout columns in the Bureau's Mine Roof Simulator (MRS) from which the load-deformation and failure characteristics were determined. The performance of grout columns is analyzed with respect to (1) the amount of roof contact established during column construction, (2) shape effects produced by the truncated cone geometry, (3) size effects in reference to grout strength and full-scale column capacity, (4) the effect of wet environments on grout strength, (5) the effect of the sodium silicate on the grout strength and full-scale column behavior, and (6) the stiffness of the grout column. Recommendations for point-support construction using sodium silicate technology and the need for additional research to develop improved containment devices for column construction are addressed in the report.

**Additional Key Words:** grout columns, point-supports, subsidence abatement.

## Introduction

Mine subsidence creates erratic and differential movement of ground that disrupts water tables and damages surface structures and subsurface utilities. In room-and-pillar mining, the subsidence is unplanned and generally occurs when unmined pillars of coal deteriorate. The subsidence may take years to decades to manifest itself, and is frequently a problem in abandoned coal mines. The lack of access to these abandoned workings makes remediation efforts difficult. The most common method for control and abatement of abandoned mine subsidence is to fill the mine voids and overburden fissures with a low-strength cementitious grout, which is pumped into the mine in slurry form through several surface boreholes (Gray et al., 1974). Ground stabilization by filling all the voids in the affected area often requires several thousand cubic yards of grout, resulting in abatement costs in the hundreds of thousands of dollars. It has long been recognized that strategic placement of point-support columns would substantially reduce the volume of material and cost of abatement (Michael et al., 1989).

Early attempts at forming point-support systems for subsidence abatement by grouting loose piles of aggregate placed down a borehole (fig. 1) were largely unsuccessful. The loose aggregate provided a poor angle of repose that required large amounts of material and inadequate roof contact area (Michael et al., 1989). It was also difficult to grout the aggregate after the pile was formed. The next generation of point supports was formed using a low-slump cementitious grout. However, placement problems were also experienced with this design, particularly in wet environments where the grout was dispersed by the water before a column could be formed. The most recent approach to grout column formation is the use of

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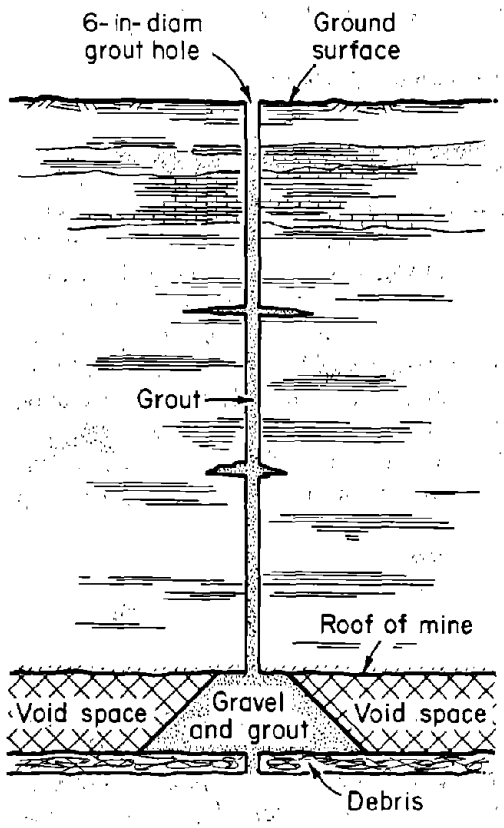


Figure 1. Early attempts at forming point supports by grouting loose piles of aggregate.

sodium silicate as an admixture to a cementitious grout. The sodium silicate accelerates the setting of the grout, providing a stiffer mix, and acts as a barrier to water, allowing truncated cone-shaped columns to be formed in completely flooded mine voids (Reifsnyder et al., 1988).

## Objectives

The purpose of this research was to determine the formation requirements and supporting characteristics of grout columns through full-scale testing. Previous research has been limited to material strength tests and observation of full-scale column construction. Full-scale strength testing of grout columns has not been attempted prior to this study.

## Scope of Work

The scope of work included the construction of four full-scale grout columns at the SAIL in water-filled pools to determine the optimum sodium silicate concentration for flooded mine environments. The scope of work also included the testing of seven full-scale grout columns in the MRS from which the load-deformation and failure characteristics of the structures were determined. The MRS load frame incorporates 6- by 6-m (20- by 20-ft) platens and can simulate loads of up to 13,334 kN (3 million lb). The material strength of the grout as a function of time was determined from controlled loading of 15-cm (6-in) diameter, 20-cm (8-in) high, cylindrical grout specimens for each of the grout mixes. Size effect relationships were evaluated by testing 61-cm (24-in) diameter, 122-cm (48-in) high, cylindrical specimens and comparing their strength to that of the 15-cm (6-in) diameter specimens. The measured strength of the 15-cm (6-in) diameter cylinders was reduced by 6 pct in accordance with ASTM specifications for specimens with an aspect ratio of less than 1.8. Load-deformation tests were also conducted on three small-scale cone specimens where the geometry of the cone was controlled by sheet metal forms. Truncated cone geometries were formed with cone angles of 15°, 30°, and 45° (angles measured with respect to vertical plane from top cone surface). The cones were all 39 cm (12 in) high with a 15-cm (6-in) diameter top surface. The variable cone angle simulates different angles of repose, which permits examination of the contribution of material outside the central core to the support capacity.

## Description of Column Construction and Test Parameters

Four full-scale columns were constructed at the Bureau's SAIL at Lake Lynn and three columns were constructed on the platens of the MRS load frame at the Pittsburgh Research Center. The columns were constructed from a fly ash-cement grout and sodium silicate. The cementitious grout was supplied by Lambert Construction Co., which was employed as a contractor to construct the grout columns. The design mix for the MRS columns consisted of 3,114 N (700 lb) of Portland I cement, 7,517 N (1,690 lb) of fly ash, and 3,558 N (800 lb) of water with a design strength of 10,276 kPa (1,490 psi) at 14 days. The grout mix for the SAIL columns was 1,672 N (376 lb) of Portland I cement, 7,993 N (1,797 lb) of fly ash, and 2,224 N (500 lb) of water with a design strength of 6,207 kPa (900 psi) at 7 days.

Figure 2. Nozzle used for application of sodium silicate and cementitious grout in column construction.

The grout and sodium silicate were delivered through a 5-cm (2-in) diameter nozzle. This nozzle is a common design in the industry for sodium silicate application. The sodium silicate is pumped through a separate line and is injected at the nozzle, where it is dispersed through the outer nozzle annulus to encapsulate the grout stream coming out of the nozzle (fig. 2). The volumetric ratio of the grout to sodium silicate is controlled by controlling the pumping rate of the grout and sodium silicate. The grout is tremied in place by initially placing the nozzle at the floor and slowly raising it as the column grows in height.

Tests were conducted under controlled-load conditions to evaluate the strength of the grout materials and the structural response of grout columns. Material strength tests were conducted on the 15-cm (6-in) diameter specimens at a load rate equivalent to 138 kPa/s (20 psi/s) in accordance with ASTM specifications. The 61-cm (24-in) diameter cylindrical specimens and the full-scale columns were tested at a controlled displacement of 1.3 cm/min (0.5 in/min), chosen from past experience with large-scale concrete specimens and the load frame limitations.

## Columns Constructed at the SAIL

The grout columns constructed at the SAIL were formed in 5.5-m (18-ft) diameter, 1.2-m (4-ft) high, water-filled swimming pools to simulate a flooded mine environment. One column was constructed in a dry environment to evaluate the interaction of the sodium silicate with the grout in the absence of water. The grout and sodium silicate were pumped from ground level to a height of approximately 9 m (30 ft) to simulate borehole drop velocities during grout placement.

The control parameter for these column constructions was the volumetric ratio of fly ash-cement grout to sodium silicate. The purpose of these tests was to determine the optimum sodium silicate concentration for a flooded environment. Four grout-to-sodium silicate ratios were evaluated: 5:1, 7.5:1, 10:1, and 12.5:1. The 10:1 concentration provided a moderate cone angle of 50° (measured from vertical) in the wet environment, while forming a solid foundation that resulted in a symmetrically shaped, truncated cone structure. The 12.5:1 ratio did not have sufficient sodium silicate to prevent dilution of the cementitious grout in the wet environment, resulting in an unacceptable cone angle. The 7.5:1 ratio provided a cone angle similar to the 10:1 ratio, but the structure was more asymmetrical. When the sodium silicate concentration was increased to 5:1 ratio, there appeared to be a surplus of sodium silicate. The cone angle provided by the 5:1 ratio was less than the 7.5:1 and 10:1 ratios, but the base area was considerably smaller. Based on these observations, a 10:1 grout-to-sodium silicate ratio is a reasonable lower limit in the required sodium silicate concentration necessary to provide a well-shaped point support in a flooded environment.

Four SAIL columns were transported to the Pittsburgh Research Center for full-scale testing in the MRS. A physical description of each column and the construction parameters are provided in table 1.

Table 1. Description of grout columns constructed at the SAIL.

| | Column 1 | Column 2 | Column 3 | Column 4 |
|---|---|---|---|---|
| Grout-to-sodium silicate ratio | 7.5:1 | 5:1 | 10:1 | 10:1 |
| Environment | Wet | Wet | Wet | Dry |
| Material volume, $m^3$ | 1.0 | 1.7 | 1.1 | 2.4 |
| Height of column, cm | 122 | 117 | 122 | 152 |
| Cone angle, deg from vertical | 45 | 38 | 50 | 40 |
| Area of base, $m^2$ | 2.3 | 3.5 | 3.4 | 2.6 |

## Columns Constructed in the MRS

Three columns were constructed on the platens of the MRS. The grout-to-sodium silicate ratio was controlled to approximately 10:1 for all three columns. The columns were constructed in a dry environment. An artificial roof with a 15-cm (6-in) diameter hole was provided to simulate a borehole and underground roof contact. A description of the three columns constructed in the simulator is provided in table 2.

Table 2. Description of support columns constructed in the MRS.

| Column | Column 1 | Column 2 | Column 3 |
|---|---|---|---|
| Roof contact area, $cm^2$ | 1,290 | 2,829 | 7,354 |
| Grout to sodium silicate ratio | 10:1 | 10:1 | 10:1 |
| Material volume, $m^3$ | 2.3 | 3.1 | 3.8 |
| Height of column, cm | 137 | 145 | 130 |
| Cone angle, deg | 42 | 42 | 45 |
| Borehole condition | Open | Closed | Closed |

# Test Results

## SAIL Specimens

The load-deformation responses of the four full-scale grout columns constructed at the SAIL are compared in figure 3. Columns constructed from the 10:1 grout-to-sodium silicate material exhibited significantly higher strength and stiffness than columns constructed with other concentrations of sodium silicate. At 15 cm (6 in) of displacement, the load capacity of the columns constructed with 10:1 grout-to-sodium silicate material was 334 kN to 445 kN (75 to 100 kips), compared with 44 kN (10 kips) for the 5:1 and 7.5:1 material column constructions.

The small top surface area, due to the lack of roof contact, and uneven bottom surface degraded the supporting capability of these columns. The application of controlled displacement to the specimen caused the top of the column to crumble, which allowed the area of the top surface to gradually increase. The larger contact area provided an increase in capacity with increased displacement. When the stress exceeded the material strength, the columns would split from top to bottom, and the column would shed load as broken segments of the column were pushed apart. When confinement reduced the lateral expansion of the

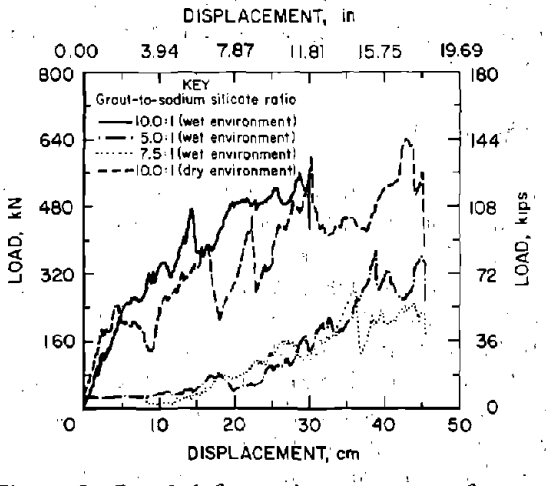


Figure 3. Load-deformation response of full-scale grout columns constructed at the SAIL.

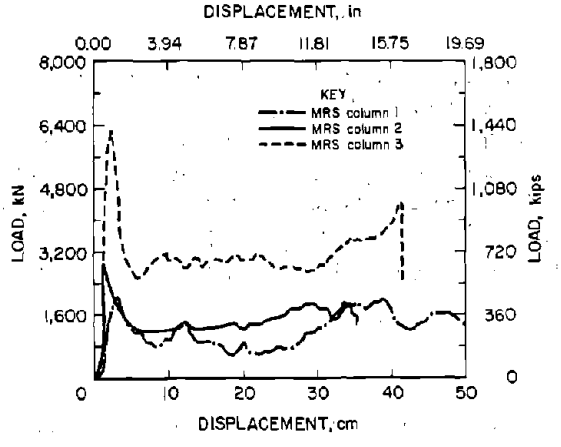


Figure 4. Load-deformation response of MRS columns.

broken segments, the load would gradually recover and continue to increase until the next failure. This cycle continued through several inches of displacement, as shown in figure 3.

## MRS Specimens

All of the columns constructed in the MRS used a 10:1 ratio of grout-to-sodium silicate. The design strength of the grout was 10,275 kPa (1,490 psi) after 7 days, which was verified by tests on 5-cm (2-in) cube specimens by the commercial supplier. However, the measured strength from the 61-cm (24-in) diameter specimens was 4,276 kPa (620 psi). Columns constructed in the MRS avoided the problem of an uneven bottom surface, which degraded the performance of the columns constructed at the SAIL. The MRS columns were also constructed against an artificial roof to provide roof contact and a measurable top surface area. Unlike the columns formed at SAIL without roof contact, the MRS columns exhibited a high initial stiffness as they developed significant resistance with little displacement, as shown in figure 4. All columns began to fail as the top surface stress approached 7,241 kPa (1,050 psi). After the initial failure, the stiffness of the support structure decreased slightly (fig. 5), while the load capacity continued to increase with increasing displacement. Maximum load occurred when localized failures (cracks) coalesced to fracture the specimen from top to bottom.

Maximum load capacities of 2,002 kN, 2,847 kN, and 6,338 kN (450 kips, 640 kips, and 1,425 kips) were observed for columns 1, 2, and 3 as referenced in table 2. These maximum capacities were attained at less than 2.5 cm (1 in) of convergence. Load resistance after reaching the maximum capacity was reduced by about 50% as the columns split through their cross section and crushed from the continued convergence. The capacity attained with the full-scale columns was lower than expected, based upon the measured 4,276-kPa (620-psi) material strength and the 350 to 400% increase in capacity above the material strength demonstrated in the small-scale cones of a similar geometry. Based upon these measures, the initial failure of the full-scale columns should have occurred at top surface stress levels of 14,966 to 17,103 kPa (2,170 to 2,480 psi), as opposed to the observed initial failure at 7,241 kPa (1,050 psi). Failure of the columns to sustain these stress levels is attributed to the presence of cold joints formed by the sodium silicate and the asymmetrical shape of the structures.

## Analysis of Test Results

Several factors have been identified that affect the performance of these support structures. These include (1) wet environment, (2) size effects, (3) shape effects, (4) deficiencies of sodium silicate, (5) roof

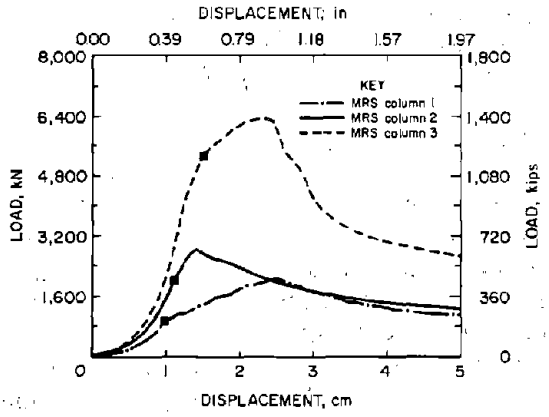


Figure 5. Decreased stiffness of grout column after first failure.

contact area, and (6) material strength. Other design considerations include the stiffness and mode of failure of the support structure.

## Wet Environments

The test data indicate that a wet environment reduces the grout strength. Both 15-cm (6-in) diameter and 61-cm (24-in) diameter grout specimens that were poured and cured in water were found to be of lower strength than samples cured in a dry environment. The effect of the wet environment was more substantial on the smaller samples, where the compressive strength was reduced by approximately 75%, compared with approximately 50% reduction for the large-scale samples. The reduced strength of the samples cured in wet environments is possibly due to the heat loss during the curing process (American Concrete Institute 1986).

## Size Effects

Previous research has shown that a 50% reduction in strength of concrete materials is observed when the specimen size exceeds 61 cm (24 in) in diameter in comparison to 15 cm (6 in) diameter or smaller specimen sizes (Mindred et al., 1981, Glucklish 1990). Tests on 61-cm (24-in) diameter grout column material specimens have shown this strength reduction can also be expected for full-scale grout column supports. The initial failure of the MRS columns occurred when the top surface stress approached 7,241 kPa (1,050 psi), which is 30% less than the grout design strength. Therefore, the material strength should be reduced by a factor of 1.5 to 2 to provide a conservative estimate of grout column capacity based on the top surface area and material strength.

## Shape Effects

Previous research in materials science and rock mechanics has shown that lateral confinement significantly increases the strength of the specimen by changing the state of stress from uniaxial to triaxial during load development (Mindress et al., 1981). Since confinement is partly dictated by the shape of the structure, shape effects are a primary consideration in large-scale structural evaluations. The conical shape of grout column structures enhances the load capacity of the support by providing lateral confinement to the core of the support column.

Figure 6 shows a 350 to 400% increase in capacity for small-scale cones with a 15-cm (6-in) diameter top surface area and a 35° to 40° cone angle as compared to a cylindrical specimen with the same top surface area. The relationship between load and cone angle is nonlinear with the strength improvement diminished as the cone increases in cross-sectional area at larger cone angles. However, this increase in capacity due to the cone geometry was not fully realized in the full-scale grout columns. Full-scale grout columns provided a strength increase of about 75% in relation to the material strength measured from 61-cm (24-in) diameter specimens, or a 5% increase in capacity in relation to material strength measured from 15-cm (6-in) diameter specimens. Failure of the columns to reach the expected level of strength is attributed to the presence of cold joints formed by the sodium silicate and to the asymmetrical shape of the structure.

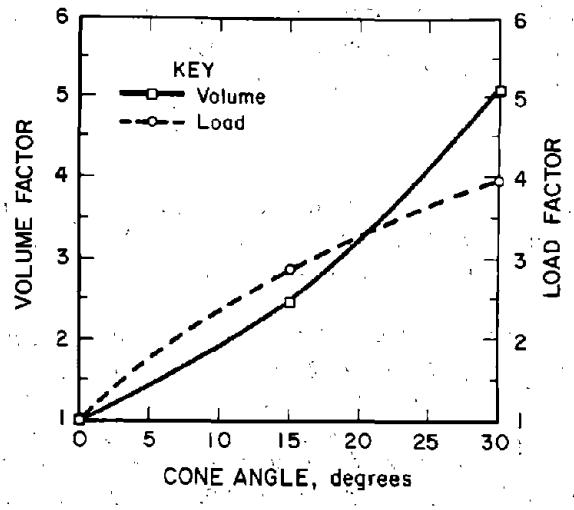


Figure 6. Relationship between material volume and specimen strength for small-scale cone specimens as a function of the cone angle.

## Deficiencies of Sodium Silicate

The use of sodium silicate enables a structure to be formed in a water-filled environment that could not be formed without it. However, there are some deficiencies associated with the use of sodium silicate. Inadequate mixing of the sodium silicate with the cementitious grout produces pockets or layers of sodium silicate gel that reduce the load capacity of the support structure. During column construction, grout flows through the path of least resistance and down a small area on the side of the cone. As this section of the cone builds, increased resistance diverts the grout flow to a different area where the grout has already set, and the newly placed grout does not fully react with the previously placed material. The resultant layering creates planes of weakness in the structure that reduce the load capacity of the column. The sodium silicate accelerates the setting time for the grout, which is advantageous for column construction, but the accelerated setting time also increases the tendency for plugging.

## Roof Contact Area

Roof contact area largely determines the capacity and stiffness of grout columns. As shown in figure 7, the initial failure of the grout columns tested in the MRS is linearly related to the roof contact area. Hence, the capacity of the support is directly related to the area of roof contact established during column construction. The material strength achieved in the full-scale column can be determined from the slope of the load versus contact area plot. For the MRS columns depicted in figure 7, the material strength was 7,241 kPa (1,050 psi).

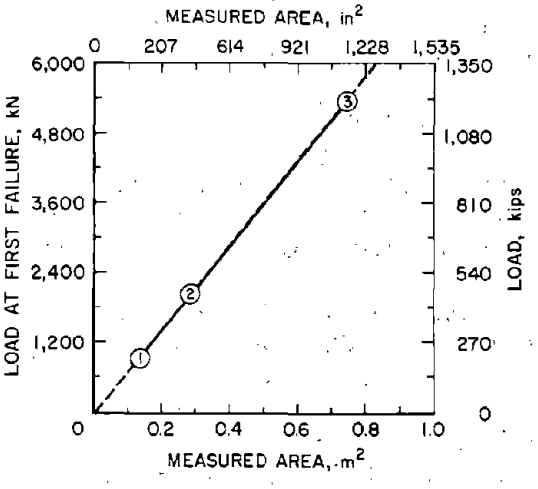


Figure 7. Initial failure of MRS grout columns as a function of roof contact area.

The stiffness of the grout column is also highly dependent on the roof contact achieved during column construction. Stiffness is an important design consideration because it indicates how much convergence must take place before the grout column provides a specified magnitude of resistance. The significance of roof contact area to the stiffness of the column was clearly demonstrated by comparison of the SAIL columns, which were formed without roof contact, and the MRS columns, which attained 1,290 to 7,354 cm$^2$ (200 to 1,140 in$^2$) of roof contact. The initial stiffnesses of the MRS columns were two orders of magnitude higher than those of the SAIL columns. The initial stiffness ranged from approximately 1,750 kN/cm (1,000 kips/in) for MRS column 1 to 6,567 kN/cm (3,750 kips/in) for MRS column 3, increasing as the top surface area increased.

## Grout Strength

The grout strength obviously affects the capacity of the support column. Grout strength is primarily determined by the amount of cement in the grout mix but is affected by many other factors. However, as

previously indicated, the sodium silicate tends to reduce the effective grout strength in the full- scale column by creating planes of weakness where the sodium silicate is not properly mixed with the grout. In high concentrations, the sodium silicate forms a gel that has very little tensile or shear strength.

## Conclusions

Sodium silicate allows a grout column to be constructed in a wet environment, provided there is sufficient concentration and proper mixing of the sodium silicate with the cement-fly ash grout. A 10:1 ratio of grout to sodium silicate provides the most effective point support in a wet environment.

The most important construction requirement to provide effective support from grout columns is to achieve adequate roof contact area. Roof contact should be as large as possible to maximize the capacity and stiffness of the support. Two important conditions must exist in order to maximize the top surface area: (1) an adequate base must be formed consistent with the angle of repose of the material to provide a foundation for achieving the desired roof contact area and (2) when initial roof contact is achieved, the grout must be able to be delivered at sufficient pressure to expand the roof contact area without plugging the nozzle or sealing the borehole.

The borehole can be sealed by using a packer or collar to prevent premature filling of the borehole with grout before adequate roof contact has been established. Grout should only be permitted to flow into the borehole when the top surface area has expanded to the required area to support the expected loads. Then grout can be used to seal the borehole and consolidate fractured roof material. Consolidation of the roof strata improves its stiffness and support characteristics.

Less sodium silicate should be used at the start of construction to allow the grout to flow freely into the voids in broken rubble on the floor of the mine opening and to consolidate the floor material to provide a solid foundation for column formation. Too much sodium silicate applied with the fly ash grout at the beginning of construction can result in a steep cone (small cone angle as measured from vertical) and an inadequate base to provide column formation with the required roof contact area.

From a construction cost perspective, the increased strength advantage of a larger cone angle is diminished by the rapidly increasing volume of material when the cone angle is greater than 20°. A 20° cone angle represents a very steep column that is difficult to construct and results in a structure with limited stability. Cone angles in the range of 30° to 45° yield more stable structures and are attainable using sodium silicate technology. Since the added strength advantage of a larger cone angle is diminished by the larger volume of material, a practical optimum cone angle is 30°. A 4-ft-high, 30° column with a 4-ft-diameter top surface will attain 91% of the capacity of the 45° column with 46% less material.

While the sodium silicate allows a column of cementitious material to be formed in a flooded mine environment, it degrades the strength and thereby reduces the capacity of grout columns as supporting structures for mine subsidence abatement. Based on these observations, it is recommended that further research be pursued to develop containment devices for grout column construction so that grout columns could be formed without sodium silicate or with a much lower concentration.

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# MINING-INDUCED SURFACE DAMAGE AND THE STUDY OF COUNTERMEASURES[1]

Cui Jixian[2]

**Abstract**: Coal constitutes China's major energy resource. The majority of the coal is produced from underground mining operations. Surface subsidence may amount to 80% of the thickness of the seam mined, while the subsided volume is around 60% of the mined volume underground. An area of 20 hectares of land will be affected with each 1 million tonnes of coal mined, thereby causing severe surface damage. Following a description of the characteristics of surface damages due to underground mining disturbance, this paper elaborates on the damage prediction method, standards applied for evaluating the damages experienced by surface buildings, land reclamation methods in subsided area, measures for reinforcing and protecting buildings in mining-affected areas, and performance of antideformation buildings.

**Additional Key Words**: ground strains, land reclamation, antideformation building.

## Introduction

Ninety-five percent of the run-of-mine coal is produced by the longwall caving method in China, causing severe ground subsidence. Investigation shows that 20 hectares of land will be affected with the mining of every 1 million mt of coal. This leads to inundation, waste and damage of land, moving of buildings due to damage, and ecological deterioration in mining areas. Great importance has been attached to these problems. Since the early 1950's, research accomplishments together with related policies, codes, and technical specifications that have been implemented have brought about marked economic and social benefits.

## Characteristics of Mining-induced Surface Damage

Mining of a large amount of coal underground leads to collapse and movement of overburden and hence surface subsidence. Therefore, the damage is characterized by spoiling and deterioration of land and damage of surface buildings.

### Damage and Deterioration of Land

This poses one of the main environmental impacts in mining area, which manifests itself in the following aspects:

(1) Presence of subsidence trough: Owing to the effect of coal mining, the originally flat land will slope and become hilly, reducing the productivity. In areas with high ground water level, the land may be waterlogged or bogged;

(2) Presence of cracks and steps on the surface due to subsidence: This usually occurs at the outer edge of the subsidence trough and fault outcrop, resulting in discontinuity of land as well as reduction of its water and fertilizer-retaining capability;

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(3) Presence of cockpit-formed subsidence trough: This generally occurs in places with karst water or where steeply inclined, shallow thick seams are mined. The land in this case is rendered inoperative, and even the land outside the subsidence trough may be affected;

(4) Loss of land due to dumping of coal refuse: Of the total output of run-of-mine coal produced underground, coal refuse shares about 20%. Though much work has been done on its comprehensive utilization, a large amount of refuse is still stockpiled, consuming land and causing pollution of the environment;

(5) Pollution of land: There are two pollution factors, i.e., use of mine waste water for irrigation and leaching of acidic refuse by rainwater;

(6) Salinization of soil: After intensive evaporation of highly mineralized ground and surface water in large areas of mining-induced low-lying land, the land is salinized due to accumulation of soluble salts.

The substantial reduction of cultivated area for farmers in mining areas has brought about severe social problems. For example, in the mining districts under the Kailuan Coal Mining Administration, the subsided area totals 14,647 ha with 3,284 ha ruined and 9,247 ha affected. Out of the total area affected, 1,151 ha are waterlogged all year round, and about 60,000 farmers are deprived of their cultivated land.

**Damage of Surface Buildings**

China is populous and has densely distributed villages. In coal-bearing areas, there is a large number of surface buildings. According to incomplete statistics, in the main mining areas in China, a total of 87.6 billion mt of coal is locked up underground, and the locked-up coal deposits under villages total 52.2 billion mt or 60% of the total. This phenomenon is severe, particularly in 10 provinces, such as Hebei, Shandong, Shanxi, and Henan, where a total of more than 1 billion mt of coal is sterilized respectively.

Under the effect of mining operations, the ground surface and hence the buildings will undergo deformation and various degrees of damage.

(1) Uniform surface subsidence is generally less harmful to buildings. However, when the subsidence is great and the ground water level is high, the surface will be waterlogged, thereby reducing the load-bearing capacity and strength of the foundations. Generally, a building undergoing even settlement is under the effect of dynamic deformation of surface. As long as the building is able to stand the dynamic effect and no water hazard is imposed, no damage will be experienced by the building.

(2) The tilt of the foundation of a building is basically similar to that of the ground surface, except that a tall building or a structure with a small base area will experience greater damage.

(3) Generally, tensile and positive curvature deformations occur simultaneously, and the same is true with compressive and negative curvature deformations. In the former case, a building would experience the following damages:

- Presence of vertical cracks at the bottom of a building, which increase in width from top to bottom;
- Presence of increasingly widening cracks in a vertical direction of the upper part of wall sills, on the eaves part above the lintel of doors or windows, in the middle of the wall body, or above the juncture of the walls;
- Presence of inverted splayfoot-formed cracks at corners, or near doors or windows;
- Presence of cracks on the roof slabs;
- Movement or even drawingout of roof trusses, beams, or purlins.

Under the effect of surface compressive and negative curvature deformations, a building would suffer the following damages:

(1) Presence of splayfoot-formed cracks on the wall near the doors or windows;

(2) Presence of horizontal cracks above or below the window openings and at the places were the masonries change in cross section;

(3) Presence of vertical cracks at the juncture of vertical and horizontal walls, and in the middle of wall bodies;

(4) Peeling off of plaster, bulging, and local damage of wall, and deformation of doors or windows;

(5) Bulging of the middle part of the roof.

## Prediction and Evaluation of Mining-induced Damages

### Study of the Law Governing Ground Movement

Since 1956, more than 2,000 observation lines and two observation networks have been established in the main mining areas across China. With the data obtained, a sound basis has been laid for the study of laws governing ground subsidence in mining areas.

Early in the 1960's, a negative exponent function method was proposed by the Tangshan Branch of the Central Coal Mining Research Institute (CCMRI) for calculating subsidence along the main cross section of a subsidence basin. Later, an equation for determining the subsidence value of other sections and a method for deriving the related parameters were proposed. The negative exponent function method is shown in figure 1 and expressed in equation 1.

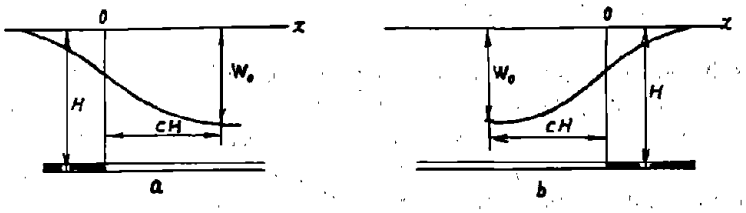


Figure 1. Subsidence curve of main profile along the strike.

$$W(x) = w_o \exp[-a\,(c + x/h)^b], \quad (1)$$

Where $W_o$ = The maximum ground subsidence under supercritical extraction conditions, mm;
H = Depth of boundary of workings, m;
a,n,c = Specific parameters involved in negative exponent function method. a = Development factor in the lateral direction; b = Profile factor; c = Position factor of the point with maximum subsidence value.

The sign "+" denotes that the x axis extends in a direction toward the worked-out area, while "-" indicates the direction is towards the coal pillar.

The prediction precision is greatly controlled by the accuracy of ground movement parameters. To reduce errors as much as possible, an accurate determination method has been proposed. Through the use of

this method in calculating parameters at nearly 200 observation stations, the variation of ground movement parameters under different lithological conditions is listed in table 1.

From the 1970's on, electronic computers were applied for predicting ground movement in mining gently to steeply inclined coal seams and for plotting sectional and isoline drawings.

Table 1. Main ground movement parameters under different overburden conditions.

| Method | Character of overburden | Subsidence factor | Lateral development factor | Profile factor | Position factor |
|---|---|---|---|---|---|
| Negative Exponent Function Method | Hard | 0.42-0.64 | --<br>$a_1 = 4.5 \sim 6$<br>$a_2 = 3 \sim 5$ | 2.9-3.2 | --<br>$c_1 = tg31°$<br>$c_2 = tg36°$ |
| | Medium-hard | 0.65-0.84 | $a_1 = 2.28 \sim 2.21 \ln \alpha / \beta$ | 2.6-2.9 | $c_1 = tg(31° - 0.5\alpha)$ |
| | | | $a_2 = 0.16 \sim 2.54 \ln \alpha / \beta$ | | $c_2 = tg(36° + 0.5\alpha)$ |
| | Soft | 0.85-1.15 | -- | 2.4-2.6 | -- |

## Evaluation of the Degree of Damages Experienced by Buildings

For safety mining of coal locked up beneath surface buildings, a preevaluation of the degree of damage suffered by buildings should be made. The degree of mining-induced damage of a building is governed by ground deformation and the antideformation capability of the building. Based on the results of a study, the classification of damages of houses of mixed brick-wood is listed in table 2.

## Study of Countermeasures and Structural Precautions

### Land Reclamation Technique Applied in Subsided Area

The land reclamation in the subsided area is designed to restore the affected land to a usable state for better environmental control.

The methods currently used in China fall into the following categories:

(1) Engineering or biological means in different stages or sequence;

(2) Land reclamation with the use of coal refuse, ash from powerplants, or by applying methods such as shallow refilling after extensive excavation, dewatering, formation of terrace land and comprehensive approach;

(3) Land reclamation for construction, agricultural, forestry, fishery, and recreational purposes or for comprehensive applications.

Table 2. Classification of damages of houses of mixed brick-wood structure as related to ground deformation.

| Classification of damage | Ground Tilt (mm/m) | Ground curvature ($10^{-5}$ 1/m) | Deformation horizontal strain (mm/m) | Description of damage | Repair requirements |
|---|---|---|---|---|---|
| Class I | <3 | <0.2 | <2 | Small fractures less than 5 mm in width appear on walls | Not necessary |
| Class II | 3-6 | 0.2-0.4 | 2-4 | Cracks 5-10 mm wide on walls. Doors & windows sloping slightly. Plaster comes off and slight dislocation of beam support. | Minor repair |
| Class III | 6-10 | 0.4-0.6 | 4-6 | 10-30 mm-wide cracks on walls. Slant of doors & windows, tilt of walls, dislocation of beam heads. Cracks or heave observed in floors and walls. | Periodic repair |
| Class IV | >10 | 0.6-0.8 | >6 | 30 mm wide cracks accompanined by vertical, horizontal, and oblique fractures in walls. Walls leaning or bulging noticeably. Severe dislocation of beam heads. Bulging of roof. | Overhaul |

**Engineering Means**

The land is backfilled, reclaimed, excavated, and leveled by mechanical means in a manner to suit local conditions and application of the land reclaimed.

(1) With the use of coal refuse as backfill material, additional benefits can be obtained in the reduction of area of land for stockpiling and improvement of environmental control.

In cases where the reclaimed land is intended for use as a construction site, backfilling should be made on a layer-by-layer basis, together with vibration compaction for increasing the load-bearing capacity and stability of the foundations. The thickness of the individual layer is determined by using the following equation:

$$H = 0.5Q + 20 \quad (2)$$

where H = Layer thickness, cm,
Q = Capacity of vibration compactor, kN.

If the refuse is backfilled once to the designed depth, intensive ramming should be applied to the foundation area. The depth to which the intensive ramming should be applied is calculated with the following equation:

$$h = (0.4 \sim 0.7)\sqrt{QH} \quad (3)$$

where h = depth rammed, m,
Q = Weight of hammer, 10kN,
H = Falling height of entire hammer, m.

If the refuse contains a considerable proportion of ferric sulfide, a layer of clay should be placed on each thick layer of refuse as an isolation layer or the topmost refuse layer should be sealed up to avoid spontaneous combustion or release of poisonous matters.

(2) Use of ash from power plant as backfilling material.

- Construction of ash yard with topsoil removed from the subsided area;
- Ash delivered hydraulically from the power plant to ash yard via pipeline;
- The water decanted in the yard is discharged from the drainage ports;
- When the ash yard is filled to the designed elevation, the dam is removed, and soil is placed on top of the ash to form farmland.

(3) Extension of the depth of severely subsided area through mechanical or manual means for raising fish, planting lotus roots, or irrigation purposes. The soil dug out is used to backfill the slightly subsided area to form paddy fields or dry farmland. The method is widely used in subsided areas with accumulated water because of its noticeable economic and ecological benefits.

(4) If the ground surface in the subsided area is still higher than the nearby lake or river water level, drainage channels are dug leading to the lake or river. An area with deep accumulated water is changed into a fish pond. If the waterflow in the river is impeded due to variations of the riverbed, pump stations may be set up in wet seasons to discharge the water into the lower section of the river. In addition to protection of the ecological environment, there is no need to move the villages in the affected area.

(5) In subsided areas with low ground water level or slopes, the land may be leveled and transformed into terrace land.

(6) Reclamation of the subsided area in a well planned way to suit local conditions with an aim to turning the reclaimed area with relatively small input into agricultural, fishery, forestry, construction, or recreational places.

**Biological Method**

This method is designed to restore the fertility and productive capacity of affected land with organisms, usually a process proceeded in the wake of an engineering approach. It consists mainly of soil improvement and optimal selection of vegetation variety.

The methods used for soil improvement involve:

(1) Biological method (green manuring method): Planting perennial or annual leguminous plants in areas under reclamation to accelerate the curing of soil;

(2) Application of fertilizer for higher content of organic materials in soil in an attempt to ameliorate the soil structure for improved physicochemical property;

(3) Blending method: For soils with excessive clay or sands, a blending method is used to adjust the clay/sand ratio for increased fertility;

(4) Chemical method: Use of lime for acid soil amelioration.

**Structural Precautions for Affected Buildings**

The structural measures adopted may fall into two categories:

(1) Addition of rigid elements, such as reinforced concrete foundation girths and columns to increase the integrity and antideformation capacity of a building;

(2) Adoption of flexible measures, i.e., provision of sliding layers, expansion joints, and selection of rational length of individual units for better adaptability to ground strain and smaller additional stress.

The structural precautions as described above may be applied on both existing and new buildings.

**Existing Buildings**

Based on experience gained in protection of buildings in mining-disturbed areas, the following technical measures can be effectively adopted:

(1) For buildings subject to Class II damage (see Tables 2 and 3), expansion joints and reinforced concrete foundation girths can be provided;

(2) For buildings suffering Class III damage, in addition to the above measures, reinforced concrete girths, steel ties, as well as reinforced concrete columns, should be provided;

(3) Buildings suffering Class IV damage should be removed and rebuilt. In particular cases where strengthening measures are required, apart from the measures as described, reinforced concrete anchor slabs should be provided for the foundations;

(4) Provision of trenches for partially absorbing the compressive deformation.

**New Antideformation Buildings**

Apart from adoption of rigid and flexible structures, one of the key measures for new buildings is provision of a horizontal sliding layer between the superstructure and the foundations for effectively reducing the effect of horizontal ground strains. Use of this new method since the late 1970's has promoted China's structural precaution technique for buildings in mining areas to a new horizon.

The technique for designing new antideformation buildings was first developed by the Tangshan branch, CCMRI in 1978, and the 1,350 $m^2$ club with this design was first built at the Zhijiang Coal Mine. Later, 36 antideformation buildings with a total area of 24,586 square meters were successively built in mining-disturbed areas. Of these, 11 were residential buildings, 13 were village houses, and 12 were public buildings. The buildings all withstood the test when three coal seams beneath (aggregate mining thickness: 4.1 m) were mined with a longwall method along strike. Since 1983, 300,000 $m^2$ of different kinds of antideformation buildings have been successfully erected prior to start of operations at mines under Yangquan, Huoxian, Pindingshan, Yima, Chenghe, Datong, Tiefa, Nanpiao, Qitaihe, Hegang, and Kailuan Mining Administrations.

The main considerations involved in designing new antideformation buildings are:

(1) Rational planning and optimal selection of construction sites: In this respect, the long-term development plan and mining activities should be taken into consideration. The axis of a new building should run as parallel or normal to the strike of seam as possible. For a building located at the edge of the subsidence basin, its long axis should run parallel to the subsidence isopleth, while a building at the flat bottom of a subsidence basin should keep its long axis normal to the direction of face advance. Buildings of any kind should not be located in areas subjected to the adverse effects due to mining. For instance, erection of a building at the deflection point of a subsidence basin should be avoided. Likewise, the construction sites should not be located in areas subjected to discontinuous cracks or in areas with water hazards or that are liable to undergo earth slump.

(2) Structural Form: Buildings to be erected in mining-disturbed areas should be designed as simply as possible. A building should preferably be separated into a number of rectangular units of identical height, structural rigidity, and uniform load. With a predicted value of $\epsilon$ greater than 6 mm/m, the length of each unit is preferably less than 20 m while at a $\epsilon$ value smaller than 6 mm/m, a length in the range of 20-25 mm should be selected. Furthermore, the longitudinal and transverse load–bearing walls should be symmetrically arranged relative to the central axis.

(3) Foundations: The buried depth of the foundations should possibly be reduced to a level just below the frozen depth and enough to meet the load-bearing requirement. On the top of foundations is a horizontal sliding layer on which reinforced concrete girths should be placed. Even in case the foundations are not on the same level, placement of sliding layers is indispensable.

(4) Walls: The partition walls, and door and window openings of equal width and height should be uniformly arranged according to height and length of walls. The distance between the edge of an opening and that of the building should be no less than 1.5 m and no smaller than 0.6 m away from the axis of T-juncture of longitudinal and traverse walls. Moreover, lintels for doors and windows should be of reinforced concrete structure, and tie bars should be provided at corners and the T-juncture of walls. Use of hollow walls should be avoided.

(5) Reinforced Concrete Girths and Columns: Reinforced concrete girths and columns must be provided for new buildings. They should be anchored with each other so as to form a skeleton system. The girths are to be placed on top of sliding layers.

(6) Slabs and Roof: The slabs, roof, and wall girths should be of cast-in-situ structure to form an integral system.

If precast concrete slabs are used, the supporting length on the wall should be no less than 10 cm. The slabs should be tied firmly with girths and walls. The beam or roof truss and walls should be reliably secured. Use of brick arch or concrete arch structures liable to induce lateral thrust force is in no case allowed.

Technoeconomic benefits obtained by applying antideformation techniques at the Zhijiang Coal Mine are as follows:

(1) Movement and deformation of buildings: Based on field observation data, the subsidence and tilt of a building and the ground where the building is erected are closely related to linearity of deflection. The deflection experienced by the building is in inverse proportion to the building's integral rigidity. Because of the provision of sliding layers, noticeably less horizontal strains were observed on walls.

(2) Change of width of expansion joint: The club's two adjacent units isolated by an expansion joint underwent a displacement of 136 mm relative to each other and the gap widened 129 mm in an east-west direction. Eighteen domestic buildings all experienced a relative displacement of 220 mm without showing any cracks in the walls. This has demonstrated the effectiveness of the provision of expansion joints.

Table 3. Maximum movement and strains of buildings and ground.

| Bldg. | Obs. Station | Sub. (mm) | Tilt (mm/m) | Pos.Curvature ($10^{-4}$/m) | Neg.Curvature ($10^{-5}$/m) | Ext. (mm/m) | Comp. (mm/m) | Structural Design |
|---|---|---|---|---|---|---|---|---|
| Guest House | Wall | 2823 | 21.1 | 1.25 | 1.34 | 0.9 | 1.2 | Anti-Deformation |
| | Ground | 2834 | 36.0 | 6.99 | 7.51 | 16.5 | 5.1 | |
| Club | Wall | 2712 | 14.0 | 1.86 | 1.19 | 2.5 | 3.4 | Anti-Deformation |
| | Ground | 2707 | 26.3 | 2.84 | 9.32 | 5.0 | 18.3 | |
| Kinder garten | Wall | 2245 | 31.4 | 1.84 | 1.60 | 1.2 | 1.7 | Anti-Deformation |
| | Ground | 2239 | 33.1 | 3.22 | 3.95 | 9.8 | 25.9 | |
| Police Sub-Station | Wall | 2571 | 18.6 | 1.82 | 3.58 | 1.4 | 2.0 | Anti-Deformation |
| | Ground | 2586 | 22.7 | 6.20 | 8.60 | 13.8 | 13.1 | |

(3) Results of observation with nine groups of stations at the interface of the reinforced concrete foundation girths and foundations revealed that the foundations underwent a displacement of 121 mm relative to the walls, demonstrating the marked performance of horizontal sliding layers.

(4) Based on predicted strains before mining, a greater ground compression was expected. Therefore, trenches were excavated on both the south and north sides of the club with less structural rigidity. Observations made after ground strains were stabilized showed that the south and north trenches experienced maximum compression strains of 229 mm and 578 mm, respectively. The successful performance of trenches in reducing horizontal strains, particularly compression, was fully borne out. For instance, compression strains of 8.3 mm/m and 14.3 mm/m were absorbed between observation stations N2-S2 and N5-S5, respectively.

(5) Conventional buildings versus antideformation buildings: The antideformation performance of village houses has been described above. Only individual houses showed slight fissures on door and windows. Six conventional houses were severely damaged due to mining, affecting their normal use because of the presence of a maximum tilt of 190 mm and 70 mm wide cracks on walls.

(6) Economic analysis: The cost incurred for applying antideformation measures on a building shares on average 14% of the total cost of its civil work.

## Concluding Remarks

The research-derived accomplishments obtained over the past 3 decades have made important contributions to the development of China's coal industry and have brought about marked economic, environmental, and social benefits. With regard to land reclamation, apart from further improvement of existing technical measures, efforts will be placed on enhancement of land utilization and economic efficiencies and development of ecological and biological measures for reducing the period required for regaining the productivity of affected land. Regarding structural precautions against damage to buildings, the study will be directed towards the development of technology and processes for applying both flexible and rigid structures, which have been so successfully used on new and existing buildings, as well as the structural pattern and design of buildings offering both deformation and earthquake-resistant performance. All the activities planned are aimed at pushing structural precaution technology to a new level.

# RETAINING WALLS ON SUBSIDENCE AREAS[1]

**Bohdan K. Boczkaj[2]**

**Abstract:** In this paper, are derived formulas for calculating the horizontal soil pressure acting on freestanding retaining walls and basement walls of structures erected on mine subsidence areas. Different work conditions resulting in different values of soil pressure in nonmined and mine subsidence areas are analyzed. An example of a structure designed using the derived formulas is also given.

## Introduction

The best results in treating structures subjected to mining-induced subsidence are achieved when the problem is divided into two parts. First, the mining engineer determines the magnitude of ground movements due to mining and transfers to the designer of the structure the value of expected subsidence and its components, such as surface curvature, tilt, and strain, both tensile and compressive. Second, the structural engineer, knowing the magitudes of subsidence components, subgrade properties, such as soil capacity, specific weight, angle of internal friction, and deformation modulus, in addition to the characteristics and service requirements of the designed structure, is able to apply proper solutions to mitigate subsidence influence or to design the structure so that damage is minimized and serviceability is unobstructed by subsidence, both for existing and newly designed structures.

In this paper, the influence of subsidence on horizontal pressure acting on vertical walls is considered, taking into account different conditions of work in mine subsidence areas in relation to nonmined areas. Freestanding retaining walls (without restrains on horizontal movements) and basement walls in buildings in nonmined areas are treated identically, as loaded by active earth pressure or by pressure at rest. In areas of mining-induced subsidence, the conditions of their work are extremely different, especially in compressive zones. The horizontal pressure increases when the subsidence basin forms under the structure and can even reach the magnitude of passive pressure. The existence of increased pressure has been proven by numerous observations of basement wall deformations, when the compression zone of the subsidence trough progresses under the buildings. In older structures designed when the influence of mining on the surface was not fully recognized, the loading on vertical walls was considered to be equal to passive pressure, was assumed to be double the active pressure. Later, based on existing structures observations, it was deduced that increased pressure falls between active and passive pressures, depending on the magnitude of compressive strain.

In the U.S. literature on the subject (Speck and Bruhn 1990) an example is given (figure 1) in which a pressure greater than passive is assumed to load the basement walls. The additional earth pressure (above passive) imposed by subsidence is defined as proportionate to the deformation modulus of soil, without giving values of coefficient of proportionality. The increase in pressure is linked to secondary factors, such as steepening the slope or weakening the wall

[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatment of Acid Drenage, Pittsburgh, PA. April 24-29, 1994.

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by subsidence below the capacity to sustain active pressure. The influence of soil strain is not considered.

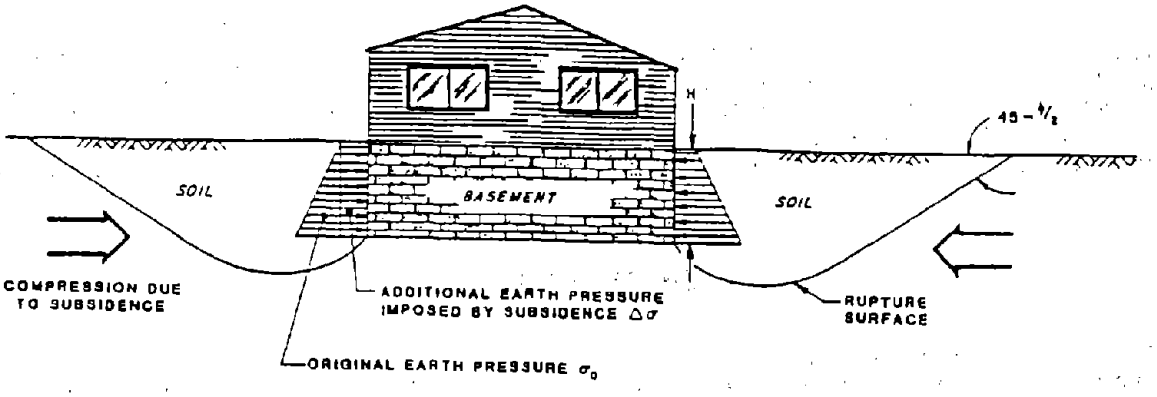


Figure 1. Earth pressure on subgrade walls of building located in compressive subsidence zone (Speck and Bruhn 1990).

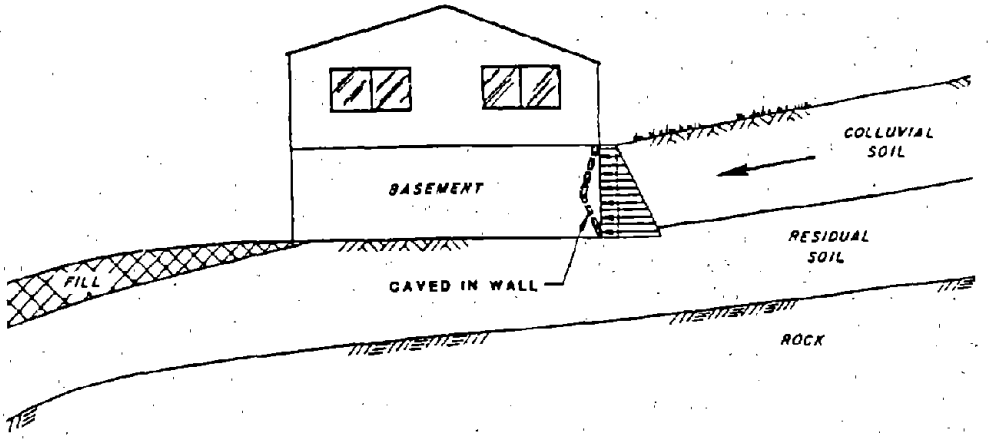


Figure 2. Earth pressure on uphill subgrade wall of building intensified by subsidence-induced hillslop instability (Speck and Bruhn 1990).

In another publication (Drumm et al. 1988) the hypothetical vertical stress distribution is considered (figure 3), derived from soil-structure interaction (i.e. dependent on relative stiffness of wall and soil). This approach also did not take into account the magnitude of compressive strain, which is of primary importance, but rather concentrate on secondary factors.

The remedial, or mitigation, means are recommended without consideration of the stresses or horizontal pressure magnitude, thus ignoring the main reason for mitigation, it means without proving the need for application of such means. Mix-up created by merging the effects of irregular or sinkhole form with regular or due trough form of subsidence implies that authors do not recognize an existence of these two forms of subsidence, which adds to the confusion in reading the cited paper. It is especially visible in the description of displacement in the tensile zone of subsidence basin.

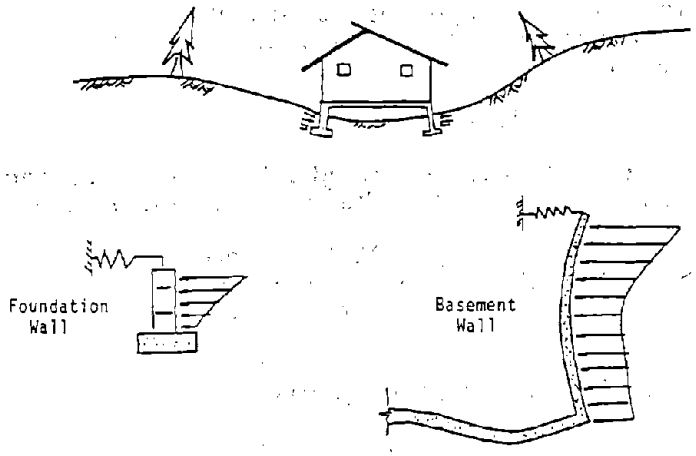


Figure 3. Schematic of plane strain idealization of compression zone (Drumm e t al. 1988).

In both of the above-mentioned papers, the increasing influence of structure dimensions on additional horizontal pressure due to compressive strain is not even mentioned.

## Soil Pressure on Vertical Walls in the Compression Zone

The exact theory of soil pressure on vertical walls in mine subsidence areas has not been derived yet. It would be necessarily based on results of research in which the spread of horizontal stresses in soil halfspace had been solved. At present an approximate solution is used. The assumption is made that soil around the structure is homogeneous and isotropic, with a constant value for deformation modulus of soil in horizontal direction $E_h$. Figure 4 was used in the of derivation of formulas for horizontal pressure.

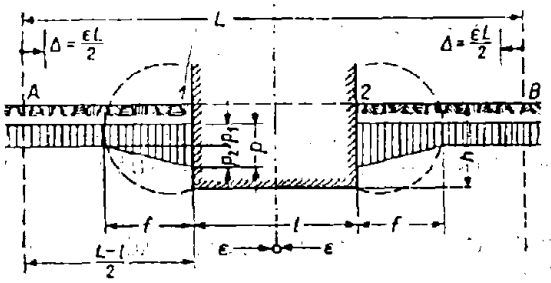


Figure 4. Disturbance zones.

The additional earth pressure (above active), due to action of horizontal strain, uniform in the entire compresive zone, and equal to $p_1$ is disturbed in the vicinity of the building walls by the stiffness of the structure, which restrains soil movement. Outside, in the distance "f" from the walls the influence of the building disappears and so does the disturbance of pressure. The distance "f" depends on building dimensions (length, foundation depth) and soil properties. In sections distant more than L≥l+2f, where l is the length of structure, the increase of pressure due to subsidence reaches value:

$$p_1 = \epsilon E_h \tag{1}$$

in which $\epsilon$ = compressive strain due to mining.
Compressive strain in soil causes the section distant L from each other to move closer for distance $\Delta$.

$$2\Delta = 2 \cdot 0.5 \cdot \epsilon L \tag{2}$$

In the disturbance zone, it means on length "f", pressure increases linearly, and close to the wall reaches value p. The comparision of displacements leads to:

$$0.5\ \epsilon L = \frac{p_1}{E_h}\ \frac{L-l}{2} + \frac{p_2 f}{2E_h} \tag{3}$$

Substituting equation 1 into equation 3 gives:

$$p_2 = \frac{1}{f} E_h \epsilon l \tag{4}$$

The additional pressure on the wall due to mining becomes:

$$p = p_1 + p_2 = \epsilon E_h (1 + \frac{l}{f}) \tag{5}$$

There are different opinions about the distance of pressure disturbence zone "f". Since additional pressure is by nature passive, it is justified to assume that the length of disturbance zone "f" is equal to the passive soil resistance wedge and:

$$f = h \tan(45° + \phi/2) \approx 1.5h \tag{6}$$

so equation 5 takes the final form:

$$p = \epsilon E_h (1 + \frac{1}{1.5h}) \tag{7}$$

The deformation modulus of soil in horizontal direction $E_h$ is generaly smaller than the vertical modulus of soil "E", which is determined by laboratory tests.

It can be taken as: $E_h = 0.4\ E$, for sandy soil and $E_h = 0.5\ E$, for cohesive soil.

The final soil pressure "$p_o$" in zones of compressive strain is equal to the sum of active pressure "$p_a$" and additional pressure "p":

$$p_a = \gamma h k_a - 2c \tan(45° - \phi/2) \tag{8}$$

where $k_a$ = coefficient of active pressure
$k_a = (1-\sin\phi)/(1+\sin\phi)$

$$p_o = p_a + \epsilon E_h (1 + 1/1.5h) \tag{9}$$

The passive soil pressure "$p_p$", is equal:

$$p_p = \gamma h k_p + 2c \tan (45° + \phi/2) \qquad (10)$$

in which $\gamma$ = unit weight of soil
$h$ = height of wall
$k_p$ = coefficient of passive pressure
$\phi$ = angle of internal friction
$c$ = cohesion of soil
$k_p = (1+\sin\phi)/(1-\sin\phi)$.

The real pressure "$p_o$" cannot be greater then passive soil pressure "$p_p$". The condition $p_o<p_p$ is always valid, so it is possible to find the zone in which $p_o$ prevails - zone I, and zone in which $p_p$ prevails - zone II, as shown in figure 5.

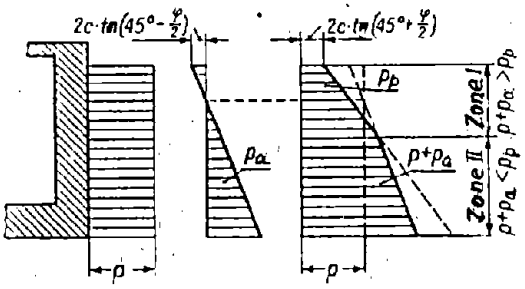


Figure 5. Pressure on wall in zone of compressive strain.

For sandy soils, it means for c=0, only first factor of sum remains, which is function of angle of internal friction $\phi$.

The above-derived equations are valid for walls in which compressive strain causes movements in a direction opposite to the direction of earth pressure. However, when movements in the direction of earth pressure are possible, the actual pressure drops even as low as to the value of active pressure, just as it does in freestanding retaining walls in nonmining areas. So, this also explains why in areas of compressive strain basement walls buckle inward a few inches under additional pressure, but do not collapse. The movement due to buckling is sufficient to reduce passive pressure to the magnitude of active pressure before the walls are forced to collapse. Buckled walls are still strong enough to safely sustain active pressure.

## Retaining Walls

Freestanding retaining walls designed for active pressure (figure 6), when pressure increases, tend to move in the direction of pressure by rotating or sliding, so actual acting pressure is reduced to the value of active pressure, for which the wall was originally designed anyway. The behavior described above is typical for nonmined areas. For the mine subsidence area, additional analysis should be performed.

The additional pressure $p= \epsilon E$ can be generated only when sliding or rotation in the direction of pressure is restrained. In most cases, the safety factor for rotation is larger than against sliding, so for simplicity it is assumed that

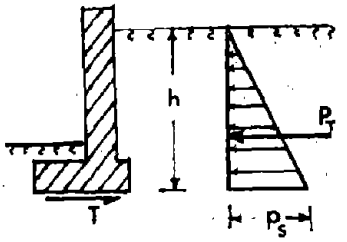


Figure 6. Retaining wall.

design the safety for sliding is critical. The maximum force the retaining wall can resist is equal to:

$$T = G\, f_m \tag{11}$$

where G = vertical load (weight) of retaining wall for unit width and
$f_m$ = largest value of coefficient between soil and foundation.

Active pressure "$p_a$" and additional pressure "p" due to compressive strain produces force $P_T$:

$$P_T = P_A + P = 1/2\ p_a h + \epsilon E_h h \tag{12}$$

Since it is always $T \leq P_T$ or $T \leq P_p = 1/2\ h\ p_p$, the earth pressure cannot grow larger than the value:

$$p_s = \min[p_a + 2\epsilon E_h,\ 2Gf_m/h,\ p_p] \tag{13}$$

As a general rule, it can be concluded that loading on freestanding retaining walls in subsidence areas cannot be greater than $p_s$. It is obvious that the factor of safety for retaining wall against sliding using minimal values of coefficient of friction $f_s$, as well as for rotation, should be checked and satisfy code requirements, as for structures on non-mined areas.

The largest values of coefficient of friction $f_m$ for applied materials should be estabilised by research and tests. Based on observation of existing

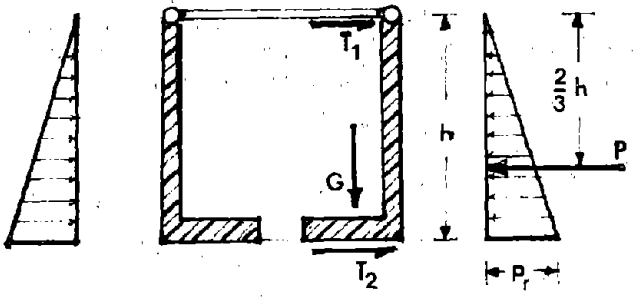


Figure 7. Schematic system of tunnel walls.

structures, mainly bridges subjected to mining subsidence, for three layers of tarpaper $f_m$=0.6 were used. No demage was reported to structures serving long term on mine subsidence areas when this value for $f_m$ was used in their design.

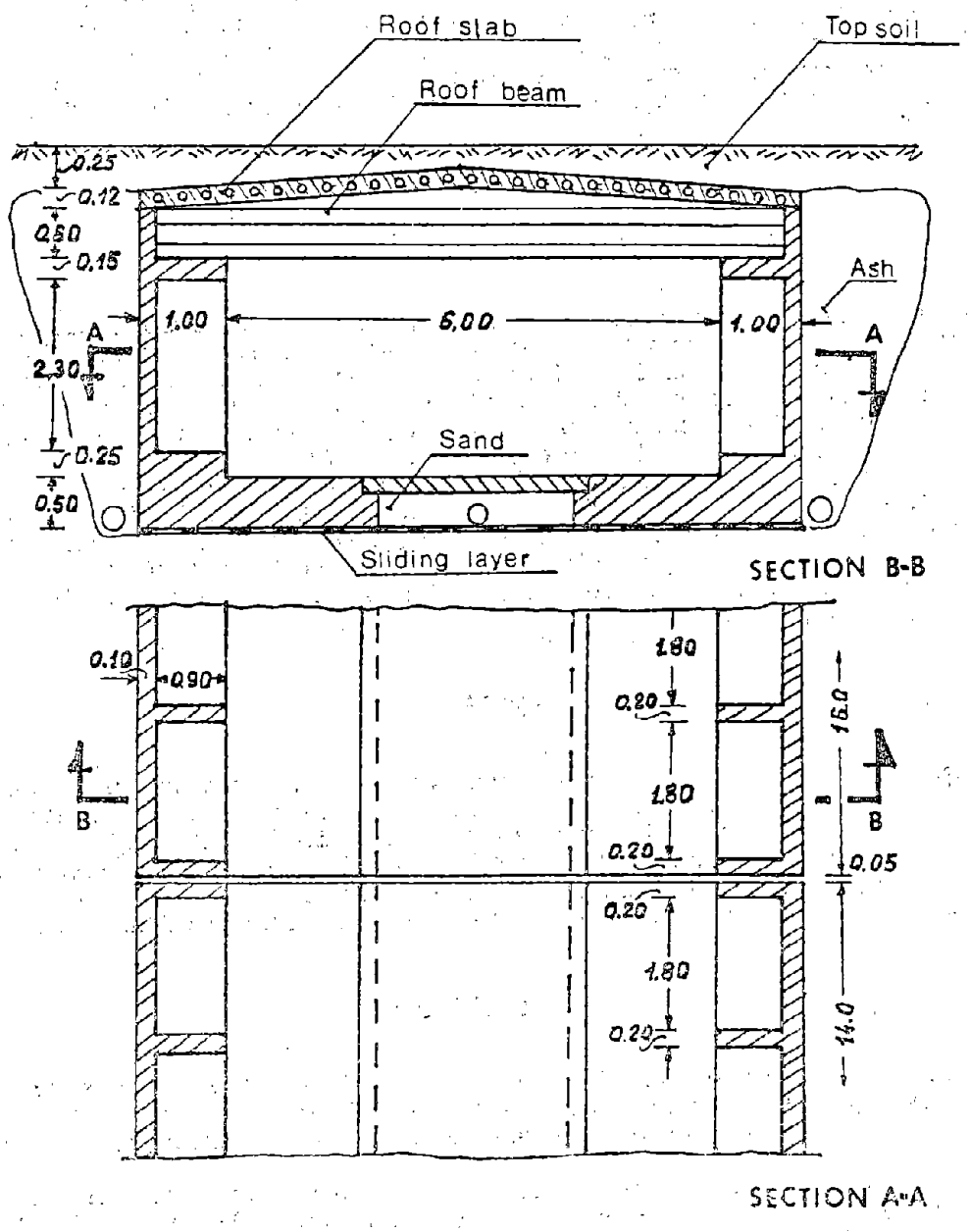


Figure 8. 6.0 m wide tunnels.

The significant reduction of loads and mitigation of mine subsidence influence can be achieved in all these cases for which the system of sliding or

rotating walls under mine-induced strain is applicable. An expample of the application of the sliding walls system, the walls in tunnels for pedestrian under a traffic circle is presented here (Boczkaj et al. 1990). Figure 7 shows the method of calculating the largest possible pressure on walls.

The safety of walls was checked for active pressure with minimal coefficient of friction. Walls against horizontal movements were supported by ceiling beams on top (force $T_1$), on the bottom movements were restrained by friction (force $T_2$). The maximum resisting force $T_2$ was calculated using equation 11. Equilibrium condition provides the condition $T_1 = 1/2\ T_2$. Extremel horizontal pressure P is equal 1.5 $T_2$, which leads to the following equation for soil pressure:

$$p_r = \frac{T_2}{3h} = \frac{f_m G}{3h} \qquad (14)$$

The pressure "$p_r$" calculated from equation 14 was almost six times smaller than pressure "$p_o$" according to equation 9, which would be valid for a case when sliding has not been possible. Figure 8 gives a more detailed view of tunnel sections and special construction joints placed in distances not exceeding 16 m to eliminate excessive bending moments in walls due to surface curvature caused by mining. Two seams of coal under the tunnels were removed and produced 600 mm (24 in) of subsidence. Tunnels were subjected to horizontal strain, compressive and tensile, equal 3 mm/m, or 0.003. The structure is periodically inspected and is reported in excellent shape after more then 25 years of service. It also attests to the fact that properly chosen structural systems on mine subsidence areas can be economically successful, with the total cost in the range of the cost for nonmined area.

## Conclusion

The problem of loading on retaining walls and basement walls requires more research and testing, until an accurate, rigorous theory is created. However, until a such theory will is presented, the simplified, approximate methods presented in this paper can serve the purpose. Utilization of experience gained in the prevention or mitigation of mine subsidence influence can lead to substantial savings when a proper system of structure is designed.

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# SERVICEABILITY CRITERIA FOR BUILDINGS IN MINE SUBSIDENCE AREA - ADJUSTMENT TO EUROCODES[1]

Marian Kawulok[2] and Zdzislaw Sulimowski[3]

**Abstract:** Due to ground deformations caused by underground mining, building structures in mining areas are frequently subjected to considerable deformations and damage to the finishing and structural elements. As a consequence, serviceable values of such structures are distinctly diminished, and in extreme cases, seriously damaged structures may be exempted from further service. These problems are not duly represented in the existing building standard codes. It is also important to determine relationship between damage stage and value of the building, and hence the strategy for reconstruction or renovation works. The paper presents proposals concerning serviceability criteria of building structures in mining areas, in terms of basic standard requirements valid in building in Poland, as well as proposals of Eurocodes. Building structures under consideration have been divided into structures designed to resist mining influences and existing structures, not adapted to conform to these influences at the design and erection stages.

**Additional Key Words:** building in mining areas, limit states, building serviceability, structure deformations.

## Characteristics of the Problem

According to Eurocode 1 (1993), calculations of building structures should be made using the limit states method, assuming possible occurrence of: Ultimate Limit States (ULS), i.e. load states that, if surpassed, cause damage to the building structure or its part; and Serviceability Limit States (SLS), i.e. load states that, if surpassed, result in excessive deformations of the structure,limiting its further serviceability as intended in the design; SLS are described by the deformations and displacements of the structure. In the application of structure design in mining areas, the standard criteria of ULS usually require a wider analysis compared to the design for structures outside the mining areas, which results from the specific character of loading and behavior of structures in subsidence - prone areas.

ULS of a structure determine its safety and must show a sufficient margin of load capacity. SLS may undergo some transitional modifications. In the narrow sense, SLS imply technical serviceability viewed from user's standpoint (e.g. doors and windows opening well) and proper functioning of machinery and equipment. In the broad sense, it implies satisfying the psychological and esthetic habits (e.g. no large tilts or deflections). Some questions of durability are also involved here (e.g. cracks in reinforced concrete). Finally, it concerns situations where damage to minor elements entailing repair works should not occur ( e.g. partition walls or linking layers).

In reference to designed structures, the analysis of SLS is generally limited to changes in some parameters (mostly movements or deformations), that are allowed by building and engineering design standards. From the viewpoint of loading and strength values accepted for verification of SLS, the probability that these states will occur is greater than that of ULS.

Extensive observations of buildings designed to withstand the effects of mining have shown that, usually

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during the serviceable period of structures (hereafter referred to as structure service), allowable values of parameters determined in standards defining the accomplishment of SLS are exceeded. Despite this fact, building failure is very rare.

In principle, existing structures behave likewise in mining areas.When more intensive ground deformations occur, the structures undergo great deformations and damage, the extent of which is usually greater than in structures designed with respect to mining effects, and may surpass many times the allowable values, as determined for SLS by appropriate design standards. Such situations do not necessarily mean building failure. Despite considerable damage to some elements (e.g. walls affected by cracking), emergency hazard may not occur with respect to the structure as a whole ( e.g. though walls are damaged, there are no floor failures). Structure serviceability conditions may be considerably deteriorated, but the building can sill fulfill its original purpose. Consequently, in both designed buildings and in buildings already existing, the interval between destruction of the building and the state of (acceptable) service standard is considerable.

The problems of structure service in mining areas are also important legally. Damage to a structure resulting in comfort deterioration and service impediments lead to a reduction in market value of the structure.

## Basis for Determination of Serviceability Criteria

### Structures Designed for Mining Loads

In general, two radically different concepts of structure building in mining areas can be considered. The first concept, which is more likely to be introduced due to regulations in force, is that new structures should be designed in such a way as to satisfy the requirements specified by ULS and SLS over the whole period of their service. With regard to the present mode of building in mining areas, this concept necessitates a considerably larger range of protection to ensure better structure preparation to compensate for subsoil deformations (e.g. increased construction rigidity compared with that used at present; rectification of habitable buildings on a general basis).

The second concept refers to the temporary design situation anticipated by Eurocodes, and would require that building design be based on the conditions of ULS and requirements of SLS mitigated temporarily during periods of mining effect activation. In the course of structure service, some damage and impediments may arise,but may pose no threats to its safety. Once the ground has settled, repair work will begin to restore basic parameters of SLS.

From the viewpoint of economic effectiveness of building in mining areas, the second concept is more rational. Less means and materials are used, and they are limited to situations that really require intervention. Time gained through shifting the intervention until later allows mineral extraction to proceed, which generates money that can be used to repair damage. On the other hand, it implies discomforts or even occasional threats to structure service.

The authors of this paper are in favor of the second concept of building in mining areas and suggest that Temporary Serviceability Limit States (TSLS) be worked out and introduced, as explained in Fig. 1a.

With program loadings typical of regular areas, the structure is designed in accordance with required standard states of ULS and SLS. While mining ground deformation effects are activated, the structure must still meet the requirements of ULS, allowing however at that time increased deformations of construction defined by TSLS. Calibration of TSLS level must result from two prerequisites as follows:
- structure deformations can cause impediments in its service to a commonly accepted degree and do not eliminate any structure part from service,
- deformations and structure damage are repairable and economical.

After most subsidence has occurred, the structure is returned to the initial state, satisfying the SLS

requirements. A structure characterized by deformations that exceed the SLS level should be repaired. It is also possible to assume a strategy advocating restoration of structure serviceability parameters, for buildings that can not withstand the effects of mining (existing structures).

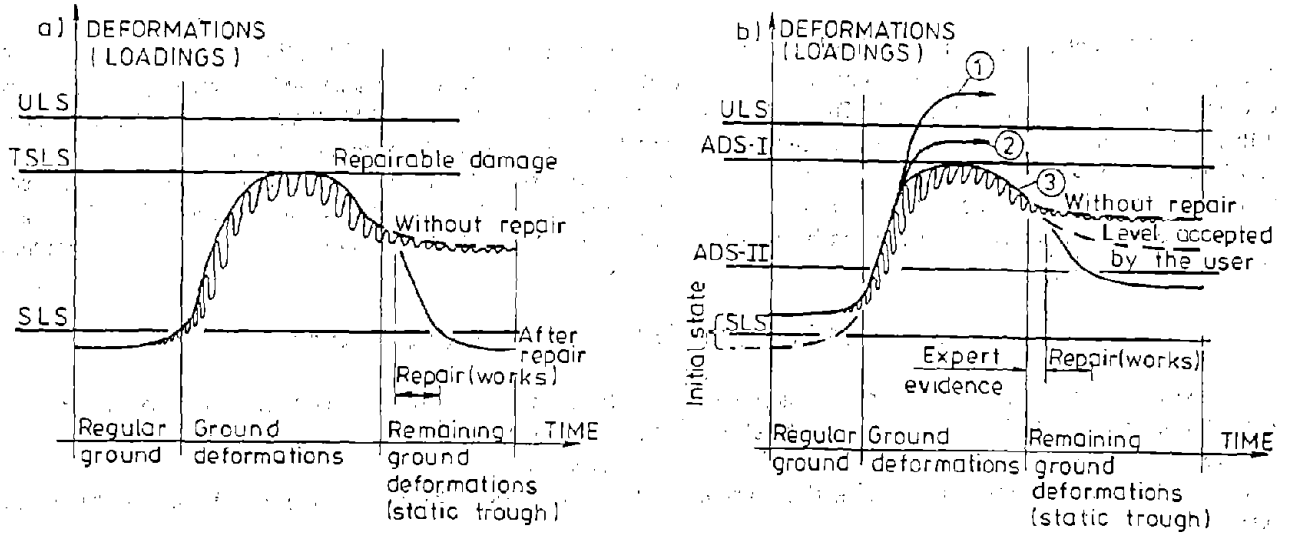


Figure 1. Deformations observed for a) structures designed to withstand effects of mining and b) existing structures.

Existing structures

Similar arguments can be carried out with regard to existing structures. In this case, ULS should include building structure, together with elements for preventing damage by mining effects (e.g. steel bowstrings, reinforcement bowstrings). In both concepts of structural maintenance, it is necessary to apply the range of precautions to hinder the destruction of the structure or its part. Yet, the range of these precautions can vary, depending on the nature and intensity of structural deformations and damage, resulting from the repair costs and damage level acceptable by the user (e.g. if certain specified damage to masonry of cellar walls is accepted, only the above-ground building part is protected). Apart from certain cases where following specific technological incentives are necessary, reasonable technical and economical action in regard to existing structures must lead to abandoning the idea of satisfying the conditions of SLS, obligatory for designed structures. Serviceability of these structures necessitates, however, providing the user a specific level of utilization comfort. This, on the other hand, is indicative of the need for introducing an extended equivalent of TSLS with regard to existing structures, that can be referred to as Allowable Deformation States (ADS). Fig. 1b illustrates this procedure. Initial state of deformations in existing structures is usually higher than the level accepted for designed structures. In individual cases, when the structure "has been constructed" with no consideration for present requirements of SLS, the structure deformations may exceed the SLS level. Mining loads resulting from ground deformation generally lead to the following situations:
(1) - ULS are exceeded, which results in destruction of the structure,
(2) - the structure has undergone so much damage that its repair is uneconomical; the structure should be demolished or its service mode changed,
(3) - structure deformations are limited, and their range justifies repair works.

As for the last situation, constituting the core of our investigation, structure deformations can not at the same

time exceed some boundary values enabling structure service in an appropriate way. This level of deformations can be determined as 1st ADS (ADS-I). This state is of transitional character, and the level of these deformations need to be determined in detail with reference to TSLS level. It can be supposed that with regard to some deformation parameters of structure not protected against the effects of underground mining, the level can exceed the TSLS level. After subsidence abates, damage and repair requirements should be estimated.

Consequently, the structure should be brought to a commonly accepted deformation state, that is, the state exempted from all negative influences on the substantial number of users (e.g. construction tilt imperceptible as major discomfort to the majority of users). This deformation level would be of permanent nature, and can be defined as 2nd ADS (ADS-II). The difference between ADS-II level and the initial state should be considered a social cost resulting from underground mining. The range of repair works that does not have to meet ADS-II level could, in such a case, be established in the form of individual agreement between the mine and the owner (user). The last stage comprising also the expert evidence can be also applied to existing buildings that have been designed to withstand the effects of mining.

## Parameters of Serviceability States in Apartment Buildings

With reference to other types of structures, the following factors describing serviceability states in mining areas should be assumed as a basis for formulation of deformation parameters :
- characteristic deformations of structures subjected to the effects of mining,
- general requirements concerning SLS and legal standard norms for structures situated on grounds outside the mining effects,
- current norms of legal standards concerning mining areas,
- recommendations of technical literature with regard to allowable deformations and damage to structures,
- possibility of calculable evaluation and calibration of accepted parameters of structure deformations.

The suggested parameters of structure deformations determining serviceability states of apartment buildings and public buildings in mining areas according to the general concept presented in Fig. 1, are discussed below.

### Designed structures

SLS are checked regardless of loading caused by mining effects; these requirements should also be met during remaining ground deformations, although it can be conditioned by the need to repair the structure.

TSLS are obligatory during the subsidence; they can be determined by corrected (mitigated) values of parameters describing SLS, including in particular:
1) Allowable ground deformations associated with the occurrence of cracks and deformations of wall supporting structure, or filling of frame constructions, defined by the characteristic (averaged) relative deflection of structure:

$$\gamma_k = \frac{2 f_k}{L}$$

where: $f_k$ - structure's deflection hogging (Fig. 2a) or sagging (Fig.2b)

2) Allowable tilt of building, $T_b$
3) Allowable width of cracks in reinforced concrete elements.

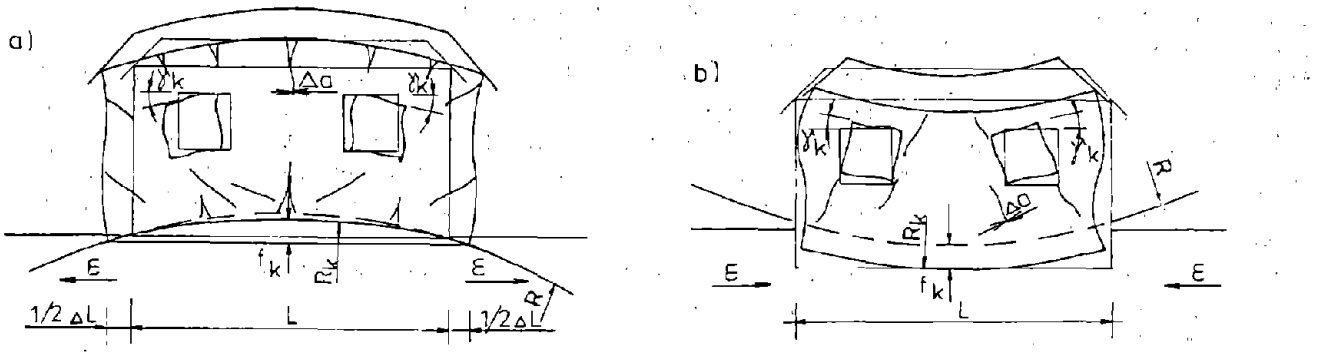


Figure 2. Deformations of a building caused by a) hogging and b) sagging

Existing structures

ADS I are required during the activation period of main mining effects, and are characterized by :
1) Allowable structure deformations connected with occurrence of cracks and deformations that attenuate climactic resistance(tightness) or other wall functions (e.g. acoustic resistance, deformations of conduits and distortion twists of windows and doors), determined by the following parameters of structure deformations (Fig. 2):
a) characteristic value of relative deflection of structure, $\gamma_k$,
b) index of total structure elongation, $\Delta L$,
c) degree of plaster cracks and other fittings, p, possible to be determined only after damage occurrence, e.g. as the ratio of cracked surfaces to the whole surface of the wall (walls) or ceiling.
2) Allowable width of single wall cracks, $\Delta a$,contributing to considerable attenuation of building tightness.
3) Allowable tilt of building, $T_b$.

ADS II are required after most mining effects are past, during the occurrence of remaining ground deformations, or when the structure is influenced by the static mining trough, and are described by:
1) Allowable - general deformation of structure defined by the characteristic value of relative deflection of structure, $\gamma_k$,
2) Allowable width of single wall cracks, $\Delta a$, contributing to the reduction of building tightness,
3) Allowable tilt of building, $T_b$.

## Factors Determining the Values of Allowable Deformations and Damage of Building

Limitations of building structure deformations result from the need to satisfy serviceability states, generally determined by :
a) the possibility of structure service respectively to its purpose,
b) the limitation or elimination of impediments in structure service,
c) the aesthetic reasons, and
d) requirements concerning durability or tightness of structure.

According to the presented concept, in structures situated in mining areas, particularly during the period of mining effects activation, and in some cases also after their regression, it is possible to mitigate standard requirements of SLS. It necessitates calibration of new values of structure deformations that can generally be referred to as allowable deformations.

Taking into consideration the above conditions, the allowable deformations must be in compliance with condition "a", but they may lead to certain lowering (mitigation) of requirements specified by conditions "b", "c" and "d". The problem of value determination of allowable building structure deformations concerns, in principle, all cases of structures built on subsiding grounds, and in particular on grounds subsiding more intensively and irregularly. Thorough and extensive research conducted in this field leads only to the determination of correlation between the deformation state and structure damage, (Skemptom and Mac Donald 1956). It is difficult,however, to define the level of allowable deformations of building structures as meant in this paper. Such a formulation implies subjective character, and is dependent on the requirements and habits of the individual user (resident).

According to Eurocode 7 (1992), the following factors should be taken into consideration in determining the allowable deformations values:

- the degree of confidence, wherewith the acceptable (tolerated) boundary values can be determined,
- type of structure,
- type of structure material,
- type of foundation,
- kind of ground,
- anticipated manner of structure service.

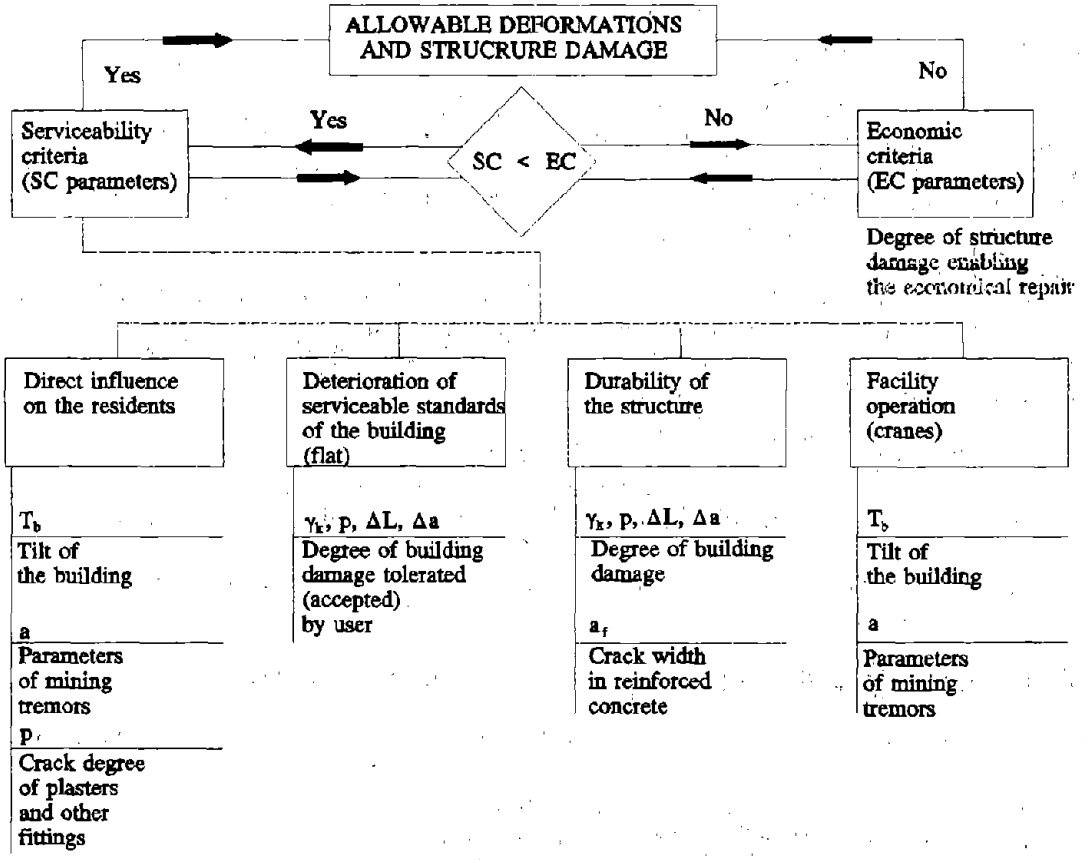


Figure 3. The scheme of allowable deformations and structure damage criteria.

The proposed criteria defining allowable deformations and structure damage, as well as corresponding deformation parameters in apartment buildings are presented in Fig. 3. Allowable structure deformations should be

determined by serviceability criteria and economic criteria. The latter should be understood as the degree of damage to the structure whereof the repair and the maintenance are still economical.

## Concluding Remarks

The presented concept necessitates definition and determination of values of allowable parameters describing service of designed structures and existing structures on subsided grounds, especially in areas subjected to the effects of underground mining.

The determination of allowable deformation values will certainly be difficult and may be controversial, depending on the criterion. Yet, the necessity to work out and calibrate such values seems to be imperative and legally necessary. Due to substantial costs of protecting of building structures in the erection stage, aimed at preventing damage occurrence not exceeding parameters of SLS, it may soon turn out that extraction of minerals is unprofitable, or location of building structures in mining areas surpasses the investors' financial possibilities. With reference to the existing building structures in mining areas, a wider range of repairs might be demanded (e.g. along with straightening of tilted buildings), even with a relatively small level of structure deformation.

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# REINFORCEMENT OF RESIDENTIAL MASONRY FOUNDATIONS TO MINIMIZE DAMAGE DUE TO MINING-INDUCED SUBSIDENCE[1]

Richard A. Allwes[2] and Clark P. Mangelsdorf[3]

**Abstract:** The differential movement of the ground during longwall mine subsidence may damage residential structures by overstressing their structural members and by tilting the structures to unacceptable levels. Three subsidence mitigation techniques are commonly practiced in the United States: trenching to alleviate ground pressure buildup on the foundation masonry walls, bracing to reinforce masonry foundation walls, and jacking the superstructure off its foundation to minimize damage due to bending and racking. Despite these precautionary measures, the foundations usually need expensive repair or replacement owing to structural failure or tilt. Information on subsidence upgrade of existing residential structures is sparse, and few subsidence upgrade-methods are laboratory and field tested. As a result, the U.S. Bureau of Mines is investigating methods of protecting and reinforcing existing residential masonry basement foundations to minimize longwall mining subsidence damage.

This paper presents five full-scale in-plane bending tests that were conducted on a single, originally plain masonry wall. The purpose of the laboratory tests was to evaluate two postreinforcement designs that were proposed for existing residential foundations. The laboratory tests showed that the combination vertical and horizontal posttensioned tendons significantly increased the resistance of the masonry wall to in-plane bending.

## Introduction

A majority of the literature available on the design of structures for protection against mine subsidence is for commercial, not residential, structures (Mautner 1948, Wasilkowski 1955, Healy and Head 1984). The designs are based upon the predicted final transverse profile of the subsided ground and do not take into account the dynamic ground movements that may be more severe and damaging (longitudinal and transverse profiles of a longwall panel). The National Coal Board's (NCB) subsidence prediction model is the most widely referenced in literature, but other subsidence models, such as Subside, SPASID, and SUBPRO, are available and are more appropriate for U.S. mining conditions (Ingram et al. 1989). The dated subsidence models usually only predict the final surface subsidence and strains for the transverse profile lines of longwall and/or retreat room-and-pillar sections, while recently developed prediction models provide three-dimensional dynamic ground movement data (Adamek et al. 1992). Simplifying assumptions are usually made concerning the soil-structure interaction, and the three-dimensional response of structures to mine subsidence is rarely reported.

Broad recommendations and guidelines have been made concerning the design and construction of residential structures in mine subsidence areas (Yokel et al. 1981, Basham et al. 1981). Suggestions for minimizing subsidence damage to existing structures are also made. Torsion of the foundation is rarely discussed, and methods of increasing the torsional rigidity of the foundation have not been pursued. No

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provisions are made in the proposed designs for leveling the superstructure and foundation following the subsidence event. None of the proposed designs have been tested in the field except for the response of a partly constructed reinforced masonry foundation (Powell et al. 1988). Laboratory and field testing of postreinforced crawl-space masonry foundations is reported in the literature (Marino 1992). The results are favorable for the crawl-space foundations tested.

Earthquake engineering research has produced a potential method of reducing subsidence damage to structures. The procedure involves the installation of an active spring system of support into the residential foundation (Hueffmann 1990). This system has only been used in Europe and no installations presently exist in the United States. The cost-effectiveness and practicality of retrofitting this system into a residential foundation are undetermined.

An investigation by the Bureau of subsidence insurance claims filed with the Commonwealth of Pennsylvania shows that 34 claims for longwall mine subsidence damage were processed during the years 1987-90. The cost to repair the residential structures as a result of structural damage and/or tilt exceeded $1.4 million. This figure would have been higher but the ceiling limit of $100,000 was imposed on two of the structures. In addition, a number of the residential structures were never releveled. Releveling the structures would have significantly increased the repair costs. It is estimated that the cost to repair or replace foundations damaged from longwall mine subsidence accounts for 70% to 85% of the total repair cost of residential structures (Motycki 1991).

The Bureau is conducting research to increase the limits of strength and deformation of residential basement foundations, since approximately 80% of the subsidence damage costs is attributed to their repair or replacement. Postreinforcement of existing residential foundations will increase their limits of strength and deformation and is a potential method of mitigating subsidence damage. Methods are also needed to reinforce foundations so that they may be releveled in a cost-effective manner rather than lifting the superstructure off its foundation and rebuilding a level foundation. The challenge is to develop postreinforcement methods that are cost-effective and feasible to retrofit onto an existing masonry foundation.

In this Bureau paper, five full-scale in-plane bending tests (hogging tests)[4] are described that were conducted on a single, originally unreinforced masonry wall at the Civil Engineering Structural Laboratory, University of Pittsburgh. The laboratory tests were conducted to evaluate two postreinforcement designs that were proposed for existing residential foundations. The types of steel reinforcement evaluated were vertical and horizontal posttensioned tendons (wire rope).

## Experimental Test Program

Plans were initially developed for testing novel posttensioning designs for masonry walls under laboratory conditions. The motivation for posttensioned reinforcement is that it is an active, not a passive, reinforcement system. Active reinforcement systems are not dependent upon the deformations within a structure to mobilize resistance to internal loads as the structure reacts to applied loads and displacements.

[4]A foundation subjected to a negative bending moment (designer's sign convention) is commonly referenced as a foundation that is hogging. A foundation that sags is subjected to a positive bending moment.

## Vertical Tendons and Anchorage System

Prior to conducting in- and out-of-plane bending tests on a posttensioned masonry wall, an evaluation was made to determine (1) the feasibility of installing wire rope in the interior (cores of masonry units) of the masonry walls[5], (2) the optimum anchorage system for wire rope reinforcement, and (3) the availability of equipment that will meet the space limitations of the foundation-floor interface. Anchorage systems evaluated were different types of bonding agents and mechanical anchor-bonding agent assisted systems.

The selection of an optimum anchorage system was based upon a series of static pull tests and an evaluation of the ease and reliability of the installation. A concrete pad (1.83 m by 1.83 m by 0.15 m) with an average unconfined compressive strength of 33.66 MPa was poured and served as the host medium for the pull tests. The pad can accommodate 121 pull tests (holes were drilled on 15.2-cm centers.) The pad rests on a 5.1-cm semirigid foam layer (0.22 MPa crushing strength). The purpose of the foam is to provide a material that will offer minimal resistance to bottom-hole blowout. In general, when a hole is being drilled through concrete with a compressive strength less than 41.4 MPa, the bottom of the hole has a tendency to blow out from the percussion blows of the rotary hammer drill. Bottom-hole blowout will reduce the effective bonding length of the anchorage system and compromise its overall pullout capacity.

The first series of tests were conducted to observe the tendency for bottom-hole blowout during drilling and to evaluate three bonding agents (Ramset, Esco, and Rawl) for the wire rope to be used to posttension the masonry wall. Fifteen pullout tests were conducted to evaluate a ceramic-filled epoxy polymer, an acrylic epoxy, and a polyester resin as a bonding agent for the wire rope. Ten tests involved wire rope with straight ends, and five tests involved wire rope with bird cage ends, in which a portion of wire rope was unraveled to increase the wire rope-bonding agent area for increased bond strength. The tests showed that bottom-hole blowout occurred but was not a detrimental problem and that the bird cage ends prevented failure of the grout-steel interface, which was observed in a majority of the tests conducted on wire rope with straight ends. In all cases with the bird cage ends, the strength of the anchorage system exceeded the load-carrying capacity (107 kN) of the wire rope. Although the tests showed that the epoxies manufactured by Rawl and Ramset were basically identical, the decision was made to use the ceramic epoxy manufactured by Ramset.

The first method used to create the bird cage ends was labor intensive. Each individual strand needed to be cleaned since the wire rope is lubricated with grease during fabrication. A small residue of grease severely compromises the integrity of the wire rope-epoxy bond. Since cleaning the individual strands was so labor intensive, a new method of constructing bird cage ends for wire rope was developed. This method consists of making a bird cage end is first to crimp a compression ring onto the twisted ends of the wire rope to hold the individual strand ends together. Then place approximately 15.3 cm of wire rope in a lathe and twist the rope in an open position (permanent diameter of 3.8 cm) to form the bird cage. This method is much simpler and less labor intensive. The bird cage end is then thoroughly cleansed with a biodegradable organic solvent degreaser to remove all grease. In all cases, the strength of the anchorage system using this method to form the bird cage end exceeded the load-carrying capacity (107 kN) of the wire rope (fig. 1).

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[5]The use of steel cables was considered early into the project but the flexural rigidity of the cable eliminated their potential for use as vertical tendons. It was nearly impossible to force the cable into a tight radius so that it could be threaded down through the masonry unit cores.

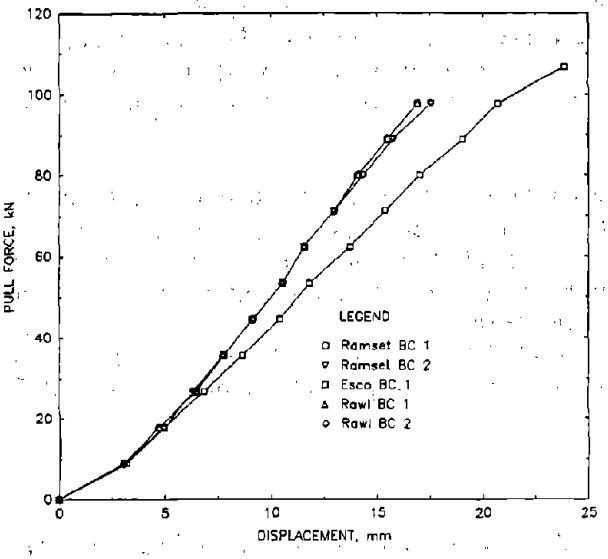


Figure 1. Pullout test results for wire rope with bird cage ends.

## Masonry Wall

The overall dimensions of the hollow masonry foundation wall were 2.64 m high by 7.32 m long (fig. 2). The concrete footing was 50.8 cm wide, 20.3 cm high, and 7.32 m long and had an unconfined compressive strength of 33.66 MPa at 28 days. The wall was constructed of 194-mm two-core concrete masonry units with flanged ends. The masonry units were laid in running bond with type N mortar. Only the face shells and flanged ends were mortared. The masonry wall was built in the laboratory by an experienced mason using normal masonry construction practices and was air cured in the laboratory for 90 days before the structural testing program was initiated. The mass of the entire wall was 6,000 kg (estimated from hydraulic actuator pressures); the mass density of the concrete was 2,323 kg/m$^3$. The mass of the masonry assemblage per unit surface area was 238.3 kg/m$^2$.

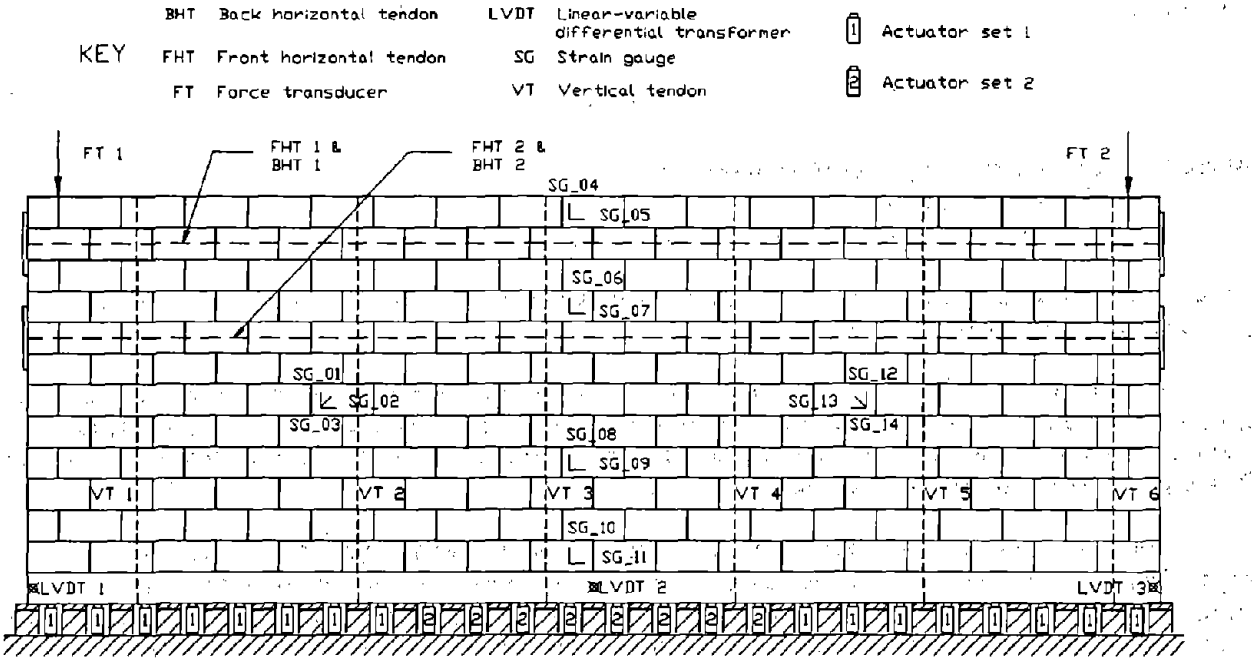


Figure 2. Masonry wall details.

The hydraulic loading system consisted of twenty-four 107-kN actuators and a hydraulic power unit with a complement of valves and hoses. Each actuator in a set shared a common hydraulic line so that uniform but different pressures could be maintained within each actuator set (fig. 2). As a result, all actuators applied an equal force within each set, but different forces could be maintained by each set. By varying the pressure within each actuator set, the magnitude of the maximum internal bending moment in the masonry wall was controlled.

The instrumentation used to monitor the hydraulic load and static frame systems and the deformations of the masonry wall were pressure transducers (PT), displacement transducers (LVDT), strain gauges (SG), and force transducers (FT). The force transducers were used to monitor the reaction forces in the two static frames that were used for two of the hogging tests, as will be discussed later in this paper[6]. The 120-ohm strain gauges were mounted on both sides of the wall and configured to form two opposite active arms of a 4-arm bridge. This doubles the output from strains due to in-plane bending and shear, and cancels out strains resulting from out-of-plane bending. A Campbell Scientific data logger and multiplexer were used as the data acquisition system.

## Posttensioned Reinforcement

The masonry wall was posttensioned with vertical and horizontal tendons (fig. 2). Table 1 provides information for each corresponding test in which the reinforcement was used and the amount of tension in each tendon. (The hogging tests are designated by the acronym HG followed by the test sequence number.) The tension in the vertical tendons was measured by monitoring the pressure in the hydraulic actuator used to tension the tendons. The values obtained were not adjusted to compensate for the additional resistance due to friction in the actuator, nor for the loss in tension when the chuck jaws seat. A reusable chuck was used to hold the wire rope during and after the tensioning process. The tension in the horizontal tendons was determined using a proximity gauge and frequency counter as the horizontal tendons were torqued to the desired tension (Mangelsdorf 1983). The tension (H) in the wire rope was determined using the expression

$$H = 4\,\omega\,L^2 f^2, \tag{1}$$

where f = fundamental frequency of vibration, Hz,
L = wire rope length, m,
$\omega$ = wire rope mass per unit length, kg/m.

Time-dependent losses, due to creep, were not considered a factor because of the short time period between poststress and the in-plane bending tests.

The wire rope used for the vertical tendons was designated as 6 x 37 classification (Bridon American Corp.). The selection of this wire rope was based upon its flexibility and strength requirement. The diameter of the rope was 11.11 mm, and its nominal tensile strength was 91 kN. The specified mass per unit length of the wire rope was 0.48 kg/m.

The wire rope used for the horizontal tendons was 15.9-mm-diam Dyform®-18 HSLR rotation-resistant rope. The wire rope was selected because of its strength and resistance to rotation when tensioned. The nominal tensile strength and mass per unit length were 202 kN and 1.18 kg/m, respectively. The ends of the wire rope were swaged and fitted with standard sleeves (3.18-cm-diam, 14-cm-long, 10.8-cm thread length).

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[6]Reaction frames are not shown in figure 2, but their point of application is indicated by FT 1 and FT 2.

Bar stock (2.54-cm-wide) was welded to the ends of the swaged fittings so that the wire rope fitting could be held and torqued to the proper tension.

To install a vertical tendon, a portion of the face shell along the first course was removed. This allowed access to the top of the footing inside the masonry unit so that a 2.2-cm-diam, 20.3-cm-long hole could be drilled into the footing. The angle of the hole was approximately 14° from the vertical and the angle was governed by the geometry of the opening in the masonry unit. The hole was filled with a ceramic-filled epoxy, and the bird cage end of the wire rope was forced into the hole after it was snaked down through the cores of the masonry wall. A reusable strand chuck (11.11 mm) was used to grip and maintain the tension in the wire rope at the top masonry course as the rope was tensioned.

## Hogging Test Program and Results

Five series of hogging tests were conducted on the masonry wall. The first hogging test (HG 1) performed was on the masonry wall with no reinforcement installed. HG 1 test conduct was as follows: The masonry wall was raised 2.5 cm off the wooden supports with all actuators operating from a common manifold (fig. 2). Actuator set 2 was locked in, and the pressure in actuator set 1 was slowly decreased at a rate of 1.4 kPa/s. (A micro-needle valve allowed such slow rates to be set). The masonry wall experienced a progressive staircase failure: tensile failure along its head joints and shear failure along its bedding joints early into the in-plane bending test (fig. 3). The moment-deflection curve is shown in figure 4. The moment resistance for the unreinforced masonry wall was calculated at its center, and the deflections of the masonry wall ends were measured with respect to the center of the masonry wall. The results of this test provided a baseline for evaluating the effectiveness of the proposed two reinforcement designs and also provided data for numerical modeling.

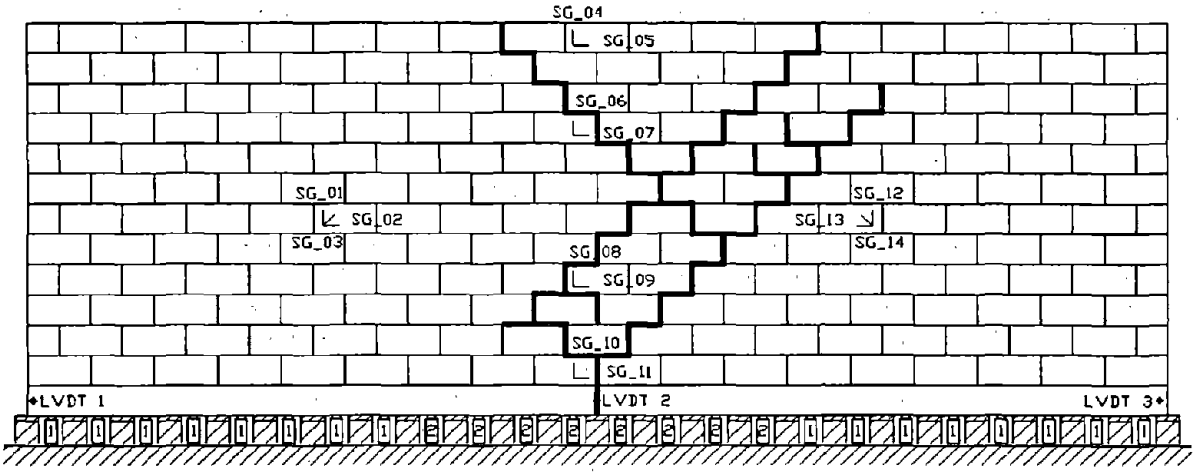


Figure 3. Failure of unreinforced masonry wall (HG 1).

Vertical tendons were then installed in the masonry wall[7] for the second test (HG 2, fig. 2) and posttensioned to the values provided in table 1. The purpose of the vertical tendons was to apply additional poststress to the wall in order to increase the frictional forces between the masonry units above those due to a superstructure load. It was believed that the additional frictional forces would increase the overall bending strength of the masonry wall. The posttension reinforcement also served as a means of repairing

[7]The masonry wall was not repaired prior to this test or any of the additional hogging tests. This was felt to be the most conservative approach for evaluating the effectiveness of the posttension reinforcement.

the failed structure. The test conduct of HG 2 mirrored HG 1. The mode of failure for HG 2 was different than for HG 1 (fig. 5). Failure initiated in the top three courses, and the failure process stopped until the applied loads increased the bending tensile stresses to the level that the tensile strength of the masonry unit (masonry unit in fourth course from the top) was reached. Once this level was reached, the entire masonry wall failed. The vertical tendons actually caused failure to occur in another portion of the wall and prevented the HG 1 failure region from failing again. The vertically posttensioned wall was less stiff than the unreinforced masonry wall; this is attributed to slippage along the previously failed bed joints (figs. 3-4). The maximum moment capacity of the posttensioned wall was nearly the same as the moment capacity of the unreinforced wall, but the limit of deformation before failure doubled. It can be concluded that if horizontal tendons or steel straps had been present at the third course from the top of the wall to carry tensile stresses, the bending resistance and capacity of the masonry wall would have been significantly increased.

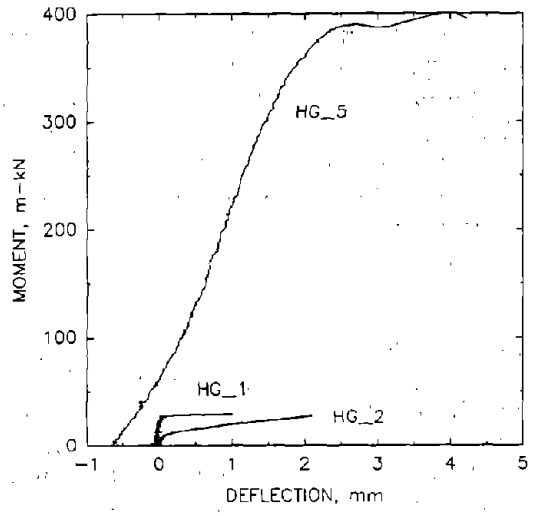


Figure 4. Moment-deflection curves for an unreinforced and posttensioned masonry wall.

For the third test (HG 3), horizontal tendons were installed (fig. 1) and tensioned to the values provided in table 1. The masonry wall was essentially converted into a truss system. The horizontal posttensioned region acts as the top cord and carries the tensile forces. The concrete footing and lower masonry courses are the lower cord and act as the compression member. The vertical members of the truss are the vertical tendons and carry tensile forces. The diagonal members are the masonry units and carry the compressional forces. The purpose of the horizontal tendons was to poststress the masonry wall and maintain it in compression across its entire cross section during the loading cycle. A hairline fracture was discovered after the horizontal tendons were tensioned at the anchorage location of VT 2. However, the hairline fracture did not affect

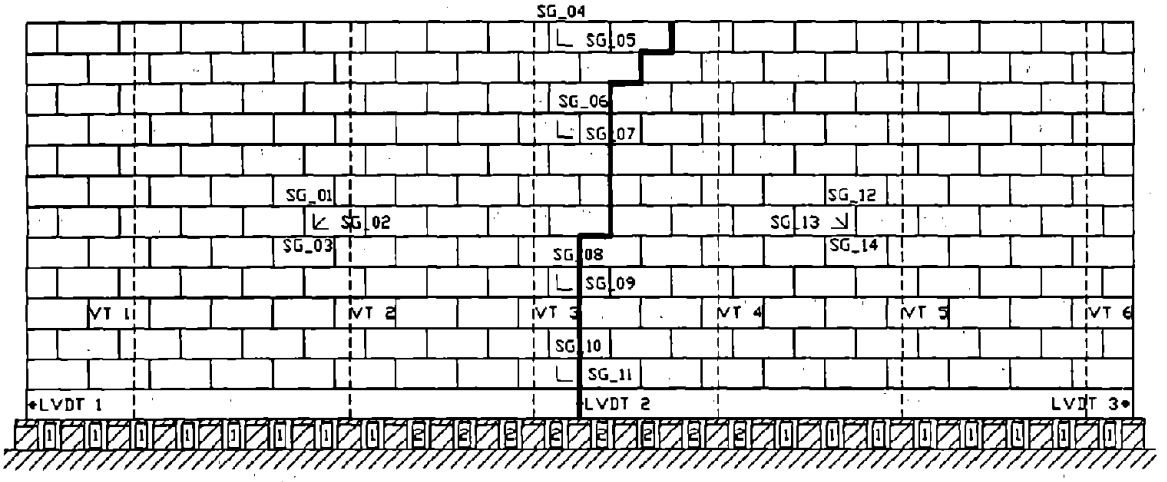


Figure 5. Failure of masonry wall reinforced with vertical tendons (HG 2).

the tensile force in VT 2. The test conduct of HG 3 was similar to that of HG 1. HG 3 was concluded when the masonry wall was able to cantilever (actuator set 1 pressure 0.0 MPa) and carry its own weight in bending. No failure in the wall was observed, including the previous failure regions HG 1 and HG 2.

Prior to conducting HG 4 the tension in each horizontal tendon was checked, and no time-dependent losses due to creep were observed. A static frame was installed at each end of the masonry wall. The reaction force locations are denoted by the locations of the force transducers (fig. 2). The hogging test proceeded as follows: The masonry wall was raised 2.5 cm, and then actuator set 2 was locked in. The pressure was slowly lowered in actuator set 1 to zero, and the wall was cantilevered. Actuator set 2 was then activated, and the pressure was slowly incremented. The test was terminated when the reaction force in each static reaction frame reached 116 kN. No failures or cracks in the masonry wall were observed. The reaction force of 116 kN represented the approximate load required for the bending moment tensile stresses to balance the poststress of the horizontal tendons.

HG 5 was conducted to determine the maximum load-carrying capacity of the posttensioned wall and to observe the failure mechanism. The test conduct of HG 5 mirrored that of HG 4, except that the test was not terminated until failure of the wall was observed. The wall failed in shear when the reaction load in the frames reached 120 kN. Despite the fact that the masonry wall failed, the wall still had a reserve of load-carrying capacity since the reaction frame forces dropped to only 49 kN. The maximum moment capacity of the vertically and horizontally posttensioned wall vastly exceeded the moment capacity of the unreinforced and vertically posttensioned masonry wall (fig. 4). The HG 5 curve is shifted because the horizontal posttensioning caused reverse flexure in the beam prior to the conduct of the test.

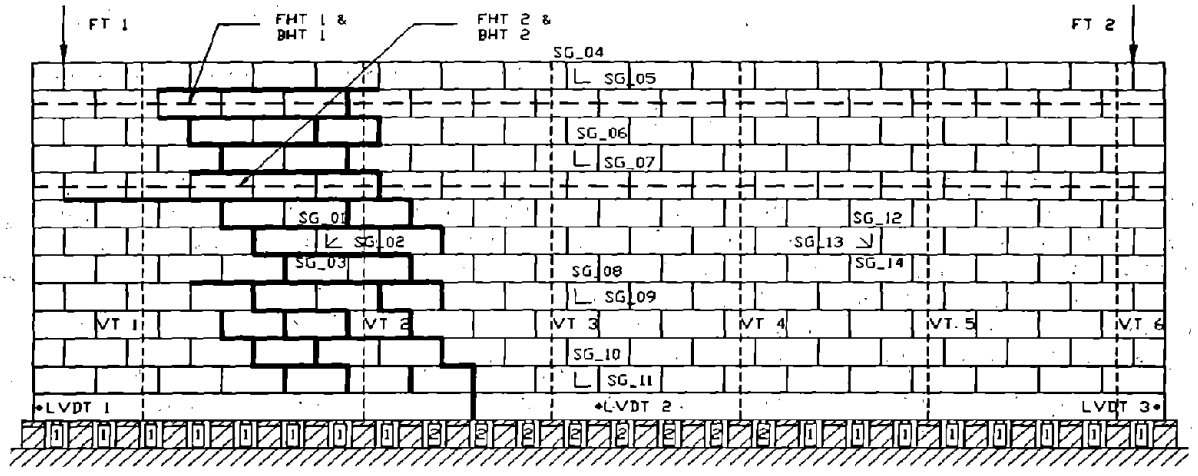


Figure 6. Failure of masonry wall reinforced with vertical and horizontal tendons (HG 5).

The sudden bond failure of vertical tendon VT 2 may have initiated the overall failure of the masonry wall since the applied vertical poststress in the area dropped to zero, and this is the area in which the wall failed in shear. It is believed that the wall would have achieved a greater load-carrying capacity had the anchorage of VT 2 not failed. The hairline fracture that was present in the footing (discussed earlier) must have propagated and subsequently compromised the integrity of the tendon anchorage.

The tension in all of the tendons was of interest, especially the vertical tendons, since their tension was determined from the setting actuator pressure. An access window was cut into the masonry, and the fundamental frequency of vibration of the vertical tendons was measured with a proximity gauge and

frequency counter. The frequency and tensile values are provided in table 2. The results show that there is a discrepancy between the initial (table 1) and final readings for the vertical tendons (table 2). The major reasons for this discrepancy are (1) the error introduced by using actuator pressure to determine the posttension values and (2) the loss in tension when the chuck jaws seat. A third reason may be the failure of the masonry wall and crushing of the bed joints; however, this is doubtful since virtually no changes in tension occurred in the horizontal tendons. Another potential explanation for the discrepancy is slippage of the anchorage system. However, the wire rope pull tests showed that this explanation is not probable since, in all cases, the wire rope failed before the anchorage system failed, and anchorage slippage would have a dramatic effect on the slope of the wire rope load displacement curves (fig. 1).

Table 1. Tendon tension of HG 5.

| Tendon | Frequency, Hz | Tension, kN |
|---|---|---|
| VT 1[1] | ND | 53.4 |
| VT 2[1] | ND | 53.4 |
| VT 3[1] | ND | 53.4 |
| VT 4[1] | ND | 53.4 |
| VT 5[1] | ND | 53.4 |
| VT 6[1] | ND | 53.4 |
| FHT 1[2] | 29.9 | 53.4 |
| BHT 1[2] | 29.9 | 53.4 |
| FHT 2[2] | 26.1 | 42.7 |
| BHT 2[2] | 26.1 | 42.7 |

Table 2. Tendon tension--completion.

| Tendon | Frequency Hz | Tension kN |
|---|---|---|
| VT 1 | 58.1 | 41.8 |
| VT 2 | 0.0 | 0.0 |
| VT 3 | 63.1 | 49.4 |
| VT 4 | 62.4 | 48.0 |
| VT 5 | 60.3 | 44.9 |
| VT 6 | 52.6 | 34.3 |
| FHT 1 | 28.6 | 54.7 |
| BHT 1 | 29.2 | 54.7 |
| FHT 2 | 25.3 | 42.7 |
| BHT 2 | 25.2 | 42.7 |

## Conclusions

This ongoing research project demonstrates the feasibility and positive benefits of posttensioning masonry walls. The combined use of vertical and horizontal tendons significantly increases the load-carrying capacity and bending strength of a masonry wall. The vertical tendons should also significantly increase the out-of-plane bending strength; this is currently being investigated in the laboratory. A method was developed to install vertical tendons in the interior of a masonry wall and to anchor the tendons into the footing. The installation of the tendons demonstrates the feasibility of posttensioning the vertical tendons within the foundation-floor interface and the availability of the equipment. Future work will concentrate on numerical modeling and evaluating postreinforcement designs for masonry walls when subjected to sagging (in-plane bending) and racking (out-of-plane bending).

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# MINE SUBSIDENCE EVENT AT WASHINGTON WEST APARTMENTS[1]

David B. Wilson[2], Mitchell W. Weber,[2] John Purdy,[2] and Patricia Acker[3]

**Abstract:** A major mine subsidence event occurred in Scranton, PA in early 1993. The initial damage included breakage of gas and water lines, cracking of pavements and sidewalks, and architectural damage to the seven-story apartment building that houses about 150 elderly persons. Visible damage included a 3/4-in dilation of the expansion joint separating the building, approximately 200 interior and exterior cracks, and distress to utility lines. The Office of Surface Mining Reclamation and Enforcement (OSM) funded an integrated geotechnical and structural engineering investigation to determine the cause of the subsidence, the limits of affected areas, and the nature of damage to the building. Work included interior surveys, exterior surveys, installation of crack gages, eight subsurface borings, review of building design drawings, review of geologic and mining data, and structural analysis of the rigid steel frame building. The surveys showed the building had undergone movements consisting of a lateral translation, a longitudinal differential settlement, and a transverse differential settlement. Preliminary structural analyses showed that the differential settlements had introduced significant additional stresses in some of the building columns.

This paper provides a case history of the cause and effects of the subsidence event. The techniques used to collect and analyze the data are presented along with the findings of the geotechnical and structural engineering investigations. The paper also describes emergency actions that were implemented, the remedial alternatives that were considered, and the method selected as the recommended alternative.

## Background and History

### Description of Structure

The Washington West Apartment building is located on Washington Avenue in Scranton, PA. The building (fig. 1) is a seven-story apartment structure approximately 50 ft wide by 280 ft long. The structure was constructed in 1969 for the present owner, the Scranton Housing Authority, and provides housing for about 150 elderly citizens.

### Mining History

Active mining of eight anthracite coalbeds occurred between 1860 and 1937. Except for the top mine, named Big Vein, all of the coal veins were mined by room-and-pillar methods. Approximately 33% to 45% of the coal was left in place to serve as support pillars. All of the mines were either partly or fully flushed in the area of the apartment building prior to 1970.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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The Big Vein Mine is 70 to 90 ft below the ground surface with an average thickness of about 15 ft within the project area. Prior studies estimated that only 20% to 30% of the coal remains in this mine, and observed roof collapses occurred in the mine within the area of the northern end of the apartment building. At the time of the last study, 25 yr ago, roof falls were reported to extend 20 to 30 ft above the floor of the mine.

## Foundation and Structural Design

The apartment building foundation consists of a modified mat that was designed to span a 20-ft-wide zone of nonsupport in anticipation of a "pothole" type of subsidence failure below any individual interior column, and a 10-ft-wide span along the mat periphery.

The Washington West Apartment building is a seven-story rigid steel frame structure with a masonry facade. Structurally, the building is divided into two separate structures by an expansion joint. The expansion joint extends through the foundation. The only connection between the two sections of the structure is through the use of tie plates attached to the column webs on each side of the expansion joint.

All beam-to-column connections are designed to develop the full-moment capacity of the beam. Though this method of design avoids the use of cross bracing and lessens the weight of steel required, a rigid frame structure is susceptible to overstressing due to differential support settlement.

## Movement History and Characterization

The history of movements preceding the January 18, 1993, subsidence event is unknown except in general terms as reported by the Housing Authority's building maintenance personnel. Minor cracking was reportedly repaired in the concrete block walls on the ground floor and stairways prior to that date. On January 18, 1993, stress cracking was noticed by Housing Authority employees, and "creaking noises" were heard by residents. The Wilkes-Barre office of OSM received a subsidence complaint and responded by dispatching technical representatives to the site for an initial evaluation.

A walk-through visual survey of the building and grounds revealed several indications of lateral and vertical movements. Inspection personnel recorded 3/4-in lateral movement at the expansion joint. About 200 cracks were observed in the walls and floors of the public areas of the building, including hallways, stairwells, storage areas, the boiler room, the day room, and office spaces. Dilation of the cracks ranged from tight to several millimeters. Some of the previously repaired cracks had reopened. Vertical or torsional movement of the building was not determinable during the site visit. Exterior evidence of ground movements included observations of tension cracks in the pavement of Washington Avenue, lateral displacement of street curbs and sidewalks, and broken gas and water lines in the immediate vicinity of the apartment building. Fig. 2 shows a tension crack in the pavement of Washington Avenue along the front of the apartment building.

## Emergency Actions Implemented Prior to Commencing Detailed Investigations.

OSM implemented a number of actions in response to the subsidence event. The agency quickly responded to the initial complaint by the Scranton Housing Authority and, recognizing the potential gravity of the situation, simultaneously implemented an emergency abatement action and began the procedures to contract outside engineering services to assist with the project.

Concurrent with the Gannett Fleming investigation, OSM undertook an aggressive mine-void-filling program to immediately slow the progression of the subsidence. The OSM program targeted the uppermost coal seam, the Big Vein, with the grouting operation. Approximately 25 injection borings were drilled in the

immediate vicinity of the Washington West Apartment building. About 3,000 cu yd of concrete and grout were injected into mine voids.

## Investigations and Findings

An instrumentation program to monitor movement of the apartment building and surrounding ground surface was established by Gannett Fleming, Inc. The instrumentation program inside the apartment building consisted of establishing and monitoring settlement points on each floor and installing and monitoring crack gages at points of visible distress. Exterior instrumentation consisted of establishing and monitoring ground settlement points and surface lateral movement points. Instrumentation monitoring was performed on a weekly basis during April. An additional survey was completed in October 1993. Details of this instrumentation program and the findings are presented below.

### Interior Elevation Surveys

The interior elevation surveys were established to provide data to evaluate measurable ongoing movement and to provide information for the structural analysis of the steel superstructure. Settlement points were established along the center line of the main hallway of the building on each floor. All surveys were performed using a digital electronic level that accurately measures elevations in a metric scale to four decimal places, i.e., 0.0001 m.

Typical movements along the hallway centerline are presented on fig. 3. This figure shows that most of the settlement typically occurs within the northernmost 60 ft of the building. The apparent total settlement is 3.36 in along the north-south axis of the building and 1.0 in along the transverse direction, with the northeastern corner of the building having settled more than the northwestern corner. A one-month downward movement of 0.1 in was recorded during April 1993. Additional downward movements of 0.36 in vertical and 0.3 in transverse occurred between April and October 1993.

### Crack Gage Monitoring

A total of 64 crack gages were installed across cracks both inside and outside the building. The primary purpose of the crack gages was to provide a simple on-site method for housing authority personnel to determine if movements were continuing.

### Exterior Elevation Surveys

Ninety vertical survey points were established outside the building. Fig. 4 is a plot of the measured settlement at each survey point indicates a trough pattern of subsidence in the area to the northeast of the apartment building. The maximum settlement of about 1.4 in occurred in the April to October 1993 monitoring period. This settlement pattern is in agreement with the observed tension cracks that have developed in this area and the deformed shape of the building, according to the interior survey and crack gage monitoring analysis.

### Structural Analyses

A baseline structural analysis of the building was performed using the as-built condition of the northern half of the structure without any settlement loads. This analysis was done to provide the baseline member stresses for comparison to an additional analysis reflecting the condition of the deformed structure at the time of the inspection.

The building was analyzed using STAAD III computer software. The building was modeled as a three-dimensional rigid frame structure consisting of 25 supports, 235 joints, and 483 members. Dead, live, and wind loads were computed in accordance with the 1990 BOCA Code.

A second analysis incorporating the April 1993 surveyed column elevations was performed to determine the magnitude of stresses that had been induced in the structural framework as a result of the movements. It was found that stresses had increased significantly in some of the columns, and that several of the columns may be overstressed beyond normally permitted design values by more than 20%, which is considered unacceptable as a permanent condition. Additional movements that occurred between April and October 1993 have further increased the column stresses.

### Geologic Exploration

The geological exploration program consisted of eight HQ-size (nominal 4-in diameter) borings drilled around the periphery of the apartment building.

The subsurface strata can be characterized as four layers consisting of the soil and the three rock intervals. The soil layer exhibits adequate densities and does not appear to be contributing to the settlement of the apartment building. The character of the lowest rock interval, between the Top and Bottom Split of the New County, appears to be relatively stable and is not contributing to the surface subsidence. The rock interval between the Big Vein and the Top Split of the New County has significant subsidence potential in the areas along the northeast portion of the building. The very poor quality of the cap rock and the conditions in the Big Vein result in this interval having the highest potential for subsidence. Subsidence related to the Big Vein in the area beneath and north of the Washington West Apartment building is the principal cause of the distress affecting the structure, with a possible secondary contribution from subsidence related to the Top Split New County Vein.

## Description of Remedial Alternatives Considered

### No Action

This alternative would involve taking no action except to continue to monitor the movement of the building. To be feasible, the monitoring program would be required to provide a warning that would be sufficiently early for remedial action be taken. This is not considered to be an acceptable alternative because of the magnitude of the damage that has occurred, and because the geologic conditions indicate significant potential for additional subsidence. A warning provided by continuing the monitoring program probably would not provide sufficient time to accomplish repairs without encountering significant additional structural damage to the building.

### Structural Strengthening of Building Frame

Column strengthening in most buildings is a major undertaking, and this building is no exception. The columns must be exposed to add strengthening steel to either reduce the unbraced length and/or increase the section properties. The preliminary structural analysis indicates that column strengthening might be required to remedy overstressing that has occurred. Column strengthening is not considered to be appropriate, however, as a general remedy to the subsidence problem because it does not address the fundamental cause of the movements.

Column Leveling

In certain buildings the effects of subsidence may be counteracted by jacking the columns back to their original position. The column can be braced during jacking and then grouted under the base plate to hold it in place. This operation could be repeated if additional subsidence occurred. This type of column leveling is generally limited to structures with simple base-plate-to-foundation connections. The columns for this building have moment connections to the foundation. A jacking operation for this type of column becomes inordinately difficult, if not impossible, since high shear loads in two directions as well as rotation at the base of the column must be resisted while allowing vertical movement.

Foundation Grade Restoration

This alternative involves leveling the foundation and columns by lifting the foundation. The restoration would be performed by lifting the foundation, using the stage down compaction grouting technique. The existing foundation of this apartment building is a mat foundation, which is typically considered a rigid foundation. In addition, it has been modified and made significantly more rigid to span up to 20 ft of nonsupport due to pothole subsidence. The load-distributing characteristics of this mat would be adverse for compaction grouting because the lifting forces would be spread out. In addition, there is a significant inconvenience to the residents and staff because all of this work is performed within the building.

Mine Flushing

Mine flushing is a technique used to fill the mine void with fine or coarse aggregate materials. This procedure involves drilling holes from the ground surface into the mine from which the flushing materials and water can be pumped into the mine. Advantages of this method are a low requirement for quality control and utilization of nonspecialized equipment and readily available materials. However, because the material is placed in a very loose condition, there is consolidation of the flushed material after placement. For this project, the magnitude of expected consolidation would allow some additional mine related subsidence to occur. This procedure is not considered to be appropriate because small additional settlements could seriously affect the structure because of the distress that the building has already undergone.

Consolidation Grouting of Mines and Cap Rock

The principal objective of this alternative is to address the fundamental cause of the problem, which is the subsidence of the bearing stratum because of mine roof failure. The consolidation grouting procedure attempts to fill the mine voids and cap rock fractures as completely as possible by low pressure grouting with concrete and/or cement-fly ash grout.

A consolidation grouting program would entail (1) developing a grouting pattern that would confine an area that includes the affected portion of the building and (2) performing secondary grouting to fill within the confined perimeter. All of the grouting would be performed from the ground surface and outside the building. Angle holes would be used for grouting under the structure. Because the Top Split of the New County is already suspect as a possible contributing factor to subsidence, grouting would be recommended for that vein as well.

Column Grouting Within Mine

This alternative involves constructing a grout or concrete column within the mine to support the roof. Basically these columns would act in a role identical to that of the coal pillars. Injection holes are drilled in a predetermined pattern, and the mine void is filled with concrete at each location. If a coal pillar is

encountered in the initial drilling, it is backfilled with grout. The grout columns would typically be installed using large air rotary rigs from the ground surface.

There are two disadvantages to this alternative: the final product cannot be readily inspected to evaluate the completeness of the treatment, and it is questionable whether the cap rock, weakened by the past subsidence, could span from support to support without causing additional subsidence.

Foundation Underpinning

This alternative, using pin piles as the most feasible method of underpinning, provides support to the foundation, but does not address the fundamental cause of the movement. A pin-pile foundation would consist of two or three piles at each structural column to transfer the building load to the rock strata below the Big Vein Mine. The pin piles would develop their capacity in the rock between the Big Vein and Top New County Mine.

The disadvantages to this alternative follow: Pin piles do not address the basic cause of the movement; therefore, the piles would be subjected to the forces associated with lateral movement of the ground caused by further subsidence. Pin piles are very weak in lateral support and probably would fail by shearing if subjected to lateral loads. The piles would all be installed from within the apartment building, disrupting the daily affairs of the building residents and staff.

## Recommended Remedial Alternative

No one alternative completely addresses all the issues that are considered necessary to remedy the present condition and to mitigate future distress to the structure. Instead, a combination of some of these alternatives taken in a phased approach provides the most reasonable solution. The recommended course of action is based on engineering judgment considering the following factors:

- Extent of current damage to the building.
- Value of the building.
- Assessment of the geological conditions.
- Apparent nature of the subsidence event.
- Construction details of the building.
- Technical feasibility and potential effectiveness of the remedial alternatives.
- Impact to the residents of the building during the remedial construction period.

The recommended remedial repairs consist of the following elements: consolidation grouting of the Top Split New County Vein, the Big Vein, and the cap rock; and implementation of a monitoring program to verify that stabilization of the structure has been achieved. A detailed structural reassessment after stabilization of the structure is complete may be performed to determine the need for structural modifications.

The objective of the consolidation grouting is to mitigate the source of the problem. By filling the mines as completely as possible with a material that is not subject to significant consolidation after placement and grouting of voids and fractures in the overburden rock, future subsidence activity is expected to be minimal. The grouting program consists of a series of vertical holes, used to create a perimeter barrier around the work area, and a series of variable angle holes oriented in a fan pattern to grout the rock strata beneath the structure. The angle holes are staggered to achieve the maximum grout penetration with the minimum number of injection holes. The selection of the type of material and mix ratios will be determined in the field, based on the conditions encountered in each injection hole.

## Discussion

Integrated engineering methods comprised of geotechnical investigations, monitoring surveys, and structural analyses were used at the Washington West Apartments to determine the cause, nature, and extent of the subsidence; the extent of damage to the building; the applicability of abatement alternatives; and the extent of abatement required. None of the engineering methods, if used alone, would provide reliable answers to all of the issues listed above. When used together as an integrated tool, each method provides part of the data necessary for a comprehensive evaluation. Data from each method complement and reinforce data obtained from the other methods. An integrated approach using these tools provides a clear image of the total problem and results in selection of the most appropriate abatement method as well as defining the minimum required limits of abatement.

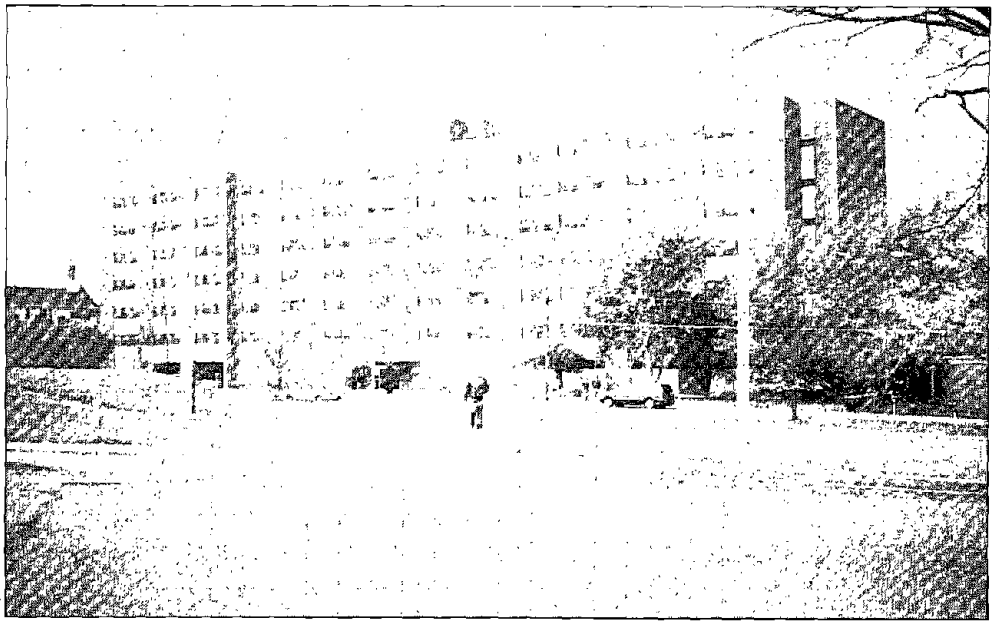

Figure 1. Washington West Apartment building in Scranton, PA.

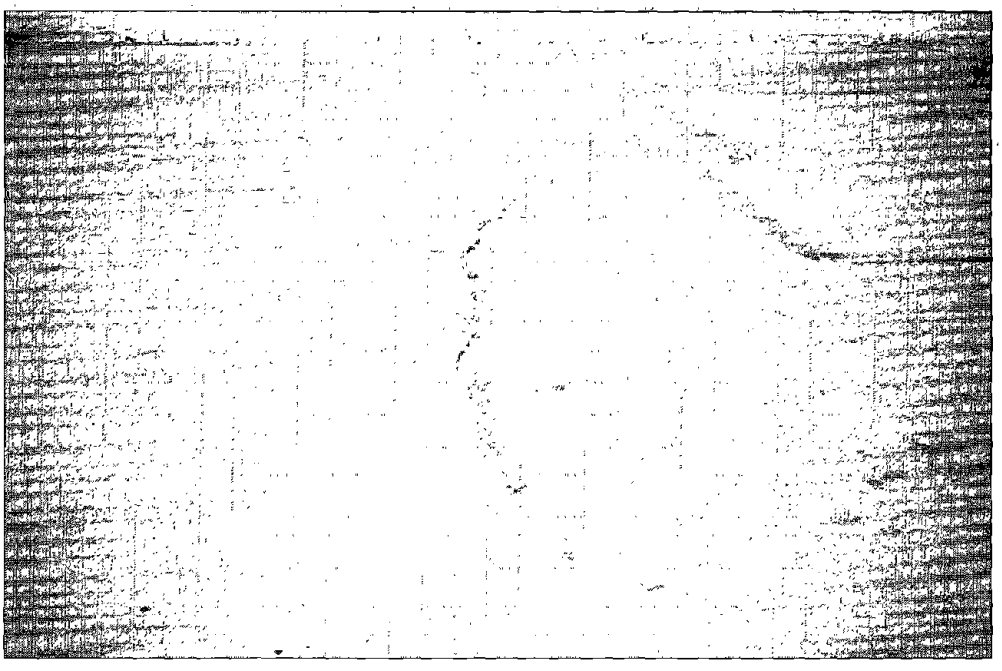

Figure 2. Typical tension crack in the pavement of the streets surrounding the Washington West Apartment building.

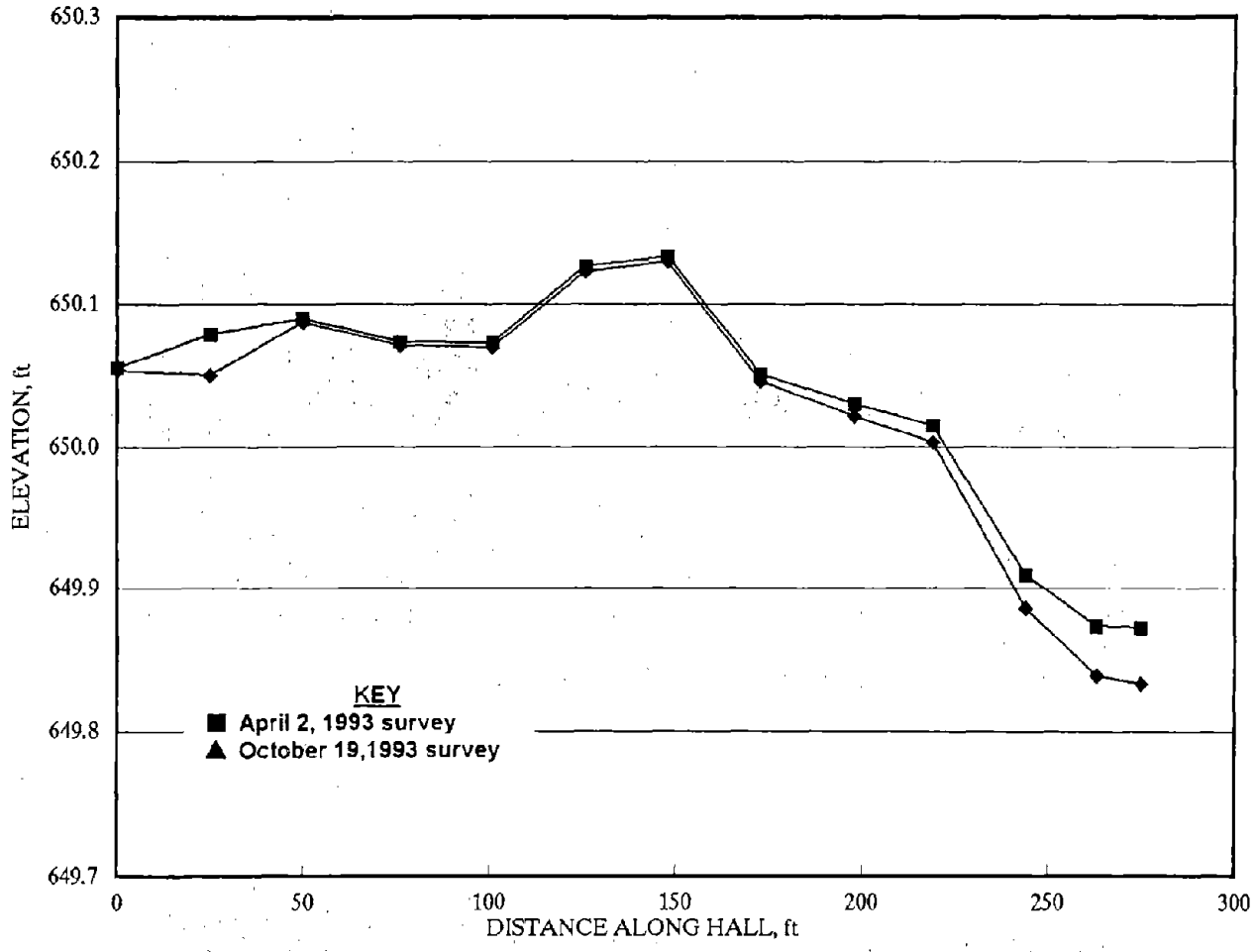


Figure 3. Plot of centerline profile showing typical vertical settlement in the northern portion of the apartment building.

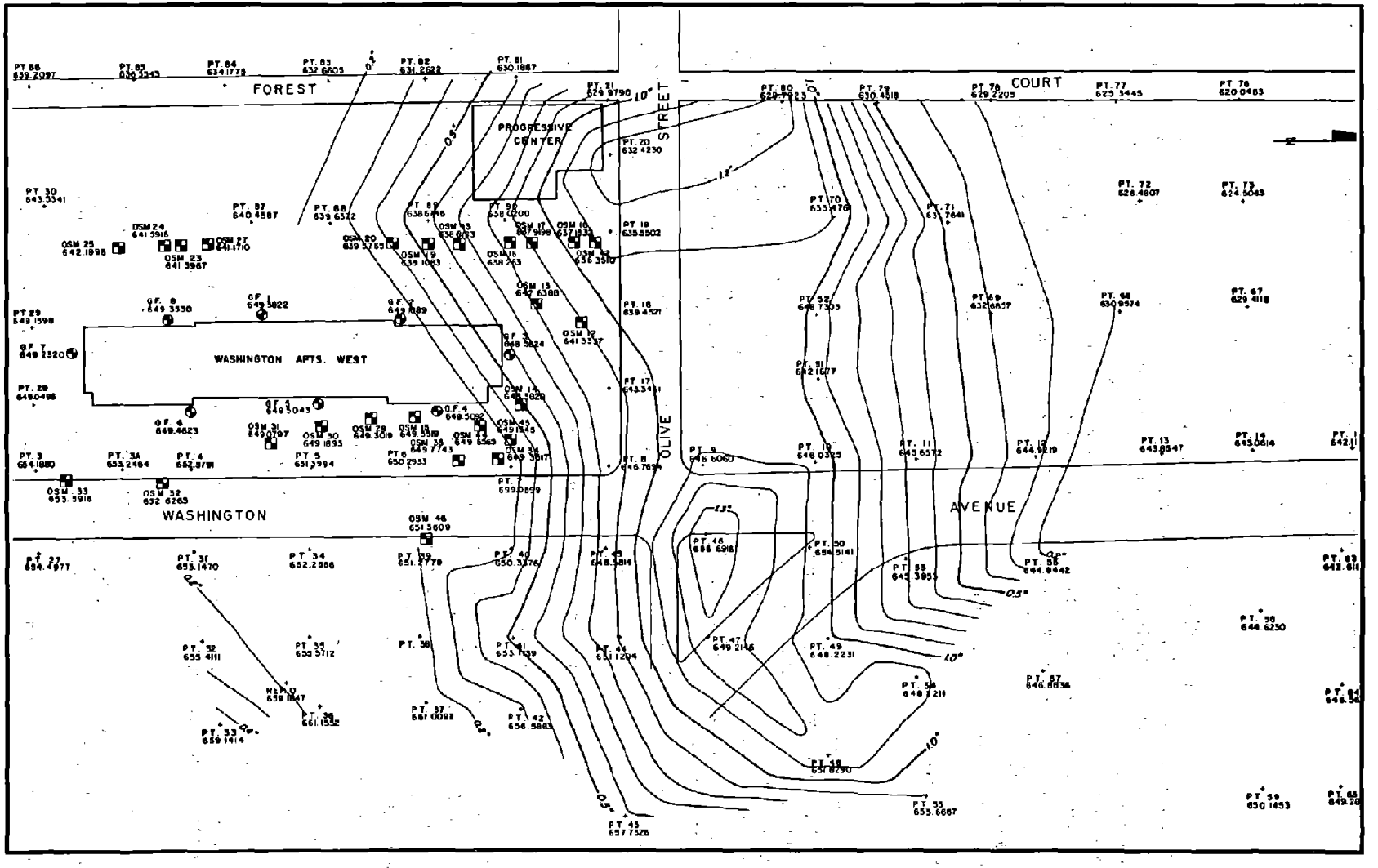


Figure 4. Settlement contour plan of the areas around the Washington West Apartments.

# PREVENTION OF DAMAGE TO HIGHWAYS AND RAILROADS IN MINING AREAS[1]

Kazimierz KŁOSEK[2]

## Abstract

Roads and railways, a distinctive part of Civil Engineering, when situated in subsidence areas, are subjected to the multiparameter reactions of soil due to subgrade deformation. The essence of technical degradation of such objects due to mining-induced subsidence entail changes in their functional and stability-strength properties. These changes take place within the structural system itself and, also in its subgrade. The linear character of such objects as a highway or a railroad creates a specific type of interaction between the subgrade and the elongated structure. The type of mining-induced contact reactions and the magnitude of service loads determine the operating capability of structures under these specific conditions. The need to maintain the proper geometrical parameters of the system (grade line, slope, superelevation, horizontal and/or vertical curvatures) arises from the necessity to maintain safe and uninterrupted traffic.
The paper is based primarily on the author's broad and lengthy research experience.

**Additional Key Words:** mining damage, transportation structures, subsidence basin.

## Introduction

Underground mining of useful minerals from protection pillars erected for transportation structures constitutes a considerable economic problem for Polish mining engineering.
40% of the total amount of minerals mined in Poland, particularly hard coal, brown coal, metal ores, and other minerals, is extracted in highly industrialized and developed areas. It is estimated that since World War II about 10% (1.5 $*10^9$ tons) of the total natural resources has been exploited, and the remaining 90% will be subject to mining in the future. It should be emphasized that the rate of mining has dropped during the last several years due to major economic changes that have occurred in this part of Europe. This reduction is especially significant for hard coal; the amount extracted in the 1980's amounted to 240 million tons per year, whereas this year it is only 150 million tons. The dense system of the highway and railroad transportation infrastructure in southern Poland converges with areas in which coalbeds occur in Silesia, which are deposited 100÷1,000 m deep, with global thickness ranging from 5 to 70 m. This creates considerable problems for mining works and also for the effective functioning of transportation systems in the region inhabited by 5÷7 million people.

## General Characteristics of Deformations in Mining Site Areas

The classification of mining sites with regard to site utility for development and construction depends on the following: character of anticipated deformations (continuous, discontinuous), type and values of anticipated deformations, type of earthen structure subsoil ground and type of structure. The next part of this paper is concerned with the analysis of deformations occurring with the emergence of continuous, regular subsidence basin with both static and dynamic character.
The basin is characterized by five basic parameters, illustrated in figure .1B:

- subsidence: $w$ [m]
- inclination: $T=dw/dx$ [mm/m] or [‰]
- curvature: $K=d^2w/dx^2$[1/km] (1)
- horizontal displacement: $u$ [m]
- horizontal strain of the subsoil: $\epsilon=du/dx$ [mm/m] or [‰] ( loosening - $\epsilon_t$, compacting - $\epsilon_c$ ); horizontal component of the displacement vector (u) related to the unit of lenght.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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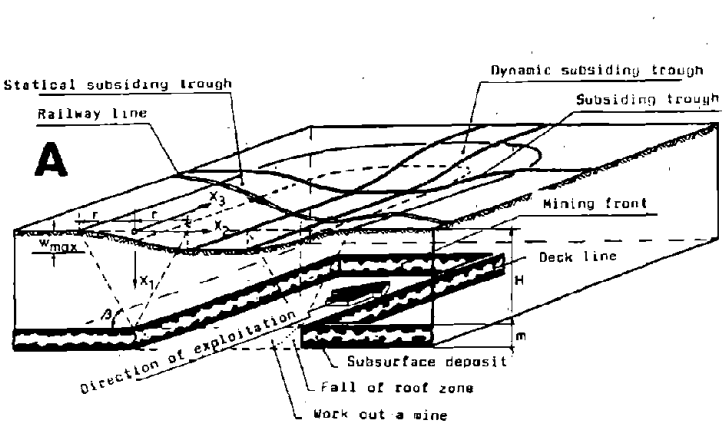


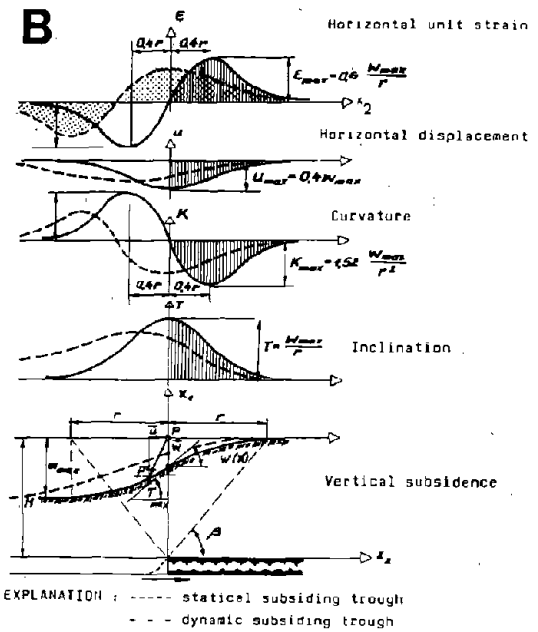


Figure 1. Basic characteristics of mining subsiding trough.
(A) a general view of mining subsiding trough,
(B) functions of surface deformation indices in the peripherial portion of trough on assuming the Gaussian effect distribution.

The analysis of the generalized components of the ground surface deformation of the rock mass zone makes it possible, for modest strains, to determine the unit strain tensor [1]:

$$L^I = \begin{vmatrix} \frac{\delta u}{\delta x_1} & \frac{\delta u}{\delta x_2} & \frac{\delta u}{\delta x_3} \\ \frac{\delta v}{\delta x_1} & \frac{\delta v}{\delta x_2} & \frac{\delta v}{\delta x_3} \\ \frac{\delta w}{\delta x_1} & \frac{\delta w}{\delta x_2} & \frac{\delta w}{\delta x_3} \end{vmatrix} = L_\epsilon^I + L_\gamma^I = \begin{vmatrix} \epsilon_{11} & \epsilon_{21} & \epsilon_{31} \\ \epsilon_{12} & \epsilon_{22} & \epsilon_{32} \\ \epsilon_{13} & \epsilon_{23} & \epsilon_{33} \end{vmatrix} + \begin{vmatrix} 0 & \gamma_{21} & \gamma_{31} \\ \gamma_{12} & 0 & \gamma_{32} \\ \gamma_{13} & \gamma_{23} & 0 \end{vmatrix}; \qquad (2)$$

where

u,v,w, - coordinates of the strain vector in the environment of a given subsoil point.

If adequate substitutions are carried out, in accordance with Cauchy's equation, it is possible to express tensor(2) by means of the components of linear strain $\epsilon_{ik}$ and angular strain $\gamma_{ik}$( or$T_{ik}$ ). By introducing substitutions in the expression of unit tensor rotation $L^I_\gamma$ and also, by transforming unit strain tensor to a diagonal determinant for three main strains occurring on its diagonal, the following equation may be obtained:

$$L^I = \begin{vmatrix} \epsilon_{11} & 0 & 0 \\ 0 & \epsilon_{22} & 0 \\ 0 & 0 & \epsilon_{33} \end{vmatrix} + \begin{vmatrix} 0 & T_{21} & T_{31} \\ T_{13}^* & 0 & \gamma_{32} \\ T_{13}^* & \gamma_{23} & 0 \end{vmatrix}; \qquad (3)$$

The state described by equation (3) relates to the case where the directions of the coordinates of the frame of reference converge with the directions of deformations of the mining site, which is illustrated in figure 1. Dissymmetrical component tensor $L_\gamma^I$ determines angles of nondilatational strain $\gamma_{ik}$ in plane (0, $x_2$, $x_3$,), inclination angles $T_{ik}^*$ characterizing angular subsoil strains, and site inclination $T_{ik}$ which is most important from the practical point of view. Tensor $L^{II}$, obtained from second normal derivatives of the unit strain tensor, may also involve the following physical sense:

$$(L^I)^* = L^{II} = \begin{vmatrix} \frac{\delta^2 u}{\delta x_1^2} & \frac{\delta^2 u}{\delta x_2^2} & \frac{\delta^2 u}{\delta x_3^2} \\ \frac{\delta^2 v}{\delta x_1^2} & \frac{\delta^2 v}{\delta x_2^2} & \frac{\delta^2 v}{\delta x_3^2} \\ \frac{\delta^2 w}{\delta x_1^2} & \frac{\delta^2 w}{\delta x_2^2} & \frac{\delta^2 w}{\delta x_3^2} \end{vmatrix} = \begin{vmatrix} \frac{\delta \epsilon_{11}}{\delta x_1} & K_{21} & K_{31} \\ K_{12}^* & \frac{\delta \epsilon_{22}}{\delta x_2} & K_{32} \\ K_{13}^* & K_{23} & \frac{\delta \epsilon_{33}}{\delta x_3} \end{vmatrix}; \quad (4)$$

Parameters on the main diagonal of matrix (4) describe the gradients of unit strains in main directions, other quantities characterize the mining site curvatures. Parameters $K_{ij}$ describe mainly the curvatures of subsidence basin.

## Classification of Transportation Structures in Mining Site Areas

Transportation structures in mining sites are of a typically linear character. The structure length parameter is much more relevant than structure width parameter, which is important to the static resistance analysis. Other features of these structures are presented in table 1.

To estimate the operating conditions of these structures, the problem of determining their real structural and functional resistance to mining site deformations should be approached. Resistance (usability) is assumed to be the capacity of a structure (or a facility) to transfer mining deformations of the subsoil (displacements and strains) maintaining, at the same time, the conditions of safety and continuous utility, relevant to the original design of this structure. Conclusions drawn from practical experience indicate that every component of a highway or a railroad system, such as subsoil, roadbed, earthen structures, pavement, contact line, drainage systems, engineering constructions (bridges, culverts, retaining walls, etc.), have different resistance to the effects of mining.

**Table 1. Classification of structures on mining sites [8]**

| Basis of classification | Structure features | | |
|---|---|---|---|
| shape of horizontal view | compact | linear | |
| structure capacity to strain with ground subsoil deformations | rigid | deformable | |
| functional and structural vulnerability to the effects of subsoil deformations | vulnerable | not vulnerable | |
| maintaining conditions of safety and utility | resistant | partly resistant | not resistant |
| occurrence | common | rare | |

Generally speaking, functional resistance results from structure vulnerability to deformations of road profile and route, changes in values of pavement longitudinal and diagonal inclination, increase of the number of profile curves, changes in horizontal and vertical arch curvatures.The effects of mining are the following: considerably lower speed of traveling, decrease of transportation line capacity, increase of delays and safety hazards concerning transportation systems, increase of noise emission, failure of the patency of drainage systems, technical deterioration of the transportation means and infrastructure, necessity of more investments to enable the functioning of transportation systems. If the allowable value of structural resistance is exceeded, the following consequences may arise: permanent deformation of the roadbed and subgrage (landslide, landslip) track surface and pavement damage ( buckling of tracks in the horizontal or vertical plane, uplift or cracks in the road pavement), damage or destruction of bridges, culverts, retaining walls, destruction of technical infrastructure, etc. Transportation structures are characterized by relatively small structural and functional resistance to the effects of mining. In particular, this applies to the basic structural element, that is, railway and road earthen structures in the form of embankments or cross-drifts.

## Transportation Structures in Mining Site Areas

The interaction of mining subgrade and structural elements of roads and railways is a complex process. The elements of a technical structure, which constitutes the pavement structure and subgrade, depend on basic ground surface deformation indices in various degrees (3-4). From a geotechnical point of view, horizontal strains of a ground surface exert a decisive influence on the state of stress in the subsoil, especially with regard to loosening strains $\epsilon_i$. They may evoke the following consequences [2,7,9]:

-loss of load capacity in the subgrade zone due to subsoil; softening; fig.2-1.
-loss of load capacity in embankment subsoil or cross-drift; foundation zone (2).
-diagonal deformation of the whole body of the embankment ("bell" cut section) (3), and additional subsidence of the embankment foundation (3*).
-landslide resulting from the diagonal loosening of the embankment subsoil (4) or from subsoil uplift pressure (5).

If inclination T is increased, there is a possibility of the diagonal shift of the whole earthen body of a structure after the tangential stress in the slide zone (C) occurs, irrespective of the permanent change of the slope inclination (B). Considerable rise of the slide height (to 20-30 m) may also be a source of stability hazard due to the need of maintaining the pavement grade line (A).Also, the change in the ground waters level caused by mining site subsidence results in the flooding of the foundation of embankments, which in turn weakens the load capacity of the subsoil and the stability of the earthen structure slopes (figure 2-A).

The factors mentioned above lead to the repositining of the stress and strain components in the subgrade. Horizontal loosening strains $\epsilon_i$ will generate the change of the state of stress in the mining subgrade $\Delta\sigma^l_{ik}$. Assuming that vertical strains $\sigma_{11}$ do not change, the repositioning of horizontal strains is described by the equation:

$$\sigma^l_{ik} = \sigma^*_{22} - \Delta\sigma^l_{ik} = \overline{\gamma}_0 * h^* \begin{vmatrix} 1 & 0 & 0 \\ 0 & (m-1)^{-1} & 0 \\ 0 & 0 & (m-1)^{-1} \end{vmatrix} - \begin{vmatrix} 0 & 0 & 0 \\ 0 & \frac{2G(m+\rho)}{m-1}\epsilon_{22} & 0 \\ 0 & 0 & \frac{2G(1+m\rho)}{m-1}\epsilon_{22} \end{vmatrix} \geq \sigma^{lim}_{22}; \quad (5)$$

where

$\sigma^*_{22}$ - total strain, including the pavement and rolling stock weight
$\Delta\sigma^l_{ik}$ - the change of the state of stress in the mining subgrade
$\sigma_{22}{}^{lim}$ - minimum value of horizontal strain $\sigma^{min}{}_{22}$ in the total active rankine state
$\overline{\gamma}_0$ - average bulk density of the subsoil, figured from the subgrade level

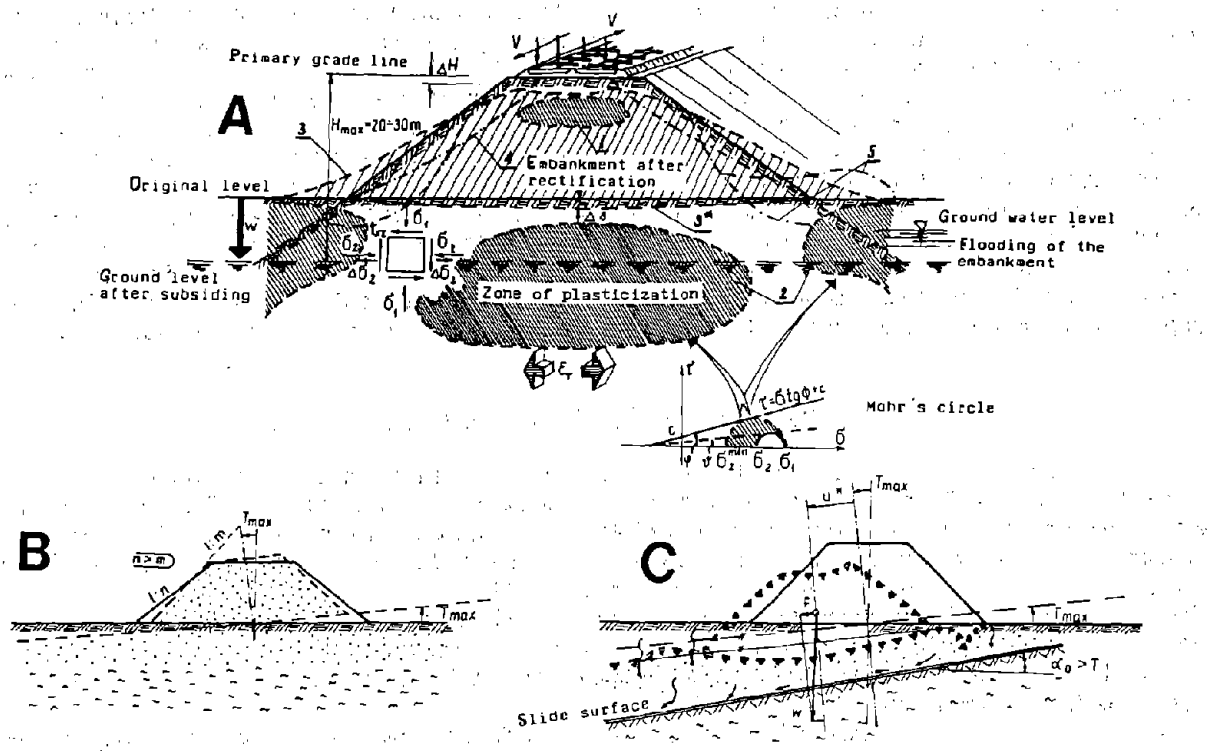


Figure 2. Load capacity and transvere stability of the embankment on mining areas:
(A) typical damages in the embankment and in earthen foundation,
(B) influence of inclination $T_{max}$ ,
(C) influence of sliding on the internal slide surface.

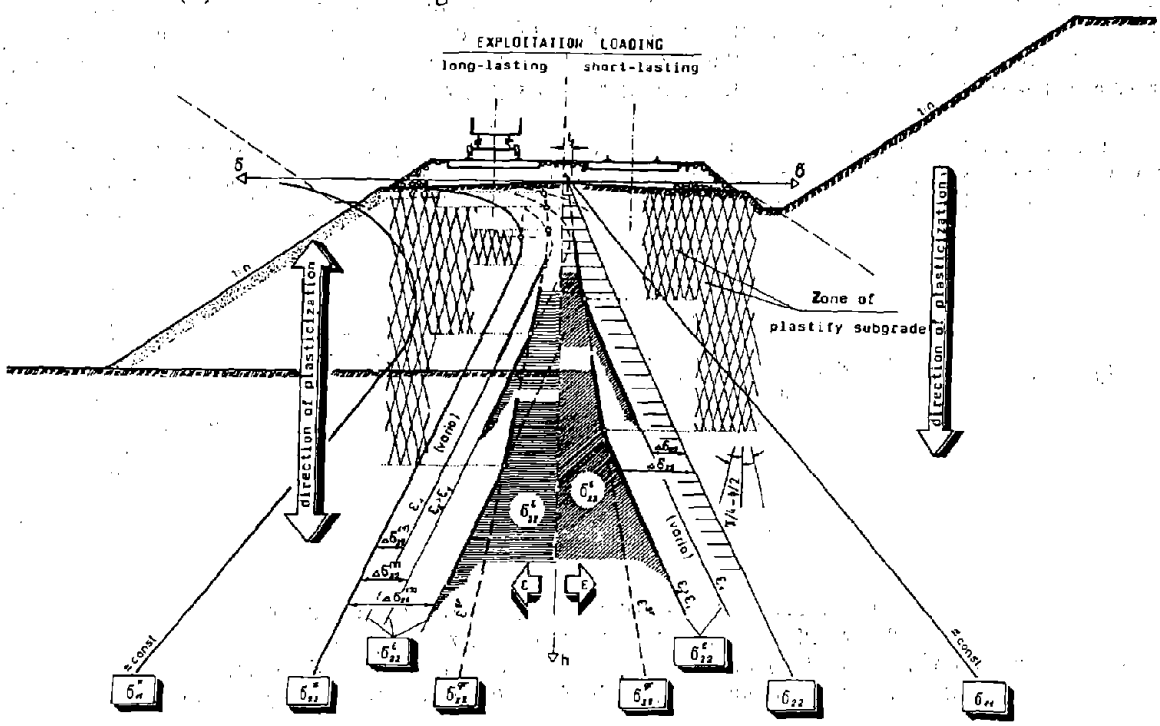


Figure 3. Development of plasticization zones in subgrade on mining areas depending on the nature of operating loads of railway subgrade.

$h^* = t_g / \gamma$ - equivalent bed thickness, including the pavement weight
$t_g$ - vertical stresses caused by the pavement weight
$m = 1/\nu$ - coefficient of diagonal subsoil deformation
$\nu$ - Poisson's coefficient
$G = E_\epsilon\ m/2(m+1)$ - subgrade diagonal elasticity module
$E_\epsilon$ - subsoil strain module in the mining deformation conditions
$\varrho = \epsilon_{33} / \epsilon_{22}$ - relation of horizontal strains in main directions.

The graphic interpretation of equation 5, based on the theory of boundary states, is presented in figure 3. Figure 3 illustrates the emergence and development of the subsoil softening zones in the subgrade, depending on the principal working load. In the case of temporary loads, the softening zone occurs at the ground surface zone, and it evolves down the subsoil with the rise of loosening strains. For long-term loads, this zone initially occurs in the subsoil (3-4 m deep) and next develops in two directions. The softening of the subsoil is associated with the temporary loss of load capacity, which practically entails the rise of additional pavement subsidence with regard to the subgrade and the whole body of a structure with regard to the subsoil, which is illustrated in figure 2A. Such kinds of subsidence occur irrespective of mining site subsidence "w". Another factor increasing pavement deformations is the permanent loss of load capacity, connected with the interference of horizontal loosening deformations into the physical and mechanical properties of the ground. The result of this process is the long-term loss of compressive strength of the subsoil ground , which is described by the equation [1][2]:

$$\tau_{max} = \sigma_{11}\ tg\Phi_w + \beta(t)\Sigma_w \geq \tau_{min} = \sigma_{11}\ tg\Phi + \beta(t)\Sigma_w\ ; \qquad (6)$$

where

$\sigma_{11}$ - ordinary component of effective stress
$\phi_w$ - angle of internal friction of the ground with humidity w
$c_s$ - structural part of ground general cohesion
$\Sigma_w$ - ground cohesion, caused by water and colloid systems
$\beta(t)$ - coefficient of mining deformation intensity; $0 \leqslant \beta(t) \leqslant 1$.

Results of experiments carried out for many years indicate that horizontal loosening displacements exert an influence on the quick reduction of the structural part of cohesion "$c_s$" (with $\epsilon_l > 6 \div 9$mm/m) and on insignificant decrease of the value of primary cohesion. The repositioning of all components of the state of stress in the subgrade is presented in figure 4 on the basis of the disintegrated media theory from the probability point of view. Figure 4A refers to the subgrade that lost its primary load capacity as a result of mining damage. Among many methods of reinforcing and protecting road earthen structures the techniques making use of geosynthetic materials should be considered as most effective [1,3,4].

## Mining Damage to Rail Transportation Systems

Rail transportation systems are of great importance to both passenger and goods transport in Polish economic conditions. This particularly applies to railways and streetcars in the District of Silesia, an area influenced by intensive mining damage caused by the operation of more than 60 mines. About 30% of all district rail lines (700-1,000 km of lines), with load capacity of 45 milion Mg/year, are situated in the zones of direct influence of underground mining. The average subsidence of many lines and stations ranges between 5-15 m, with intensity reaching the range 0.5- 1.5 m/year. The District of Silesia constitutes only 4% of Poland's total surface area, but about 50% of home goods transport and 40% of home passenger transport falls to Silesia. Failure of the elements of railway lines on mining sites increases three to four times more in comparison with lines in other parts of the country. The resulting operational inefficiency of transportation systems constitutes a serious technical and traffic problem.

Streetcar lines in Silesia are 370 km long, which is 15.5% of all Polish streetcar lines. The average speed of travel is not greater than 30 km/h due to intensive mining influence, including discontinuous deformations in the form of pits, crevasses, and cavings. So far, subsidence of the 15% of lines has reached 0-6.5 m. By the year 2010, the rate of subsidence will have reached 6-8 m for 30% of the lines. Also, about 40% of all engineering structures

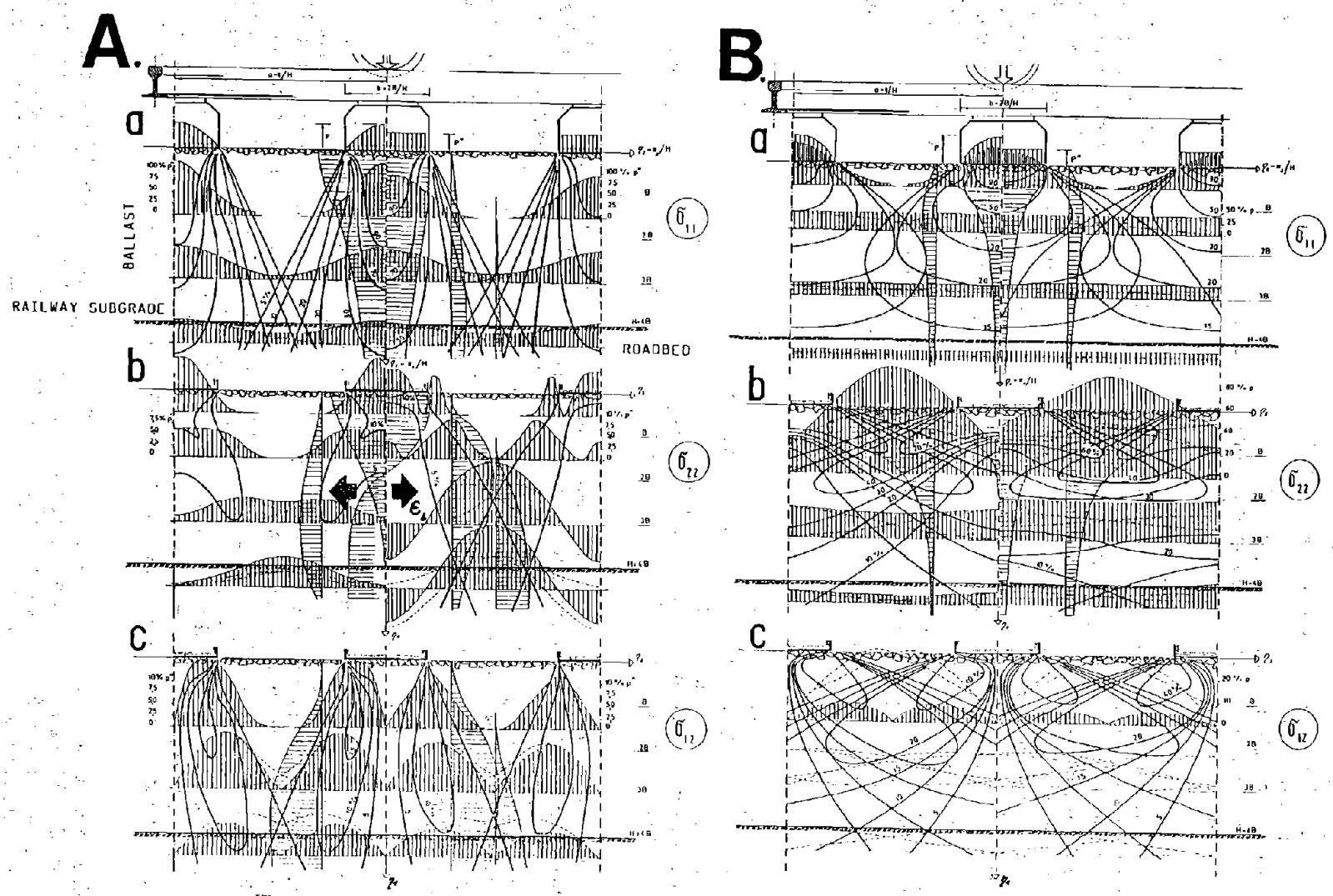


Figure 4. Components of stress state in homogeneous subgrade subjected to horizontal unit strain:
(a) vertical stresses and their isobars,
(b) horizontal stresses and their isobars,
(c) shearing stresses and their isobars,
(A) with influence of horizontal unit strain on mining subsiding area,
(B) without mining damages.

is exposed to intensive influence of mining damage.

The influence of ground surface deformations entails changes in the form of tracks in both route and profile. Ground subsidence causes changes in line profile, which in turn limits the maximum train weight allowable. Because the longitudinal profile inclination of a line is a sum of the primary track inclination"i" and of the profile inclination of mining subsidence basin "T", the allowable value of line inclination (for stations $i_{max}$=0.5-1.5‰, for lines of I/II/III category $i_{max}$=6/12/20‰ respectively) is exceeded in some line segments. These deformations are followed by changes of track tilt, warping, unacceptable increase of horizontal profile curves, considerable horizontal displacements of railway axes, variable curvatures of horizontal arches, and transition curves.

Prevention of mining damage consists of recommendations for mining to be carried out by means of long and continuous working front, perpendicular to track axes (figure 1).
Preventive treatment of structures is focused on minimizing the range of essential leveling works. In practice this means that tracks are not lifted to their primary grade line. The types of grade lines are the following (see figure5):
- primary (designed) grade line, before the time of mining subsidence occurrence ($L_1$),
- preventive grade line, taking into account the anticipated subsidence that secures continuous and safe traffic of railway vehicles ($L_2$),
- temporary (transitory) grade line, entailing the time of subsidence occurrence and also intervals in the successive stages of subsidence ($L_3$),
- permanent grade line, this is the ultimate grade line, taking into account the anticipated subsidence in relation to primary grade line ($L_4$).

Preventive grade line can be applied in designing new railways or modernizing railways of special importance. Permanent grade line can play a part in reducing the range of essential leveling works and investment necessary for their execution. The general principle of transportation systems protection is the proper coordination of mining and civil engineering preventive techniques applied to long railway line segments and stations, together with railways junctions. Experiments carried out by the author of this paper indicate that the elastic flexibility coefficient concerning the railroad tie subsoil is reduced, as a result of horizontal loosening strains, by the value:

$$\Delta C_{el} = C_0 (1-\alpha_l^{4/3}); \tag{7}$$

where

$C_0$=p/y - output value of the elastic flexibility of the railroad tie subsoil
(p - stress in railroad tie foundation, y - subsidence)

$\alpha_l$ - coefficient of variable compressive module of the loosened ($0 \leqslant \alpha_l \leqslant 1$) subsoil [1,5].

The influence of parameter C on the flexure stress of rail $\sigma$ and on the deflection value is expressed by the equations:

$$\sigma = \Phi_1 C^{-0.25} < \sigma_{permis}; \tag{8}$$

$$y = \Phi_3 C^{-0.75}; \tag{9}$$

$$\Phi_1 = P/4W(4EJ/b)^{0.25}; \ \Phi_2 = 0.5Pl(4EJ/b)^{-0.25}; \ \Phi_3 = \Phi_2(bl)^{-1}; \tag{10}$$

where

EJ - rail flexural rigidity
W - index of rail flexure strength
b - railroad tie width ( equivalent)
l - railroad tie spacing
P - railroad tie load force
$\sigma_{permis}$ - allowable value of rail steel flexure stress.

Site investigation results determined the value of coefficient $5 \leqslant C_\epsilon \leqslant 15$ MPa/m, due to cyclical railway stock operation (fatigue), the allowable value of flexure stresses may be exceeded: $\sigma_{real} \geq \sigma_{permis}$. The cracking of rails and turnout elements occurs more and more often. In Silesia, the number of such cracks is five to eight times greater than in other parts of Poland. Figure 6 illustrates results of calculating flexure stresses for conventional rail surface structure elements, for values of coefficient C measured before ($C_0$ = 130 MPa/m) and after ( $C_\epsilon \cong$3MPa/m) mining influence occurred. It should be pointed out here that the number of flexure stresses in rails increases with

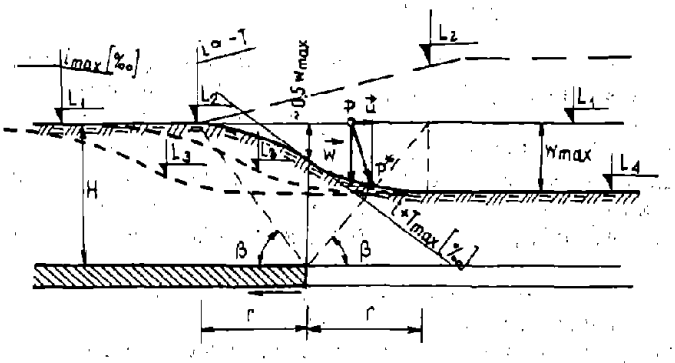


Figure 5. Types of grade lineas on mining subsidence areas.

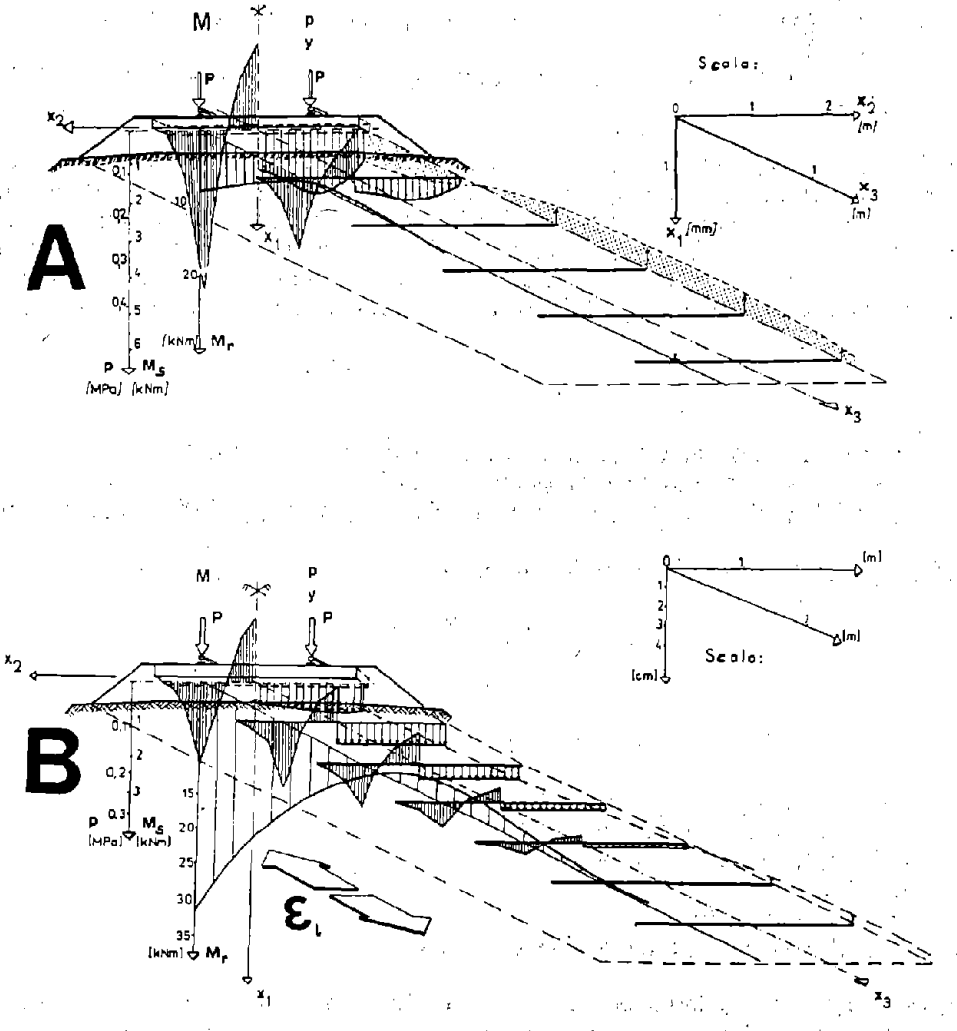


Figure 6. The effect of quasi-static loading of track structure by a single axle of wheel set on its elastic strains, bending moments in rails $M_r$ and sleepers $M_S$, and a distribution of normal contract reactions "p" in the basic of track layer framwork
(A) on nonmining grounds for: $C_0$ =130 MPa/m, track structure R-49, load P=98kN,
(B) on mining ground for $C_{\epsilon l}$=3 MPa/m.

the reduction of contact stresses in railway tie foundations. Also, rail surface deflection rises, together with the number of railway ties transmitting the external influence.

## Mining Damage to Roads

In Silesia the road transportation system is the densest and busiest in Poland (60 km/100 km$^2$). More than 50% of district roads are influenced by mining damage, and the average occurrence of subsidence ranges within 1÷10m. In some given segments, subsidence may reach the range of 20-30 m in the future. For some internationally important roads, deformations with the following parameter values are anticipated: $T > 15$mm/m, $\epsilon = 9$mm/m, $R = K^{-1} = 4$km. The actual protection of roads in comparison with railways is in a considerably worse state. There are no generally accepted procedures for the execution of mining underneath the roads and, in addition, there are no proven methods of increasing road resistance to the effects of mining.

The mechanism of the occurrence of mining damage in the subgrade may be assumed to apply to roads. Damage to road pavements indicate that structures vulnerable to strains of rigid pavement show better resistance than those that are less vulnerable. Changes in road geometry in route and profile are particularly disturbing, for example: short, local vertical irregularities of surface (pavement bulging), pavement gradation and cracking, loss of the patency of drainage systems, numerous damages to engineering structures such as bridges, overpasses, trestle bridges, culverts. Discontinuous deformations in the forms of crevasses and pits are particularly dangerous, together with the paraseismic influence of the rock mass. In such conditions, roads are becoming less functional and traffic less efficient. The resulting losses of time and energy decrease the economic parameters of roads situated in mining site areas.

## Summary

The synthetic analysis of the causes of mining damages occurring in structural elements of roads and railways presented in this paper is based on years of practical experience in this field and also on theoretical analyses of problems explaining the mechanism of the observed damages and the inefficiency of transportation systems. Mining deformations considerably reduce the structural and functional resistance of roads and railways and often create serious hazards to their safety and utility. The resulting damages and defects of pavement and subsoil should not be allowed and therefore require adequate preventives and repairs. The damage reduces the safety of travel, pavement live, and often changes the geometry of roads and railways to the degree that makes effective functioning of traffic virtually impossible. The only possible guarantee of the effective protection of transportation systems in areas influenced by underground mining is the coordination of mining and civil engineering protective techniques.

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# MINE SUBSIDENCE CONTROL PROJECTS ASSOCIATED WITH SOLID WASTE DISPOSAL FACILITIES[1]

Randy M. Wood[2]

**Abstract:** Pennsylvania environmental regulations require applicant's for solid waste disposal permits to provide information regarding the extent of deep mining under the proposed site, evaluations of the maximum subsidence potential, and designs of measures to mitigate potential subsidence impact on the facility. This paper presents three case histories of deep mine subsidence control projects at solid waste disposal facilities. Each case history presents site specific mine grouting project data summaries which include evaluations of the subsurface conditions from drilling, mine void volume calculations, grout mix designs, grouting procedures and techniques, as well as grout coverage and extent of mine void filling evaluations. The case studies described utilized basic gravity grouting techniques to fill the mine voids and fractured strata over the collapsed portions of the deep mines. Grout mixtures were designed to achieve compressive strengths suitable for preventing future mine subsidence while maintaining high flow characteristics to penetrate fractured strata. Verification drilling and coring was performed in the grouted areas to determine the extent of grout coverage and obtain samples of the in-place grout for compression testing. The case histories presented in this report demonstrate an efficient and cost effective technique for mine subsidence control projects.

**Additional Key Words:** subsidence, grouting, solid waste, deep mines.

## Introduction

As required by the Pennsylvania Department of Environmental Resources (DER) regulations for municipal solid waste (MSW) disposal facilities, an applicant for a solid waste disposal permit must provide information regarding the extent of deep mining under the proposed landfill site. In fulfillment of the regulations, the applicant is required to provide evaluations of the maximum potential for subsidence at the site and designs of measures to be utilized to mitigate the potential for adverse impact on the facility due to subsidence. This paper presents case histories of deep mine subsidence control projects associated with the development of three solid waste disposal facilities. Each case history presents site-specific mine grouting project summaries that include evaluations of the subsurface conditions from drilling, mine void volume calculations, grout mix designs, grouting procedures and techniques, grout coverage, and extent of mine-void filling evaluations.

The case studies presented utilized basic gravity grouting techniques to fill the mine voids and associated subsidence fractured strata over the collapsed portions of the deep mines. Based on the anticipated conditions of the deep mine voids as determined from preliminary drilling information, grout mixes were designed to maximize filling of the mine voids and fractures. The grout mixtures were designed to achieve compressive strengths suitable for preventing future mine subsidence while maintaining high flow characteristics to penetrate fractured strata. A variety of grout mix designs were developed using varying proportions of sand, fly ash, cement, and water. Control of the grout mix designs during grouting operations was maintained by compression testing of grout test cylinders. In addition, coring was performed in the grouted areas to determine the extent of grout coverage and obtain samples of the in-place grout for compression testing.

## Regional and Local Geology

All three of the case study sites are located in the Appalachian Plateau physiographic province of southwestern Pennsylvania. Regionally, the geologic structure of this area is characterized by subhorizontal bedrock strata and gently plunging northeast-southwest-trending folds. Geologic strata encountered at the sites consist primarily of interbedded sandstone, siltstone, and shale and range in age from Lower to Upper Pennsylvanian. These case studies deal with subsidence control projects associated with deep mining on the Upper Kittanning and Pittsburgh coal seams.

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## Subsidence Evaluations

As required by the MSW regulations, an evaluation of the maximum potential for subsidence at landfill sites underlain by deep mines must be performed. In addition, the design measures incorporated into the landfill design to mitigate the potential for adverse impacts on the integrity of the landfill facility must be presented. Subsidence predictions were performed for each of the sites presented in this paper during the landfill permitting process. The subsidence calculations were typically generated using the National Coal Board (NCB) method for subsidence predictions (Peng, 1986). This method of subsidence prediction bases the calculation on the minable thickness of the coal (void height) times a subsidence factor. The determination of an appropriate subsidence factor is critical to the prediction. In England the NCB has developed empirical subsidence factors based on observation of subsidence over active coal mining operations. Subsidence factors in the United States are generated based on research and field observations. A typical maximum subsidence factor for U.S. bituminous coalfields is 0.5 (GAI, 1974). Other assumptions regarding the type of subsidence (sinkhole versus trough) relative to the depth of overburden are also applied to the subsidence predictions (Jones & Kohli, 1985). The end result of the subsidence evaluation is that if the site is underlain by deep mines, some form of subsidence mitigation under the waste disposal area to protect liner and leachate control systems is typically required by the regulatory agencies. In Pennsylvania, typically mine subsidence mitigation is required where overburden thickness is less than 30.5 m (100 ft). The determination of the areas requiring subsidence control includes factors such as angle of draw, in addition to the overburden depth criteria.

## Subsidence Control Projects

In central and southwestern Pennsylvania two basic mine subsidence control techniques have been utilized: overexcavation-backfilling and grouting. Since the Pennsylvania MSW regulations require that no waste be placed within 7.6 m (25 ft) of a coal seam, overexcavation and backfilling is the most common mine subsidence control technique for shallow-overburden deep-mined areas. Often the overexcavation techniques will be extended to some economical overburden depth limit. Areas of deep mining beyond the excavatable limits which require subsidence control are then grouted.

Grouting projects for the mitigation of deep mine subsidence potential have been performed on a number of landfills in southwestern Pennsylvania. The overexcavation and grouting techniques are the predominate, if not the only, subsidence mitigation methods known to have been used. These mine grouting projects typically use fly ash and cement grout mixtures placed into the deep mine voids and subsidence fractured strata via boreholes. Grouting is typically performed using gravity placement techniques. To minimize environmental and logistical problems at the landfills, the grouts are typically mixed at local commercial concrete plants and delivered to the sites via standard concrete ready-mix trucks. Three mine grouting projects representative of this type of mine subsidence control technique are presented in the following sections.

# Case Studies

## Site 1

The mine subsidence control project at Site 1 was performed as part of a permitted expansion of the landfill to mitigate potential subsidence effects on the landfill liner and leachate collection systems. The liner system in this area was to be installed on an approximately 167.6 m (550 ft) long 3 horizontal to 1 vertical (3H:1V) slope, along the southern perimeter of the waste disposal area.

Four coal seams, the Upper Freeport, Lower Freeport, Upper Kittanning, and Lower Kittanning, occur within the strata in the project area. The Lower Kittanning Coal and Upper Kittanning Coal are the two prominent coal seams within the vicinity of the site. Both beds have been stripped and deep-mined. Based on available mine map information, it appears that both the Lower and Upper Kittanning Coal were mined using the room-and-pillar method. The mine maps also indicate that extensive retreat mining was performed in this area. Only the Upper Kittanning Coal required grouting in the area below this portion of the landfill expansion. The seam thickness of the Upper Kittanning averages 92 cm (3 ft) within the project area, with a maximum thickness of 107 cm (3.5 ft). Previous geologic investigations described in the landfill permit application indicated that substantial collapse of the mine voids had occurred. Based on these investigations, it appeared that in most of the project area, the mine voids were completely collapsed or gob-filled, with open voids averaging less than half of the mine void volume based on seam thickness and typical mining extraction ratios.

Substantial excavations were required in the project area to remove coal within 7.6 m (25 ft) of the liner systems and to achieve the proposed liner grades in the 3:1 slope area. To facilitate the grouting operations, the blasting and excavation operations were designed to create two wide flat benches at elevations just above final landfill grades. Grouting operations were performed from these two benches. The lower bench, which was created by required overexcavation of the Lower Freeport Coal (to provide minimum 7.6 m (25 ft) separation between the landfill liner and coal seams), occurs approximately 10.6 to 15.2 m (35 to 50 ft) above the Upper Kittanning deep mine. The upper bench associated with the overexcavation of the Upper Freeport Coal occurred approximately 28.9 to 41.1 m (95 to 135 ft) above the Upper Kittanning deep mine. After overexcavation of the coal within 7.6 m (25 ft) of the liner grades, approximately 3.2 ha (8 acres) of Upper Kittanning deep mines remained which required grouting.

**Grout Hole Drilling.** The perimeter of the grouting area was located and staked by survey. Locations for grout barrier holes were marked at 4.6 m (15 ft) centers along the perimeter of the grouting area. The actual locations of the grout barrier wall grout placement holes, installed around the perimeter of the grout area, were adjusted during drilling operations to achieve the minimum 30.5 m (100 ft) liner subgrade-mine separation requirements.

Production grout hole locations were staked within the grouting area on approximately 12.2 m (40 ft) centers. The 12.2 m (40 ft) spacing was based on observations of the travel distance of the grout during previous mine stabilization programs at this site. Some of the production holes could not be drilled on the 12.2 m (40 ft) grid spacing due to the highwall created by coal isolation barrier excavations. To achieve grout coverage in these areas, production hole locations were adjusted and additional production holes were drilled around the locations of known voids in an attempt to target high grout take areas. As a result, many production holes were drilled at spacings less than 12.2 m (40 ft).

A total of 541 barrier and production grout holes were drilled within the project area. All production and barrier wall grout holes were drilled using air rotary drill rigs equipped with a 16.5 cm (6-1/2 in) diameter air percussion hammer drill bit. Drilling records were maintained by the field engineer documenting the depths at which the coal, void, or fractured strata were encountered and the thicknesses of these zones. The intact coal seam (assumed coal mine pillars) was encountered in only about 20% of all drill holes. This suggests that the extraction rate in this area was much greater than that normally associated with room-and-pillar mining and confirms that retreat mining was common in the area. Collapse of the mine roof in the retreat mined area substantially reduced the mine void volume to be grouted. The extent of the collapsed mine voids and remaining void spaces is variable throughout the stabilization area. However, of the 541 holes that were drilled, only 26 (5%) encountered open void space at the Upper Kittanning Coal elevation.

The Upper Kittanning mine conditions interpreted from the drilling activities were observed and documented visually at the base of the highwalls created by the overexcavation of coal in the project area. The exposed deep-mined areas were observed to be totally collapsed with a few, highly deformed pillars and a small amount of void space remaining as delaminations along bedding planes within 3 to 6 m (10 to 20 ft) above the mine floor.

**Grout Production.** A fly ash-cement grout mixture was used for this mine stabilization project. The fly ash was supplied from a local coal-fired powerplant. This fly ash source was preapproved for use by the DER, based on chemical analyses to identify pollution potential and physical characteristics. A grout mixture of 5 parts fly ash to 1 part cement (by weight) was used throughout the mine grouting project. This mixture was selected to meet a minimum grout strength of 0.7 Mega-Pascal (MPa) (100 psi) as required by the landfill permit. Test cylinders for compressive strength testing were made from the grout mixture delivered to the site on a daily basis. Compression testing of the grout cylinders was performed after a 7-day curing period. Compressive strengths of the grout ranged from 0.8 to 6.1 MPa (113 to 877 psi) with an average of approximately 3.8 MPa (556 psi).

The fly ash, cement, and water were mixed at an offsite commercial concrete batch plant and delivered to the job site in standard concrete ready-mix trucks. The dry weights of fly ash and cement were recorded and enough water was added at the batch plant to provide a uniform mixture. Additional water was added at the site to provide the correct consistency (slump) depending on the conditions of the area being grouted. Grout barrier holes were filled using a stiffer grout mix, typically with a slump in the range of 20.3 to 25.4 cm (8 to 10 in). Production grout holes were filled with a more fluid grout mix, usually with slumps ranging around 30.5 cm (12 in), so as to promote greater coverage and more complete filling of the fractured overburden conditions encountered in many of the boreholes.

**Grout Placement.** Grouting of the deep mine voids and associated fractured rock strata was performed by gravity placement. The fly ash-cement mixture was delivered directly into the uncased barrier-production grout boreholes by the ready-mix trucks. The barrier grout holes were filled first, beginning at the lowest portions of the mined area (downdip end) and continuing around the perimeter of the grout area. This order of filling was observed so as to construct a barrier grout wall around the perimeter of the project area to contain the production grout. Grout was placed into each barrier hole until it was filled to the surface. The grout level was observed for at least one-half hour after initial placement; if the grout had settled, more grout was added until the hole was again filled to the surface. All barrier holes were filled in this manner with records kept of the hole number, date(s) of grouting, and grout take volumes.

Production grout holes were filled in the same manner as the barrier holes, except that the slump was typically 27.9 to 30.5 cm (11 to 12 in). The higher slump grout was used to increase the distance that the grout would flow within the area contained by the barrier holes to promote filling of fractures and smaller spaces within the gob-filled seams and fractured overburden. The production grout holes were filled systematically beginning at the downdip or lower portions of the deep mine within the project area. Once grouting placement began in a grout hole, every attempt was made to complete the grouting of that hole before moving to the next hole. This procedure was adopted since discontinuous grouting typically resulted in the grout holes sealing off prematurely. If this condition was encountered, additional grout holes were drilled in the immediate area to complete the grouting of that area.

**Grout Take.** The records of grout used in this mine stabilization program indicate that a total of approximately 6291 $m^3$ (8,228 $yd^3$) of grout was placed within the project area. Approximately 60% of the grout was used in the northern half of the project area and 40% in the southern half. Both areas were approximately 1.6 ha (4 acres), so grout takes averaged 589.7 $m^3$/ha (1,234 $yd^3$/acre) in the northern area and 392.8 $m^3$/ha (822 $yd^3$/acre) in the southern area. The grout takes for the northern portion of the project area are very similar to those that were seen in previous mine stabilization projects on the Upper Kittanning at this site, while those in the southern area are somewhat lower. A review of the deep-mine maps indicates that the southern area contained more retreat-mined areas than the northern area. More complete subsidence in the southern area associated with the retreat-mining would explain the lower grout takes observed.

**Verification Coreholes.** Verification coring was performed at 49 locations within the project area. NX-sized cores were obtained from approximately 6 m (20 ft) above the mine elevation to at least 0.6 m (2 ft) below the bottom of the coal seam. As indicated on the corehole logs, grout was observed filling voids within the collapsed mine roof materials and filling fractures in the strata above the level of the Upper Kittanning Coal Seam. Where suitable lengths of intact grout cores were obtained, they were used for compressive strength testing. These core samples had compressive strengths averaging 7.1 MPa (1,025 psi), significantly exceeding the permit requirements. These higher compressive strengths are a direct result of the age of the samples, which was in excess of 28 days.

Some of the verification coreholes did not encounter grout in the rock cores. The majority of the coreholes that did not encounter grout showed the presence of solid rock, indicating a complete roof collapse where whole blocks of mine roof collapsed into the void space. These coreholes were characterized by core runs with good core recoveries of 80% to 100%. Only two verification coreholes encountered small voids or highly fractured rock strata that had not been sufficiently grouted. In these instances, the verification coreholes were grouted and additional production holes were located around the coreholes and grouted to ensure good grout coverage. Additional verification coreholes were then drilled, which showed the area to be grouted completely.

## Site 2

The mine stabilization program at Site 2 was also performed as part of a permitted expansion of the existing landfill and was similar to Site 1 in that the mine grouting area was associated with a lined area on a 3H:1V slope. This mine stabilization project took approximately 6 weeks to complete.

Based on geologic investigations conducted as part of the landfill permitting efforts, two coal seams are found within the project limits. These coal seams are identified as the Pittsburgh Coal and the Redstone Coal. The Pittsburgh Coal has been both surface-mined and extensively deep-mined in the area, and the Redstone Coal was only surface-mined. This mine subsidence control project deals with deep mines on the Pittsburgh Coal Seam. According to information contained in the landfill permit, it appears that the Pittsburgh Coal was mined using the room-and-pillar method, with some retreat mining. According to drilling information, the thickness of the

Pittsburgh seam ranges from 2.4 to 3 m (8 to 10 ft). Under the permit area, it was determined that an approximate 8-ft thickness of the Pittsburgh Coal was removed at extraction rates of approximately 70%.

On this project it was decided that the majority of the deep-mined areas would be overexcavated in conjunction with removal of coal within 7.6 m (25 ft) of the landfill liner system. The remaining deep-mined areas, consisting of approximately 0.6 ha (1.5 acres), which occurred along the perimeter of the expansion area at the top of the 3H:1V slope, were to be grouted. This mine grouting project was performed from two benches prepared as part of the overexcavation operations. These two benches ranged from 7.6 to 24.4 m (25 to 80 ft) above the deep mine.

**Grout Hole Drilling.** The drilling program was started at the eastern (downdip) end of the mine stabilization area. Drilling progressed from this point around the perimeter of the area at 4.6 m (15 ft) spacings. These perimeter (barrier) holes were located so that relatively stiff grout injected into these areas would form a barrier around the mine stabilization area and prevent the loss of the thinner grout that was to be injected in the interior (production) holes. Production grout holes were drilled in a 4.6 m (15 ft) grid pattern across the interior of the mine stabilization area. Drilling progressed across the area from east to west to the western corner of the mine stabilization area, where previously excavated and backfilled mined areas were encountered. All production and grout barrier holes were drilled using air rotary drill rigs equipped with a 16.5 cm (6-1/2 in) diameter air percussion hammer drill bit. Drilling records were maintained by the field engineer documenting the depths at which the coal, void, or fractured strata were encountered and the thickness of these zones. These records were used to guide and assist in the evaluation of the grout program.

A total of 510 grout holes were drilled within the project area. Coal pillars were encountered in about 27% of all drill holes (barrier, production, or verification holes). This suggests that the extraction rate in this area was comparable to that normally associated with room-and-pillar mining (about 70%). Of the 510 holes that were drilled, only 61 (about 12%) encountered open void space at or above the Pittsburgh Coal elevation. The drilling information suggests that mine roof failures have occurred over much of the area and the associated subsidence has substantially reduced the remaining open mine void space. It is also believed, based on drilling observations, that many of the mined rooms were backfilled during mining with mine tailings or gob. These conditions were confirmed by the downhole video camera survey (discussed later in this section).

The mine conditions described above were observed and documented at the site in the Pittsburgh Coal overexcavation areas. The majority of the deep-mine voids in the area exposed by overexcavation were observed to be totally collapsed and/or backfilled with gob materials. The extent of the collapsed mine voids and remaining void spaces was variable throughout the stabilization area. However, based on the drilling information, it was determined that the Pittsburgh mine voids were substantially collapsed and/or backfilled and only a fraction of the original mine void volume remained to be grouted.

**Grout Production.** A sand-cement grout mixture was used for this mine stabilization project. Prior to the start of the project, mix designs and test batches of grout were prepared to determine the appropriate proportions of sand, cement, and water needed to achieve a minimum compressive strength of 0.7 MPa (100 psi). A grout mix consisting of 1193 kg (2,630 lb) of sand, 213.2 kg (470 lb) of cement, and 219.5 l (58 gals) of water per cubic yard of grout at a 25.4 cm (10 in) slump was determined to be optimal for the required strength. The sand, cement, and water were mixed at a local offsite batch plant and delivered to the job site in ready-mix concrete trucks. Weights and volumes of the mixture were recorded for each load. Additional water was added, as needed, at the site to provide the correct consistency (slump) prior to injection.

Grout test cylinders were collected from each 38.2 $m^3$ (50 $yd^3$) of grout mixture delivered to the job site. The 28-day compressive strengths of the grout ranged from 6.4 to 12.8 MPa (928 to 1,863 psi), with an average of about 9.7 MPa (1,405 psi).

**Grout Placement.** The grout was poured into each barrier or production grout hole using a small hopper with a funnel end to direct the material into the grout holes. The barrier grout holes were filled with grout having a 20.3 to 25.4 cm (8 to 10 in) slump, which was determined to be appropriate to allow the mix to flow down the hole and fill the void space. Grout was placed into each barrier hole until it was filled to the ground surface. The grout level was observed for at least one-half hour after initial placement; if the grout had settled, more grout was added until the hole was filled to the surface.

Production grout holes were filled in the same manner as the barrier holes, except that the slump was typically 27.9 to 30.5 cm (11 to 12 in). The higher slump grout was used to increase the distance that the grout would flow within the mine voids and to promote filling of fractures and smaller spaces within the gob-filled seams and fractured overburden.

**Grout Take.** Estimates of the amount of void space existing in the mine stabilization area were made utilizing the information collected during the drilling portion of the project. The drilling logs indicated that the average height of the void based on the cumulative height of all voids encountered was approximately 1.2 m (4 ft). It was also determined that only about 12% of the holes contained voids. Assuming an average 1.2 m (4 ft) void space, over 12% of the area, and a grout area of 0.6 ha (1.5 acres), the existing void volume can be estimated to be approximately 886.9 $m^3$ (1,160 $yd^3$). Additionally, some of the holes logged as voids during drilling did not have the larger grout takes typical of boreholes drilled into an open mine void. It was determined by the downhole video camera survey that, in fact, some holes logged as having voids actually were localized fractures or soft gob zones. This means that the actual void volume is somewhat less than the 886.9 $m^3$ (1,160 $yd^3$) estimated above. The records of grout used in the mine stabilization program indicate that a total of 730.9 $m^3$ (956 $yd^3$) of grout was placed, which is comparable to the estimated void volume based on drilling information.

**Verification Drilling and Grouting.** Air rotary verification drilling and grout takes were used to evaluate the effectiveness of the grouting program. Following completion of production grouting, the subsurface conditions and grout takes recorded were compared. Based on this comparison, areas with the potential to have ungrouted void space were identified. Additional drilling was performed in these areas and the holes were subsequently grouted. If the grout take in a particular air verification hole was less than 0.76 $m^3$ (1 $yd^3$), the area was considered to be sufficiently grouted. Following this procedure, if an area was still suspected to contain ungrouted void space, additional verification holes were drilled and grouted. This procedure was followed until the entire mine stabilization area was sufficiently grouted. In all, 91 air verification holes were drilled in the mine stabilization area.

**Verification Coring.** Verification coring was completed at seven locations within the mine stabilization area. NX-sized cores were obtained, beginning approximately 6 m (20 ft) above the mine elevation and extending at least 0.6 m (2 ft) below the bottom of the coal seam. The majority of the verification coreholes encountered grout. Grout in the corehole was observed mostly as filling of voids within the collapsed mine and fracture fillings in the rock above the level of the Pittsburgh Coal Seam. Suitable lengths of intact grout cores were collected and used for grout compressive strength tests. The core samples retrieved had compressive strengths ranging from 10.8 to 17.1 MPa (1,560 to 2,480 psi) and averaging about 13.3 MPa (1,925 psi).

**Downhole Video Survey.** A downhole video camera survey was undertaken to determine the pregrouting conditions within the mine stabilization area and verify the filling of the void space after grouting. The pregrouting downhole video camera survey confirmed that the mine voids were generally collapsed. Many of the intervals that were logged as voids during drilling appeared to be localized, large fractures or soft zones of mine backfill or caved material that were mistaken to be open voids.

A postgrouting video survey of the verification holes was also performed. During these surveys, intervals of grout were observed in previous mine void intervals and as fracture fillings in broken or fractured overburden zones. The postgrouting video survey provided visual confirmation that complete void and fracturing filling had been achieved.

## Site 3

The original mine stabilization plan for this site entailed the complete excavation of the underlying Pittsburgh Coal Seam and deep mine workings during excavation for the development of the waste disposal areas. Horizontal placement of backfill materials, by pneumatic stowing, was proposed to fill any mine voids encountered along the base of the exposed highwalls. Previous landfill development construction activities used the horizontal, pneumatic stowing technique to fill mine voids encountered in the base of the highwall in the landfill area adjacent to this project area. However, the horizontal stowing method proved to be difficult, costly, and not completely effective. In addition, work required to prepare the highwall area for stowing created instabilities in the excavated highwall slopes. For these reasons, it was proposed to eliminate the underground mine voids and stabilize the overburden within this project area by grouting, prior to excavation of the highwalls.

Based on previous geologic investigations, it is known that the project site is underlain by the Pittsburgh Coal and the Redstone Coal. Within the project area the Pittsburgh Coal occurs at a depth of 39.5 m (130 ft) and the Redstone Coal occurs approximately 27.4 m (90 ft) above the Pittsburgh Coal. The Pittsburgh Coal has been both surface mined and extensively deep-mined in the area, while the Redstone Coal has been only surface mined. The thickness of the Pittsburgh seam is generally about 2.4 to 3.7 m (8 to 12 ft), and it was determined that approximately 1.8 to 3.7 m (6 to 12 ft) of the Pittsburgh Coal was removed within the project area. An evaluation of available mine maps revealed that the Pittsburgh Coal was mined at the site using the room-and-pillar method with typical extraction ratios of approximately 70%. The mine maps also indicated that a main entryway for the mine existed in the vicinity of the proposed mine stabilization project area, along with large coal pillars which were left inplace to support the roof along this main entry. Excavations in the adjacent project areas also encountered large amounts of in-place coal, as well as large, open, intact mine voids confirming this condition.

This mine stabilization project was conducted along the top of a 39.6 m (130 ft) highwall excavated as part of the landfill expansion and on a bench cut at the Redstone coal elevation. Grouting of the deep mines was performed on mine voids located under and immediately adjacent to the highwall and included areas within the 30° angle of draw. This mine stabilization project took approximately 5 weeks to complete.

**Grout Hole Drilling.** The drilling plan called for six rows of grout injection holes running parallel to the northern perimeter of the landfill expansion and highwall excavation. The rows were labeled A through F (south to north) and were spaced approximately 4.6 m (15 ft) apart. Rows D and E were spaced 6 m (20 ft) apart to make allowances for the slope of the partly excavated highwall. Holes were drilled along each row at a 4.6 m (15 ft) spacing, and hole alignment was staggered between each row.

Grout hole drilling was performed using air rotary methods and a standard 17.1 cm (6-3/4 in) rock bit. Work was started initially on rows A and F and progressed around the perimeter of the area at 4.5 m (15-ft) spacing. These perimeter (barrier) holes were used to emplace relatively stiff grout to form a barrier around the mine stabilization area and prevent the loss of the thinner grout that was to be grout in the interior (production) holes. Due to the existing surface conditions of the mine stabilization area, access to the exact hole locations as stipulated by the drilling plan was difficult or unsafe due to the highwall. These field conditions dictated slight adjustments of the drilling plan to cover the entire mine stabilization area and maintain adequate spacings. Some of the hole locations along the top and the toe of the highwall in rows D and E were completely inaccessible, or undrillable because of safety concerns. Many of these holes were "angle" drilled. Typically, the angles were less than 6°. Drilling records were maintained by the field engineer documenting the depths at which the coal, void, or fractured strata were encountered and the thickness of these zones.

The drilling defined a very distinct trend of large, open voids within the project area. These voids roughly correlated with the main haulageway as shown on the deep-mine map. Coal pillars were encountered in about 40% of all injection holes, suggesting that the coal extraction rate in this area was less than that normally associated with room-and-pillar mining. Much of the unmined coal appears to be large, closely spaced chain pillars, which protected the mine haulageways from roof collapse. These chain pillars have apparently provided more long-term roof support for the haulageways, resulting in larger, more intact mine voids existing in these areas. Of the 255 holes that were drilled, 92 (36%) encountered open void space at or above the Pittsburgh Coal elevation.

The mine conditions described above were observed and documented at the site in the adjacent Pittsburgh Coal overexcavation areas. The mine in the exposed areas was observed to be partly collapsed with many in-place coal pillars and a few intact rooms and haulageways. However, the extent of the collapsed mine voids and remaining void spaces is variable throughout the stabilization area. Based on these observations and the drilling information, it was determined that much of the Pittsburgh mine voids are intact and a considerable amount of the original mine void volume remained to be grouted.

**Grout Production.** Two different grout mixes were utilized during this mine stabilization project: a sand and cement mix; and a fly ash, sand, and cement mix. The sand and cement mix consisted of approximately 1338.1 kg (2,950 lb) of sand, 136.1 kg (300 lb) of cement, and 189 to 227 liter (50 to 60 gal) of water. This mix was used for the first 3 days of the project, mostly in the barrier holes. The remainder of grouting was performed with a fly ash, sand, and cement grout mix consisting of approximately 226.8 kg (500 lb) of fly ash, 907.2 kg (2,000 lb) of sand, 147.4 kg (325 lb) of cement, and 189 to 227 l (50 to 60 gal) of water. The mix components and part of the water were mixed at the batch plant and delivered to the job site in ready-mix trucks. Additional water was added, as needed, at the site to provide the correct consistency (slump) prior to injection.

At least one test cylinder was collected each day during the grouting operations. The cylinders were tested and showed compressive strengths ranging from 0.8 to 3.4 MPa (115 to 495 psi), with an average of about 1.4 MPa (196 psi) after a 3-day cure period.

**Grout Placement.** The grout was placed into each barrier or production grout hole using a small hopper with a funnel end. The barrier grout holes were filled with grout having a 15.2 to 25.4 cm (6 to 10 in) slump. Grout was placed into each injection hole until it was filled to the ground surface. The grout level was observed for at least one-half hour after initial placement; if the grout had settled, more grout was added until the hole was filled to the surface.

Production grout holes were filled in the same manner as the barrier holes, except that the slump was typically greater than 25.4 cm (10 in). The higher slump grout was used to increase the distance that the grout would flow within the project area and to promote filling of fractures and voids within the mine workings and fractured overburden.

**Grout Take.** The records of grout used in the mine stabilization program indicate that a total of 2153.5 $m^3$ (2,816.5 $yd^3$) of grout was placed. Estimates of mine void volumes can be made based on typical mining extraction ratios and coal seam thickness. These estimates are typically greater than the actual remaining mine voids that require grouting, owing to mine roof collapse and subsidence. However, utilizing the information collected during the drilling portion of the project, a relatively accurate estimate of the void volume can be determined. The drilling logs indicate the thickness of all voids encountered, and it was determined that the average height of the void based on the cumulative height of all voids encountered was approximately 1.04 m (3.4 ft). It was also determined that about 36% of the holes contained voids. Assuming an average 1.04 m (3.4 ft) void space, over 36% of the area, and a grout area of 0.44 ha (1.1 acres), the existing void volume can be estimated to be approximately 1586.5 $m^3$ (2,075 $yd^3$), which is approximately 40% less than the actual volume of grout placed into the mine voids. This discrepancy is most likely attributed to grout being placed outside the project area during construction of the barrier walls in areas where large voids were encountered. This occurred in rows A and F, where several adjoining barrier holes encountered large, connected voids and had very large grout takes.

**Air Rotary Verification Drilling and Grouting.** Following completion of production grouting, the subsurface conditions and grout takes recorded were analyzed. From this evaluation, areas with the potential to have ungrouted void space were identified. Additional drilling was performed in these areas, and the subsurface conditions were logged to determine the effectiveness of the grouting program. The verification holes were monitored for the presence of grout, coal, open voids, or fractured rock. Of the 24 air verification holes drilled, 21 encountered grout, 3 encountered fractured material, and none hit voids or coal. Additionally, all of the air verification holes were subsequently grouted. If the grout take in a particular hole was less than 0.76 $m^3$ (1 $yd^3$), the area was considered to be sufficiently grouted. Following this procedure, if an area was still suspected to contain ungrouted void space, additional verification holes were drilled and grouted. This procedure was followed until the entire mine stabilization area was determined to be sufficiently grouted.

**Verification Coring.** The intent of the verification coring program was to retrieve grout samples to verify the grout coverage as well as the insitu strength of the grout. Verification coring was completed at eight locations within the mine stabilization area. NX-sized cores were obtained, from approximately 6 m (20 ft) above the mine elevation and extending at least 0.6 m (2 ft) below the bottom of the coal seam.

Grout was observed in the core as filling of voids within the mine and as fracture fillings in the strata above the level of the Pittsburgh Coal Seam. Seven of the verification coreholes encountered grout, and one encountered coal. Suitable lengths of intact grout cores for testing were obtained from most of the holes that hit grout. The core samples retrieved had (unconfined) compressive strengths ranging from 6.5 to 7.6 MPa (940 to 1,100 psi) and averaging about 7.1 MPa (1,035 psi).

The unique opportunity to observe the grouted zone was presented at this site when the excavation for the highwall in this area was performed. Overexcavation to provide the required 7.6 m (25 ft) coal separation zone for the landfill exposed the grouted zone at the base of the highwall. Visual observations of the grouted mine voids showed complete filling of the open voids by the grout. In addition, the grout was observed to have filled in the small voids within the caved materials, and fractures created by roof falls and subsidence. The extent of the void filling was notably very complete, and only a few small voids could be observed.

## Project Comparison

The three grout projects described were designed to utilize readily available materials, locally available standard equipment and services, and simplistic grouting techniques in order to minimize cost and increase productivity. The drilling services employed for the grouting programs were shothole drillers from the coal mining industry who specialize in high production drilling at low cost. Grout materials were selected based on local availability, low cost, and suitability. The decision to mix the grout offsite at local concrete batch plants was based on previous experiences with grouting projects. Grouting projects that stockpile materials and mix grout onsite require large areas for storage, material handling, and mixing equipment. In addition, these projects are difficult to supply, and typically the large water supplies required to mix the grout are not available. Mixing the grout offsite eliminates material storage problems, mobilization and setup of substantial amounts of equipment, and onsite logistical problems.

Grout mixtures were developed based on the availability and cost of the materials. Fly ash was used in the grout mixtures, if readily available, owing to its physical properties, which enhance the flow characteristics of the grout. Fly ash grouts penetrate readily into fractures and loose caved materials in the mine voids providing a greater degree of void filling.

Grout placement techniques were by gravity through uncased boreholes. Since the drilling services were relatively inexpensive, it was more cost effective to drill additional grout placement holes to achieve grout coverage or to replace a collapsed or blocked borehole than to use the more conventional methods of installing casing in boreholes, tremie pipes, and injection of grout under pressure.

A summary of the size, drilling, grout mixtures, drilling for the three grout projects is presented in Table 1.

Table 1. Grouting project summary.

| Site | Hectares (Acres) | Grout holes m (ft) | Grout mixture | Grout take $m^3$ ($yd^3$) | Verification drilling |
|---|---|---|---|---|---|
| 1 | 3.2 (8.0) | 541<br>10,567 (34,668) | Fly ash-cement | 6291<br>(8228) | 49 coreholes. |
| 2 | 0.6 (1.5) | 510<br>9,555 (31,350) | Sand-cement | 731<br>(956) | 91 air rotary,<br>7 coreholes. |
| 3 | 0.44 (1.1) | 255<br>7,160 (23,491) | Sand-cement fly ash-sand- cement. | 2154<br>(2817) | 24 air rotary,<br>8 coreholes. |

The mine subsidence control projects described in the preceding sections, while different in terms of site specifics, have one basic similarity. This similarity is the subsided conditions of the mined voids. Typical estimates of the mine void volumes requiring backfilling to prevent subsidence invariably overestimate the amounts of materials that can actually be injected. These overestimations occur as a result of the failure to consider past subsidence or because the amounts of subsidence that have occurred are underestimated owing to the lack of enough site-specific information. The projects discussed in this paper suggest that mine void volumes remaining in older deep mines requiring grouting to prevent future mine subsidence are in the range of 30% to 40% of the estimated mine void volume, based on typical extraction-mine seam thickness techniques. This percentage is lower at Site 2 due to the fact that a portion of the project area had not been deep-mined.

## Cost

The cost of mine subsidence control grouting projects will vary depending on the size of the area to be grouted, thickness of the coal seam, volume of the mine voids to be grouted, and drilling depths. The most significant cost items in the grouting projects presented were the drilling and the grout. Labor cost can be a major item if more conventional grouting techniques are used. However, in these projects the labor costs were low since the majority of these costs are included in the unit prices for the drilling and the grout. A summary of the project costs for the three mine subsidence control projects is included in table 2.

Table 2. Grout project cost summary.

| Site | Hectares (Acre) | Drilling cost | Grout cost | Labor cost | Total cost | Cost per hectare (acre) |
|---|---|---|---|---|---|---|
| 1 | 3.2 (8) | $52,000 | $288,000 | $25,000 | $365,000 | $114,062 (45,625) |
| 2 | 0.6 (1.5) | 47,000 | 41,000 | 15,000 | 103,000 | 171,666 (68,700) |
| 3 | 0.44 (1.1) | 23,000 | 110,000 | 12,500 | 145,500 | 330,681 (132,272) |

From the data presented in table 2 it can be seen that the cost per acre for mine grouting can vary greatly. The most variable cost is the grout cost, which is directly related to the mine void volume to be grouted. The degree of mine collapse, relationship of the project area to the mine layout (room-and-pillar areas, main entries, retreat-mined areas), and thickness of the coal seam (mine void height) vary significantly among mined areas and can greatly impact the grout volumes and cost.

## Conclusions

The deep mines associated with these projects are representative of the deep mining in this region. Conditions in the deep mines were well documented by the high density of the drilling and visual observations. These investigations show that the mine voids are typically substantially collapsed and are filled with gob or caved material. As a result, the void volume remaining to be grouted is substantially less than estimated by conventional conservative estimation techniques. In addition, subsidence prediction techniques that use original mine void heights and volumes to calculate potential subsidence and estimate potential future damage from these old deep mines will overestimate the potential and be conservative.

Conventional mine grouting projects typically specify high-strength grout mixes; cased grout injection boreholes, grout tremie pipes, and pump or pressure injection systems. These requirements increase the cost of the grouting projects significantly with respect to material costs and are labor intensive. The use of these techniques typically limits grout placement production rates, which increases the length of the project and ultimately the cost. The simplified grout placement methods used in the case studies presented in this paper increase productivity by eliminating labor-intensive activities and decrease cost by reducing labor, materials, and equipment. Another consideration for reducing project cost is the use of low-cost, locally available materials and low-strength grout mixtures. Unless high loads are expected to be superimposed on the project area in the future, high strength grouts are not warranted. As demonstrated by these projects, the use of fly ash in the grout mixes, with minimal amounts of portland cement, produces a grout with considerable strength and good flow characteristics at a reasonable cost.

The simplistic methods used to perform these mine subsidence control projects are the key to their success with respect to cost effectiveness. These projects relied upon inexpensive, high-production drilling as the key to providing the required grout coverage. If a grout production hole collapsed or became blocked during grout placement, an additional hole (or holes) would be drilled in that area to achieve complete grout coverage in the area. An important aspect of this grouting technique is the maintenance of good field records. If drilling records indicated substantial voids were encountered in a particular area and grout takes were low, these areas were targeted for additional drilling and grouting during the verification drilling phases of the projects. The success of this technique is verified by the observations made of the grouted zones in excavations, the degree of void filling observed by the air rotary verification drilling procedures and verification coring activities, and the borehole video camera surveys.

Conventional grouting techniques utilizing more sophisticated procedures, such as cased injection holes, tremie systems, and pressure injection systems, have a definite application in specific situations. However, for large-area mine subsidence control projects, the simplistic grouting techniques used in these projects provide an efficient and effective alternative mine subsidence control technique.

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# THE USE OF CONTAINMENT BARRIERS AND FIRE FIGHTING FOAMS FOR THE EXTINGUISHMENT OF COAL WASTE BANK FIRES: A LABORATORY STUDY[1]

J. R. Jones[2], A. G. Kim[2], and A. M. Kociban[2]

**Abstract**: The U.S. Bureau of Mines studied the integrated use of grout barriers as a potential method to isolate and fire fighting foams to extinguish coal waste bank fires. Small scale tests were conducted in a 2.5 $m^3$ gravel pile utilizing a series of electric heating elements within the pile to simulate a burning coal waste bank. Testing focused initially on a polyurethane grout mixture that is relatively inexpensive, could be readily pumped through coal waste, and would reduce the permeability and porosity of the waste material effectively to contain the subsequently injected fire fighting foam mixture. The grout mixture that best satisfied these conditions was a mixture of one-third water and two-thirds polyurethane, which had an expansion ratio of about 15:1, reduced the permeability of coal waste from 50 darcys to less than 1 darcy, and the porosity from 20% to less than 1%. In small-scale heating tests, injection of a medium-density, medium-expansion foam reduced temperatures 0.5° C/min without the grout containment barrier and 0.75° C/min with the containment barrier about two hours after the respective foam injection tests.

## Introduction

Burning coal waste banks can cause a variety of environmental problems in both active mine areas and abandoned mine lands. These can include the emission of noxious or toxic fumes, the potential of the fire spreading to adjacent and nonaffected areas of the waste bank or in some cases to areas outside the confines of the waste bank, and other safety hazards associated with surface fires. For subsurface mine fires, many methods have been used as a means of extinguishment, including water injection, the use of cryogenic gases and liquids, and excavation. In many cases, these extinguishment projects have been used in conjunction with some type of constructed internal barrier within the mine in order to isolate the fire as well as contain the extinguishment material within the fire zone. An effective barrier should also prevent the transfer of heat from the burning zone to the nonburning areas of the mine. The materials used to form these underground barriers have generally been cement-type grouts.

Previous research has demonstrated that induced gas flow patterns through coal waste or materials with size characteristics similar to those of wastes can be reasonably predicted (Jones and Chaiken 1990, Jones et al. 1992). Prediction of flow patterns within waste materials is essential in order to maximize the potential for successful extinguishment of fires in burning waste banks. Because these flow patterns represent the paths the heated gases will travel from the burning areas toward unburnt areas, as well as the likely route the extinguishment materials will follow during the extinguishment phase, the permeability and porosity of the waste bank material should be understood. The physical behavior of the extinguishment materials during the extinguishment process will also influence the effectiveness of heat reduction, and how well, and for how long the material can be contained at or near the burning zone.

To test a possible alternative method for the extinguishment of waste bank fires, the U.S. Bureau of Mines undertook a series of integrated bench-scale experiments using coal waste material, limestone gravel, polyurethane grout, and medium-density fire-fighting foams as testing materials to determine (1) the reduction of permeability and porosity after polyurethane grout injections, (2) the durability of the injected grout material

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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with respect to atmospheric weathering conditions, (3) the flow characteristics of a variety of polyurethane mixtures through the waste materials, (4) the effectiveness and flow characteristics of the medium-density, medium-expansion fire-fighting foam as an injected material in transferring heat and cooling a heated zone during an extinguishment process, and (5) estimated material costs associated with the combined use of polyurethane grout as a containment barrier and medium-density foam as the extinguishment material for field-scale testing. The results summarized in this report indicate that based on bench-scale testing, the integrated approach has potential application as a means of extinguishing burning waste bank fires.

## Methods, Materials and Testing

### Waste Materials

The coal waste material for this test was collected from an active coal preparation plant in western Pennsylvania. It was partially weathered. The size of the waste material was determined in the laboratory by standard methods (fig. 1). Other characteristics of the waste are presented in table 1. Permeability of the material was measured at 50 darcys within a six-sampling port 0.3 $m^3$ permeameter under constant- head conditions. The porosity of the material was found to average 20%. The limestone gravel used for the heated series of tests was roadbed material averaging about 5 cm across its longest (a-axis) dimension. The permeability of the gravel was about 500 darcys with a porosity of about 28%..

Table 1. Refuse characteristics

| | As-received | Dry | Dry ash-free |
|---|---|---|---|
| Proximate Analysis, %: | | | |
| Moisture | 2.78 | -- | -- |
| Ash | 67.37 | 69.29 | -- |
| Volatile matter | 12.49 | 12.85 | 41.85 |
| Fixed carbon | 17.36 | 17.86 | 58.15 |
| Ultimate Analysis, %: | | | |
| Hydrogen | 1.84 | 1.57 | 5.11 |
| Carbon | 19.23 | 19.78 | 64.42 |
| Nitrogen | 0.27 | 0.28 | 0.91 |
| Sulfur | 5.29 | 5.44 | 17.72 |
| Oxygen | 6.00 | 3.64 | 11.84 |
| Ash | 67.37 | 69.29 | -- |
| Heating value (Btu/lb) | 3,344 | 3,440 | 11,203 |

### Polyurethane Grout Injection

To test the effectiveness of polyurethane grout as a potential containment barrier for burning waste banks, a series of injection tests were conducted with a variety of polyurethane grout and water mixtures. The test series included the injection of 100% polyurethane grout, 80% polyurethane grout - 20% water, 67% polyurethane grout - 33% water, 50% polyurethane grout - 50% water, and 10% polyurethane grout - 90%

water into 1 m long by 7.6 cm diameter polyvinyl chloride pipes filled with the coal waste. The various mixtures were injected at 0.6 m depths from the top of the pipes through a 0.6 cm rod. Injection pressures were 345 $kN/m^2$ for each test. A volume of 0.2 L of each grout mixture was used for the injection tests.

The particular mixture that appeared to best satisfy the goals of this part of the experiment outlined in the introduction contained about 2 parts polyurethane grout - 1 part water. The coal waste - polyurethane grouted mixture cured within 5 min, has an expansion ratio of about 15:1, and maintained its integrity after being exposed to atmospheric conditions for over a 1-yr period. The permeability of the grouted coal waste using this mixture was reduced from 50 darcys to less than 0.01 darcys, and porosity was reduced from 20% to less than 1% within the confinement of the polyvinyl chloride pipe.

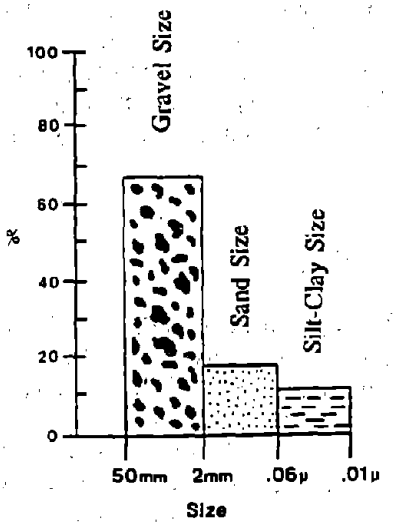

Figure 1. Histogram of sediment size distribution of coal waste material.

Additional testing of this grout mixture under less confined conditions also showed satisfactory results. For this test, another 0.2 L of the grout mixture was injected at 345 $kN/m^2$ through 0.6 cm rods at eight injection points into a 3 by 1.5 by 1.2 m box containing similar coal waste. The curing time was about 8 to 10 mins. Porosity was reduced from 20% to about 3%. Air flows induced by a suction fan through the waste measured with magnahelic gauges at nine monitoring ports within the box showed an average flow reduction of about 50% from 1 cm-$H_2O$ to 0.52 cm-$H_2O$. The expansion ratio of the 2 parts polyurethane - 1 part water grout mixture in the unconfined box was about 8:1.

## Foam Injection

Removal of heat from burning coal requires the passage of some heat- absorbing agent through a burning zone (Chaiken et al. 1984). Attempts in the past to use water have demonstrated that although water is capable of removing heat, its distribution through the waste pile cannot be readily controlled. To test alternative methods, a medium-density, medium-expansion fire fighting foam was evaluated as a heat transfer agent.

Foam is a dispersion of gas bubbles in a liquid. It is produced when a mixture of a surfactant and water encapsulates a small volume of air. Contiguous bubbles create a foam. A high-density (0.15 $g/cm^3$), low-expansion (10:1) foam is a wet foam with a lower ratio of gas to water. A low-density (0.002 $g/cm^3$), high-expansion (100:1) foam is a dry foam with a high ratio of gas to water. A medium-expansion, medium-density (about 0.05 $g/cm^3$) foam falls between these two extremes, and is considered most applicable to mine fire control type problems (Kim et al. 1992). A medium-expansion foam combines desirable qualities of high- and low-expansion foams and has an apparent density of about 0.05 $g/cm^3$. A typical foam in this range would contain 95% gas and 5% liquid. The liquid ratio of water to surfactant is about 99:1.

Foam used as an extinguishment material will be sensitive to the permeability and porosity of the burning waste bank. According to the Hershel-Bulkey model, the effective viscosity of the foam is more sensitive to changes in the width of flow channels than to the height of the material into which it is being injected (Kim and Chaiken 1993). This indicates that in a nonhomogeneous porous bed in which there is a distribution of channel widths and heights (i.e., the permeability "network" of the material), the effective viscosity and the

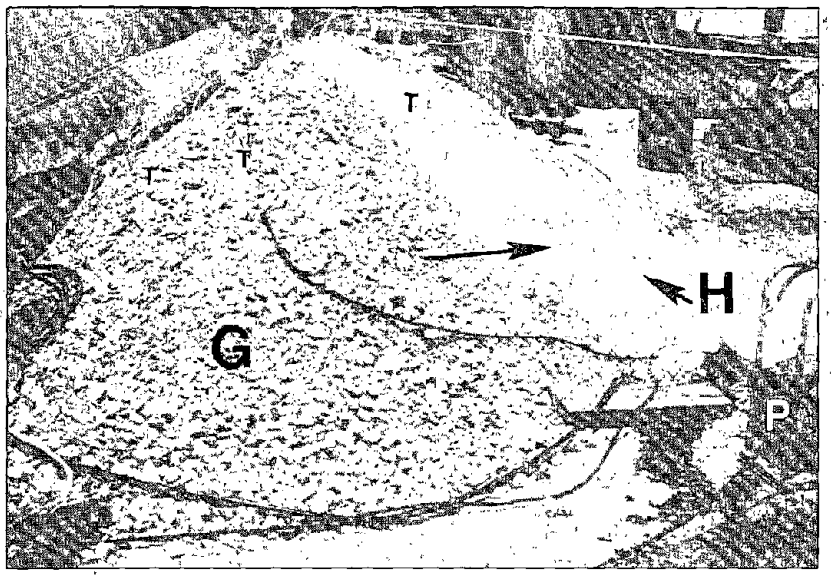


Figure 2. Simulated coal waste pile. G, gravel material, P, foam pump, H, heated zone, T, thermocuple. Arrow denotes foam.

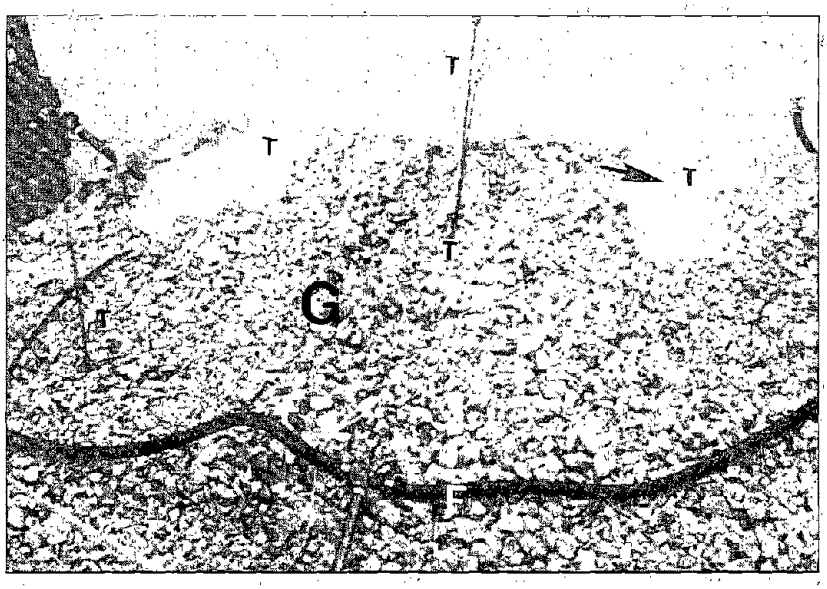


Figure 3. Simulated coal waste pile illustrating foam moving upward from gravel pile. G, gravel material, R, foam injection rod, F, foam injection tubes, t, thermocouples. Arrow denotes foam.

buoyant force will favor the upward flow of foam through a wide channel preferentially to lateral flow through a narrow channel. On the basis of this reasoning, foam injection rods should be placed below the level of the burning waste bank zone to ensure maximum contact of the upwardly moving foam through the burning zone. The amount of foam that is needed to effectively cool and transfer heat within a burning waste bank depends on the area affected by the fire, and the temperature of the heated zone.

To test the potential of foam as an extinguishment and cooling material, a 3 by 1.2 by 0.6 m limestone gravel pile was used to simulate a bench-scale waste bank fire (fig. 2). Two 1 m, 240 V heating elements were placed within the center of the gravel pile. A series of 14 thermocouples were placed at points within the pile to monitor temperatures during the heating process before and after foam injection into the pile was completed. Two 2.5 cm pipes inserted into the pile below the heating elements were connected to a foam pump. These served as the foam injection points (fig. 3).

Temperatures varied from about 180° C at the center to about 40° C at the edges of the heated zone after about 6 h of heating. A mixture of 18.9 L. of medium-expansion foam (99% water - 1% surfactant) was injected into the pile at 30 s intervals for about 30 min until the entire 18.9 L mixture had been pumped through the two injection rods. Injection pressures were about 35 $kN/M^2$. Visual observations during the injection process showed an initial conversion of the foam into steam, which escaped from the top and sides of the gravel pile. As injection continued, hot water and steam were observed flowing from the base of the pile. About 5 min after injection, intact foam was observed moving upward and flowing out the top and sides of the gravel pile (fig. 3). Temperature changes about 2 hr after foam injection showed an average rate of decrease of about 0.5° C/min (fig. 4a).

To test the effect of foam injection on reducing temperatures within a confinement barrier, the 2 part polyurethane - 1 part water grout mixture was injected into the pile at 345 $kN/m^2$ through a series of 0.6 cm rods. Six rods were placed in two rows about 0.7 m away from the center encircling the heated zone. A volume of 0.2 L of the polyurethane grout mixture was injected through each rod and allowed to cure for about 12 hr. This essentially constructed a barrier within the test pile.

The heating and foam injection procedure for this test followed those described above. Steam-water-foam flows were similar to those observed during the foam injection into the ungrouted heated pile were produced (fig. 3). Temperature reduction 2 hrs after foam injection within the containing grout barrier averaged 0.75° C/min (fig. 4b).

**Material Costs**

The material costs of extinguishing a burning waste bank would be dependent on the extent of the fire within the waste bank. The material costs incurred during our experiment can be used to provide cost estimates for the integrated polyurethane grout barrier and foam injection method. Assuming a 20% porosity and a 10:1 grout expansion ratio, it would cost about $\$10/m^3$ to form an effective barrier within coal waste material. A medium-density foam surfactant costs about \$6/L. It took about 0.2 L of surfactant to cool 0.5 $m^3$ of material from a temperature of 180° C to about 100°C. So the surfactant cost to cool a 1 $m^3$ of hot or burning coal waste would be less than \$4. By way of comparison, it has been estimated that it would take about 30,000 L of medium-density foam surfactant and 3,000,000 L of water to cool an underground mine fire and proximal heated zone with a heat content of 1.5 trillion calories (Kim et al. 1992).

A potential field application of the process is illustrated in figure 5. After the fire is located within the waste pile, grout is injected to form a containment barrier around the fire zone. Foam would then be pumped at points below the fire zone. The amount of grout and foam necessary to successfully extinguish a waste bank fire would be dependent upon the area of burning zone as well as the heat content of the burning coal waste.

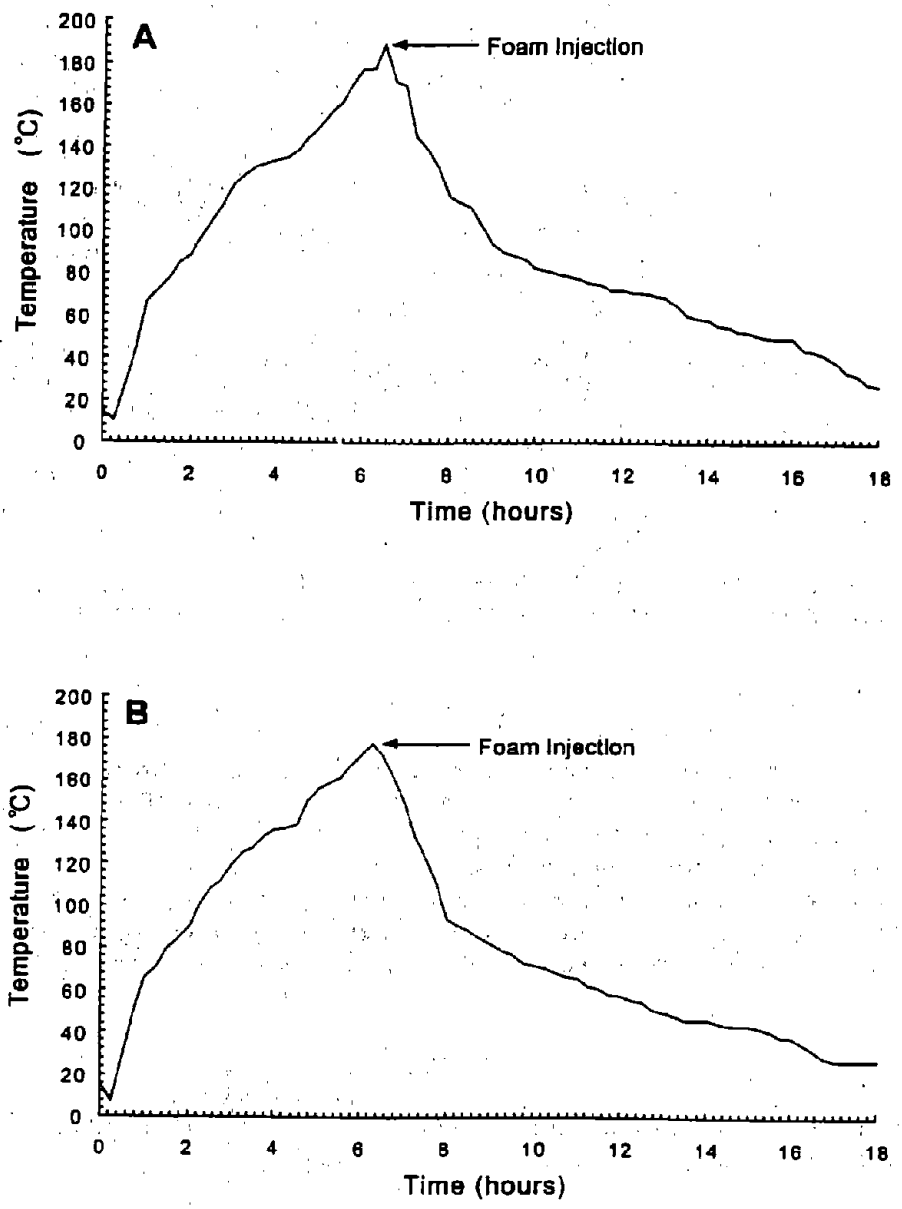


Figure 4. Temperature Reductions. Temperature reduction after foam injection near heated zone in simulated waste bank fire (A). Temperature reduction after foam injection within containment barrier near heated zone in simulated waste bank fire (B).

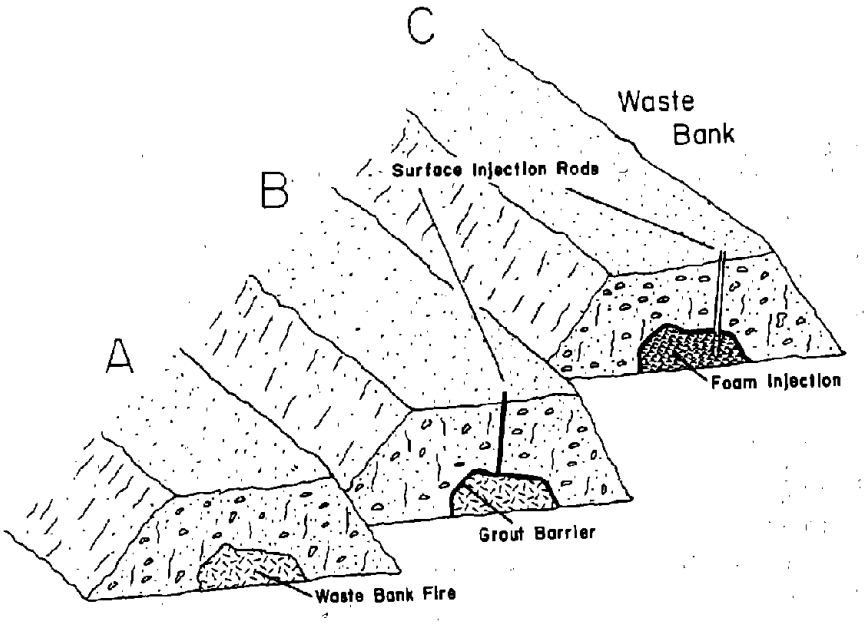


Figure 5. Idealized model of extinguishment process. Fire zone within waste bank (A). Grout containment barrier encircling the burning waste (B). Foam injection below fire zone (C).

## Conclusions

Bench-scale testing indicates that a medium-density, medium-expansion fire fighting foam combined with a polyurethane grout containment barrier appears to be an efficient and cost-effective means of extinguishing a waste bank fire. A containment barrier composed of 2-part polyurethane - 1-part water grout effectively reduces the permeability and porosity of coal waste material. This mixture's flow characteristics, ease of injection, and cost make it superior to cement-type grouts that have been used to form containment barriers in underground mines. The use of the foam is also cost effective in transferring heat from within a burning waste pile. The use of the grout barrier to contain the foam increases the heat transfer rate from 0.5° C/min to 0.75° C/min.

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# CRYOGENIC SLURRY METHOD TO EXTINGUISH WASTE BANK FIRES[1]

Ann G. Kim and Andrew M. Kociban[2]

**Abstract.** The U. S. Bureau of Mines (USBM) has developed a cryogenic heat transfer method to extinguish waste bank fires, which features an injected slurry of liquid $N_2$ and granular $CO_2$. The heat-absorbing character of the cryogenic slurry is due to its low temperature (-180° C), and to changes in state from the solid or liquid to the gas phase. The phase change also produces a 500-fold increase in volume, creating a cold pressure wave that moves radially from the injection point. The cold wave absorbs heat, produces an inert atmosphere, and forces smoke and fumes from the combustion zone to the surface. Thus, the injected slurry causes a relatively quick cooling of the burning material, while the expansion of the evaporating gas maintains the cool atmosphere for an extended period. The movement of the inert gas is controlled by pressure and buoyancy. In injection and heat transfer tests, the slurry was effective at lowering the temperature of an enclosed bed of burning coal to below -100° C. Low temperatures were maintained for 1 month. The USBM is currently planning a full-scale test to control a waste bank fire with the cryogenic slurry. Preliminary estimates indicate that the cost of injecting a cryogenic slurry is comparable to the cost of more conventional control methods for abandoned mine land (AML) fires. The cryogenic slurry may be also applicable to extinguishing fires in abandoned mines and to other subsurface fires.

## Introduction

Waste bank fires on abandoned mined land negatively affect safety and environmental quality in many communities in coal-mining States. A survey conducted in 1968 (McNay 1971) listed 292 waste bank fires in 13 States, affecting over 1,200 ha. For 1988, the Office of Surface Mining Reclamation and Enforcement (OSM) listed 225 surface fires (table 1) on the AML Inventory (OSMRE 1989). These affected an area of over 405 ha and had an estimated reclamation cost of $42 million. The average estimated cost to control a waste bank fire was over $200,000 in the eastern United States and less than $100,000 in the west.

In the past, mining companies dumped waste in any convenient area within or near the mine property. Subsequent abandonment of the mine and development of nearby real estate have placed many of these waste banks in populated urban or suburban areas. Waste bank fires, often caused by indiscriminate dumping of trash or by spontaneous combustion, are sources of noxious odors and toxic fumes. The subsurface combustion of the waste can also cause surface subsidence. These AML fires degrade the environment, can create human health problems, and can depress the value of nearby property, frequently affecting a much larger area than the waste bank.

At present, there are few effective methods for extinguishing such fires at a reasonable cost. Excavation was used in 80% of waste bank fire control projects between 1970 and 1990 (Kim and Chaiken 1993). The minimum cost per project was less than $10,000, while the maximum was over $1 million. At least 25% of the excavation projects were unsuccessful. Excavation is both expensive and hazardous. It requires men and equipment to work in close proximity to a fire, exposing them to a potentially dangerous environment. Excavation requires moving large amounts of hot material, large areas to spread and quench the material, and an area to dispose of the cooled material. It also exposes hot material to an essentially unlimited supply of oxygen and can propagate the fire. The failure of excavation projects is usually due either to not removing all the burning material or to replacing material that has not completely cooled.

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[1]Paper presented at The International Land Reclamation and Mine Drainage Conference and the Third International Conference on Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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Surface sealing with incombustible material is frequently used as a control technique because of its relatively low cost. Although surface seals suppress evidence of combustion, they do not extinguish fires. The surface seal is intended to exclude oxygen, limiting the rate of combustion. However, waste bank fires usually smolder and can continue to propagate with as little as 2% oxygen. Therefore, the air present in the bank at the time of sealing can support a fire for a relatively long period of time. The seal must be maintained so that no additional oxygen is introduced while the heated material cools below the reignition point. A conservative estimate of this time period is 20 yr. Since sealing does not address the problem of removing heat, unless properly constructed and well maintained, surface seals fail within 1 to 3 yr after construction.

The use of heat removal methods to extinguish waste bank fires requires delivering a heat transfer medium to the combustion zone and then removing the heated medium. The use of an extinguishing agent like water has not been routinely successful because of problems with controlling its distribution (Dalverny 1988, McNay 1971). When water is introduced into a subsurface fire, either by injection or by surface infiltration, it usually does not saturate the combustion zone and is therefore ineffective in controlling the fire.

The movement of a liquid through a nonhomogeneous porous bed (i.e., an abandoned mine or waste bank) is controlled by gravity. A liquid extinguishing agent moves downward through the larger branches of a nonuniform fracture network with little lateral dispersion; erosion makes the branches larger, causing more of the liquid to flow through the same restricted volume. The same constraints apply to surface application of a liquid extinguishing agent. Uniform distribution of a liquid through the combustion volume is the primary problem in using any liquid as a heat transfer agent.

Table 1. Surface fires on abandoned mine lands, 1988.

| State or Tribe | Surface fires | Cost, $K |
|---|---|---|
| Alabama | 14 | 601 |
| Colorado | 9 | 173 |
| Iowa | 1 | 97 |
| Illinois | 2 | 260 |
| Indiana | 3 | 833 |
| Kansas | 2 | 400 |
| Kentucky | 43 | 5,698 |
| Missouri | 6 | 474 |
| Montana | 23 | 2,589 |
| Navajo | 3 | 320 |
| New Mexico | 4 | 686 |
| North Dakota | 1 | NA |
| Ohio | 4 | 280 |
| Pennsylvania | 33 | 11,963 |
| Tennessee | 2 | 5 |
| Utah | 9 | 731 |
| Virginia | 10 | 838 |
| Washington | 1 | 10 |
| West Virginia | 48 | 16,351 |
| Wyoming | 7 | 138 |
| Total | 225 | 42,447 |

NA Not available.

Ideally, a fire control method for waste banks would be remotely emplaced, thus reducing the exposure of men and equipment to potential danger; would be effective in a relatively short period of time; would involve the use of a heat transfer agent that would move equilaterally from a centralized point; and would be at least no more expensive than conventional methods. Water-based liquid extinguishing agents, no matter how they are applied, normally do not satisfy these criteria.

However, a cryogenic liquid or liquefied gas, such as nitrogen or carbon dioxide, is a potentially efficient heat transfer medium. Because of the extremely low temperature of cryogenic liquids, they absorb large quantities of heat. Changes in state from the solid or liquid to the gas phase also absorb heat and produce a

600-fold increase in the volume of the material. The expansion of the gas from the point of injection produces uniform distribution of the heat-absorbing gas, forces hot combustion gases out of the pile, and also displaces oxygen (fig. 1).

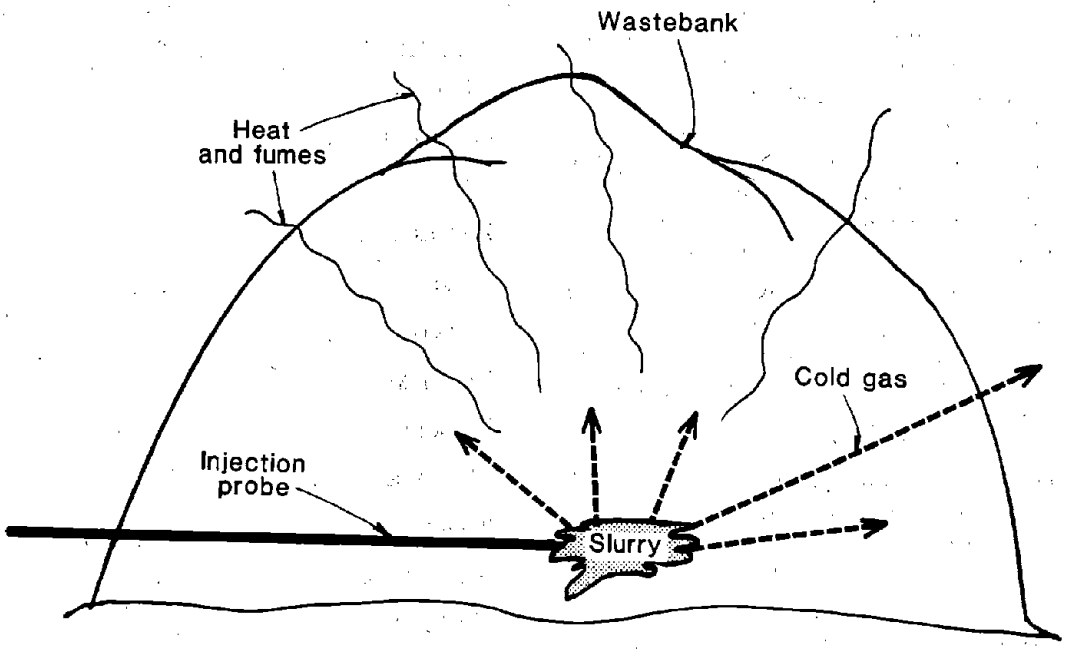


Figure 1. Schematic of heat removal related to cryogenic slurry injection.

Despite its theoretical advantages, the injection of cryogenic liquids as a heat removal technique for controlling subsurface fires has not been investigated. The injection of cryogenic $N_2$ has been used to control fires in active mines, but in such cases, the gas is warmed and vaporized before injection, and the control of combustion is based on oxygen exclusion. In AML fires, the smoldering combustion and the amount of stored heat, as well as the physical characteristics of waste banks or abandoned mines, limit the effectiveness of oxygen exclusion as a fire control method. The injection of a cryogenic liquid or very cold gas, however, has the advantages of heat removal capacity and uniform distribution of the extinguishing agent. Cryogenic injection is also theoretically superior with respect to safety, effectiveness, and efficiency, and it appears to be comparable in cost to other methods of extinguishing AML fires.

## Experimental Work

To determine the limiting parameters and operating conditions for cryogenic injection as a method of extinguishing waste bank fires, the USBM conducted a series of small-, medium- and large-scale tests. In the initial small-scale tests, a 200-L drum was filled with approximately 310 kg of coal waste. The injection point was in the middle of the drum, and thermocouples were positioned at the same level and above and below it.

In the first test, $CO_2$ was obtained from a standard cylinder, in which it existed as a liquid at a temperature of 21° C and a pressure of 58 kg/cm$^2$. When the liquid flowed from the cylinder, evaporative cooling caused the temperature of the $CO_2$ to drop to -78.5° C. As the pressure dropped, the liquid $CO_2$ was converted to a solid. In the small-scale test, the drop in pressure at the injection point caused the liquid $CO_2$ to solidify immediately. The formation of this $CO_2$ snowball blocked the point of injection and limited the distribution of the cold $CO_2$.

Ten tests were conducted using liquid nitrogen, injected at a temperature of -196° C. The initial tests with liquid nitrogen showed better distribution than with $CO_2$, but indicated high heat transfer losses in the delivery lines. With a modified injection nozzle (perforated sides and a closed end) and insulated (standard pipe insulation) lines, the distribution was improved and more rapid cooling was achieved. In all tests, the effect of injection on other thermocouples was observed within minutes and temperatures below 0° C were sustained for approximately 12 hr. For three tests, the refuse material in the barrel was moistened, and the injected liquid nitrogen apparently floated on the interstitial water, enhancing the lateral movement of the nitrogen.

Medium-scale tests of liquid nitrogen injection were conducted in a 4.5 $m^3$ box constructed of steel grate and filled with coal waste. Thermocouples were inserted into the bed of coal waste 0.3, 0.6, and 1 m below the surface. The injection point was at the horizontal and vertical midpoint of the waste bed. These tests indicated that if the refuse was wet, the formation of ice during cryogenic injection could contain and direct the flow of the liquid nitrogen. However, at any point where the refuse was dry, the liquid nitrogen would act like a liquid and flow to the bottom of the box. It was noted that as the liquid nitrogen evaporated, the expanding gas acted like a piston, forcing warm air out of the porous bed.

The USBM's Surface Trench Burn Facility was used for large-scale tests. It is a refractory-lined concrete structure, 2 m by 2 m by 6 m, that can hold approximately 40,000 kg of coal or coal waste. Temperature profiles from an array of 63 thermocouples (fig. 2) were used to characterize the liquid distribution and cooling efficiency.

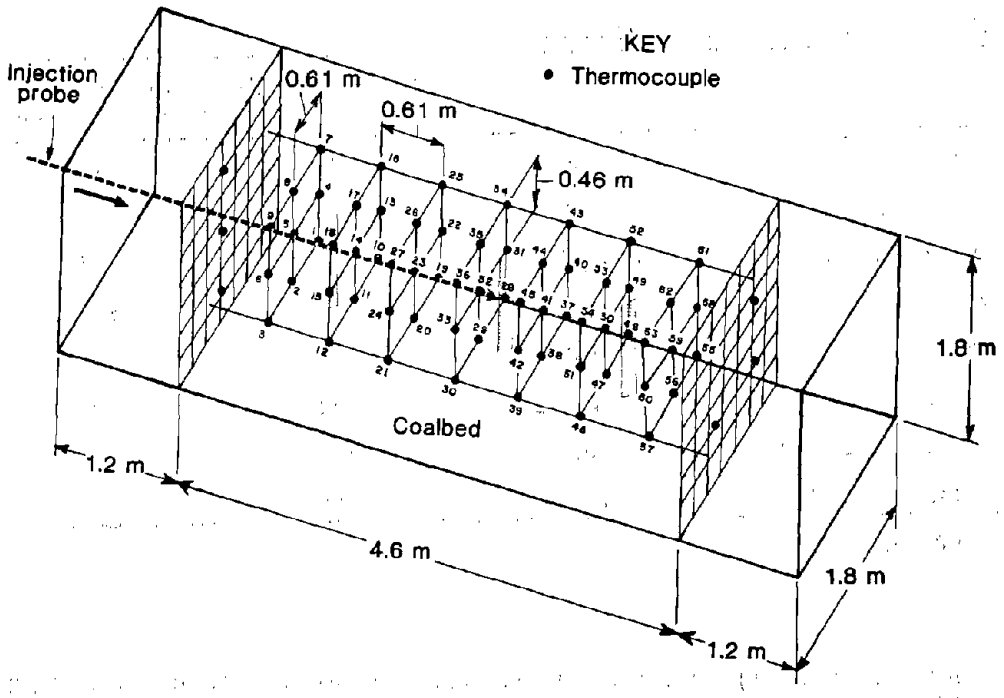


Figure 2. Diagram of thermocouple array in Surface Trench Burn Facility. Coal, 12,700 kg, run-of-mine high-volatile bituminous; volume, 15 $m^3$.

In the first test in the Surface Trench Burn Facility, liquid nitrogen (5,000 kg) was injected into the center of the coal at the rate of 950 L/h. Prior to injection, the maximum temperature of the coal was approximately 100° C (fig. 3). Within a few minutes of the start of injection, the temperature near the injection

point was -68° C. In one-half hour, the temperature of half of the trench was less than 0° C. When injection was completed, approximately one third of the bed was at -170° C. After 12 days, the temperature in the trench ranged between -60° and 5° C. It was approximately 8 weeks before the temperature at all stations in the trench was above 0° C.

To improve the distribution of the heat transfer medium, tests were conducted on producing and injecting a slurry of solid $CO_2$ in liquid nitrogen (table 2). As noted in earlier tests, $CO_2$ exists in a liquid state only when it is under pressure. When the pressure is released, the $CO_2$ solidifies. By producing a granular $CO_2$ snow and combining it with liquid nitrogen, a pumpable slurry is obtained (Chaiken et al. 1991). Deposition of the solid $CO_2$ partially inhibits the downdip movement of the cryogenic liquid.

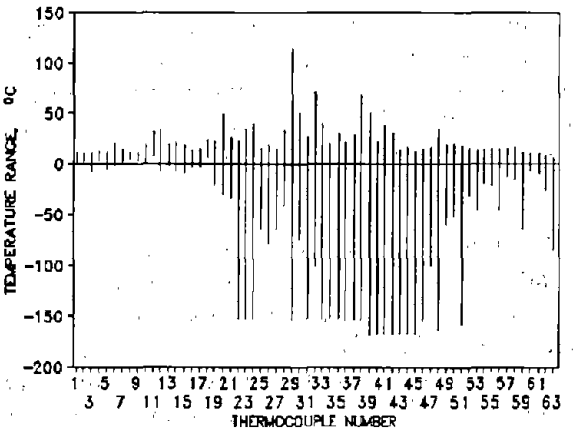


Figure 3. High and low temperatures measured during liquid nitrogen injection in Surface Trench Burn Facility.

Table 2. Physical properties of cryogenic fluids.

| | Nitrogen ($N_2$) | Carbon dioxide ($CO_2$) | Slurry, 50/50 by weight |
|---|---|---|---|
| Molecular weight ..... g/mole.. | 28 | 44 | 36 |
| Boiling point ..... °C.. | -196 | -78 | -180 |
| Heat capacity, solid or liquid ..... cal/g/°C.. | 0.475 | 0.5 | NAp |
| Heat capacity, gas ..... cal/g/°C.. | 0.25 | 0.20 | NAp |
| Heat of vaporization ..... cal/g.. | 48 | 136 | NAp |
| $-dH_i$ ..... cal/g.. | 98 | 190 | 138 |
| Specific volume, liquid or solid ..... $cm^3/g$.. | 1.25 | 0.66 | 1.04 |
| Specific volume, gas ..... $cm^3/g$.. | 800 | 509 | 622 |

NAp Not applicable.

A prototype apparatus to produce the $CO_2$ snow and combine it with liquid nitrogen has been designed and assembled (fig. 4). Liquid or gaseous $CO_2$ is pumped through a bell shaped-nozzle; the rapid expansion forms finely powdered $CO_2$ particles. At the same time, liquid $N_2$ is injected into the nozzle to cool the $CO_2$ to the temperature of the nitrogen, approximately -196° C. The liquid nitrogen and solid $CO_2$ form a slurry in the mixing tank. A vent in the mixing tank releases the pressure caused by the vaporizing gases. The slurry flows by gravity from the mixing tank to a holding tank. Another stream of liquid nitrogen flows through a jet

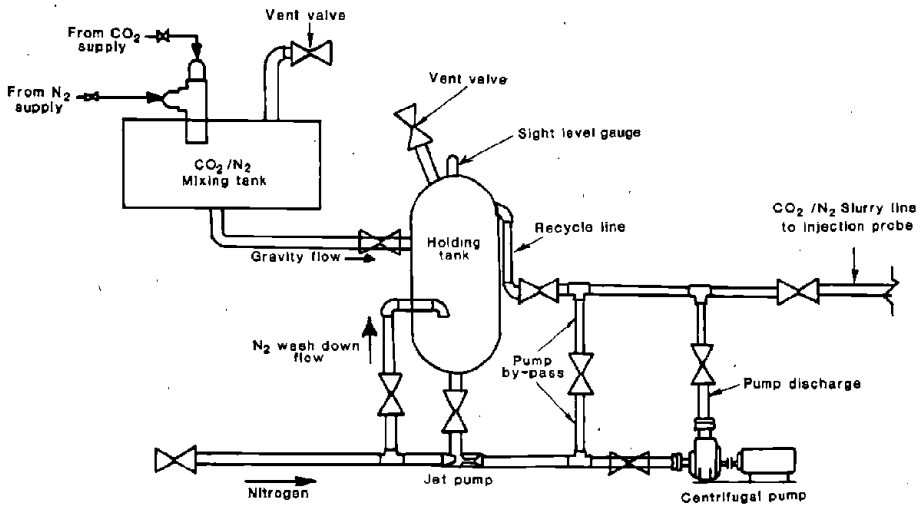


Figure 4. Prototype apparatus for making and injecting cryogenic slurry.

pump, entraining the slurry from the holding tank and carrying it through the injection line[3]. The injection line ends in a pointed injection probe (fig. 5) with 0.24- and 0.64-cm holes around the circumference. When the slurry of liquid $N_2$ and solid $CO_2$ is pumped into a bed of coal or coal waste, the nitrogen vaporizes and the carbon dioxide sublimes. The temperature near the probe drops to -180° C, and the expanding cold gas lowers the temperature in the surrounding volume.

Two tests of slurry injection in the Surface Trench Burn Facility were conducted. In the first test, the apparatus was cooled with liquid nitrogen, then operated in a batchwise, alternating mode. The slurry was produced in one tank, then this portion of the system was isolated and the nitrogen vapor pressure was used to force the slurry through the delivery lines into the bed of coal particles. Six batches, of approximately 100 L each, were injected. At the end of the injection phase, the remaining liquid nitrogen was injected to completely cool the bed of coal. During injection, increasing the $CO_2$ content of the slurry increased the probability that the delivery lines would become plugged. Slight warming and nitrogen pressure were used to clear the lines. Maintaining sufficient liquid $N_2$ in the slurry and eliminating sharp bends in the delivery lines minimized this problem.

During the second injection test at the trench, again using an alternating system of making the slurry in one tank while injecting it from the other, ten 100-L batches of the cryogenic slurry were injected. Fires had been started at several locations in the top third of the bed. Prior to injection, the temperature in parts of the bed exceeded 400° C (fig. 6). One day after injection, temperatures in the bed ranged from 100° to -180° C. One week after injection, 25% of the bed was still below 0° C, and after 1 month, the average temperature was 5° C (fig. 7). The variations in temperature observed at two thermocouples, TC 54 in the top of the bed and TC 56 in the bottom of the bed, indicate the effect of the movement of hot combustion gases over time and

[3]The original design included a cryogenic pump. However, it did not function effectively and was bypassed in all tests.

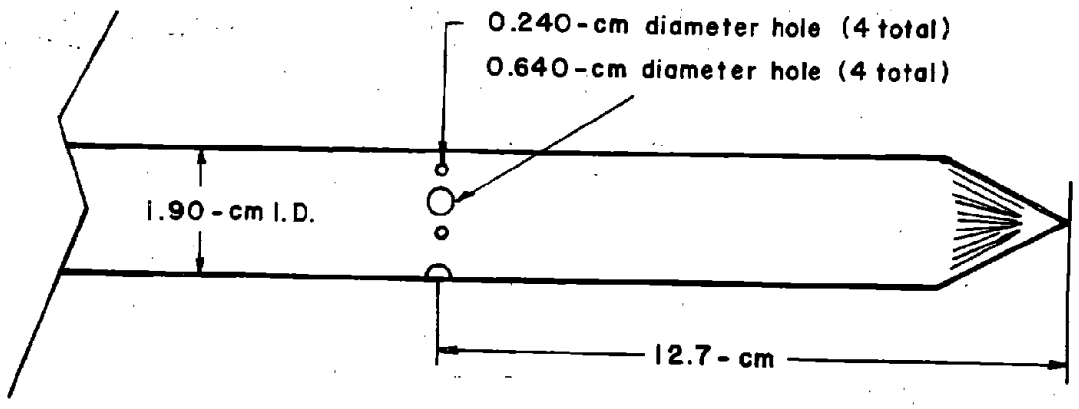


Figure 5. Injection probe for cryogenic injection.

the insulating properties of the coal. Temperatures at both thermocouples were relatively high after 1 week, but at the end of a month they were below the 100° C reignition point.

The tests in the Surface Trench Burn Facility indicated that the injection of cryogenic slurry is an efficient heat transfer method for remote fire control. The expansion of the cold gas produces a cold wave that forces heated air and gaseous combustion products out of a porous bed, displaces $O_2$, and lowers the temperature of the material below the reignition point.

## Potential Applications

To evaluate the application of the technique to potential sites, the amount of cryogenic slurry needed to extinguish fires was estimated for two waste banks and an underground mine. The first consideration was the amount of material and the total heat content. A map or survey of the site was used to estimate its volume and temperature distribution. Based on the total heat capacity ($-dH_t$) of the slurry and the heat content of the site, amounts of cryogenic fluids with a comparable heat content were calculated.

In a typical waste bank fire, less than 5% of the material may be actively burning at any given time. Some of the material will be heated by the fire, and some will be at or near normal temperatures. To extinguish the fire, it is necessary to stop combustion and to remove stored heat. The heat content is a function of the mass of material, its heat capacity, and the temperature distribution.

The first site evaluated was a bituminous coal waste bank. The bank was approximately 70 m long, 30 m across at its widest point, and a maximum of 12 m high (table 3). It was a valley fill with an approximately oval cross section. The waste bank had an estimated volume of 19,600 $m^3$ (700,000 $ft^3$), and contained 35 x $10^6$ kg (38,500 st) of coal refuse. At this bank, although there was visual evidence of extensive combustion, data on the temperature distribution at depth were not available. For purposes of estimating the heat content, it was assumed that the average temperature of material in the bank is 20° C above normal. The heat capacity of bituminous coal refuse is 0.4 cal/g/° C. On this basis, the bank has a heat content of 1.8x $10^8$ kcal. The cryogenic slurry of liquid nitrogen and solid $CO_2$ has an estimated heat removal capacity of 138 kcal/kg. Approximately 1.4 million kg of slurry would be needed to remove the stored heat.

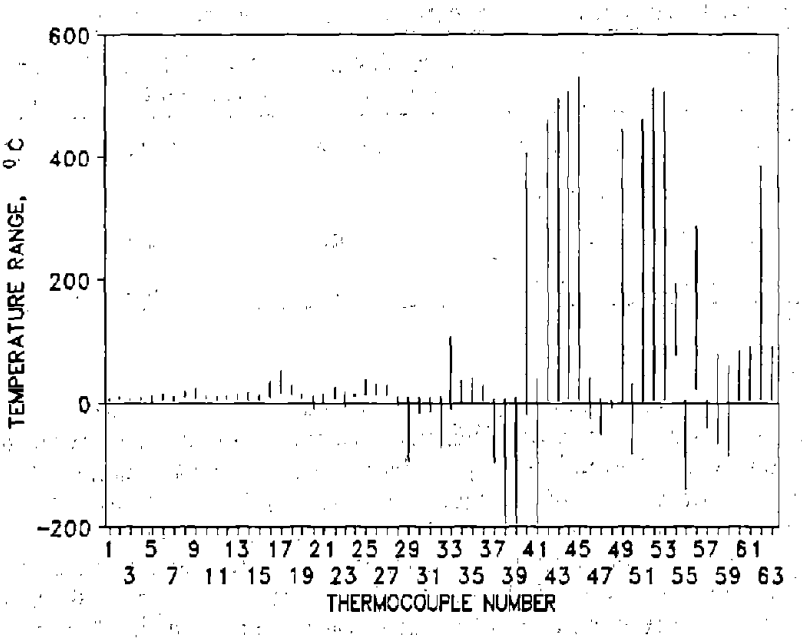


Figure 6. High and low temperatures measured during injection of cryogenic slurry (Test No. 2) in Surface Trench Burn Facility.

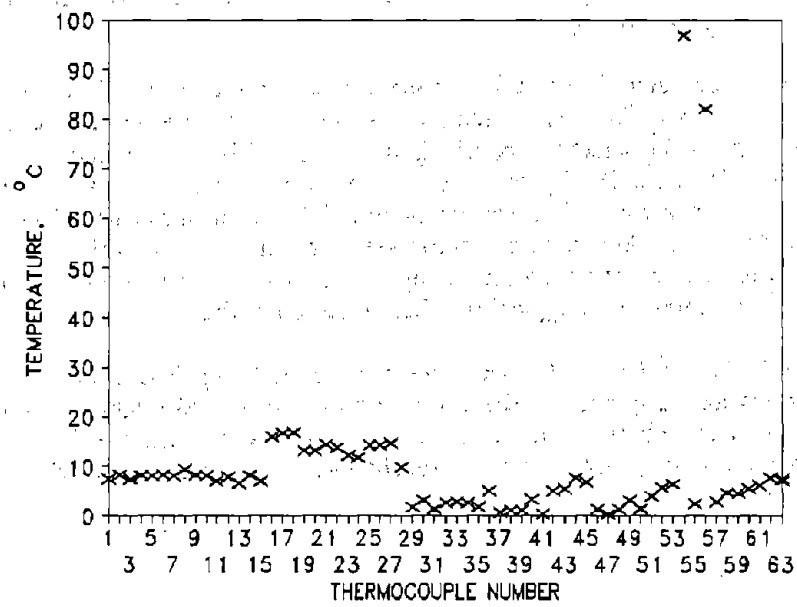


Figure 7. Thermocouple temperatures 1 month after slurry injection (Test No.2) in Surface Trench Burn Facility.

Table 3. Estimated slurry requirements for fire extinguishment

| | Bituminous waste bank | Anthracite waste bank | Underground mine |
|---|---|---|---|
| Volume ..... $m^3$ | 19,600 | 2,250 | 428,000 |
| Mass ..... $10^6$ kg. | 35 | 2.9 | 830 |
| Total heat content . $10^6$ kcal. | 180 | 84 | 1,200 |
| Slurry equivalent .... $10^6$ kg. | 1.4 | 0.6 | 8.7 |

At an anthracite waste bank, a topographic survey and a temperature survey were used to estimate the material requirements for cryogenic injection. This bank was in the form of a truncated cone and had a volume of 2,250 $m^3$. With a density of 1.6 $g/cm^3$ and a porosity of 20%, the bank contains 2.9 x $10^6$ kg of anthracite waste. At this site, a temperature survey was conducted by inserting 15-m sections of 1-cm stainless steel tubing horizontally into the bank. Temperatures, measured as a function of distance from the surface, ranged from 19° to 850° C. Treating the measured temperatures as a normal distribution, it was estimated that 70% of the bank was below 100° C, 17% was between 100° and 300° C, 4% between 300° and 500° C, and less than 9% was above 500° C. These values were integrated to estimate a total heat content of 84 x $10^6$ kcal, equivalent to 600,000 kg of cryogenic slurry.

In the underground mine scenario, both the volume of material in the mine and the overlying rock were considered. Temperature data were available for both materials. Considering 2.4 m of coal and 15 m of overlying shale, the heated volume contained 830 x $10^6$ kg. The temperature distribution indicated that 97% of the coal was at or below 100° C; the remaining 3% had an average temperature between 100° and 300° C. The temperature of the heated rock was 10° to 25° C above normal. The total heat content was 1.2 x $10^9$ kcal, the equivalent of 8.7 x $10^6$ kg of cryogenic slurry.

A cryogenic slurry injection project could be implemented in three phases. The first phase would include the acquisition of data about the site. These data would be used in determining the amount of cryogenic materials needed, in designing the injection manifold, and in establishing an injection schedule. The second phase would be a small-scale test of the injection equipment. The purpose of this test would be to verify the design parameters and to check for unforeseen problems. The third phase would be the full-scale slurry injection to extinguish the fire. Assuming no unforeseen or major delays, this type of project could be completed within 6 months to 1 yr. Although the example given here is hypothetical, the bank dimensions and the projected costs are considered to be typical of AML waste bank fires.

In tests at the USBM, smoke and hot gases were observed being forced from the burning pile. The expansion of the inert gas may displace volatile combustion products and might also produce a low oxygen environment in the immediate vicinity of the waste bank. The low temperature of the expanding gas appears to minimize the combustion odor problem, but appropriate safety equipment should be available in case of deteriorating ambient air quality.

## Conclusion

The work the USBM has performed to date indicates that the use of a cryogenic slurry of liquid nitrogen and solid carbon dioxide may be a significant improvement in the methods of extinguishing AML waste bank fires. The slurry can be produced with relatively simple equipment and injected with a jet pump. The expansion of the cold gas absorbs heat and displaces air and combustion gases. Cryogenic injection should be safer and more effective than conventional methods of controlling AML fires and would appear to be no more expensive to implement.

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# MINE FIRE DIAGNOSTICS AT THE LARGE MINE SITE[1]

A. G. Kim and L.E. Dalverny[2]

**Abstract**: The U.S. Bureau of Mines (USBM) has developed a Mine Fire Diagnostic Methodology to determine the location and extent of combustion zones in abandoned mines, to monitor the progress of extinguishment, and to identify the point at which a fire can be considered extinguished. In the USBM's method, a characteristic fire signature is based on the ratio of higher molecular weight hydrocarbons ($C_2$ to $C_5$) to total hydrocarbons. Initially, samples are obtained at the bottom of boreholes under baseline or static conditions. A second set of samples is obtained under communication, i.e., when a suction fan is used to influence the direction of gas movement at the base of each borehole. Temperature and pressure data are also obtained to define the degree of communication between boreholes. The value of the ratio under communication conditions is taken as a measure of subsurface fire activity related to a particular flow direction. Using a Venn diagram technique, the results are mapped as quadrants on a borehole map of the site. Repetition of the communication tests provides overlapping quadrants that define hot, cold, and indeterminate areas. In a field study at the Large site in Allegheny County, PA, Mine Fire Diagnostic Methodology indicated that a mine fire extended for approximately 152 m parallel to the buried outcrop. The combustion zone also extends into the mine, possibly more than 180 m. Additional, noncontiguous, heated areas were detected in the project area.

## Introduction

Fires in abandoned mines and waste banks are a relatively common occurrence in coal-producing areas (Johnson and Miller 1979, Kim 1993, McNay 1971). Abandoned mine fires present serious health, safety, and environmental hazards due to the emission of toxic fumes, subsidence, and the deterioration in air quality. In most abandoned mine fires, the application of any control method is made more difficult by the inability to accurately locate the fire. Standard indicators such as borehole temperatures or surface venting are rarely indicative of subsurface heated or cold areas. The USBM's Mine Fire Diagnostic Methodology is based on controlled subsurface sampling for low molecular weight hydrocarbons desorbed from heated coal and can be used for remote evaluation of the combustion status of areas accessible only through boreholes. Using the results of the diagnostic method, a cold boundary can be defined. Monitoring during and after extinguishment can be used to determine if the fire is being propagated to other areas of the mine and when the area has cooled below the reignition point.

Under an agreement with the Pennsylvania Department of Environmental Resources (DER), the USBM used its Mine Fire Diagnostic Methodology to determine the location and extent of combustion zones at the Large fire site. Administrative delays, problems in obtaining drilling services, and the unsuspected extent of the fire, made it necessary to perform the diagnostic work in two phases over a 4-yr period (1986-88, 1990).

## Mine Fire Diagnostic Methodology

To determine the location of a subsurface fire, it is necessary to (1) identify a parameter that is characteristic of the fire, (2) measure the parameter through appropriate sampling methods, and (3) have a logical and consistent method of interpreting the results. In the Mine Fire Diagnostic Methodology developed by the USBM (Dalverny and Chaiken 1991, Kim et al. 1992), borehole measurements of temperature, pressure,

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combustion products, and desorbed hydrocarbon gases obtained under natural or baseline conditions are compared to those obtained when a suction fan is used to impose a pressure gradient. It is assumed that mine gases will flow radially from the surrounding underground areas toward the point at which the suction is being applied. Hydrocarbon desorption from coal is temperature dependent; by relating the movement of gases to assumed flow directions, a Venn diagram technique can be used to define heated areas.

A concentration ratio of hydrocarbon gases desorbed from heated coal is the characteristic fire signature in the USBM's diagnostic method. Research by the USBM (Kim 1991) has shown that the desorption of low-molecular-weight hydrocarbons, methane ($CH_4$) to pentane ($C_5H_{12}$), is strongly temperature dependent, and that changes in such gas emissions are observable at temperatures as low as 100° C. For most coals and coal wastes, as the temperature increases, the concentration of heavier hydrocarbons in the desorbed gas also increases. A relative index, R1, was defined as

$$R1 = \frac{1.01\,[THC] - [CH_4]}{[THC] + c} \times 1{,}000,$$

where [THC] = concentration of total hydrocarbons, ppm
$[CH_4]$ = concentration of methane, ppm
c = constant, 0.01 ppm.

Based on laboratory tests of bituminous coals, R1 can be empirically related to coal temperature:

| R1 | Interpretation |
|---|---|
| 0 | No hydrocarbons detected |
| 10 | Only methane detected |
| 11 to 50 | Normal temperature |
| 51 to 100 | Possible heating |
| > 100 | Heated coal. |

The use of hydrocarbon emission as a diagnostic indicator has several advantages: (1) Changes in the rate of hydrocarbon emission are caused by changes in temperature, (2) changes in hydrocarbon emission can be detected for temperatures far below the "ignition" temperature for coal, thus increasing the sensitivity of the method, (3) hydrocarbons can be routinely detected by gas chromatographic analysis at concentrations as low as 1 ppm, and (4) the use of a ratio of gas concentrations can minimize the effect of dilution. R1 can also be used for time-dependent monitoring of relative changes in temperature, such as during remote extinguishment, after extinguishment to obtain early warning of reignition, and after surface sealing.

To determine changes in the combustion signature, access to the coal horizon and a method for remotely obtaining uncontaminated samples are required. In the Mine Fire Diagnostic Methodology, boreholes are drilled to the base of the coal bearing strata and cased to within 1 m of the coal mine roof. A thermocouple and sampling line extend to near the bottom of the borehole from a borehole instrumentation cap. A diaphragm pump and evacuated sample tube are used to sample the mine gases.

Two types of data are obtained during a fire project. Baseline measurements (temperature, pressure, and gas composition) are made under natural conditions. Communications measurements are made while suction is being applied to the underground environment. Differences in pressure, between baseline and communication values, indicate the extent to which the suction fan affects the mine at any sampling point. Positive or negative changes in the gas composition, as indicated by R1 values, are related to heated or cold areas.

The data analysis involves several steps. A change in the pressure at the bottom of the boreholes of at least 5 Pa (.02 in-$H_2O$) indicates communication between the boreholes and the suction hole, and defines the area for which the data are valid. A 90° shaded quadrant having a radius of 15 m is drawn on a scaled site map at each borehole. The quadrant apex is centered at the borehole, and the quadrant appears to be pointing towards the suction borehole. The quadrant shading based on the R1 value represents areas that are cold, warm, or hot. Areas for which there are no data are also defined. A composite of all test results is constructed. Combining the results of all tests effectively bounds the possible fire zones through successive approximation. The method defines probable combustion zones and also indicates the location of the cold boundary, an important consideration in implementing an extinguishment plan.

## Site Description

The Large mine fire site is located in Allegheny County, PA. The fire is in the Pittsburgh coal seam, which at this location outcrops at the base of a hill. Approximate depth of overburden ranges from 6 to 55 m. The surface area affected by the mine fire is approximately 1 hectare (fig. 1). The slope averages 20°. Five gas pipelines and three sets of high voltage power lines cross the property.

The fire is located in what is believed to have been the Walden Pool #2 Mine. Active underground mining probably ceased in the 1920's. Between 1953 and 1964, the Pittsburgh and Redstone coal seams were partially strip mined. The property was restored to the then-current standard of the law, and the mine operator was released of liability for the mine site.

The fire apparently started at an unknown date on the stripped area and was first reported in December 1976. At that time, the fire could be seen burning along the highwall and in an entry. Apparently, some of the burning material was removed and the entry was sealed. This assumption is based on the fact that the entry is no longer visible and a small pile of burning material was located near the old mine as recently as 1986.

The elevation of the Pittsburgh Coalbed at the Large site is approximately 253 m above mean sea level. The Redstone Coalbed occurs approximately 20 m above the Pittsburgh Coalbed. Shales, sandstone, and limestone occur between the Redstone and Pittsburgh Coalbeds.

Over a 10-yr period, seven 6.35-cm (2.5-in) and sixty-four 20-cm (8-in) boreholes (BH) and four 5-cm (2-in) coreholes were drilled at the Large site. Fifty of the 20 cm holes, seven of the 6.35 cm holes, and two coreholes were used as monitoring points for the mine fire diagnostic tests. The other boreholes were not used for diagnostic testing because they terminated in solid coal and had been backfilled, because they contained more than 0.6 m of standing water, or because they had sustained well bore damage during drilling. Seven of the boreholes were terminated below the Redstone Coal instead of the Pittsburgh Coalbeds. These were retained as temperature-monitoring points but were not used for diagnostic testing.

## Diagnostic Results

Baseline values for temperature, static differential pressure, and gas composition were determined on a regular basis at the Large site. The sampling point was approximately 1 m below the casing and 1.5 m above the bottom of the borehole. During the first diagnostic phase, the average baseline temperature for three boreholes was over 100° C (fig. 2); most of the holes had an average baseline temperature between 30° and 100°C. Ten of the holes had an average baseline temperature of less than 20°C. During the second diagnostic phase, the average baseline temperature at borehole 43 decreased from over 350° C to less than 100° C. Average baseline temperatures at other boreholes exhibited little variation; temperatures at the holes drilled near the perimeter of the work area were generally in the 30° to 50° C range. Temperatures measured in the Redstone Coal, 18 to 27 m above the mine, were also above normal subsurface values, in the 20° to 40° C range. The normal subsurface temperature in the mine would be expected to be in the 11° to 15° C range.

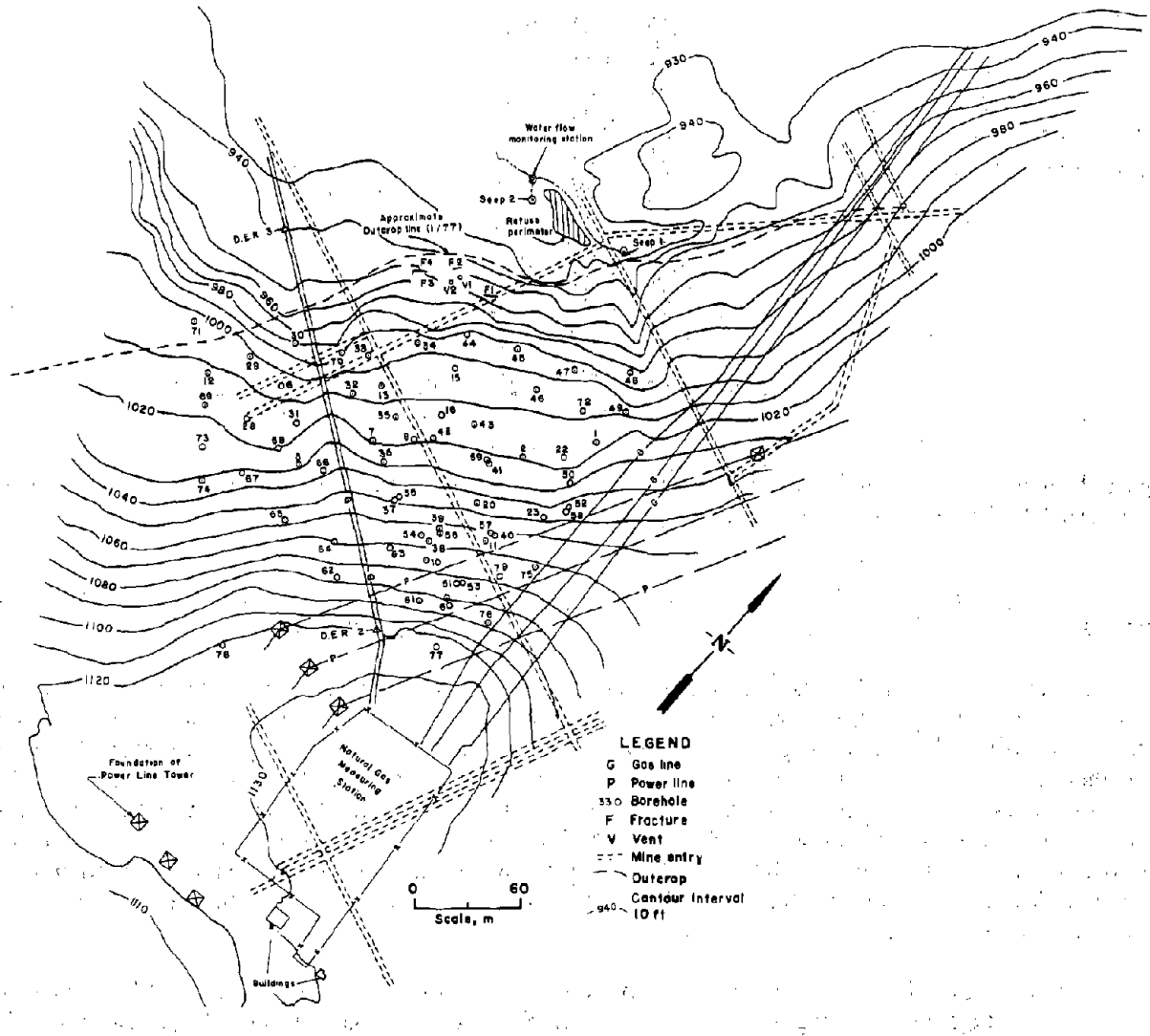


Figure 1. Plan view of the Large site with locations of boreholes, gas pipeline rights of way, power lines, and site boundaries.

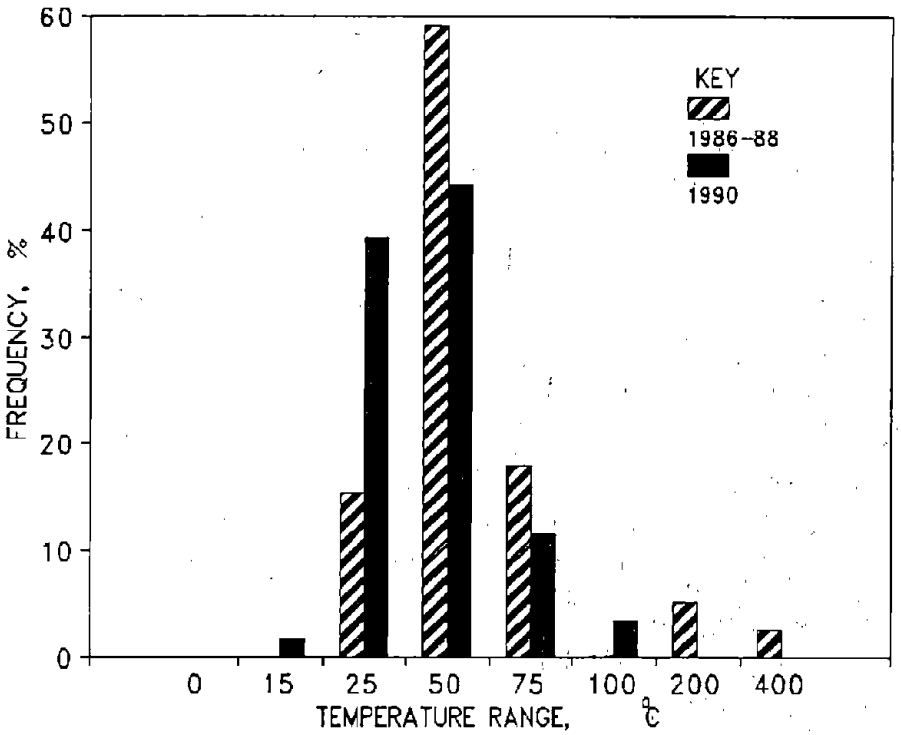


Figure 2. Relative frequency distribution of baseline temperatures at Large site for all boreholes during phase 1 (1986-88) and phase 2 (1990) of diagnostic testing.

Since the temperature at only one borehole was what would be considered a normal value, the baseline temperatures indicated that large areas of the mine were being heated by the fire.

The static differential pressure (the difference between the ambient surface barometric pressure and the measured pressure in the mine) was also determined. A differential pressure of 0 indicates that the mine is at the same pressure as the atmosphere. In the absence of a fire, the differential pressure should be uniform throughout the mine. An average baseline differential pressure greater than 0.05 cm $H_2O$ was measured in 10 of the boreholes. Elevated pressures may be due to mass addition of combustion products to the airstream and to thermal expansion of the hot gases. Eleven of the boreholes had average differential pressures between 0 and -0.20 cm $H_2O$. Areas that exhibit negative differential pressures may indicate that air is being drawn into the mine. Variations in differential pressure may indicate that the mine fire is causing the formation of a convection cell within the mine.

The mean baseline hydrocarbon diagnostic ratio (R1) was greater than 100 at eight boreholes during the first diagnostic phase. During the second diagnostic phase, the average baseline R1 value was over 100 at 22 boreholes, including 4 of the holes drilled near the perimeter of the site (fig. 3). The baseline diagnostic ratio was between 50 and 100 at five boreholes and less than 50 at two boreholes. The total hydrocarbon concentration was less than 20 ppm at the remainder of the boreholes.

For the majority of boreholes, the mean baseline oxygen ($O_2$) concentration was above 12% (fig. 4), a relatively high value for an abandoned mine. The average carbon dioxide ($CO_2$) concentration was inversely correlated with the $O_2$ concentration (fig. 5) Only 10 of the boreholes had average baseline carbon monoxide (CO) concentrations greater than 0. The baseline gas composition indicated that combustion was occurring in a relatively oxygen-rich environment.

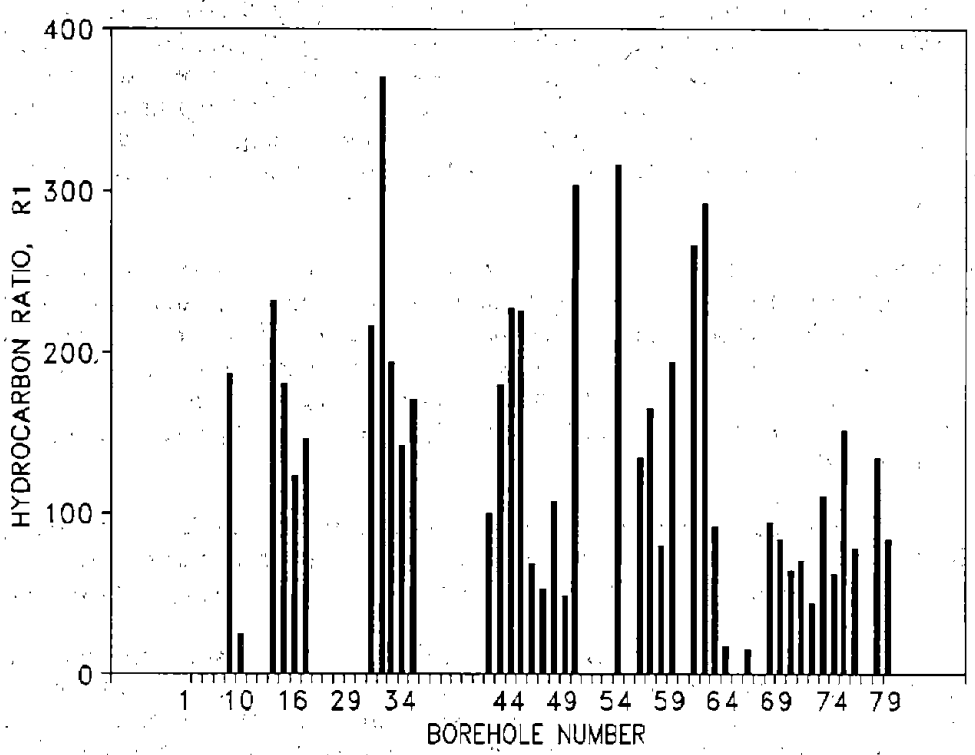


Figure 3. Baseline hydrocarbon ratio (R1) at Large site. Boreholes without values had total hydrocarbon concentration < 20ppm.

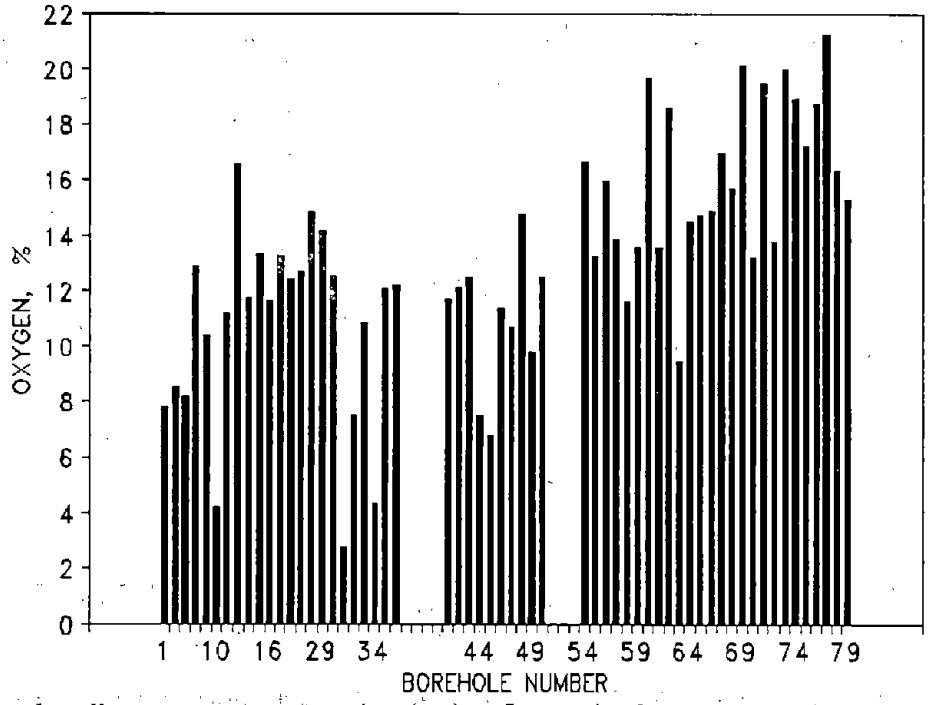


Figure 4. Mean baseline oxygen concentration (pct) at Large site for all boreholes to Pittsburgh coalbed.

Over a 4-yr period, 61 communication tests were performed at the Large site. The earliest communication tests detected a heated zone around BH 15 and 16 and BH 6 and 20. Cold areas were detected at BH 8, 10, 11, 12, and 13. At BH 1, 2, 7, and 23, the concentration of hydrocarbons was too low to determine a valid signature. From the data available at that time, it was inferred that the primary combustion zone was located near the outcrop, with possible heated areas in the interior of the mine and to the west of the northwest gas transmission lines. It was presumed that a cold boundary existed from BH 12 eastward to BH 10 and 11 and northward to BH 2.

In the next year, 25 communication tests located an extended combustion zone along the outcrop and in the interior of the mine beyond BH 20. Signature values obtained from boreholes to the west of the northwest gas transmission lines indicated an isolated heated zone around BH 6 and a cold boundary extending from BH 12 through BH 28. Although the cold boundary values were again obtained at BH 10 and 11 and at BH 52, there was an indication that the combustion zone was extending toward this area. Five communication tests in the third year confirmed the earlier tests. The only significant change was the apparent decrease in combustion activity around BH 6.

In phase 2 the results of 23 communication tests indicated a more extensive combustion zone along the outcrop. The heated area previously detected around BH 20 was determined to extend to BH 76. Heated areas were also detected beyond what had been presumed to be a cold boundary from BH 12 to BH 10.

Based on the results of all communications tests at the Large site, it was apparent that a combustion zone extended parallel to the buried outcrop for approximately 152 m (fig. 6). This zone also apparently extended 182 m into the mine, possibly along a set of main entries. There is evidence of heating to the left and right of the combustion zone that is perpendicular to the outcrop. Areas of low hydrocarbon concentrations exist to the right and left of the combustion areas. In the area around BH 47, combustion signatures were detected where the concentration of hydrocarbons in previous samples had been below the detection limit.

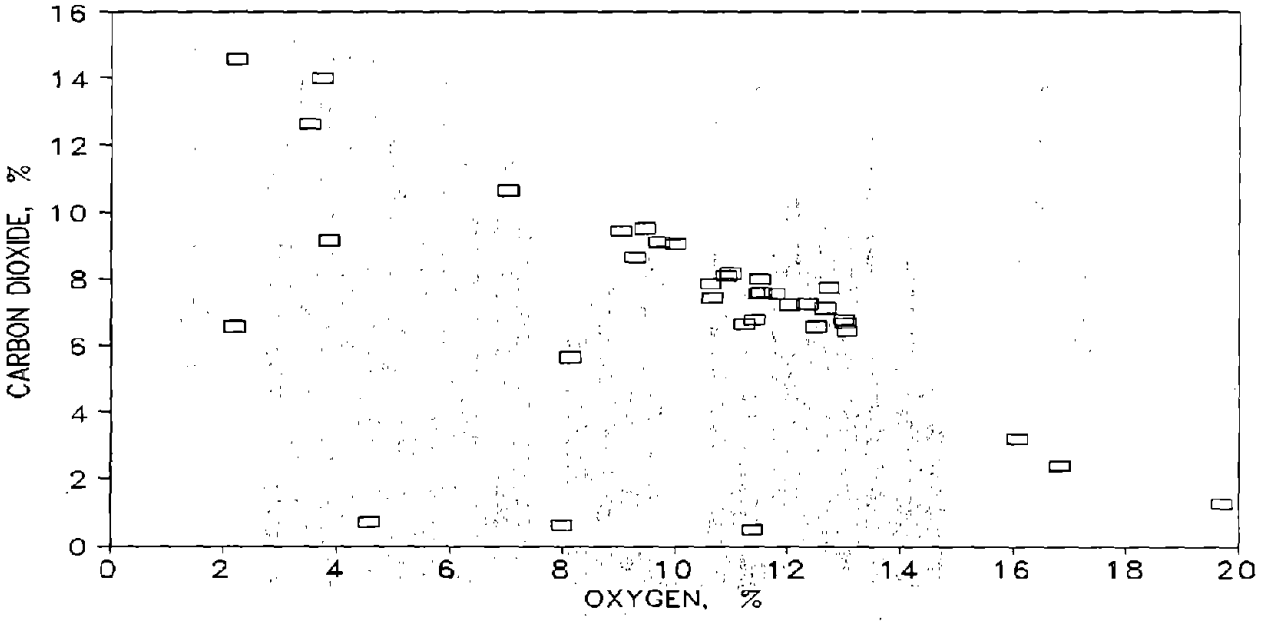


Figure 5. Variation in mean carbon dioxide concentration with mean oxygen concentration (volume pct) for baseline testing at the Large site.

The application of the USBM's Mine Fire Diagnostic Methodology at the Large mine fire indicates that the fire has propagated at least 152 m parallel to the outcrop. All surface venting occurs between the buried outcrop and the boreholes used to locate the fire. The fire also extends 180 m or more into the mine, possibly along a set of main entries. It extends beyond the area that can be sampled through the existing boreholes. Combustion in the Large Mine is occurring in a relatively oxygen-rich environment. At the majority of boreholes, the average oxygen concentration is at or above 8%. In an abandoned mine, bacterial activity and other ambient temperature processes normally reduce oxygen levels below this value. The relatively high oxygen concentration indicates that there is a source of fresh air into the mine.

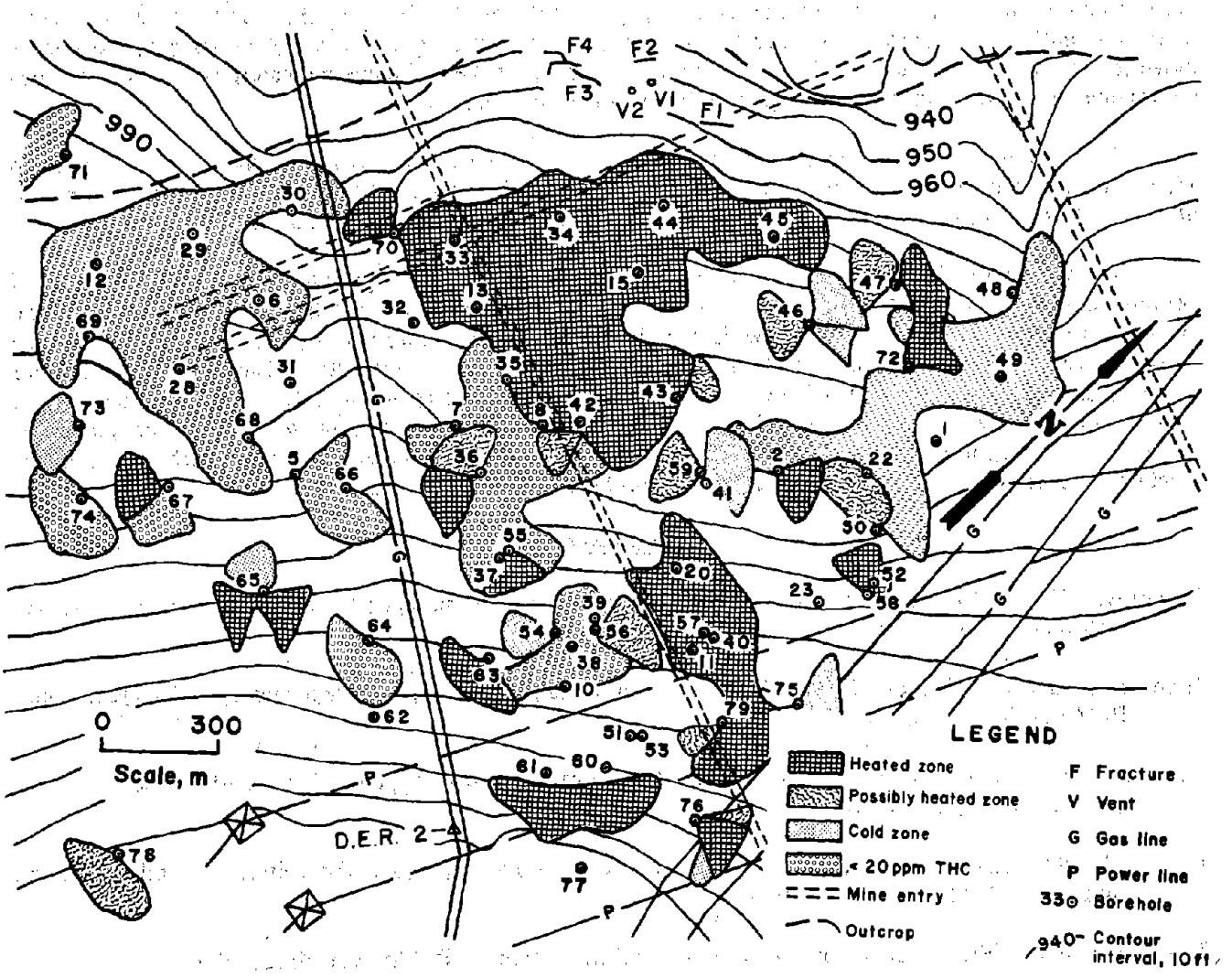


Figure 6. Map of Large Mine Fire Site showing heated and cold zones, with surface contours and approximate locations of mine entries.

## Summary

The USBM has used its Mine Fire Diagnostic Methodology to determine the subsurface location of a fire in an abandoned mine. Areas of venting were confined to a relatively small area and indicated propagation along the burned outcrop. Temperature monitoring indicated that large areas of the mine were heated but did not define a combustion zone. As defined by the USBM's Mine Fire Diagnostic Methodology, the mine fire at Large extends for approximately 150 m parallel to the buried outcrop. The combustion zone also extends into the mine, by 180 m or more. This in-mine area may coincide with a set of main entries. Noncontiguous heated ares were detected in the southwest quadrant of the project area. Diagnostic testing indicates that the heated zone extends beyond the boundary of the current work area.

The apparent direction of propagation into the mine appears to coincide with the location of a set of main entries. Heated areas have been detected to the north and south of this area. By overlaying the surface map on an old mine map, it appears that the fire may be in the entries that extend beyond BH 79. This entry intersects another set of entries that extend north and south under the adjacent road. Subsurface fires generally move toward a source of air, which may be either a portal that the old maps indicated existed in the valley approximately 1 mile from the site or permeable zones in the outcrop.

A cold boundary may be presumed to exist in the area between BH 30 and 69 and in the area of BH's 48, 49, and 72. A cold boundary has not been located for the inby portion of the mine. Determining the limit of the combustion zone would require extending the borehole array and additional diagnostic testing.

The use of the USBM's Mine Fire Diagnostic Methodology at Large indicated that it is capable of defining subsurface heated and cold zones. The work at Large also indicated that the method requires an extended array of 20 cm cased boreholes. In the absence of natural features to bound the fire zone, locating a fire may require several rounds of drilling to determine the extent of combustion. The method also is labor intensive, and repeated iterations of the communication tests are time consuming. The use of more advanced instrumentation may reduce the time and labor involved in communication testing. A more efficient diagnostic system could incorporate the preliminary use of a remote noninvasive technique to indicate the extent of the combustion area, followed by the more accurate borehole diagnostic testing.

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# TEMPERATURE-INDUCED CHANGES IN MAGNETIC SUSCEPTIBILITY OF PITTSBURGH COAL SEAM STRATA[1]

Louis E. Dalverny[2]

**Abstract:** U.S. Bureau of Mines researchers are investigating geophysical methods for the remote detection of fires in abandoned coal mines and waste banks. All phases of fire control projects need inexpensive, noninvasive techniques for rapid and reliable evaluation of subsurface combustion location and extent. Heat from combustion causes mineral phase changes, altering the remanent magnetization and magnetic susceptibility of iron-containing minerals in carbonaceous strata. Paramagnetic pyrite can oxidize to strongly magnetic magnetite or to other oxides of iron with smaller positive magnetic susceptibilities. The minerals become more detectable as magnetic properties increase. The Bureau's laboratory project determined correlations between temperature and magnetic susceptibility for samples of coal mine roof strata from core drillings in the vicinity of a burning portion of an abandoned Pittsburgh seam mine in southwestern Pennsylvania. The magnetic susceptibility of each sample was measured before and after each trial at a furnace setpoint temperature (150°, 300°, 440°, 600°, 660°, 825°, or 980° C). Fifteen 20-to-25-g samples (triplicates of five carbonaceous materials) were placed in ceramic combustion boats; the heating rate was 60° C/h with a 3-h hold at the setpoint value. Oxygen concentration also affects changes in magnetic susceptibility. The furnace atmosphere source gas consisted of about 0%, 1%, or 10% oxygen in nitrogen, flowing at 1.4 to 2.4 L/min. Mossbauer spectroscopy and x-ray diffraction spectrometry identified iron compounds in the samples. Changes in magnetic susceptibility occurred as functions of both temperature and oxygen concentration.

## Introduction

Researchers of the U.S. Bureau of Mines have been investigating methods for remote detection of underground fires in abandoned coal mines and waste banks. Residential, business, or recreational structures and their occupants can be adversely affected by the fumes or subsidence associated with the hidden combustion. Determining the location and extent of subsurface combustion is critical to minimizing the time, cost, and surface disruption of fire control and/or extinguishment projects. This assessment usually is done by evaluators standing on the surface, as much as several hundred feet above the combustion activity. Often, the extent of burning is estimated by observing surface venting locations and looking at available mining maps. There is a growing awareness of the need to apply sensitive and cost-effective diagnostics to determine accurately the extent and level of combustion in an inaccessible mine before control measures are initiated. Inexpensive, noninvasive techniques are also needed for use during and after control actions. Altogether, there is a need for more diagnostic alternatives, applicable to different situations, that will obtain sufficient information for choosing appropriate control actions while minimizing time and cost. Investigators have been trying various remote detection techniques to locate subterranean anomalies, including combustion, and enable them to map these phenomena. They have used electrical conductivity, electrical resistivity, ground penetrating radar, aerial photography (visible and infrared), seismometry, and boreholes (for corings, temperature measurements, and/or gas sampling). Magnetometry is one of the geophysical techniques being evaluated.

Magnetometry has had limited use as a remote detection method for establishing locations of underground fire zones. Magnetic anomalies have been associated with combustion metamorphism of oil-bearing sediments and burning or burnt Western U.S. coal-associated strata (Castleberry 1979, Cisowski and

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Fuller 1987, Geissman et al. 1983, Hooper 1987). Heat from combustion causes mineral phase changes that alter iron-containing materials in the overlying strata, affecting their remanent (permanent) magnetization intensities and magnetic susceptibilities.[3] Heating of these materials can increase their magnetic properties, thereby causing them to become more detectable. Therefore, the effects of underground combustion should be detected as changes in the intensity of magnetization measured above ground. One procedure for detecting subterranean magnetic materials uses the portable proton precession magnetometer (Breiner 1973). A limited magnetometer survey study of a burning abandoned underground coal mine site in Pennsylvania was done in 1988 and partly repeated in 1989 (Schueck 1990). Visual observations along the outcrop areas that had burned and cooled were correlated with positive magnetic anomalies (i.e., local increases in magnetization). These deductions and detected changes in magnetic intensity in the (inaccessible) mine workings permitted mapping of magnetic intensities relative to background. The map provided some idea of the progress of the fire zones after a year. Some correlations of magnetic properties of pyrite in coal with temperature and air partial pressure changes have been made (Thorpe et al. 1984). Correlations of magnetic properties of coal mine roof strata with temperature and oxygen concentration are not available in the literature. This paper describes a laboratory project to determine the effect of temperature on the magnetic susceptibility of strata components associated with the Pittsburgh coal seam. The results of this work should contribute to improving remote detection and delineation of underground abandoned coal mine fire zones using magnetic methods.

## Experimental Approach

Laboratory studies were conducted on coal and roof shales from core drillings obtained in the vicinity of a burning portion of an abandoned underground Pittsburgh coal seam mine in southwestern Pennsylvania. These strata contain iron compounds that when heated change chemically with consequent alterations of their remanent magnetization. The primary independent variables are considered to be temperature and oxygen concentration because mine fires can smoulder at very low oxygen levels and exhibit local temperature increases owing to the low thermal conductivity of rock strata.

Portions of the core materials were pulverized so all pieces passed through a 2.36-mm dry sieve mesh. Each portion was divided into a minimum of nine representative samples; the remainder was stored in a glass jar. Each sample was weighed prior to the susceptibility measurement. The approximately 20-to-25-g, 15.5-$cm^3$ samples were put into cylindrical plastic containers that fit into the measuring coil of the magnetic susceptibility meter (Model SI-2, Sapphire Instruments, Canada).[4] The meter was controlled with a personal computer that recorded the magnetic susceptibility values of the samples in each test (fig. 1). The magnetic susceptibility of each sample was measured before and after heating in a tubular combustion furnace (fig. 2). Furnace setpoint temperatures were 150°, 300°, 440°, 600°, 660°, 825°, and 980° C. The heating rate was 60° C/h with a 3-h hold at the setpoint value. Each sample was poured into a ceramic sample boat (5.5 by 8.3 by 1 cm) able to withstand temperatures to 1,000° C. Five triplicate samples were randomly placed in the furnace at the beginning of a test series. The placement was retained for that series. After 3 h at the setpoint temperature in each test, the furnace's power was turned off and the equipment was allowed to cool prior to removal of the samples. A thermocouple at the center of the furnace was located between firebricks that provided horizontal support for the sample dishes. The brick temperature was expected to be closer in value to the sample temperature than was the gas temperature measured above the samples. The brick temperature attained equilibrium more slowly than did the setpoint and gas temperatures; this was the primary reason for the 3-h hold at the setpoint temperature. While both brick and gas final temperatures were less than the setpoint value,

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[3] The equivalent terms remanent magnetization and permanent magnetization are applied to rocks and to metals, respectively (Breiner 1973). Magnetic susceptibility is the measure of how easily a material can be magnetized. It is a ratio of the magnetic moment per unit volume to the product of applied magnetic field and the density of a substance (Senftle and Thorpe 1987).

[4] Reference to specific products does not imply endorsement by the U.S. Bureau of Mines.

the brick value was at least 95% of the setpoint value from 440° to 980° C.

Furnace atmosphere was varied to determine any correlation between oxygen concentration and magnetic susceptibility of the heated carbonaceous materials. Approximately 0%, 1%, or 10% oxygen ($O_2$) in nitrogen ($N_2$) source gas was used in each test series. The gas flowed through the furnace at a rate of 1.4 to 2.4 L/min. An activated carbon filter installed at the vent end of the furnace adsorbed organic vapors from the heated materials. A particulate filter was placed between the vapor filter and the outlet flowmeter. Representative portions of the unheated samples were analyzed for their moisture, ash, volatile matter, fixed carbon, total iron, and forms of sulfur concentrations. Solids analyses using Mossbauer and x-ray diffraction spectrometry determined initial iron mineralogy and phase changes associated with heating.

Figure 1. Magnetic susceptibility meter and personal computer system with sample container.

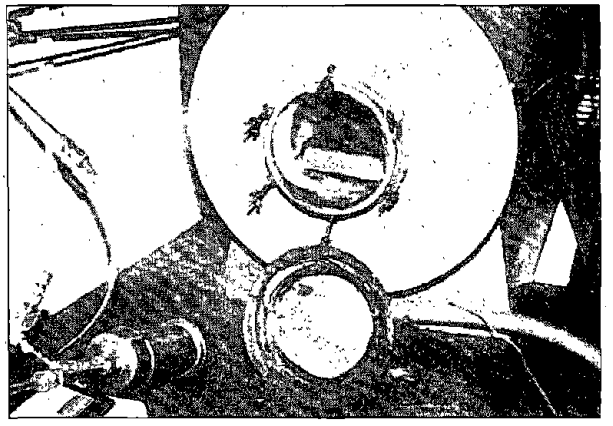

Figure 2. Tubular combustion furnace.

Two initial sets of heating tests with $N_2$ used furnace temperatures of 150°, 600°, and 980° C. For these samples, the data indicated significant increases in magnetic susceptibility at 600° C and subsequent decreases to near original values at 980° C. Four more temperature setpoints (300°, 440°, 660°, and 825° C) were added to complete the test protocol. Literature values for expected iron-containing compounds were considered when choosing the seven setpoint values. The susceptibility values at 150° C in the initial two trials had not changed from the ambient value. The first setpoint in the third set (also using $N_2$) was 300° C, whereas all subsequent sets began at 150° C. The third through fifth trials were run as identically as possible, each set with a different $O_2$ concentration. The following discussion and comparisons pertain to the three sets with the added setpoints and furnace source gas $O_2$ concentrations of about 0%, 1%, and 10%, respectively.

## Discussion of Experimental Results

Gas sample analyses indicated that the lowest oxygen concentration attained at the furnace vent was 0.1% in tests conducted in nitrogen. It is possible that the "residual" oxygen desorbed from the firebricks used as props in the ceramic furnace tube or from the tube itself, or was generated during decarboxylation reactions in the heated carbonaceous matter.

Figure 3 depicts how the magnetic susceptibility of the material heated in $N_2$ (averaged over all the samples in the set) varied with temperature. These data were taken as the baseline for evaluating the effect of oxygen concentration, although there may have been as much as 1,000 ppm of $O_2$ in the furnace gas during heating. While there was a small positive change in magnetic susceptibility from the ambient temperature value to that at 300° C, there was an increase of greater than one order of magnitude when the temperature was raised to 440° C. The decrease in magnetic susceptibility as temperature increased to 825° C implied that

magnetically weaker mineral phases were being formed. The magnetic susceptibility increase at 980° C to between the average values at 600° and 660° C is interesting because it indicates that magnetically stronger phases were again formed.

Both temperature- and oxygen-related changes in magnetic susceptibility are shown in figure 4. There were significant increases, from the values at the next lower setpoints in each set, at both 440° and 980° C for both 1% and 10% $O_2$ sets, as noted for the $N_2$ atmosphere data. Except at the 440° C setpoint, the 1% $O_2$ data values were greater than those of either of the other two sets. It can be inferred that at the 10% $O_2$ concentration, the oxides formed were magnetically weaker than those formed in the 1% $O_2$ atmosphere. These results may be partly explained by comparing the quantities of iron (Fe) and $O_2$ available during a heating trial. The material in each of the sample boats (18 to 27 g) contained a low percentage of total iron (3% to 14% in the original samples) for a maximum of 1.02 mol Fe per test. The flow of gas (1.4 to 2.4 L/min) through the furnace provided a (minimum) total of 0.11 mol $O_2$ in 1% $O_2$ and 1.1 mol $O_2$ in the 10% $O_2$ tests.

A different way of viewing the data to determine how the magnetic susceptibility changes could affect field measurements is depicted in figure 5. The average ratios of magnetic susceptibility values (after heating to unheated) were plotted versus the various setpoint temperatures for each set of oxygen concentrations. The maximum ratio, at 600° C and 1% $O_2$, was about 128 times the unheated magnetic susceptibility. For individual sample ratios between 440° and 980° C at 1% $O_2$, the smallest ratio was about 15 (at 440° and 825° C) and the largest was about 174 (at 600° C).

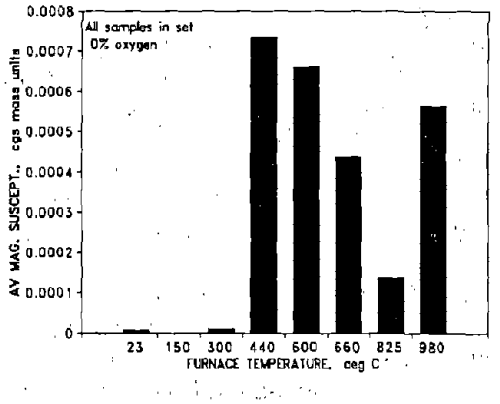


Figure 3. Average magnetic susceptibility (15 samples) versus temperature in nitrogen atmosphere.

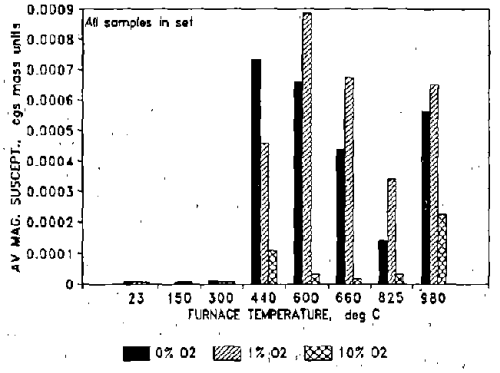


Figure 4. Average magnetic susceptibility (15 samples per set) versus temperature for three oxygen concentrations.

## Summary

Laboratory experiments show that there is a thermal effect on the magnetic susceptibility of Fe-containing minerals in carbonaceous strata overlying the Pittsburgh coal seam. Oxygen concentration during heating also affects the magnetic susceptibility. Magnetically stronger mineral phases apparently formed at about 440° C and again when the materials were heated to about 980° C; ongoing solids analyses should define the phases produced at the several temperature and oxygen conditions. The experimental data indicate that the increase in the magnetic susceptibility after heating of the carbonaceous materials is less at 10% $O_2$ than at 1% $O_2$, but the increase over the value at ambient temperature is significant in either case. The inference is drawn that sensitive field measurements of another magnetic property of iron-containing materials-the

magnetic intensity-may allow remote detection for detailed mapping of both burning and burnt coal. The Earth's magnetic field in an area is sufficiently constant that sensitive magnetometers could differentiate among the magnetic intensity levels it induces in unburnt, burning, and burnt coal seam strata.

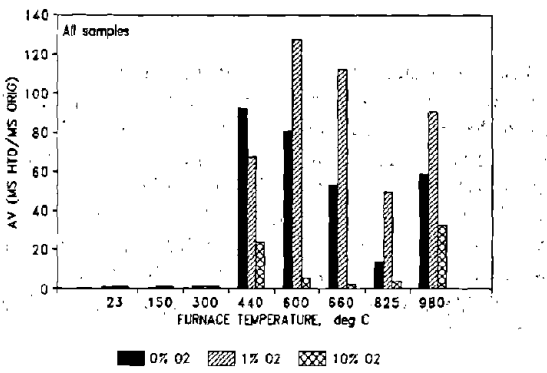


Figure 5. Average ratio of magnetic susceptibilities: heated/original versus temperature for all samples in each set.

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# PHYSICAL PROPERTIES OF KENTUCKY'S AML LANDSLIDES: CASE STUDIES ANALYZED[1]

Anthony T. Iannacchione,[2] John Sefton,[3] and Luis E. Vallejo[4]

**Abstract**: Once an abandoned mined land (AML) landslide occurs and is identified as an emergency, engineers must rapidly implement a slope stabilization design. Correct slope remediation solutions are generally derived from well-executed geotechnical examinations. This paper summarizes a large body of geotechnical data compiled by the U.S. Office of Surface Mining Reclamation and Enforcement (OSM) from AML landslides in eastern Kentucky. Special attention is placed on the examination of subsurface failures, phreatic water levels, soil profiles, and soil composition information from numerous borehole exploration programs. Strength properties calculated from laboratory procedures and stability analysis techniques were also reviewed. Laboratory-determined soil shear strength values were found to be higher than those inferred from stability analysis. This suggests that postfailure determinations of the phreatic surface may be largely inappropriate when used in stability analysis or that laboratory-measured shear strengths are ineffective in replicating in situ colluvium/spoil slope properties.

**Additional Key Words:** landslides, slope stability, coal mine spoil.

## Introduction

Prior to the Federal Surface Mining Control and Reclamation Act of 1977 (Public Law 95-87), rock, soil, organic material, coal, and even discarded mining equipment from surface and underground mines in Kentucky were simply cast over the sides of strip and adit benches. This coal mine spoil was generally poorly mixed, uncompacted, and allowed to settle at the material's angle of repose. The spoil generally rested on top of the preexisting colluvium, alluvial, and residual soil layers. Many of these slopes were already close to failing or, in some cases, had already experienced some slope instability. Weigle (1966) estimated that 12% of eastern Kentucky's strip mine spoil had failed by the mid-1960's. Often, runoff from the abandoned mines drains through the adjacent soil cover, increasing its degree of saturation. Increasing moisture contents caused by poor drainage conditions, reduces the shear strength of the soil and enhances the development of subsurface failures. This type of instability has become one of the Nation's most critical AML problems. From 1979 until February 1992, 425 landslides in 9 States were stabilized at a cost of $64,625,044. Kentucky had the most AML landslides, with a total of 268 at a cost of $41.5 million. The average cost of each stabilization project was $152,059.

Depending on the landslide's rate of movement and proximity to civil structures, different reactions are required by State and Federal agencies responsible for implementing the AML program. A landslide is classified as a "dangerous slide" if it represents an imminent hazard to humans or, if unchecked, it could inflict heavy damage on civil or residential structures. These types of landslides are managed under the AML Emergency Program. Attempts are made to remedy this type of landslide as quickly as possible (Roberts and Spadaro 1992). Approximately 10% of the AML emergency landslide remediation actions in eastern Kentucky have been designed using extensive geotechnical information. However, even with this geotechnical information, the soil's shear strength was still difficult to establish. Accurate shear strength characterization of landslide material is absolutely essential for reliable designs. Uncertainty about the range of material strengths experience from different categories of AML landslides affects the engineer's judgment. Under these conditions, the remediation effort must often be conservatively designed. Underestimating material strength leads to increased remediation cost, while overestimating could lead to inadequate slope stabilization.

## Causes of AML Landslides

Characterization of AML landslides in eastern Kentucky was accomplished by analyzing 37 geotechnical reports from OSM's project data base (table 1) dispersed over much of Kentucky's eastern coalfields (fig. 1). In all of the cases analyzed, past mining activity was associated with AML landslides and generally produced what is technically referred to as a rotational

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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[4]Luis E. Vallejo, Associate Professor of Civil Engineering, University of Pittsburgh, Pittsburgh, PA.

Table 1. Characteristics of eastern Kentucky AML landslides.

| | Landslide name/soil layer[1] | Soil type | Description | In situ moisture | Consistency | USCS[2] |
|---|---|---|---|---|---|---|
| (1) | Adkins/1 | Spoil | Sandy lean clay | Moist to saturated | Soft | CL |
| | Adkins/2 | Residual | do | Moist | Medium to stiff | CL |
| | Adkins/3 | Colluvium | do | Moist to wet | Soft to medium | CL |
| (2) | Allen-Banner/1 | – | Clay sand | – | – | SC |
| | Allen-Banner/2 | – | Inorganic silt | – | – | ML |
| | Allen-Banner/3 | – | Silty sand | – | – | SM |
| (3) | Anna Miller/1 | Colluvium | Sandy silt with rock fragments and large boulders. | Moist to wet | Soft to very stiff | ML |
| (4) | Bear Branch/1 | Spoil | Sandy lean clay with rock fragments and boulders. | Damp to moist | Stiff to very stiff | CL |
| | Bear Branch/2 | Spoil | Clayey sand with rock fragments and boulders. | do | Medium dense to dense. | – |
| | Bear Branch/3 | Colluvium | Sandy lean clay with rock fragments. | Damp to moist | Stiff to very stiff | CL |
| (5) | Bentley/1 | – | Silty sand with sandstone fragments. | – | Moderately stiff | SC-SM |
| (6) | Betty Trent/1 | Colluvium | Clayey sand with gravel and coal. | Moist | Stiff to very stiff | SC |
| | Betty Trent/2 | Colluvium | Clayey sand with gravel-size rock fragments. | Moist | do | SC |
| | Betty Trent/3 | Spoil | Silty clayey sand with gravel. | Moist | Stiff | SC-SM |
| (7) | Birchfield/1 | Colluvium | Sandy lean clay with rock fragments. | Damp to moist | Stiff to very stiff | CL |
| | Birchfield/2 | Colluvium | do | do | do | CL |
| (8) | Bonnyman II/1 | Colluvium | Sandy lean clay | – | – | CL |
| (9) | Bryant/1 | Spoil | Lean clay with sand and gravel. | Moist | Soft to medium | CL |
| | Bryant/2 | Residual | do | Moist | Stiff | CL |
| (10) | Carol Begley/1 | Spoil | Sandy lean clay with gravel. | Damp to moist | Medium to stiff | CL |
| | Carol Begley/2 | Colluvium | Sandy lean clay | Moist | Stiff to very stiff | CL |
| (11) | Church/1 | Colluvium | do | Damp | Stiff | CL |
| | Church/2 | Spoil | Sandy lean clay with gravel | Damp to wet | Medium to hard | CL |
| (12) | Clear Creek/1 | Spoil | Sandy lean clay with rock fragments and boulders. | Moist to wet | Soft to stiff | CL |
| | Clear Creek/2 | Colluvium | Lean clay with gravel-size rock fragments. | Moist | Medium to very stiff | CL |
| | Clear Creek/3 | Colluvium | Clayey sand with silt | Moist to wet | Stiff to very stiff | SC-SM |
| (13) | Cubert Spence/1 | Residual | Sandy lean clay | Damp to moist | Very stiff to hard | CL |
| | Cubert Spence/2 | Spoil | Sandy lean clay with rock fragments. | do | Stiff to very stiff | – |
| (14) | Denver Newsome/1 | Spoil | Sandy with coal shale fragments. | – | Medium | SM-SC |
| | Denver Newsome/2 | – | – | Wet | Very soft | CL |
| (15) | Eva Hamilton/1 | Colluvium | Sandy lean clay with gravel and boulders. | Moist | Soft to very stiff | CL |
| | Eva Hamilton/2 | Colluvium | Sandy lean clay with gravel. | Moist to wet | Medium to stiff | CL |
| | Eva Hamilton/3 | Colluvium | Sandy lean clay with shale fragments. | Moist | Stiff | CL |
| (16) | Everage/1 | Colluvium | Sandy silt with rock fragments. | Damp to wet | Stiff to very stiff | ML |
| (17) | Frazier/1 | Colluvium | Sandy lean clay | Damp to moist | Medium to hard | CL |
| | Frazier/2 | Colluvium | Lean clay with sand | do | Stiff to hard | CL |
| | Frazier/3 | Residual | Clayey sand with gravel | Moist | Medium dense to dense. | SC |
| (18) | French III/1 | – | Silty sand | – | – | ML-CL |
| | French III/2 | Spoil | Sandy | – | – | – |
| (19) | Grose/1 | Colluvium | Sandy lean clay with rock fragments. | Damp to moist | Soft to very stiff | CL |
| | Grose/2 | Residual | Lean clay | Moist to wet | Stiff to very stiff | CL |
| (20) | John Kennedy/1 | Fill | Lean clay with sand | do | Medium to stiff | CL |
| | John Kennedy/2 | Spoil | do | do | Soft to medium | CL |
| (21) | Kodak/1 | – | Clayey sand with gravel and coal fragments. | Dry to wet | Loose to medium compact | SC |
| | Kodak/2 | Spoil | Clayey sand with rock fragments and boulders. | Damp to wet | Soft to medium | – |
| | Kodak/3 | Colluvium | Sandy lean clay with gravel and coal fragments. | Damp to moist | Stiff to very stiff | CL |
| | Kodak/4 | Colluvium | Clayey sand with sandstone fragments. | do | Very stiff to hard | SC |
| (22) | Leroy Jewell/1 | – | do | Moist | Medium to stiff | SC |
| | Leroy Jewell/2 | Colluvium | Sandy lean clay | Moist | do | – |
| (23) | Loftis/1 | Fill | Clayey sand with rock fragments and boulders. | Damp to moist | Medium to compact | SC |
| | Loftis/2 | Fill | Sandy lean clay with rock fragments. | do | Medium to very stiff | CL |
| | Loftis/3 | Colluvium | do | do | Very stiff. | CL |
| | Loftis/4 | Residual | Silty clay with shale partings. | do | Medium | – |
| (24) | Maynard/1 | Spoil | Sandy clay with gravel and rock fragments. | Dry to wet | – | SC |
| | Maynard/2 | Colluvium | do | do | – | SC |
| (25) | Moore II/1 | Residual | Sandy lean clay | Damp to moist | Medium to very stiff | CL |
| | Moore II/2 | Spoil | do | Moist to wet | Soft to medium | CL |
| (26) | Pinky Lee/1 | Spoil | do | Moist | Medium | CL |
| (27) | Ray/1 | Colluvium | Clayey sand with sandstone fragments. | Damp to moist | Stiff to very stiff | – |
| (28) | Ritchie/1 | Spoil | Silty to clayey sand with gravel and boulders. | Moist | Medium to stiff | SC-SM |
| | Ritchie/2 | Residual | Sandy lean clay | Moist | Stiff | CL |
| | Ritchie/3 | Colluvium | Clayey sand with gravel | Moist | Stiff | SC |
| (29) | Ronnie Anderson/1 | Spoil | Clayey sand with gravel and boulders. | Moist to wet | Soft | SC |
| | Ronnie Anderson/2 | Colluvium | Sandy lean clay with gravel | Moist to saturated | Very soft to medium | CL |
| | Ronnie Anderson/3 | – | Sandy lean clay with sandstone fragments. | Moist to wet | Medium to stiff | CL |
| | Ronnie Anderson/4 | Colluvium | Lean clay with sand and gravel. | do | Soft to stiff | CL |
| | Ronnie Anderson/5 | Alluvial | Clayey sand with gravel | do | Medium | SC |
| (30) | Sanders/1 | Spoil | Sandy lean clay with rock fragments. | Damp to wet | Soft to stiff | CL |
| | Sanders/2 | Colluvium | Lean clay | Damp to moist | Stiff to very stiff | – |
| (31) | Sharp/1 | – | Sandy lean clay | Moist to wet | Soft to very stiff | CL |
| (32) | Spradlin Branch II/1 | Spoil | Organic material and boulders. | Wet to saturated | Very soft to medium | CL |

| | Landslide name/soil layer[1] | Soil type | Description | In situ moisture | Consistency | USCS[2] |
|---|---|---|---|---|---|---|
| | Spradlin Branch II/2 | Residual | Gravelly lean clay with sand. | Moist to wet | Medium | CL. |
| | Spradlin Branch II/3 | Colluvium | Clayey sand with gravel | – | – | SC. |
| (33) | Stone/1 | Spoil | do | Damp to wet | Soft to stiff | SC. |
| | Stone/2 | Spoil | Sandy lean clay | Moist to wet | Soft to medium | CL. |
| | Stone/3 | Alluvial | Clayey sand | Saturated | Medium | SC. |
| (34) | Sue Taylor/1 | – | Sandy silty clay with gravel | – | – | CL-ML. |
| | Sue Taylor/2 | – | Sandy silt | – | – | ML. |
| (35) | Wells-Pennington/1 | Spoil | Sandy lean clay | Damp to moist | Soft to stiff | CL. |
| | Wells-Pennington/2 | Colluvium | Sandy lean clay with gravel | Moist | Stiff | CL. |
| | Wells-Pennington/3 | – | Sandy lean clay | Moist | Medium to stiff | CL. |
| (36) | Winfield-Weaver/1 | – | – | – | – | – |
| (37) | Wooton/1 | Spoil | Silty sands with boulders and logs | Wet | Very soft to very stiff | SM. |

do. Same as above.
SC Clayey sands, sand-clay mixture.
SM Silty sands, sand-silt mixture.
SC-SM Clayey sands and silty sands.
CL-ML Sandy clays and sandy silts.
CL Inorganic sandy, silty, or lean clays of low to medium plasticity.
ML Inorganic clayey silts or clayey, very fine sands with low plasticity.
[1]Numbers in parentheses refer to landslide locations shown in fig. 1.
[2]Unified Soil Classification System.
NOTE.–Dash indicates not determined.

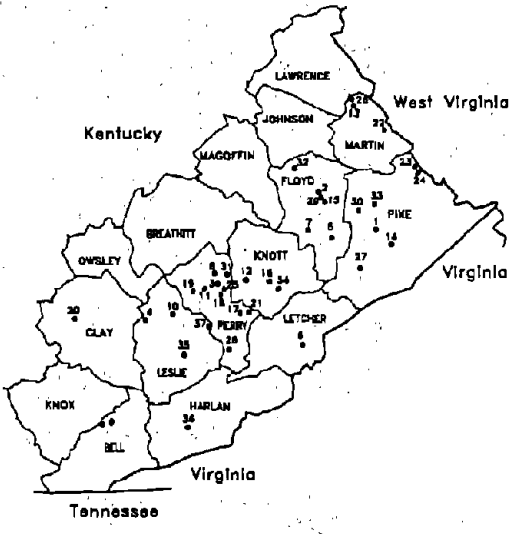


Figure 1. Eastern Kentucky counties with AML landslide locations where geotechnical data was available.

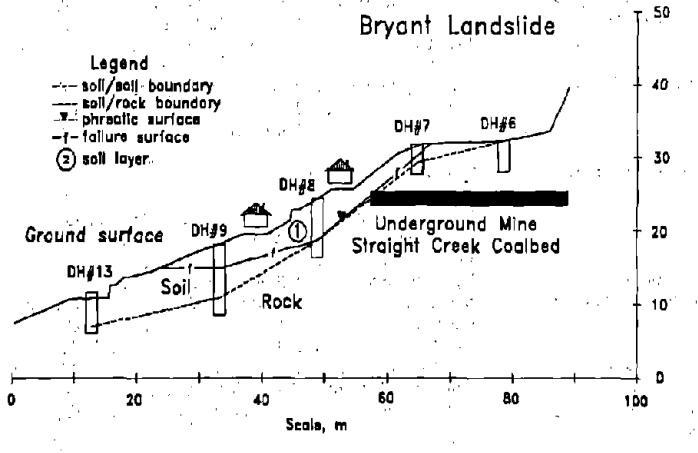


Figure 2. Typical cross-section through an eastern Kentucky AML landslide consisting primarily of spoil.

or translational debris flow. Typical landslide characteristics are multiple soil layers, soil thicknesses, and the position of phreatic surfaces and mined coalbeds.

Three general categories of mining-induced landslides were recognized from this data base: 1) failure of outcasted spoil material; 2) failure of natural soil slopes due to increased water saturation from mining-induced sources; and 3) failure of natural soil slopes due to increased surcharge loading from outslope spoil material. Of the three causes, the first two were much more prevalent. A more detailed description of each cause is given below.

## Failure of Outslope Casted Spoil Material

Considering the composition and manner of placement, it is understandable that outslope casted mine spoil has been involved in extensive slope instabilities. Spoil is the dirt and rock material overlying the coalbed that is removed to gain access to the coal. Of the 37 landslides examined, approximately one-half were attributed directly to failures within the casted spoil material with conditions similar to those shown in fig. 2. Kimball (1974) noted that prior to the Federal Surface Mining Control and Reclamation Act of 1977, almost all of eastern Kentucky's strip mines dumped or pushed the spoil over the bench with little or no compactive effort, allowing it to assume its angle of repose. Because the spoil has been placed recently in geologic terms (10 to 40 years), much of this material is still disintegrating with time, producing higher concentrations of fines. Therefore, temporary increases in porewater pressure from heavy storms and progressive deterioration of the strength of soil from weathering have caused and will continue to cause sporadic slope instabilities throughout eastern Kentucky.

## Failure of Natural Soil Slopes Due to Increased Water Saturation From Mining-Induced Sources

Natural soils found on eastern Kentucky's slopes are predominantly comprised of colluvium deposits, with lesser amounts of residual and alluvial soil layers. Typically, colluvium consists of unstratified, locally derived, and randomly oriented angular blocks of bedrock in a clayey matrix. In geologic terms, colluvium is actively moving (Rahn 1986). Residual soils are the disintegrated materials above the bedrock, which have been subjected to weathering process influenced primarily by changes in moisture content and the freeze-thaw cycle (Taylor 1948). In general, eastern Kentucky's hillsides have a relatively thin layer of residual soil (0 to 7 m thick). Fills are compacted soil moved in place by mechanical means. Alluvial soil includes stratified layers of silt, sand, gravel, and clay deposited within a stream's flood plain. Alluvial soil is only found along the lowest of eastern Kentucky's hillsides and is therefore rarely found in landslide areas. Landslides were found to occur most frequently in colluvium, which is consequentially most like the spoil material in character.

When reclamation is not practiced, rainfall and/or mine water is allowed to drain directly into the natural soil (fig. 3). As noted previously, increased water saturation raises the phreatic surface, increasing the porewater pressure and lowering the shearing resistance of the soil. The shearing resistance of the soil counteracts the tendency of the soil on a slope to move downward and outward under the influence of gravity.

## Failure of Natural Soil Slopes Due to Increased Surcharge Loading From Outslope Spoil Material

Sometimes landslides in the naturally steep slopes of eastern Kentucky are activated by surcharge loading from casted spoil somewhere upslope from the scarp of the resulting failure surface (head). The weight of the spoil increases the driving force exerted on the existing soil, overcoming the resistance force. Surcharge loading from outslope spoil was found to be responsible for only one landslide.

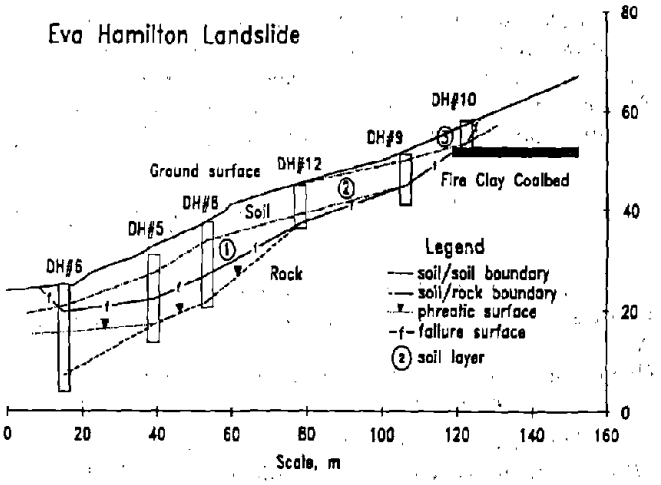


Figure 3. Cross-section of a typical eastern Kentucky AML landslide where underground mine has emitted water in the natural soil slope.

## Classification of AML Landslide Material

Two to three distinct soil layers were found within each of the 37 landslides examined (table 1). Thirty-one colluvium and 24 spoil layers were identified from a total list of 83. Nine other soil layers were classified as residual, 3 as fill, 2 as alluvial, and 14 were not identified.

It is understood that the type of soil found on eastern Kentucky's slopes directly depends on the lithologic type of the subjacent rock strata (Davies 1973). In most of the slopes studied in this paper, the strata found immediately below the soil belong to the Breathitt Formation. This formation contains numerous minable coalbeds and alternating units of sandstones, siltstones, and shales. The siltstones and shales readily weather into soils, which form the colluvium and residual layers. The same weathering process takes much longer to break the resistant sandstone down to its granular components. Therefore, rock fragments ranging in size from gravel (4.75 to 75 mm) to boulders (>300 mm) are generally derived from sandstone formations. The percentage of rock debris within eastern Kentucky slopes is believed to be much higher than that of other landslide regions in Ohio, Pennsylvania, and northern West Virginia principally because of increased occurrence of sandstone strata.

Of the 31 colluvium layers examined, 20 were described as sandy lean clays or clayey sands with gravel, rock fragments, or boulders. A lean clay is one that is only slightly plastic because it contains a larger proportion of silt or sand than clay. Seven samples were simply described as sandy lean clays. Two samples were sandy silt with rock fragments or boulders. The spoil layers were described in a similar fashion. Sandy lean clay or a clayey sand with gravel, rock fragments, and boulders were the dominant descriptions. Only seven soils were designated without a larger fragments component. Spoil

layers differ from colluvium in the occasional occurrences of mining machinery, topsoil and organic material such as trees and brush, and the uneven distribution of rock fragments based on the distance from the spoil dump point. Weigle (1966) and Kimball (1974) described similar occurrences of spoil in eastern Kentucky. Clearly, a significant portion of the colluvium and spoil contains large particles varying in size and shape and floating in a matrix of clay and sand mixtures. The significance of this will be discussed later in this paper.

Soils studied in this paper were classified by the Unified Soil Classification System (Casagrande 1948). Sixty-six percent of the layers were classified as either CL or ML, both fine-grained soils (table 2). The dominant classification was CL (lean clays), accounting for 65% of the colluvium, 65% of the spoil, 88% of the residual, and 67% of the fill layers. All of the fine- grained soils had low to medium plasticities. Next to the alluvial, spoil had the highest concentration of coarse-grained soils (SC to SM) with 35%, while residual had the lowest concentration with 12%. All of the coarse-grained soils were dirty (clayey or silty) sands. In another study, Weigle (1966) found that spoil varied from clayey silt (ML) to silty sand (SM) with relatively low plasticity.

Table 2. Classification of soils found in eastern Kentucky's AML landslides.

| Soil Layer | Fine-grain | | | Coarse-grain | | | Not Defined |
|---|---|---|---|---|---|---|---|
| | CL | CL-ML | ML | SC | SC-SM | SM | |
| Colluvium | 17 | | 2 | 6 | 1 | | 3 |
| Spoil | 13 | | | 3 | 3 | 1 | 4 |
| Residual | 7 | | | 1 | | | 1 |
| Fill | 2 | | | 1 | | | |
| Alluvial | | | | 2 | | | |
| Not determined | 4 | 2 | 2 | 3 | 1 | 1 | 1 |

## Material Properties Affecting AML Landslides

Certain material properties are important in estimating the strength of AML slopes. Two of the most critical properties are *moisture content* and *consistency/density*. In general, the rapid weathering of the shales and siltstones have produced well-graded colluvium and spoil layers. Well-graded soil is often low in permeability and variable in moisture content. Okagbue (1984) noted a correlation between spoil moisture content and the percentage of fine-grained soils, suggesting that water contributes to material degradation.

Below the ground water or phreatic surface, soils are completely saturated with water. Eastern Kentucky's AML slopes have a range of phreatic surface conditions, some of which may be associated with seasonal changes in rainfall. Swanson et al. (1983) found, however, that phreatic surface fluctuations were not directly comparable to rainfall intensity. Exploration drilling at many of the study sites found some evidence of phreatic surfaces above the soil-rock interface, especially where an abandoned mine was emitting water directly into the slope. Several geotechnical reports mentioned phreatic surfaces thought to extend close to the ground surface during periods of excessive rainfall, but none measured such a condition. This is, of course, a serious consideration because as the phreatic surface rises through the soil layers, the ability of the soil mass to resist failure is greatly diminished. Additionally, Swanson et al. (1983) noted that after the phreatic surface is lowered below a newly developed failure surface, the spoil layers may soften with a permanent loss of strength.

Most of the landslide slopes studied had phreatic surface well below the ground surface and most often occurred at or below the rock line. Soils above the phreatic surface were found to have a wide range in moisture contents. The degree to which the soil was saturated can be expressed in terms of the percentage of water filling void spaces: dry (0%), humid (1% to 25%), damp (26% to 50%), moist (51% to 75%), wet (76% to 99%), and saturated (100%). Colluvium, spoils, residuals, and fill layers within this study were found to span the entire range of saturation. However, the average saturation was in the moist category (fig. 4). Not surprisingly, the alluvial layers, which are found near the flood plains, were closer to the saturated state.

Another important material property, as it relates to slope stability, is *consistency and/or relative density*. Consistency of fine-grained soils corresponds in some respects to the relative density in coarse-grained soils (Holtz and Kovacs 1981). In most of the 37 AML landslide projects studied, the consistency/relative density characteristics were evaluated with data generated from the Standard Penetration Test (SPT). The SPT measures the resistance supplied by different soil layers as a tool is driven downward. The penetration resistance is reported as the number of blows to drive the tool 0.3 m. Lambe and Whitman (1969) discuss techniques for correlating penetration resistance with the relative density of sand and the consistency and strength of clay.

Consistency is a measure of the degree of adhesion between clay particles and the resistance offered against forces that tend to deform or rupture the soil aggregate (Terzaghi and Peck 1967). The consistency of clays can be categorized as very soft, soft, medium, stiff, very stiff, and hard. All of these consistencies were encountered in the AML landslides of eastern

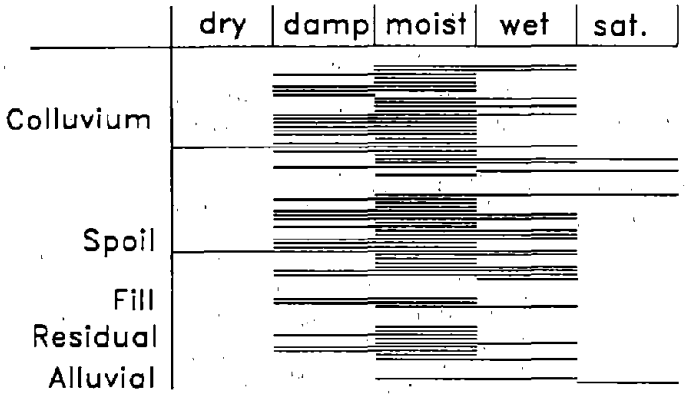


Figure 4. Saturation ranges of eastern Kentucky AML landslide soil layers.

Kentucky (fig. 5). Very soft or hard clays, which correspond to soils of little or great strength, were rarely encountered. In many cases, extremely high resistance rates (>100 blows/0.3 m) were experienced when the penetrating tool encountered large rock fragments or boulders. In general, colluvium, residual, and fill CL-ML layers have stiff consistencies, while spoil CL-ML layers are slightly less stiff, averaging medium consistency.

Coarse-grained SC-SM soils have relative density categories which are similar to those of the consistency categories of clay: very loose, loose, medium, dense, and very dense. These categories are also a crude indicator of strength in that very loose sand could be excavated by hand, while very dense sand would require power tools. Because of the inconsistencies in reporting relative density, analysis of this property was not accomplished.

Density characteristics can also be examined by weighing equal volumes of soil. It is generally assumed that increases in density of similar materials produce corresponding increases in strength. For instance, when spoil is first placed it can have a dry unit weight of 1,137 to 1,442 kg/m$^3$ (Weigle 1966). Over time, consolidation, which is a form of densification, can occur through dewatering or surcharge loading from overlying soil layers, increasing the dry unit weight. In eastern Kentucky, an average dry unit weight of 1,725 kg/m$^3$ was measured. A comparison of dry unit measurements with the optimum values as calculated by the Standard Proctor test was also performed. This analysis indicated that nonuniform consolidation is typical for both the colluvium and spoil soil layers. It was determined that colluvium had compacted an average of 92% of its optimum value and the aging spoil an average of 94%. Generally, a compaction of 95% is viewed as a minimum for mechanically compacted soils placed in civil engineering applications. These numbers compare well with Swanson et al. (1983) where the percent of compaction ranged from 81% to 107%, with a median value of 89% for spoil recently reclaimed from active strip mines.

## AML Landslide Shear Strengths

The most important design input parameter needed to engineer remediation efforts is the soil's *shear strength*. Its primary use is to define the shape of the Mohr-Columb failure envelope. Soil shear strength is delimited by two parameters: the friction angle ($\phi$) and the cohesion (C). These shear strength parameters were analyzed for eastern Kentucky's AML landslides by comparing laboratory-derived shear strength data with those found by performing stability analysis calculations.

### Laboratory Determination

One of the most accepted techniques for determining soil shear strength parameters is by laboratory means. The two commonly used laboratory tests are the *direct shear* and the *triaxial* test. It is generally accepted that the direct shear test is simpler to perform but less accurate than the triaxial test. Thirty-eight shear strength tests were analyzed from 28 different AML landslides (table 3). The triaxial method accounted for 36 tests.

Triaxial tests are generally performed under one of three conditions: unconsolidated-undrained (UU), consolidated-undrained (CU), and consolidated-drained (CD). The consolidation term refers to the preparation of the sample prior to testing, and the drainage term refers to conditions during testing. When a CU triaxial test is performed, measurement of the porewater pressure permits the data to be reduced to the pressure that is borne only by the soil particles. This produces what is referred to as the effective shear strength parameters: $\phi'$ and C'. In general, the effective shear strength parameters best mirror long-term conditions and are the most beneficial in analyzing landslides.

All of the laboratory tests were performed on selected fine- and coarse-grained soils. Material of coarse gravel size (19 mm and greater) and above was removed prior to laboratory runs owing to the constraints imposed when testing material

in direct shear boxes or triaxial chambers. Generally, a standard 71-mm-diameter triaxial specimen can have particles no larger than one-sixth its diameter, or approximately 12 mm.

Soil, from which testing was accomplished, was collected from exploratory drill holes. Eleven of the 38 tests were performed on undisturbed soil recovered primarily from Shelby tubes. Twenty-three of the tests were performed on remolded soil. Remolding is the process of kneading or working a soil to produce higher densities. These samples were remolded from 85% to 95% of their maximum dry unit weight.

Data compiled from all the triaxial CU tests give average shear strength parameters of $\phi' = 29.7$ and $C' = 4.9$ kPa. Some variations exist among different soil types as shown from the average values listed below:

colluvium, $\phi' = 30.6$ and $C' = 3.1$ kPa

spoil, $\phi' = 27.8$ and $C' = 9.7$ kPa

residual, $\phi' = 28.6$ and $C' = 1.1$ kPa

fill, $\phi' = 27.9$ and $C' = 6.2$ kPa

The above parameters represent peak strength condition. Swanson et al. (1983) noted similar values for spoil $\phi'$ with an average of 29°. Residual strength values for the spoil are less, averaging approximately 20°. Residual values are more representative of a failed material's strength characteristics. The friction angle for the residual state in clays is generally much lower than the peak strength values. Unfortunately, residual strength data were not determined for this data set. Consideration should be given to utilizing residual data when stabilizing failed landslides comprised largely of clays.

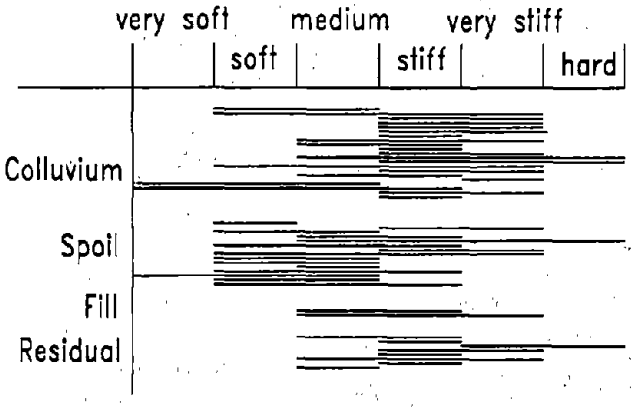


Figure 5. Ranges of clay consistency encountered in eastern Kentucky's AML landslides.

Table 3. Laboratory determined shear strengths.

| Landslide name/soil layer | Effective friction angle ($\phi'$) | Effective cohesion, (C', kPa) | Test | Sample type | Percent remolded | Maximum dry unit weight, kg/m³ | Optimum moisture, % |
|---|---|---|---|---|---|---|---|
| Allen-Banner/1 | 27 | 2.4 | CU | Remolded | 90 | 1,897 | 13.3 |
| Bear Branch/3 | 29.9 | 0 | CU | Undisturbed | – | 1,845 | 14.3 |
| Bentley/1 | 39 | 0 | CU | Undisturbed | – | 1,796 | 11.1 |
| Birchfield/1 | 30.1 | 1.0 | CU | Remolded | 91 | 1,839 | 14.1 |
| Birchfield/2 | 30.6 | 1.0 | CU | Remolded | 91 | 1,954 | 11.6 |
| Bonnyman II/1 | 24.2 | 29.7 | DS | – | – | – | – |
| Bryant/1 | 30.7 | 3.4 | CU | Undisturbed | – | 1,826 | 15.3 |
| Bryant/2 | 25.5 | 2.4 | CU | Remolded | 90 | 1,892 | 14.5 |
| Carol Begley/1 | 28.7 | 2.4 | CU | Remolded | 92 | 1,808 | 13.9 |
| Carol Begley/2 | 28.7 | 2.4 | CU | Remolded | 92 | 1,877 | 13.9 |
| Clear Creek/1 | 27.3 | 3.8 | CU | Remolded | 92 | 1,800 | 12.7 |
| Clear Creek/2 | 26.7 | 5.3 | CU | Remolded | 92 | 2,111 | 11.8 |
| Clear Creek/3 | 32 | 0 | CU | Remolded | 92 | 1,938 | 11.1 |
| Cubert Spence/2 | 27.8 | 1.0 | CU | Remolded | – | 1,720 | 17.3 |
| Denver Newsome/1 | 23 | 71.9 | CU | Undisturbed | – | – | – |
| Eva Hamilton/1 | 29.9 | 2.4 | CU | Undisturbed | – | 1,849 | 13.9 |
| Eva Hamilton/2 | 28.6 | 0 | CU | Undisturbed | – | 1,784 | 14.4 |
| Everage/1 | 34.5 | 10.1 | CU | Undisturbed | – | 1,778 | 17.5 |
| Frazier/1 | 32.3 | 7.2 | CU | Undisturbed | – | 1,866 | 13.7 |
| French III/1 | 28.7 | 0 | CU | Remolded | 90 | 1,865 | 16.3 |
| Grose/1 | 35.3 | 2.4 | CU | Undisturbed | – | 1,776 | 16.5 |
| John Kennedy/1 | 27.4 | 10.1 | CU | – | – | 1,732 | 18.1 |
| John Kennedy/2 | 28.7 | 9.6 | CU | – | – | 1,688 | 17.4 |
| Loftis/1 | 28.4 | 2.4 | CU | Remolded | 95 | 1,967 | 12.1 |
| Maynard/2 | 5.2 | 10.4 | UU | – | – | – | – |
| Moore II/2 | 27.4 | 2.9 | CU | Remolded | 90 | 1,834 | 14.4 |
| Pinky Lee/1 | 27.5 | 1.0 | CU | Remolded | 92 | 1,776 | 16.6 |
| Ray/1 | 30/36 | 1.9 | DS | Remolded | – | – | – |
| Ritchie/2 | 32.6 | 0 | CU | Remolded | 92 | 1,975 | 11.2 |
| Ritchie/3 | 32.5 | 10.5 | CU | Undisturbed | – | 1,861 | 14.2 |
| Ronnie Anderson/4 | 27.5 | 2.9 | CU | Remolded | 91 | 1,901 | 12.7 |
| Sanders/1 | 26.5 | 3.8 | CU | Remolded | 85 | 1,901 | 13.3 |
| Spradlin Branch II/3 | 32 | 0 | CU | Remolded | 88 | 1,964 | 11.4 |
| Stone/1 | 29.7 | 1.0 | CU | Remolded | 92 | 1,903 | 12.7 |

| Landslide name/soil layer | Effective friction angle ($\phi'$) | Effective cohesion (C', kPa) | Test | Sample type | Percent remolded | Maximum dry unit weight kg/m³ | Optimum moisture, % |
|---|---|---|---|---|---|---|---|
| Stone/2 | 27.9 | 3.8 | CU | Remolded | 92 | 1,815 | 15.2 |
| Wells-Pennington/1 | 28.4 | 3.4 | CU | Remolded | 95 | 1,780 | 17.6 |
| Wells-Pennington/2 | 28.9 | 1.4 | CU | Remolded | 95 | 1,807 | 15.9 |
| Winfield-Weaver/1 | 33.7 | 0 | CU | Undisturbed | – | – | – |

CU Consolidated undrained triaxial with pore pressure measurements. UU Unconsolidated undrained triaxial.
DS Direct shear. NOTE.–Dash indicates not determined.

It has become general practice to assume C′ equals zero for eastern Kentucky AML landslides. Analysis of laboratory data suggests this may not be prudent. An average of 4.9 kPa was obtained from all tests. This value seems reasonable since near surface soils are subjected to considerable wetting and drying which can cause overconsolidation (C′ > 0). Sensitivity studies, using analytical slope stability procedures, indicate that a 5-kPa cohesion value could produce consequential increases in the slope's safety factor.

## Stability Analysis

When a landslide occurs, the shear strength of the soil is mobilized along the full length of the failure surface. An estimate of the safety factor at this state can be made by performing a technique known as *stability analysis*. If the geometry of soil layers and the position of the phreatic and failure surfaces are known, the uniquely determined laboratory values of $\phi'$ and C′ can be evaluated. Safety factors greater than 1 would indicate the laboratory values of $\phi'$ and C′ are overestimating in situ soil strength. Safety factors below 1 would indicate the opposite.

In theory, the effective shear strength parameters could be determined by utilizing the back analysis technique. This technique assumes the safety factor of the failed slope is equal to 1. Strength values for the entire soil mass are evaluated by altering $\phi'$ and C′ until the safety factor equals 1. Unfortunately, these material properties cannot be uniquely defined since there are two effective shear strength parameters for each layer and several layers often exist for each landslide.

Sufficient data does exist to perform stability analysis on four landslides: Bryant (fig. 2), Eva Hamilton (fig. 3), Wells-Pennington (fig. 6), and Clear Creek (fig. 7). Each of these sites had suitable geotechnical data for determining the precise safety factors along observed failure surfaces. When uniquely determined laboratory values for $\phi'$ and C′ were input into stability analysis, safety factors between 1.3 and 2.3 were calculated (table 4). The stability technique used was the Spencer Method contained within the SB-SLOPE program.

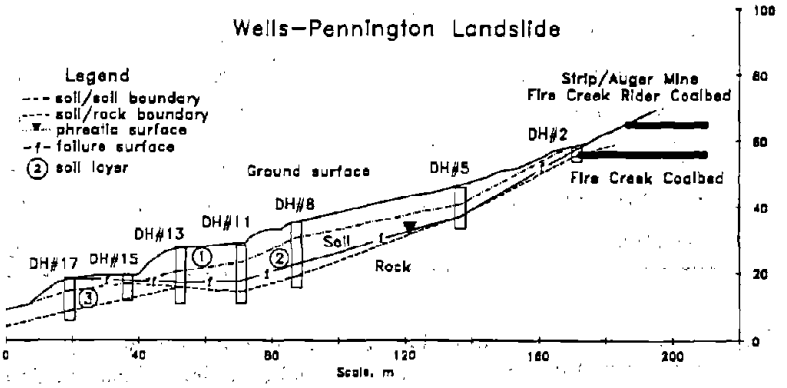


Figure 6. Cross-sectional view of the Wells-Pennington landslide.

Two solutions seem plausible to explain this discrepancy. The first would be that the field-determined locations of the phreatic surfaces are incorrect. Problems with evaluating phreatic surfaces after landslide initiation have already been discussed. The second is that laboratory-determined strength values of colluvium/spoil material are unreliable. One possible explanation for this is that laboratory testing does not consider the influence of particles greater than approximately 12 mm in dimension. Considerable mention has been made of the rock fragments occurring within both the colluvium and spoil layers (fig. 8). These large particles may influence the performance of the soil mass to a greater extent than had been previously believed. This could diminish the importance of testing soil void of rock fragments.

## Summary and Conclusions

Using data gathered from drilling, observational, and laboratory techniques, the following important factors that affect AML landslides in eastern Kentucky were determined:

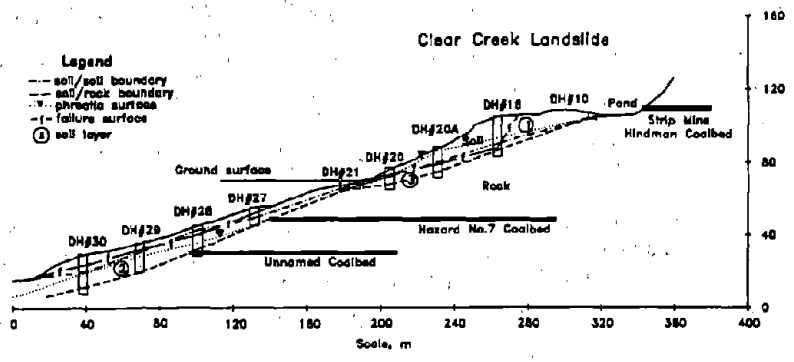


Figure 7. Cross-sectional view of the Clear Creek landslide.

1. A lack of reclamation has allowed rainfall and/or mine water to drain directly into the AML slopes, increasing the porewater pressure and lowering the shearing resistance of the soil.
2. Sixty-six percent of the landslide layers were classified as fine-grained soils (clays or silts). All of the coarse-grained soils were classified as dirty (clayey or silty) sands.
3. A significant portion of the colluvium and spoil contains large particles ranging from gravel to boulders in size and floating in a matrix of clay and sand mixtures.

Table 4. Safety factors of landslides using laboratory test data.

| Landslide Name/soil layer | Laboratory Test | | | Stability Analysis | | | Safety Factor |
|---|---|---|---|---|---|---|---|
| | $\phi'$ | C' | Unit Weight, kg/m³ | $\phi'$ | C' | Unit Weight, kg/m³ | |
| Bryant/1 | 30.7 | 3.4 | 1922 | 31 | 3.4 | 1922 | 1.9 |
| Clear Creek/1 | 27.3 | 3.8 | 2082 | 27 | 3.8 | 2082 | 1.3 |
| Clear Creek/2 | 26.7 | 5.3 | 1999 | 27 | 5.3 | 1999 | |
| Clear Creek/3 | 32 | 0 | 1986 | 32 | 0 | 1986 | |
| Eva Hamilton/1 | 29.9 | 2.4 | | 30 | 2.4 | 2002 | 2.1 |
| Eva Hamilton/2 | 28.6 | 0 | | 29 | 0 | 2002 | |
| Eva Hamilton/3 | | | | 29 | 0 | 2002 | |
| Wells-Pennington/1 | 28.4 | 3.4 | 1991 | 28 | 3.4 | 1986 | 2.3 |
| Wells-Pennington/2 | 28.9 | 1.4 | 1994 | 29 | 1.4 | 1986 | |
| Wells-Pennington/3 | | | | 29 | 1.4 | 1986 | |

4. Eastern Kentucky's AML slopes have a widespread of phreatic surface conditions ranging from below the rock line to close to the ground surface. Phreatic levels and/or drainage from abandoned mines are generally controlled by seasonal changes in rainfall.
5. Soils above the phreatic surface were found to have a wide range in saturation.
6. Wide variations were encountered in clay consistency. Spoil layers are less stiff than the colluvium, residual, and fill layers.
7. Colluvium and spoil are nonuniformly consolidated. Colluvium had compacted an average of 92% of its optimum value and the aging spoil an average of 94%.
8. Thirty-eight shear strength tests were analyzed from 28 different AML landslides. Particles greater than 12 mm were removed prior to laboratory runs. Data compiled from all the triaxial CU tests give average shear strength parameters of $\phi' = 29.7$ and $C' = 4.9$ kPa.
9. Stability analysis performed on four landslides using laboratory-determined $\phi'$ and C' produced safety factors between 1.3 and 2.3.

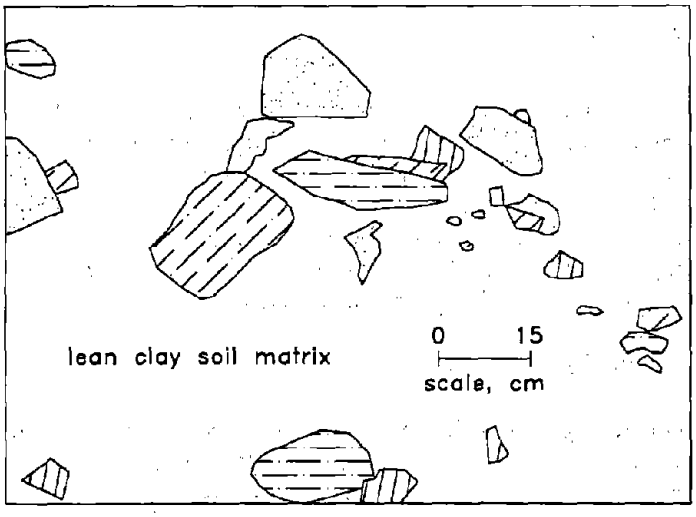


Figure 8. Image of a photograph showing the concentration of rock fragments within the colluvium of an AML landslide, near Hazard, KY.

Conclusions drawn from this investigation are:

Most of the AML landslides occurred either in the outcasted spoil material or within the saturated natural soil slopes. These slopes contain high concentrations of shale, siltstone, and sandstone rock debris. The shales and siltstones weather rapidly, producing well-graded colluvium and spoil with poor permeability. Because the spoil (and to a lesser degree the colluvium) is relatively young geologically, much of this material is still disintegrating with time, producing higher concentrations of fines. This implies that AML landslides will continue to occur in eastern Kentucky as strength conditions adjust to changing soil compositions.

Soil strength conditions were examined using laboratory and stability analysis techniques. The lack of agreement between these techniques presents difficulties. More attention needs to be focused on the most probable location of the phreatic surface during slope failure. The research community must also better establish the role of rock debris in slope strength.

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# RECLAMATION OF ABANDONED UNDERGROUND MINES IN THE UNITED KINGDOM [1]

David Brook [2]

**Abstract**: Since 1980, the Derelict Land Grant program has supported reclamation of abandoned mines in the United Kingdom. The stabilization of large-scale limestone mines in the West Midlands has stimulated the development of new methods of bulk infilling using waste materials as thick pastes. Colliery spoil-rock paste develops strengths of 10 to 20 kPa to support roof falls and prevent crown hole collapse. Pulverized fuel ash rock paste develops strengths over 1 MPa where lateral support to pillars is required. Smaller scale mine workings in the West Midlands and elsewhere have been stabilized using conventional grouting techniques, hydraulic and pneumatic stowing, foamed-concrete infill, bulk excavation with controlled backfill, and structural support using bolts, mesh, and shotcrete.

## Introduction

The United Kingdom has a long history of mining dating from prehistoric flint mines such as those at Grimes Graves in Norfolk. The bulk of British mining activity has been during and since the Industrial Revolution. Britain was at various times the principal world producer of copper, tin, lead and coal. Mining continues today on a much reduced scale.

Mining has taken place at some time in every county in England and Wales and, except for the Western Isles, every region in Scotland (fig. 1), although in some areas it has only been of minor extent. However, reliable records exist only from the last part of the 19th century, and much development has taken place over areas that were previously undermined. The methods of mining (fig. 2) and the general absence of any attempt to stabilize the mines when they were abandoned have resulted in a widespread legacy of potential subsidence.

Britain also has a long history of mining subsidence including the collapse of a large area above a limestone mine in Walsall in 1828 and the sudden collapse of the floor of a house in Norwich in 1853. More recently, on October 19, 1990, an earthquake caused by the collapse of the Tennant salt mine at Carrickfergus, Northern Ireland had a Richter magnitude of 2.5; an area of about 200 by 150 m subsided by up to 8 m in a single event (Griffith 1991).

Examples from Department of the Environment (DOE) research projects illustrate the scale of the problems. In the former limestone mining area of the West Midlands (Ove Arup and Partners 1983), 100 subsidence events in 150 yr have been identified, the majority from mines shallower than 30 m. In the South Wales coalfield (Statham and Treharne 1991) the majority of over 400 subsidence events identified between 1960 and 1993 are associated with mine entries (shafts or adits). In the tin and copper mining area of the Kerrier District in Cornwall (Arup Geotechnics 1990), 41 out of 74 subsidence events recorded between 1974 and 1990 were due to shaft collapse.

Conventional grouting methods have long been used to stabilize land prior to development and following subsidence events. In very few cases, however, was mine filling used to remove the risk of subsidence on a wider scale. In the early 1970's parts of limestone mines beneath Dudley in the West Midlands were infilled by flushing in sand through boreholes to prevent subsidence affecting housing and other development. This was only partly successful, and one or two subsidence-damaged houses over parts of the mines that were infilled have been demolished subsequently. More recent investigation has also shown that there are significant voids within the infilled areas, and a risk of subsidence still remains.

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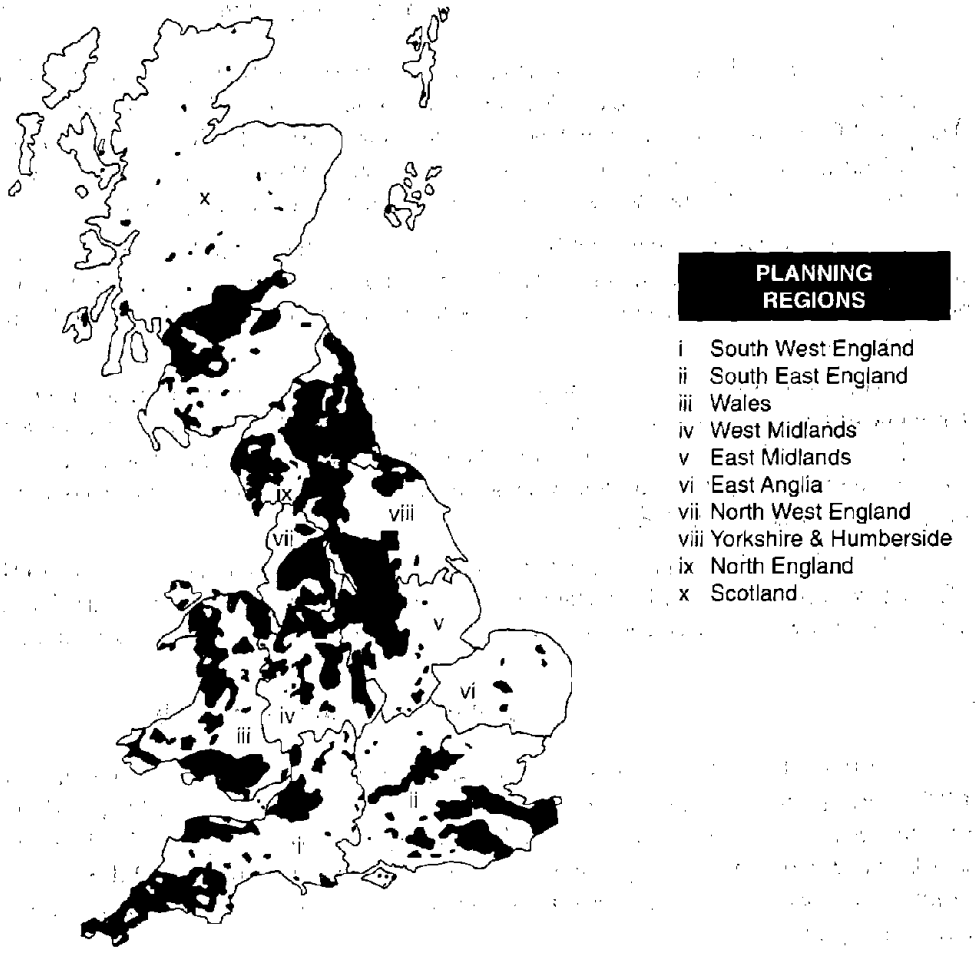


Figure 1. The extent of mining in Great Britain.

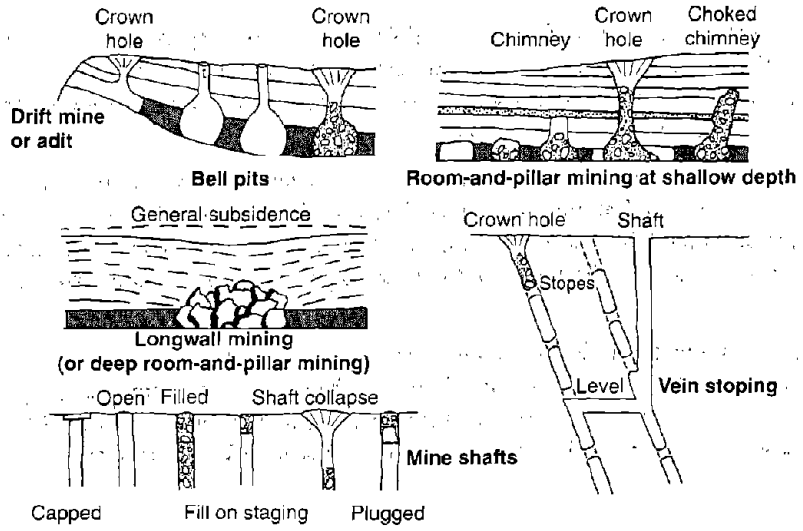


Figure 2. Mining methods and effects on the surface.

## Derelict Land Grant

The National Parks and Access to the Countryside Act 1949 conferred powers on local authorities to acquire land and to carry out works to enable derelict, neglected, or unsightly land to be reclaimed, improved, or brought into use. The Local Government Act 1966 extended Derelict Land Grant (DLG) coverage throughout England and allowed a national program of reclamation to be set up based on schemes carried out by county and district councils; in Scotland and Wales, separate Development Agencies carry out such works among others.

DLG is restricted to land that is so damaged by industrial or other development as to be incapable of beneficial use without treatment. The DLG program has achieved the reclamation of substantial areas of derelict surface mineral workings and of spoil heaps and tips, especially those from disused collieries, as well as areas of land already damaged by mining subsidence.

The Local Government Planning and Land Act 1980, now consolidated in the Derelict Land Act 1982, extended the scope of DLG to include land which, though not derelict, is likely to become so by reason of actual or apprehended subsidence due to the collapse of underground mine workings. Abandoned coal mines were excluded because there was already statutory compensation for subsidence damage; their inclusion would also have increased the potential scale of the problems faced by the DLG program.

The 1980 legislation had essentially been triggered by the situation in the West Midlands, where almost 500 ha of the conurbation, including land near the town centers of Walsall and Dudley and substantial residential and industrial development, were potentially at risk from subsidence due to abandoned limestone mines.

## The Development of Rock Paste

In 1981, the DOE, together with the local authorities concerned, funded a research study (Ove Arup and Partners 1983) to assess the extent and degree of risk of surface disturbance attributable to old limestone workings in the West Midlands and to consider what remedial action was required. Subsequently, DLG funds were earmarked to permit work on site investigation, monitoring, and remedial works to preserve surface stability above the mines. This Limestone Program has formed a major element in the national DLG program since 1983, and expenditure is currently about £10 million per year.

The scale of these old limestone mines, with rooms up to 11 m high, extraction ratios of the order of 75% to 80% and individual mine volumes of up to 500,000 $m^3$, at depths of up to 200 m, meant that conventional grouting techniques would be prohibitively expensive and alternative means of stabilization were needed.

Since there were considerable amounts of waste materials from coal mining in spoil heaps in the West Midlands area, which could be available at a nominal price, it was suggested that these should be examined as a potential bulk filling material. This would have dual environmental benefits in enabling the risk of subsidence to be removed at the mine sites and in reclaiming derelict spoil heaps at the source sites.

Flushing of this material into the mines with water would require large numbers of boreholes and the disposal of large amounts of contaminated water, and might not achieve the desired objectives. It was therefore proposed that the colliery spoil should be injected through boreholes as a thick paste able to flow considerable distances within the mine (fig. 3) and to achieve complete filling to the mine roof even at high points where some roof collapse had already occurred.

While the pumping requirements would dictate the need for a low strength paste, consolidation within the mine would increase the strength sufficiently for its use to be effective. The objective was not that the paste should be load-bearing but that it should fill the mine voids and provide sufficient strength to support material falling from the roof, enabling developing chimneys, which could otherwise propagate as crown holes to the surface, to become effectively self-choking.

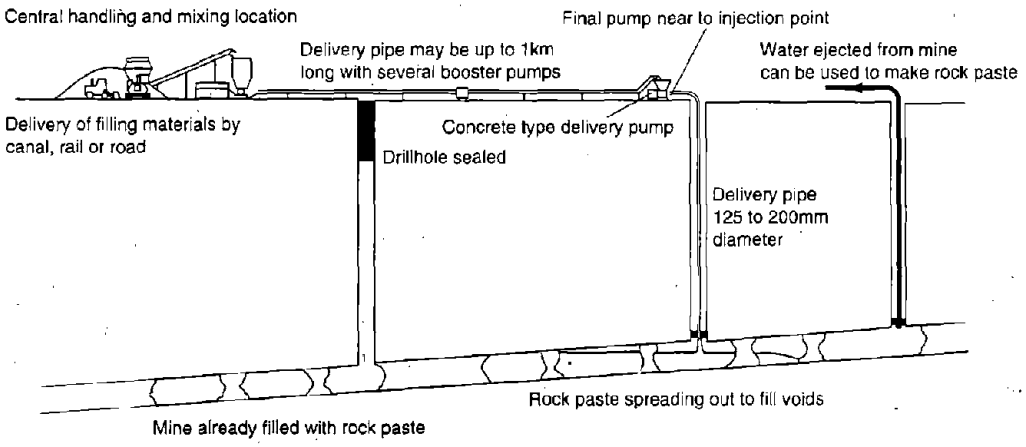


Figure 3. Rock paste infilling as originally proposed.

Through its research program and the DLG program, the DOE has funded the development of this novel method of mine infilling, designated rock paste, since 1982. Small- and large-scale pumping trials and heap spreading experiments were carried out by the Building Research Establishment (Hills 1983, Hills and Ward 1985, Ward and Hills 1985).

## Small-Scale Pumping Tests

Exploratory trials included the measurement of spoil properties before and after mixing and pumping, methods of mixing the spoil and water, and the pumping of mixes of different water contents and size gradings round a 26-m-long, 125-mm-diameter pipeline at about 15 $m^3/h$ using a small concrete trailer pump. These trials showed that spoil could be mixed satisfactorily in a pan mixer to produce pumpable mixes with water contents in the range 13% to 38%, shear strengths of 1 to 6 kPa and consolidation of 4% to 20%.

## Large-Scale Pumping Tests

In the large scale trials, 1000 mt of colliery spoil was mixed with 30% to 45% water and pumped through pipelines of 125- and 260-mm diameter at rates of up to 80 $m^3/h$ and over distances of up to 220 m using standard construction equipment. Handling, mixing and pump and pipeline performance were examined, and the results provided the basis for the specification of a subsequent field trial at a mine site in the West Midlands.

## Spreading Tests

During these trials, small heaps were pumped out onto an approximately level surface, and measurements taken of the central height and horizontal spread of the heap, with penetrometer measurements to indicate the shear strength of the rock paste. At the end of the trials, two larger heaps (up to 100 $m^3$) were pumped out, one onto an approximately level road surface and one onto a sloping grass field. These experiments showed that the paste flowed over surfaces as a self-weight mass obeying the theory of plasticity, except when its undrained shear strength was exceptionally low (<0.3 kPa). Flow within the mine could be expected to be up to 60 m. Based on small-scale laboratory tests, it was estimated that typical pastes would consolidate <0.5 m with 90% consolidation in 2 to 12 yr.

Examination of the capacity of the paste to prevent roof collapse suggested that when the paste is in contact with the mine roof, even the weaker pastes would provide adequate support. However, as the gap increases to 0.5 m the weaker pastes would not provide support even with a large bulking of the collapsed rock. With a stronger paste or a smaller gap, the simple model used for analysis showed that collapse would terminate within about 2 m of the original roof.

## Trial Infill

A fully instrumented trial infilling of part of the Castlefields Mine in Dudley was carried out in 1985. This site was chosen because the workings were at shallow depth (18 to 24 m) and drained, enabling access to inspect the mine, install barriers to contain the rock paste, and install instruments to measure the progress of infilling.

About 6,600 $m^2$ of 4-m-high workings with an extraction ratio of about 80% were isolated by a reinforced concrete wall flanked by grouted gravel dams (where the roof condition was too dangerous to allow access). Colliery spoil, screened to minus 50 mm, was transported 35 km to the site by road, mixed with water extracted from the flooded part of the mine, and injected into the mine via two boreholes at a mean rate of 53 $m^3/h$. Over a period of 3 months, 29,100 $m^3$ of rock paste were placed, and the practicability of the method was successfully demonstrated. Rock paste within the mine behaved broadly as predicted, flowing around mine pillars and rising into chimneys in the mine roof.

## Modified Rock Paste

Postinfill monitoring of the Castlefields trial showed excessive consolidation (15%) of the rock paste within the mine, probably due in part to underdrainage due to a layer of sand (overflow from earlier infilling by sand flushing in an adjacent area) on the floor of the mine. The results also indicated that the long-term strength would not be sufficient to prevent surface collapse and that topping up of voids due to consolidation would be required.

To reduce consolidation and ensure reliable long-term strength, testing of rock paste mixes with various additives was carried out between 1985 and 1987. The results led to the development of a modified rock paste (Cole and Figg 1987) comprising colliery spoil with about 4% pulverized fuel ash (pfa) and 1.5% to 3.0% lime, depending on the ultimate strength requirements. For straightforward bulk infilling (i.e., with no requirement to provide lateral support to pillars), the design strength after one year ranges from about 10 to 20 kPa compared to the strength of 2 to 4 kPa required for the paste to be pumped and achieve maximum dispersion within the mine.

## Mine Infilling Using Modified Rock Paste

The concepts described were first applied to flooded workings at the Littleton Street Mine in Walsall. In 1986-88, 500,000 $m^3$ of 4- to 8-m-high room-and-pillar workings at 35 to 60 m below surface with a plan area of about 13 ha were filled with modified rock paste via only 13 injection holes at a cost of £6 million. The mixing process is shown diagrammatically in figure 4. Full monitoring of all materials and paste produced was undertaken, and the infilling process was continually monitored from the surface. Postinfill monitoring using piezocone tests (Braithwaite et al. 1988) confirmed that the 20-kPa design strength was achieved. The 600,000 mt of colliery spoil required was won from three derelict spoil heaps, which, on completion of operations, were restored for future amenity use.

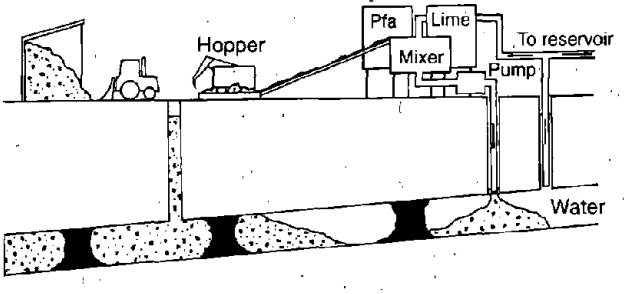


Figure 4. Modified rock paste infilling

Modified rock paste has also been used to infill 77,350 $m^3$ of the Castlefields Mine, Dudley (including topping up the adjacent trial infill area) in 1987-88 and 110,000 $m^3$ at the Arboretum and Moss Close Mines in Walsall in 1989-91. Together with the Littleton Street infilling, these works have allowed the removal of the threat of subsidence from about 30 ha of land variously used for residential, industrial, and commercial purposes.

## Pfa Rock Paste

Where the quantity of infill required has been too low to justify the cost of plant at source and infilling sites required for colliery spoil rock paste, a pfa rock paste with about 4% cement has been used. This method has been used successfully to infill small mines at Lincoln Hill (20,600 $m^3$), Ironbridge in 1987; at Bear Pit cavern and part of the East Castle Mine (58,000 $m^3$); Dudley, in 1988; at Broadway Cavern (94,000 $m^3$), Dudley, in 1989; and at the Cinder Hill Mines (20,000+ $m^3$), Wolverhampton, in 1992-93.

Pfa rock paste can also achieve far greater strengths, in excess of 1 MPa, and has been used in situations where pillars are over-stressed and support other than preventing roof falls is required. It has been used successfully at the Castlefields Mine (160,000+ $m^3$), Dudley, and three large contracts are currently in progress to infill the Wolverhampton Street Mine (360,000 $m^3$), Walsall, the Cow Pasture Mine (216,000 $m^3$); Sandwell, and the remainder of the Castlefields Mine (about 250,000 $m^3$).

With time, it has been found that colliery spoil is less readily available than was previously thought. It is also a very variable material, particularly in comparison with pfa, which is essentially homogeneous, and it is currently proving to be more expensive than pfa, in contrast to the position a few years ago. Pfa rock paste is, therefore, now the preferred material. Each infilling scheme is, however, assessed on its merits, and the choice of infill material requires a full evaluation of the scheme requirements and the availability and cost of materials.

# Other Methods of Reclamation

The rock paste infilling of limestone mines in the West Midlands has been the major focus of reclamation of underground mine workings in Britain. However, the problem of potential subsidence is widespread, and a number of other methods have been used in different areas. Many of these have been funded through the DLG program, whilst others have been funded by the landowner or developer.

## Grouting of Coal Mine Voids

As part of a study of mining subsidence (Statham and Treharne 1991), 39 grouting contracts carried out throughout the South Wales coalfield were examined. Site areas ranged from 250 to 70,000 $m^2$ with grout takes from nil to 29,000 mt (i.e., 0 to 20,700 $m^3$). Typical grout mixes were 9:1 or 10:1 pfa-cement with 5:4:1 pfa-sand-cement at the site perimeter and additional pea gravel when significant voids were encountered. Borehole spacing was largely on a 6-m primary grid with infill at 3-m centres but the primary grid ranged from 3 to 10 m and the infill grid from 1.5 to 5 m. Borehole depths were generally in the range of 10 to 30 m but some grouting was undertaken at depths as shallow as 5 m and some as deep as 60 m. Borehole diameter was generally 50 mm though occasionally 100-mm boreholes were used through superficial deposits.

There was little or no grout take in 11 of these contracts, raising serious doubts about the justification for grouting. In those contracts with significant grout takes, residual voidage was calculated to vary from <10% to almost 60%. Grout takes ranged from <1 to >100 mt per borehole, suggesting that collapsed (or back-stowed) stalls accepted very little grout, while roadways, which tend to remain open for many years after mining, were capable of accepting large quantities. This confirms the analysis of subsidence events referred to earlier.

## Hydraulic Sand Infilling

At Corbridge, Northumberland, galleries in a fireclay mine, typically 1.5 to 2.0 m wide but up to 3 to 4 m, underlay a site proposed for housing with about 0.5 m of intact limestone cover supporting about 6 m of quarry waste fill (Witherington 1988). Conventional grouting, excavation, and bulk replacement and supporting the houses on deep pile foundations were considered and rejected, and hydraulic sand infill was used to stabilize the land prior to development.

Grouted gravel barriers were used to seal the site and to provide additional support in areas with large roof spans, and the mine was divided into four zones by ungrouted gravel barriers. A uniformly graded fine sand byproduct resulting from coarse aggregate production at a local quarry was hydraulically flushed into the mine via 200-mm-diameter boreholes. Progress was checked by visual inspection, probing, and downhole video inspections, and piezometers were installed to check the drainage of the sand. On completion of sand filling, closely spaced check holes were grouted to infill any residual voids. The total cost was £20/m$^3$ compared with the estimate of £30/m$^3$ for conventional grouting techniques.

**Pneumatic Stowing of Sand Mines**

At Reigate, Surrey, lightly cemented sands of the Lower Greensand (Cretaceous) were mined by room and pillar methods in the 18th and 19th centuries. Two-metre-high rooms with a cover ranging from 2 to 8 m occur beneath roads, houses, and commercial properties in the town centre. In the light of the subsidence history in the town and following a stability survey, parts of the mine were infilled in 1987-88.

To preserve the stability of the surrounding sand, it was necessary to maintain the natural moisture levels of 10% to 12% and a free-draining granular material was specified to be placed by pneumatic stowing. Early trials with waste sand were unsatisfactory owing to clogging of placement hoses. Waste gravel (minus 20 mm) from a nearby fullers' earth quarry was used with success but could not be supplied at a sufficient rate. Finally crushed concrete from demolition work in London was successfully used. The investigation, infilling, and other treatment works (retaining structures, strengthening of pillars) were jointly funded by Reigate and Banstead District Council and the DOE using DLG at a cost of £500,000. Subsequently a wine bar located within the mines beneath a busy main road was acquired and filled.

At Pontefract, West Yorkshire, a previously unknown mine in Lower Permian Sands at 10 m below surface was investigated following subsidence damage to houses. It was found to extend beneath much of the housing estate. In 1991 a trial infill was carried out using pneumatic stowing of 10-mm clean limestone chippings down previously drilled 100-mm-diameter investigation boreholes. The delivery pipe could be rotated to ensure even filling, and up to 95% fill was achieved. 230 mt of material was stowed in 1 week at a cost of £10,000 using DLG funding.

The entire mine was subsequently filled by this method, using approximately 5,000 mt of limestone chippings, together with grouting of collapsed areas, at a cost of about £200,000 using DLG funding.

**Foamed Concrete Infilling of Chalk Mines**

In the City of Norwich, the collapse of a road beneath a double decker bus and other incidents led to the investigation and treatment of ancient chalk mines at shallow depth. Foamed concrete, a stable preformed foam mixed with sand, cement, and water, was used in preference to grout because it was fluid when pumped but also thixotropic, thus minimizing seepage losses into fissures in the chalk. At a cost of about £90,000, 762 m$^3$ of foamed concrete was injected. The work was funded jointly by the City Council and the DOE using DLG, with a small contribution from a commercial property owner whose building is above the mine.

**Bulk Excavation of Shallow Coal Mines**

In many parts of the coalfields, ancient workings are sufficiently near the surface that the most economical method of stabilization is to excavate down to the floor of the workings with controlled backfill to allow development to proceed. The value of the coal extracted is often sufficient to meet the costs of excavation and backfill. At Coatbridge, near Glasgow, this method has saved £4 million over conventional ground treatment techniques in the reclamation of 12 ha of derelict land. The £2.5 million Scottish Development Agency contract is being subsidised by coal sales of about £500,000.

## Stabilisation of Voids for Other Uses

A viable alternative to infilling in some cases has been the stabilization of underground voids to allow the underground space to be used. A number of show mines where coal, lead, copper, tin, and limestone were formerly mined have been so treated. In the West Midlands, the Singing Cavern, Castle Hill, Dudley, was stabilized using rock bolts and mesh, and new canals were constructed from the 18th century Dudley Canal to enable a popular tourist attraction to be added to the Black Country Museum. Across the road at Dudley Zoo, a similar walk-in attraction is being created.

## Proposals for Treatment of Bath Stone Mines

At Combe Down, in the City of Bath, about 400,000 $m^3$ of room and pillar workings extend to some 16 ha beneath residential areas at 4 to 8 m below the surface. Following a survey and hazard assessment, a proposal to infill these workings with pfa faced objections from the local community because of the disruption it would cause, the perceived threat to ground water from using pfa, the damage to ecologically valuable open space above the mine if used as a plant site, the destruction of the industrial archaeological heritage, and the threat to a significant colony of Greater Horseshoe Bats, which use the mine for roosting and hibernation; bats are a protected species under the Wildlife and Countryside Act 1981.

Consequently, a feasibility study and environmental assessment were commissioned by Bath City Council (50% funded by DLG). Subject to the environmental assessment, it recommends that 220,000 $m^3$ (11 ha in plan area) need to be infilled; structural support using fiber-reinforced shotcrete sprayed in place by machine, supplemented in some areas by grouting roof cavities and dowelling, would be feasible over about 3 ha; and, subject to monitoring and any necessary remedial action, a low-risk area of about 2 ha beneath the open space area would require only localized work. This strategy would reduce the impact of the works, particularly with respect to the disruption to the local community and the preservation of the industrial archaeological heritage. Leaving parts of the mine open would also allow the bats to continue to use the mine and would be cheaper than a total infill solution. The estimated cost is £20 million if pfa is used and £29 million for a non-pfa infill, compared with £21 million and £38 million for a total infill operation.

## Summary and Conclusions

Britain's long history of mining has led to a widespread legacy of potential subsidence. Legislation enacted in 1980 has enabled the reclamation of underground mines to progress from the individual development site or subsidence event basis to the treatment of mines on a wider scale to remove the threat of subsidence through the Derelict Land Grant program.

The major effort has focused on old limestone mines in the West Midlands, where the scale and potential threat are large, and it has involved the development of new methods of infilling using rock paste based on colliery spoil or pfa as opposed to conventional grouting techniques. These methods have enabled waste materials to be injected into mines through widely spaced boreholes to remove the threat of subsidence beneath existing development and areas for redevelopment. Where the scale of workings is smaller, both in the West Midlands and elsewhere, a number of other techniques have been used successfully.

The DLG program has been administered to date by the Department of the Environment. In 1994, however, administration will pass to the newly formed Urban Regeneration Agency, now known as English Partnerships. This agency will aim to promote development by regenerating land and buildings, to create employment through local enterprise and inward investment, and to improve the environment. It will honour the DLG commitments it inherits and ensure that these projects are completed properly, and it is expected to maintain a commitment to the reclamation of underground mine workings.

## Acknowledgments

This paper is based on the work of many people in the DOE, local authorities, and consultant and contractor organisations through the Department's research and DLG programs. Gratitude is due to all these people. The views expressed, however, are those of the author alone and cannot be taken as representing the views of the Department of the Environment or any other organisation.

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# SOME APPROACHES TO DETERMINE THE POTENTIAL INFLUENCE OF LONGWALL MINING ON GROUND WATER RESOURCES[1]

Derek Elsworth, Jishan Liu, and Zhihua Ouyang[2]

**Abstract:** An approach is developed to quantify the impact of longwall mining on the potential disruption of ground water resources. The underlying premise of the method is to assume that the primary mechanism causing dewatering is the development of new fractures, or the dilation of existing fractures, as a result of mining-induced displacements. Numerical simulation, using the finite-element method, is used both to determine the mining-induced distribution in strain and to complete hydrologic budget calculations with the revised permeability field. This approach is applied to mining geometries representative of the Appalachian Plateau to define the anticipated sensitivity of these systems to changes in overburden thickness, base level location, topography, and other relevant parameters. Results indicate that observed trends in dewatering behavior may be explained on the basis of zones of contiguous extensile strain, induced within the overburden as a result of mining. The systems are more sensitive to the sense of mining-induced strain (extensile versus compressional) than to the absolute magnitude. This is due to the extreme sensitivity of permeability to even relatively modest changes in extensile strain. Correspondingly, observational rules developed to predict the extent and location of potential aquifer dewatering are confirmed and may be extended to evaluate the potential long term influence of mining on the ground water system.

**Additional Key Words:** subsidence, aquifer dewatering, numerical modeling.

## Introduction

Determination of the probable impact on ground water supplies is of key importance in the successful development of a longwall mine. The principal effect is that of dewatering, where disruption of the strata overlying the seam results in the development of hydraulic connections that had not existed prior to mining. The severity of failure, together with its spatial and volumetric extent, regulates the potential impact on ground water resources. The development of fracturing may result in the depressing of the existing ground water surface, which may remain depressed or may rebound with time. Where aquifers are used for potable or industrial water resources, this may result in the dewatering of wells, or in the degradation of water quality with the development of new hydraulic connections and the renewed mobility of oxygen within the subsurface. A considerable body of observational evidence has been compiled (eg. Leavitt and Gibbens 1992) to define the potential severity and extent of ground water influence in association with longwall mining. Analysis of this observational data set typically incorporates the indices of mining depth and the location of valleys to describe the possibility of dewatering and subsequent potential for recovery. Our approach, in the following, is to represent the phenomenological behavior of the system to reproduce these observed effects and enable extrapolation of the results both to nonstandard mining situations and in the evaluation of long-term effects.

## Method of Analysis

Observational evidence indicates the form of potential dewatering above mined panels to be quite localized and limited to zones of induced extensional strain. Since the primary conduits for ground water flow in the sedimentary rocks of the Appalachian Plateau are the secondary fractures, hydraulic conductivities are strongly influenced by mining-induced extensional displacements. Our approach is to use numerical models to define the anticipated influence of mining on the ground water system through the following straightforward steps:

- The strain field that develops around a longwall panel as a result of mining is defined using a finite-element model that accommodates the influence of material failure and self-weight.

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[1] Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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- From this predicted strain field and from knowledge of the premining hydraulic properties of the overlying strata, the change in hydraulic conductivity that results from the strain field may be determined.
- With the modified conductivity field determined, the postmining hydrologic system may subsequently be defined through application of a ground water model. Again, this ground water model utilizes the finite-element method to determine the postmining hydrologic system where the position of the piezometric surface indicates changes in well or aquifer yields.

## Determination of the Strain Field

The subsidence field that develops around a longwall panel may be determined directly from charts available in the Subsidence Engineers' Handbook (National Coal Board 1966). These charts relate the subsidence and surface strain profiles to parameters describing, mining depth, $h$, mining width, $w$, and seam thickness, $t$, alone, as illustrated in figure 1. The insensitivity of the resulting subsidence profile to the material properties of deformation modulus and strength results from the overriding influence of geometric controls on deformation. Following mining, the panel span is sufficiently large that closure between panel floor and roof is unavoidable. Consequently, the resulting strain field, $\epsilon_x, \epsilon_y$, is uniquely defined in terms of the parameters,

$$\epsilon_x, \epsilon_y = f\left[\frac{w}{t}, \frac{w}{h}\right] \tag{1}$$

where the influence of topography is also included in the term, $w/h$. Consequently, the strain field may be uniquely defined given that these geometric parameters are known, a priori, and the analysis is now completely decoupled from the material parameters describing the overlying strata. This lack of sensitivity has important considerations in reducing to a minimum the data required to complete an appraisal of the subsidence effect.

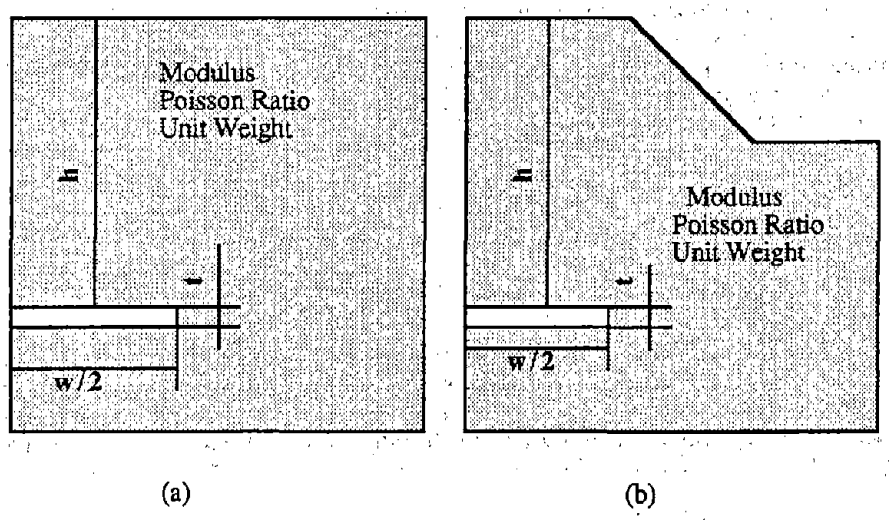


Figure 1. Idealized mining geometry relating seam thickness, $t$, and panel width, $w$, to the overburden thickness, $h$. Of importance is the location of the panel with respect to the topographic features for the (a) horizontal ground surface, and for (b) hillslope and valley base setting.

## Determination of Conductivity Changes

With the strain distribution determined throughout the surrounding medium, changes to the permeability distribution may also be defined, provided an appropriate model is accommodated. A reasonable model is one that assumes the primary influence on hydraulic conductivity is provided by the secondary porosity, or fractures. The hydraulic conductivity, $K$, of a set of fractures of aperture $b$ and spacing $s$ may be defined as

$$K=\frac{gb^3}{12\nu s}, \tag{2}$$

where $g$ is gravitational acceleration and $\nu$ is dynamic viscosity of the fluid. If it is assumed that a uniform strain field is applied to the fracture and matrix system, then the directional hydraulic conductivities are modified as

$$K_x=\frac{g}{12\nu s}[b+[s(1-R_m)+b]\varepsilon_y]^3 \tag{3a}$$

and

$$K_y=\frac{g}{12\nu s}[b+[s(1-R_m)+b]\varepsilon_x]^3, \tag{3b}$$

where the added factor, $R_m$, represents a modulus reduction ratio (mass modulus to intact modulus) that apportions the changes in strain between the fracture and matrix material. When $R_m=1$, the mass and intact material moduli are identical and the strain is uniformly distributed between fractures and matrix. This results in the smallest extensional change in conductivity. When $R_m=0$, the extensional strain is applied entirely to the fracture system and precipitates the largest possible change in conductivity. These values bound the behavior of the system. This representation of conductivities is extremely useful, since the modulus-reduction factor, $R_m$, may be readily evaluated from rock mass classifications (Ouyang and Elsworth 1993) defining structural behavior as a function of readily observable factors of rock structure. This avoids the difficulty, of defining conductivity enhancement in terms of the component moduli of fractures and matrix, parameters that are unlikely to be available in practice.

If the premining fracture spacing and conductivity distribution are known, or may be estimated, a priori, then an equivalent hydraulic aperture may be determined directly from equation 2. This initial aperture may be substituted directly into equations 3 to define the modified conductivities for use in the subsequent analysis.

The assumption necessary in this evaluation is that strains are uniformly distributed at the scale of a single element. These assumptions seem reasonable where strains are moderate, but may be questionable where significant strain localization occurs. The potential to represent this critical behavior is presently under evaluation.

## Determination of Postmining Ground Water Regime

With the modified conductivity distribution determined from an evaluation of the strain field, and equations 3, the influence on the postmining ground water regime may be evaluated. A forward finite element model is applied that accounts for the influence of a continuously distributed conductivity field, defined as a result of the previous analysis. With boundary conditions applied to represent ground water and surface recharge, the change in elevation of the phreatic surface may be defined for the postmining regime. This enables the influence of mining on well yields, aquifer yields, and flow patterns to be identified.

Some examples of application of this form of analysis are outlined in the following paragraphs.

## Ground Water Flux Charts

The basic dimensionless variables that identify geometry and material parameters may be used to describe the behavior of the system using a minimum set of nondimensional variables. Using the hierarchy that deformation controls flow through the distribution of body strains, it is possible to define flow rates, $Q$, in terms of the functional relations

$$\frac{Q\nu\gamma}{gs^2E_m}(\frac{s}{b})^3=f\,[\frac{b}{s};R_m;\frac{w}{t};\frac{E_m}{\gamma h}], \tag{4}$$

where $\gamma$ is the unit weight of rock, $E_m$ is the mass modulus, and all other parameters are as previously defined. To further simplify the analysis, the ratio $b/s$ may be sensibly linked to the modulus reduction ratio, $R_m$, enabling a single chart to describe behavior, as illustrated in figure 2. These may be used to define flux volumes into excavations that result from mining.

## Topographic Controls on Behavior

Observational evidence within the Appalachian coalfields supports the contention that topography exerts a significant influence on the potential for dewatering (Leavitt and Gibbens 1992, Donohue 1993). This dewatering may be short lived, or it may be longterm. Water supplies located in valley bases are less susceptible to interruption than those located at hilltop, and shallow wells are less likely to be influenced than deeper wells. These observations may be explained relative to the prevalent mechanisms of conductivity enhancement that develop within the overburden strata in response to mining-induced displacements. This treatment is pursued in the following paragraphs.

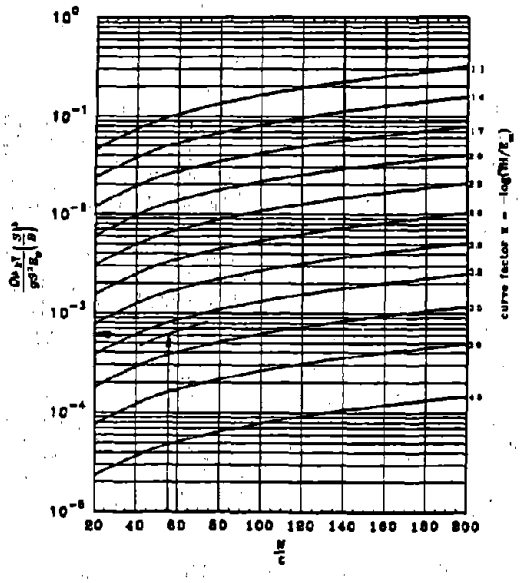


Figure 2. Design chart representing the variation in dimensionless flow rate, $Q_d$, as a function of mining width to seam thickness ratio, $W/t$. Parameters describing the system are unit weight of rock, $\gamma$, fracture spacing, $s$, initial fracture aperture, $b$, and rock mass modulus, $E$.

## Model Assumptions

The mining-induced strain field is evaluated through use of a nonlinear finite element model developed to represent large displacements in strain softening strata. The nonlinearity is realized through changing the elastic constants for overburden elements subject to extension and in-panel elements subject to full compression and closure. These behavioral models are based on stress and strain criteria, respectively. The model has been carefully validated against subsidence profiles for Appalachian coalfields (Liu 1993), and in a more generic sense against empirical models for subsidence [National Coal Board 1966].

Because the processes representing the failure and subsequent deformation of the overburden are accurately represented, the resulting subsidence profiles are relatively insensitive to the material parameters of strength and deformation modulus. Instead, the constraints on subsidence, as applied through the maximum convergence between roof and floor, result in a small range of feasible displacement fields. Under this constraint, the induced strain field may be evaluated without need for complex material parameters and still represent a reliable depiction of the true situation. The model has been extensively tested and calibrated

against a variety of available data (Liu 1993).

The form of the mining-modified conductivity field is determined through the unique relationship linking induced strains and hydraulic conductivities, as represented by equations 3. It is important to note that this modified conductivity field may develop both extreme heterogeneity and anisotropy, as conditioned by the induced strain field. This modified conductivity field may then be used to define the potential for strata dewatering as conditioned by the topographic setting. This behavior is evaluated in the following paragraphs.

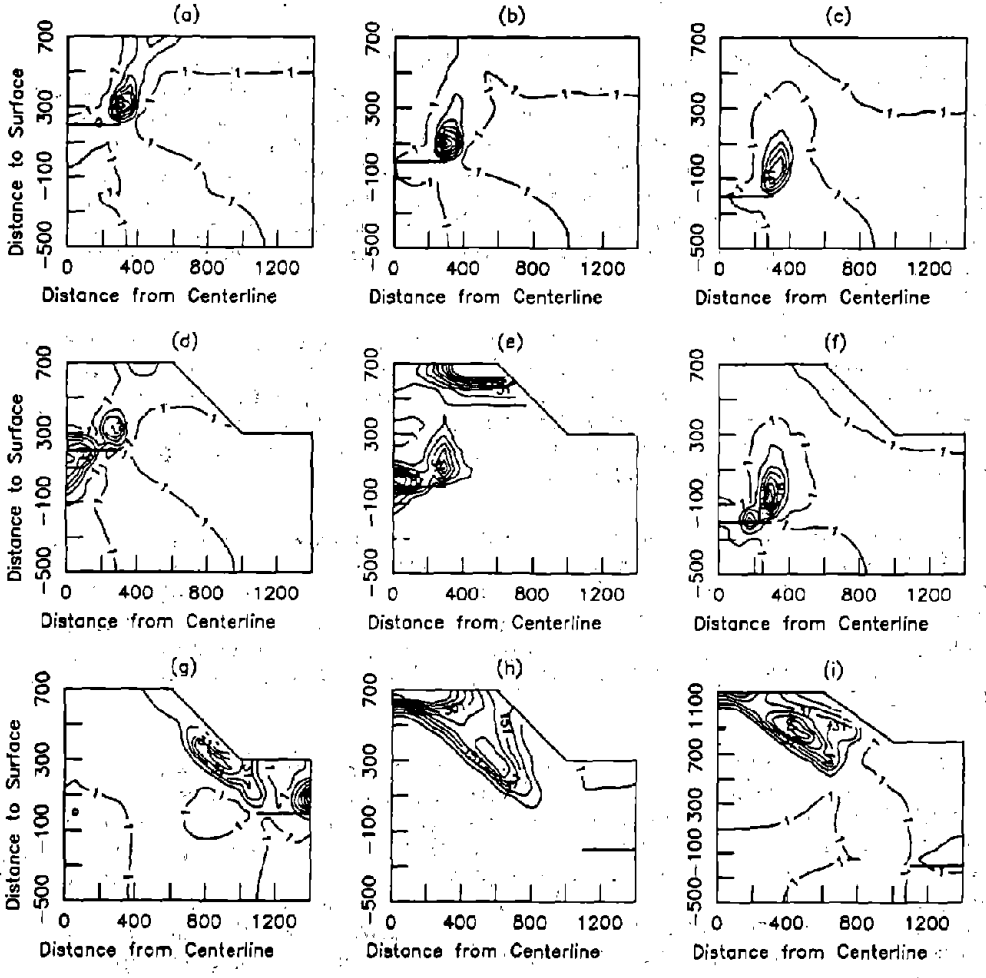


Figure 3. Transverse vertical sections showing contours of the ratio of postmining vertical hydraulic conductivity to premining hydraulic conductivity for mining at depths of 500 to 900 ft in topographic environments representing plateau (top), subhilltop (center), and subvalley (base). The null influence contour (representing zero strain) is represented by a ratio of unity. Ratios greater than unity represent mining-induced conductivity enhancement. Horizontal and vertical axes are marked in ft.

## Parametric Results

To investigate the impacts of topography on domestic water supplies, the influence of mining in three topographic settings, namely, plateau, subhilltop, and subvalley settings, is evaluated using three different mining depths for each of the idealized geometries. The plateau setting comprises a horizontal ground surface, with the subhilltop and subvalley settings exhibiting the topography of figure 1. The changes in hydraulic conductivities generated by subsidence of the overburden are evaluated. Since change in water supply will be most influenced where conductivities are increased, the most obvious index to define zones of potential dewatering is the ratio of postmining to premining hydraulic conductivity. These ratios are reported for vertical and horizontal conductivities in the nine distinct mining geometries in figures 3 and 4, respectively. The following conclusions may be drawn from these results.

According to figure 3, the enhancement in vertical hydraulic conductivityis limited to a narrow region around the panel for the plateau mining settings. This zone appears coincident with the area of high lateral extensional stresses that develop at the abutment and slope upwards at approximately the angle of draw. Where mining is shallow, the zone may extend to, or close to, the surface. As the mining depth increases, the region of vertical conductivity enhancement becomes encapsulated in a zone of reduced conductivity. Where topographic influences are included, this zone may spread into the near-surface regions of the surface layer for the subhilltop mining settings, or into upslope areas for the subvalley mining settings. The zone of vertical hydraulic conductivity enhancement may extend up to 400 ft above the mined coal seam for both plateau and subhilltop mining settings.

For subhilltop mining settings, a zone of vertical hydraulic conductivity enhancement may develop within the surface layer in addition to that around the panel. The zone of vertical hydraulic conductivity enhancement shifts into upslope areas for the subvalley settings. The influence of topography diminishes for subhilltop settings as the depth of the overburden increases. This critical depth may be about 900 ft. However, topographic influence still plays a dominant role for the subvalley mining settings even as the depth of overburden exceeds 1,000 ft below the valley surface.

The lateral extent of the zone of vertical hydraulic conductivity enhancement is relatively limited and is typically confined within 200 ft of the panel edge. This is true for the plateau and subhilltop mining settings alike. The other zone of vertical conductivity enhancement may extend up to 400 ft within the surface layer from the edge of the coal seam for subhilltop mining settings. For subvalley mining settings, the lateral extent of the zone of vertical conductivity enhancement extends farthest from the mine panel edge. The zone extends along the surface layer, especially in the upslope direction. This feature appears to develop as a result of, and is controlled by, the topography.

The zones of mining-induced horizontal conductivity enhancement are illustrated in figure 4. Conductivity enhancement develops both around the panel and within the surface layer for the plateau mining settings. For subhilltop mining settings, this zone forms only within the surface layer when the overburden depth is less than 900 ft, with deep conductivity enhancement developing immediately above the panel. An auxiliary zone forms within the surface layer when the overburden depth is greater than 900 ft. For subvalley mining settings, these zones are formed only (1) around the panel when the overburden depth is greater than 500 ft or (2) within the surface layer when the overburden depth is less than 500 ft. The vertical extent of horizontal conductivity enhancement may be less than 200 ft for all the mining settings.

## Conclusions

According to these model results, the distributions of the hydraulic conductivity enhancement zones are quite different for the three mining settings. These differences appear to be controlled by the combination of both mining geometry and topography. Two parameters may be recognized as the most important, namely, the elevation difference between the well base and the mine panel, and the overburden depth, for both the plateau and subhilltop mining settings. For subvalley mining settings, one parameter may be recognized as the most important, namely, the depth of the overburden. These three parameters can be used to evaluate the potential for strata dewatering. Each parameter exhibits a critical value which can be used in evaluating the potential for strata dewatering. For subhilltop mining situations, the critical overburden depth is about 900 ft. For

depths of overburden greater than 900 ft, the influence of topography on the potential for dewatering diminishes, as would be anticipated. The critical value of mine-to-well elevation separation is about 500 ft for both plateau and subhilltop settings. As defined previously, if the value is more than 500 ft the well is defined as a shallow well, and otherwise as a deep well. These two situations exhibit different responses to longwall mining. For the subvalley mining settings, the critical value for the overburden depth is about 500 ft. If the depth of overburden is greater than 500 ft, wells completed within the valley may be recharged by ground water discharge from upslope areas; otherwise, it may possibly be dewatered due to longwall mining.

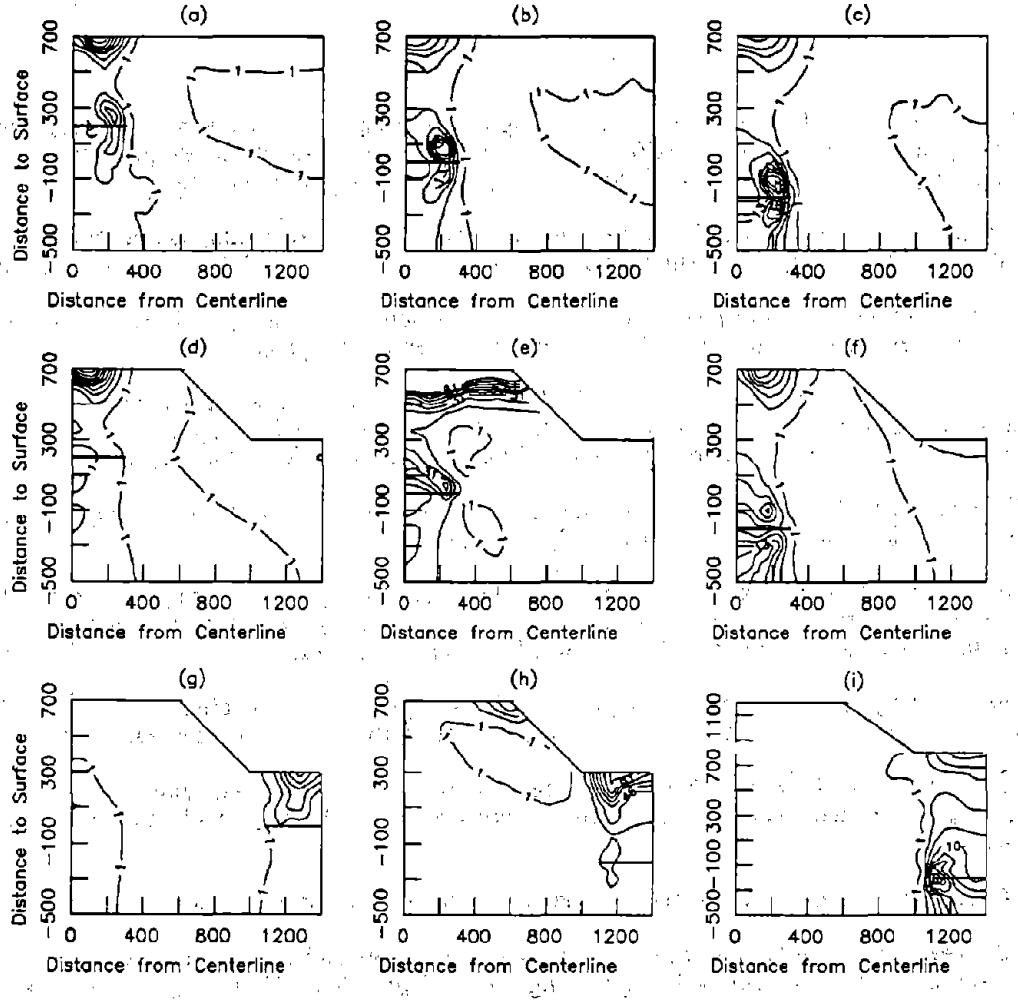


Figure 4. Transverse vertical sections showing contours of the ratio of postmining horizontal hydraulic conductivity to premining hydraulic conductivity for mining at depths of 500 to 900 ft in topographic environments representing plateau (top), subhilltop (center), and subvalley (base). The null influence contour (representing zero strain) is represented by a ratio of unity. Ratios greater than unity represent mining-induced conductivity enhancement. Horizontal and vertical axes are marked in ft.

These results appear in general agreement with observations and analyses reported for the Appalachian coalfields (Ciffeli and Rausch 1986, Hasenfus et al. 1990, Leavitt and Gibbens 1992, Matetic and Trevits 1992, Donohue 1993).

## Acknowledgements

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# EVALUATION OF THE LONG TERM IMPACT ON DOMESTIC AND FARM GROUNDWATER SUPPLIES UNDER PENNSYLVANIA LONGWALL MINING CONDITIONS[1]

Timothy D.A. Donohue[2] and Richard R. Parizek[3]

**Abstract:** Longwall coal mining is a dynamic method of coal extraction that disrupts overlying geologic strata and develops zones of increased permeability. This study was undertaken to determine what long-term effects longwall mining has on overlying domestic and farm water supplies under thick overburden conditions in Pennsylvania. The study region was comprised of the Greene and Washington County area in the southwestern corner of the State where overburden ranges from 700 to 900 ft thick. Investigative methods included analysis of coal company water level monitoring data, interviews with company representatives and State officials, and field contacts with concerned property owners. A new comparative approach for analyzing water level data led to these general conclusions: Hilltop and hillside topographic settings tended to lose water supplies, while valley settings gained new ones in the form of springs and increased water well levels. Deeper drilled wells found in higher topographic settings lost water, while shallower ones in the valley settings remained reasonably unaffected. Three parameters, topographic setting, well depth, and overburden thickness, are recognized as the most important aspects for whether a water supply will be affected by longwall mining under these overburden conditions. Though water supplies were affected by mining, coal companies in the area exhibited good success in finding replacement water supplies by drilling new, deeper wells nearby for those affected property owners.

**Additional Key Words:** wells, springs, coal mining, control wells, topographic position, overburden thickness

## Introduction

The increased use of longwall mining has resulted in a growing opposition as environmental consequences are feared or realized. Objections often cited to longwall mining and its potential adverse environmental influences include (1) land subsidence, (2) the fracturing and differential subsidence of overburden strata that enhances its vertical permeability;, (3) the loss of springs and other shallow groundwater supplies, (4) the capture of surface water by vertical fractures that extend to the land surface, (5) loss of soil moisture relied upon for crop production, (6) changes in surface drainage, (7) undesirable ponding of surface water, (8) damage to utility lines, transportation networks, buildings, etc., (9) releases of methane gas, (10) enhanced acid mine drainage production and other water quality changes, and (11) uncontrolled release of mine water following closure and similar other concerns.

Longwall mining has caused an increase in public concern in coal mining districts throughout the country. This increasing concern has led to, among other things, legislative efforts to regulate and control longwall mining impacts or to eliminate longwall mining entirely. This project was undertaken in response to the emergent need to better understand one aspect of the problem: the hydrogeological impacts of longwall mining in the southwestern portion of the Commonwealth of Pennsylvania. In particular, there is a need to address questions concerning the long-term post-longwall mining impacts on domestic and farm ground water supplies under the complete range of overburden conditions present in various mining districts.

Noteworthy studies that address aspects of the longwall mining in the Eastern U.S. shall include Nutter et al. (1980), Stoner (1983), Cifelli and Rauch (1986), Bruhn (1986), Hiorfdahl (1987), Johnson (1991), Trevits and Matetic (1991), and Leavitt and Gibbens (1992).

## Objective

The objective of this project was to develop a method to discriminate between the effects of seasonal weather and droughts on domestic and farm well and spring yields and ground water levels from changes induced by longwall mining. This study was to consider site-specific data under thick overburden conditions in Washington and Greene

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[1] Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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Counties, PA (fig. 1).

A major portion of this project included analysis of monitoring data made available by three coal companies. All data presented are also contained in mine permit files, which have public access, at the Department of Environmental Resources Office, McMurray, PA with the exception of personal communications.

## Geologic and Hydrogeologic Setting of Site Location

Greene and Washington Counties are located in the extreme southwestern corner of Pennsylvania and border West Virginia on two sides. As part of the Appalachian coal basin (Eastern Coal Province), the two-county area, especially Greene County, contains much of the nation's high-volatile bituminous coal reserves. These counties cover a very rugged section of the Allegheny Plateau. Topography of the area is hilly with broad rounded ridges and intervening valleys. Stream erosion has created a complexly dissected area, having as much as 620 ft of relief between hilltops and valley bottoms with almost 90 % of Greene County considered nondevelopable owing to slopes in excess of 15 % (Stoner et al 1987).

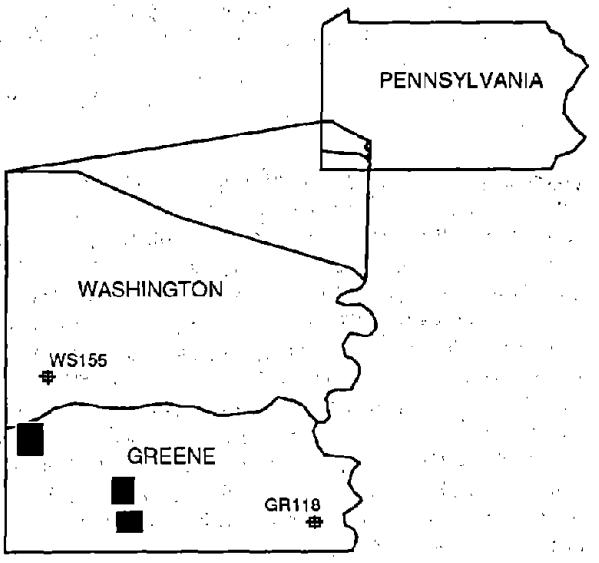


Fig.1 Location of study area, mines, and control wells.

Overburden strata considered for this project consist of stratigraphic units overlying the Pittsburgh coalbed, which is the most valuable coal in the two-county area. It consists of two to four benches separated by clay or shale partings; the lower, or main, bench ranges from 36 to 104 in. in thickness, including clay partings up to 1 in. thick (Stoner et al 1987). Bedrock in the study region consists of interbedded sandstone, siltstone, shale (or mudstone), limestone, and coal. Sandstone and limestone form the thickest lithologic units, exceeding 90 ft in some areas (Stoner 1983). Carbonate units are more prevalent in the lower section of the overburden strata, while sandstone becomes more dominant in the upper portion of the strata.

Water resources for the study area are comprised dominantly of shallow wells and springs. The hydrogeologic characteristics of the overburden strata reflect relatively low hydraulic conductivities and secondary permeabilities in the form of fractures important for sufficient domestic well yields. Fracture flow within bedrock coupled with quaternary alluvium comprises a shallow aquifer system adequate for domestic and farm ground water supplies (Newport, 1973).

## Water Well Data Collection

Communication with and cooperation of coal companies in the study area was integral for substantial conclusions to be realized. A random suite of monitoring well sites was selected early in the study from data sets made available by three coal companies. This data acquisition process reduced chance for bias. Owing to time considerations and property owners' fear of jeopardizing their settlements with the coal companies, the study was broken into two parts: analysis of coal company monitoring data, and field contacts made with both company personnel and home owners.

Company data included premining and postmining monitoring records of both wells and springs. In the case of wells, sampling points were either preexisting domestic wells or company-drilled monitoring wells. Data supplied for a property owner's site included water level, yield, and quality before, during, and after longwall mining took place within close vicinity of the well or spring as well as rate of face advance and position of face during mining. Company-drilled monitoring wells provided data during the operational life of each well, which can include data long after mining occurs nearby. This allows for more extensive postmining data to be collected.

Effort was made to contact as many property owners as possible whom would have likely been affected by mining in the study area. Most of the property owners were glad to talk about their situation and about what they may have experienced since mining had occurred in their vicinity but were hesitant about specific documentation of their problem and allowing field measurements. Their concern rested on whether such documentation might in some way jeopardize their situation with the coal companies regarding settlement of claims to repair wells or replace wells and water supplies. Hence, these more general observations and site-specific data are omitted from this report.

Nine wells and 18 springs comprised the monitoring database. An average data set included 2 yrs of monthly measurements for water levels, yields, and quality. Only water level and yield data are considered here. After a database was established, parameters examined for each water resource during analysis included its classification by topographical setting, position of resource above longwall panel, overburden thickness, timing when panel passed and/or undermined, and depth of well.

Preliminary analysis of yield and water level monitoring data indicated that significant fluctuations in static head values occurred when mining was in the vicinity of a well location. However, large fluctuations in precipitation for the area, most noticeably the drought that began in the spring of 1991, also seemed to coincide with head changes. Since all wells in question tapped shallow water-table aquifers, static heads responded to the natural variability of seasonal changes in precipitation and ground water recharge. The problem was finding some way to filter out the effects of variable precipitation from well fluctuations to determine if longwall mining was altering performance of wells. A time series method of analysis was not possible given the limited data set.

The analysis procedure devised, required use of control wells located in areas not impacted by mining or other human influences. Such wells should be constructed similar to local domestic wells which have 15-20 feet of surface casing followed by an open borehole to total depth. They should also have static heads that fluctuate only according to changes in precipitation and natural ground water recharge and discharge. Such wells have been established and are monitored by the U.S. Geological Survey with one well in each county of Pennsylvania. The wells for Greene (GR118) and Washington (WS155) Counties have provided daily static head measurements since the early 1970's (table 1). These control wells were ideal for comparison and proper analysis of on-mine wells. By comparing the static head of the monitoring well with that of the control well for each particular month, it was anticipated that the data would reveal the effect of mining.

Table 1. USGS control well specifications (Water Resources Data - PA).

| | USGS Well GR118 | USGS Well WS155 |
|---|---|---|
| Location | Greene County, PA | Washington County, PA |
| Latitude | 39°46'55" | 40°02'33" |
| Longitude | 80°01'43" | 80°26'13" |
| Local Site | State Game Land #223 | State Game Land #245 |
| Aquifer | Sh & SS of lower member of Waynesburg Formation | Washington Formation |
| Well characteristics | diam. 6 in, depth 104 ft, cased to 22 ft, open hole. | diam. 6 in, depth 160 ft, cased to 19 ft, open hole |

## Normalizing Control Well Data

It was necessary to devise an appropriate way of comparing water level responses within control wells remote from longwall mines with those in the monitoring wells located near or above longwall panels. Each control well is shallow, and its water level fluctuates according to water-table aquifer conditions. This compares favorably with water levels within shallow monitor wells located next to longwall panels. Because the control wells are completed in different formations, it is inevitable that each would react somewhat differently according to varying hydrogeological characteristics that dominate ground water occurrence and recharge at a particular site. Therefore, the maximum depth to water or lowest static head of each month for each control well was singled out. The mean of the lowest, monthly static heads for the two control wells was then computed. This allowed for a mean control head value to be determined for each month that represented an average of hydrogeological conditions and maximum fluctuations due to varying precipitation independent of mining. Because the mean control head is based on minimum static heads for each control well, a conservative approach to the effects of mining is taken. This approach allows for a maximum effect of precipitation to be filtered from the monitoring data, hence its conservative stance.

The next step was to combine the static heads of the monitoring wells with those of the mean control. After various attempts, a unique approach was found by plotting the static head of the monitoring well versus a ratio: {static head divided by the control mean value for the corresponding month of sampling}. This was applied for each subsequent month of available water level data up until April 1992, the extent of U.S. Geological Survey data and available monitoring data at time of this writing. By plotting data using this approach, a scatter plot was produced, which in turn was analyzed using a best-fit linear regression approach.

An average slope value of 38 was obtained for the control wells. This allows one to compare monitoring well data for longwall panel hydrogeological settings to this control well value. Logically, a steep decline in monitoring head with

little change in the control head will produce a larger ratio, which will transpose into a steeper slope. Therefore, monitoring data plots that have a slope during and after mining that is significantly above 38 are considered to be degradationally affected by mining (fig. 2). Likewise, premining and quite possibly postmining data displays that have slopes below or at 38 are said to be unaffected and quite possibly enhanced by mining.

## Analysis of Spring Data

Monitoring data for springs were analyzed using the same control mean technique. Spring yield fluctuations behaved similarly to hydraulic head variations for monitoring and control wells. Using the same graphical analysis, monthly spring yield was plotted versus a ratio of spring yield divided by control mean value for the corresponding month. After applying best-fit lines to the scatter plot, a slope comparison analysis was used to determine which springs were affected by longwall mining. This slope analysis procedure for springs is similar to the well analysis procedure except that the average slope for an unaffected spring is around 35, instead of 38 as described above. As in the well analysis, this technique proved to be extremely useful for filtering out the effects of variations in precipitation on spring yields.

Since changes in slope for spring data are dominated by higher yield changes, it is also helpful to plot spring yield values along with control well data versus the monitoring period to look at transient changes in spring yields. This additional analysis provided further insight into how spring yields varied according to precipitation changes and mining impacts.

## Results

Several trends in the monitoring data appeared after applying the control mean method of analysis described above. These data observations reveal that water level responses within some wells and springs were subjected to external forces other than precipitation effects. Longwall coal mining is recognized as the cause of these degradational changes for water levels within domestic and farm wells and springs. No other wells were known to have been drilled nearby that might have caused these well interferences.

### Wells

Of the many field variables that were analyzed in determining changes in well performance, three specific ones relate efficiently to data trends: depth of well, topographic position of well, and location of well above panel workings. This is in agreement with observations made by other investigators.

**Depth of Well.** A clear relationship exists between the depth of a well and the degree to which it is affected by mining. As shown in table 2, shallow wells remain relatively unaffected up to about 50 ft deep. Then, deeper wells show

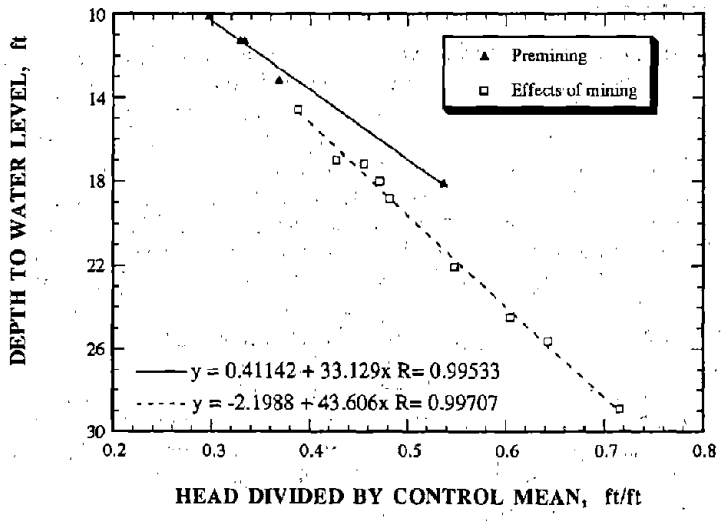


Fig. 2 Control mean evaluation of well 179-W1 affected by longwall mining.

Table 2. Water well construction, topographic setting, location with respect to longwall panel, and response to mining.

| Site[1] | Slope of best-fit Premining | Slope of best-fit Postmining | Impact of mining | Well depth, ft | Topo setting | Elevation, ft | Topo position, ft | Overburden thickness, ft | Position in panel[2] |
|---|---|---|---|---|---|---|---|---|---|
| 214-W1 | 36.33 | 51.1 | Affected | 105 | hilltop | 1324 | 119 | 615 | ge |
| 88-W1 | 36.78 | 41.26 | Affected | 75 | hillside | 1038 | 3 | 416 | 219 ft inside |
| 179-W1 | 33.13 | 43.61 | Affected | 62 | hillside | 1140 | -22 | 549 | ge |
| 57-W1 | 29.12 | 41.87 | Affected | 66 | valley | 1068 | 42 | 451 | 219 ft inside[3] |
| 58-W1 | 46.6[4] | 32.44 | Affected | 76 | hilltop | 1141 | 105 | 529 | 188 ft inside[3] |
| 88-W3 | NC[5] | 45.15 | Affected | 65 | valley | 992 | -33 | 440 | 156 ft inside[3] |
| 56-W1 | 32.44 | NC | Unaffected | 50 | valley | 1017 | 7 | 465 | 344 ft inside |
| 56-W2 | 39.70 | 35.61 | Unaffected | 17 | valley | 1018 | 41 | 498 | 281 ft inside |
| 88-W2 | 38.60 | 39.11 | Unaffected | 10 | valley | 996 | 26 | 498 | 125 ft inside |

[1] W = well
[2] ge : gate entry  inside: located inside mined panel
[3] Located above mined-around block.
[4] High number indicative of mining in adjacent panel.
[5] NC=No change in data trend.

NOTE: all elevations given as feet above sea level.
Casing lengths unknown, generally avg. 20 ft of surface casing.
Topo position = elevation of resource - well depth - base level elevation.
Overburden thickness = elevation of resource - well depth - coal bed elevation.
Base level : elevation of local drainage level.

considerable degradational changes, with the greatest change occurring in the deepest well, 214-W1. A general observation can be made: The deeper a well is, the more likely it is to be affected by longwall coal mining practices. Because most water production is obtained from fracture-borehole intersections, a deeper well is more likely to be affected by increased fracture development in both aquifer and aquiclude strata. Deeper fractures are more likely to be interconnected with mine voids that serve as a free drain. This would cause fractures at the base of the well to act as drains for water cascading into the well from above but that is free to flow out of lower fractures that are interconnected to the underlying mine or mine-subsidence-affected strata. The mine acts as a free drain as long as the mine workings are being dewatered. Fluid potentials in shallow soil and rock are higher than at greater depths as long as the mine subsided strata are being dewatered. Ground water flows toward the mine in response to this head gradient. A drop in fluid potentials is to be expected as monitoring and water supply wells are drilled and cased deeper in the direction of ground water flow. This drop in head will continue until fractures directly interconnected to the mine opening are capable of transmitting ground water into the mine at a rate faster than can be supplied by the overlying aquifers or confining beds. At this stage, the well may be completely dewatered.

There is also a greater chance for borehole restrictions to develop in a deeper well where there is a greater number of fractures present than in a shallow well. Looking at the data displays generated in this study, there is a close relationship between the depth of wells and the topographical position.

**Topographical Position of Wells.** As with the depth of wells, there is a clear relationship present between the topographical position of a well and the degree of degradation to that well. As seen in table 2, water levels in all of the wells found on hilltops or hillsides were affected to some degree. Levels in valley wells remained relatively unchanged except for well 57-W1. Well 57-W1 is a special circumstance because it is by far the deepest well among those in the valley setting at 66 ft. This depth puts it into the category of deep wells that are affected by mining. Well 57-W1 is also a special case in its location above the panel workings. Generally, a trend exists for hilltop and hillside wells to be degradationally affected by longwall mining and for valley wells to remain essentially unaffected despite special cases.

**Location of Well Above Panel Workings.** Matetic (1993) and other researchers have conducted detailed studies of water level changes across a profile of a longwall panel to determine changes related to different positions above a panel: gate-entry, one-third inside panel, two-thirds inside panel, and midpanel. This study was limited to data obtained from preexisting wells. Three very interesting observations can be made from data presented in table 2. Well 214-W1 showed the greatest degradational change and is located above a gate entry. Wells 56-W1 and 56-W2 showed no change with possible enhancement in water levels and are located directly above the midpanel. Finally, well 57-W1 is located above a mined-around block of support coal. However, it is located near enough to the edge of the block to be contained within a typical 24° angle of draw. This means that strata below well 57-W1 is in a tensional stress state. Strata surrounding the well at depth have been disrupted, and recharge to the well is affected because mining has occurred around this block. A tensional zone location and loss of recharge combined with its depth of 66 ft would explain its changes despite being located in a valley setting. Of these three interpretations, the fact that wells located above mined-around support blocks are not necessarily free from mining effects is valuable information. This will have a bearing when selecting barrier sizes intended to protect water wells and well fields under a wellhead protection strategy.

## Springs

Applying the same control mean method of analysis, springs also show rather similar comparative results to those of wells. However, spring flow data rather than water level elevation data are used in this analysis. Springs are used both as domestic drinking water sources and for farm animal watering troughs in the study area. Springs and wells are both equally important sources of farm and domestic ground water supplies in the two-county study area.

**Topographical Position of Spring.** Springs primarily occur along hillside locations as well as near hilltops. After examining monitoring data, topographical position in itself is not a clearly defined or independent means of examining trends in spring data as was the case for wells (table 3).

**Overburden Thickness.** Some springs exhibited different results for the same topographical position, hillside or hilltop. Hence, it was important to examine individual overburden thickness for each spring (table 3). Overburden thickness represents the difference between the topographic elevation of a spring and the Pittsburgh coal seam being mined directly below the water source. Conceptually, the thicker an overburden interval is, the less strata disruption is likely to disturb a spring. Springs 208-S1 and SP-5 are essentially at the same high/topographic elevation, 1,271 ft and 1,300 ft, respectively. However, 208-S1 is affected by mining while SP-5 remains unchanged. A closer examination of their respective overburden thicknesses reveals a clue as to why there is a different reaction to mining in each spring. 208-S1 has an overburden thickness of 670 ft while SP-5 has one of 880 ft. It appears clear from this example that the greater an overburden thickness is for a spring, then the less chance it has of being affected by mining. Obviously, two springs do not comprise a basis for setting standards, but they do help to illustrate the importance of overburden thickness.

**Yield Peaks.** When analyzing springs for effects of mining, a logical finding would be that a spring went dry or dropped off in yield in comparison to precipitation levels. However, another pattern was noted during spring data analysis and included high yield peaks up to a month prior to having a spring undermined or passed by an adjacent panel (fig. 3). These peaks are considered directly related to mining because plots of spring yields versus control mean head levels over a particular monitoring period show that these yield peaks are not contemporaneous with precipitation increases represented by higher control heads. Though these flow peaks or bursts occur, they are short-lived, and spring yields often drop below premining values following a burst. A yield peak represents a strata disruption and quite likely an increase in hydraulic conductivity of strata surrounding and nourishing a spring. Since the dilation of fractures in strata overlying a spring increases the transmissivity of a perched aquifer, these yield peaks signify a purging of stored water in the aquifer.

## Analysis

Two parameters emerge as important quantitative ways to combine data for both spring and well water resources. These were defined after addressing wells and springs individually.

**Topographic Position.** Topographic position is realized as the elevation of a resource above the drainage elevation or base level for a particular drainage subbasin. This drainage elevation is the elevation of the main stream for a subbasin. A topographic position is represented by the difference between the resource elevation and the respective drainage elevation. Since depth of wells also contributes to whether a well will be affected, topographic position of wells is treated as the difference between the elevation of the bottom of a well and the respective drainage elevation. This analysis provides a quantitative number for the vertical elevation of a resource above the local base or drainage level.

**Overburden Thickness.** This quantitative analysis is the same method that was introduced in the earlier section for springs. Overburden thickness represents the difference between the resource elevation and the elevation of the Pittsburgh coal seam directly below the resource. As in the topographic position analysis, the overburden thickness for a well is measured from the bottom of the well to the coal seam directly below the resource.

**Topographic Position Versus Overburden Thickness.** Having determined which monitoring sites were affected or unaffected by mining, a graphical analysis of these sites includes plotting them on a scatter graph with topographic position on the y-axis and overburden thickness on the x-axis. Each point is either an affected or unaffected water resource. There is a clear linear relationship between a resource's position above the drainage plane, its height above the coal seam, and whether it is affected by mining (fig. 4). These two parameters may help to define probability fields for whether a particular water resource will be affected by longwall mining in the Appalachian coal basin. This

Table 3. Location of springs with respect to topographic setting, mine panel location, and observed impact of mining.

| Site[1] | Slope of best-fit Premining | Postmining | Impact of mining | Topo setting | Elevation resource, ft | Topo position, ft | Overburden thickness, ft | Position in panel[2] |
|---|---|---|---|---|---|---|---|---|
| 214-S3 | 35.35 | went dry | Affected | hillside | 1350 | 250 | 742 | 313 ft inside |
| 208-S1 | 34.90 | 37.47 | Affected | hillside | 1271 | 311 | 670 | 25 ft inside |
| 214-S4 | 33.93 | 32.66 | Affected | hillside | 1330 | 230 | 720 | edge |
| SP-5 | 35.37 | NC[3] | Unaffected | hilltop | 1300 | 200 | 880 | ge |
| 208-S3 | 35.45 | 37.51 | Affected | hillside | 1215 | 255 | 613 | ge |
| 179-S9 | 35.33 | NC | Unaffected | hillside | 1265 | 165 | 765 | 260 ft inside |
| 208-S2 | 35.04 | went dry | Affected | hillside | 1190 | 230 | 583 | 94 ft inside |
| 208-S6 | 34.52 | NC | Unaffected | hillside | 1232 | 272 | 630 | ge |
| 179-S10 | 34.58 | NC | Unaffected | hillside | 1250 | 150 | 756 | ge |
| 179-S4 | 34.86 | NC | Unaffected | hillside | 1256 | 156 | 699 | 156 ft inside |
| 179-S3 | 34.41 | NC | Unaffected | hillside | 1238 | 138 | 670 | 94 ft inside |
| 179-S2 | 34.78 | NC | Unaffected | hillside | 1225 | 125 | 671 | 266 ft inside |
| 12-S1 | 35.16 | NC | Unaffected | hilltop | 1235 | 275 | 810 | ge |
| 12-S2 | 30.05 | NC | Unaffected | hilltop | 1221 | 261 | 796 | ge |
| 12-S3 | 34.25 | NC | Unaffected | hilltop | 1244 | 284 | 819 | edge |
| 12-S4 | 30.00 | NC | Unaffected | hilltop | 1240 | 280 | 815 | ge |
| 214-S1 | 35.53 | 30.81 | Affected | hillside | 1206 | 106 | 608 | 313 ft inside |
| 214-S2 | 35.47 | NC | Unaffected | hillside | 1200 | 100 | 600 | 47 ft inside |

[1] S = Spring
[2] ge : gate entry; inside: located inside mined panel; edge: located on edge of panel
[3] NC=No change in data trend.

NOTES: all elevations given as feet above sea level.
Topo position = elevation of resource - base level elevation.
Overburden thickness = elevation of resource - coal bed elevation.
Base level : elevation of local drainage level.

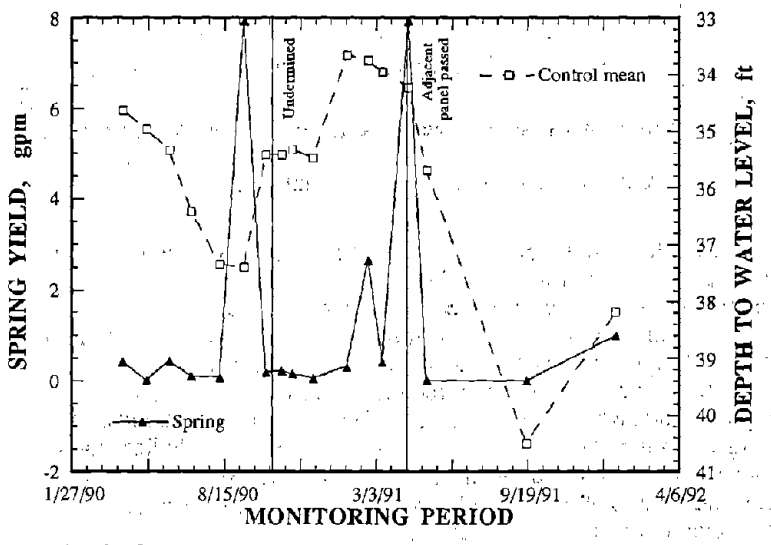


Fig. 3 Graph showing spring yield peaks for 208-S1 along with control mean water levels to evaluate spring response to mining.

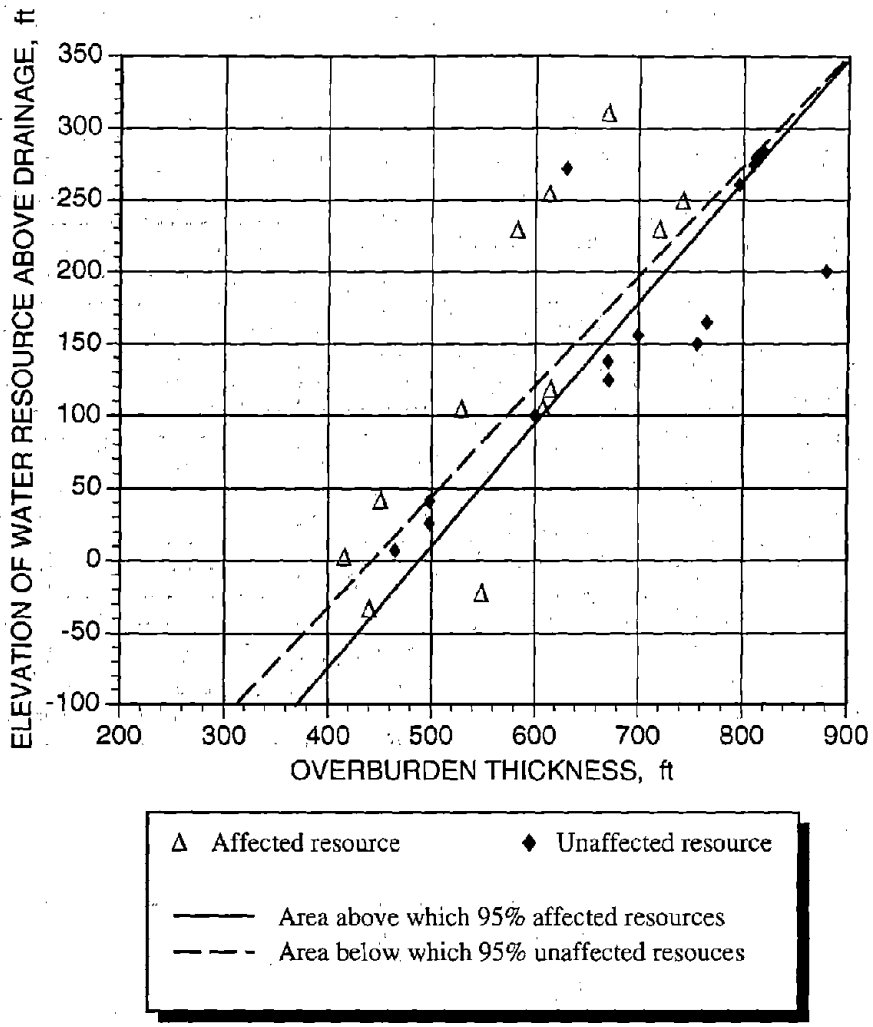


Fig. 10 Resultant graph for affected and unaffected water resources.

relationship applies specifically to the overburden characteristics of the northern portion of the Appalachian coal basin. These results are so encouraging that they warrant further refinement for the full range of overburden conditions expected within various mining districts. As new data are added to figure 4 or its variant to allow for change in topographic relief, overburden rock type, presence of various structural features such as fracture zones, faults etc., it should be possible to assign a water supply failure frequency or risk value to wells and springs likely to be adversely impacted by longwall mining in each hydrogeologic setting. A family of graphs representing various overburden conditions, climate, etc., may be developed. Such graphs would be based upon actual field experience determined on a case-by-case basis and hence should be of value in longwall mine planning, assist in wellhead protection studies, and have similar other uses.

The water level responses of domestic and farm water wells and flow variations of springs influenced by longwall mining under the Greene and Washington County hydrological and longwall mine settings need not be similar to the response observed or expected in other mining districts.

## Other Field Observations

General field observations for the effects of mining in this study were in agreement with those of other authors. Property owners located on hilltops and hillsides tended to lose water supplies or saw extreme shifts of their spring locations downslope. Valley bottoms were virtually unaffected in the long run in terms of relocating water. Strata around

wells may still shift and cause wells to become inoperable in the valleys, but new wells were very successful in returning reasonable yields to meet replacement water supplies. Conversely, hilltop and hillside topographic and hydrogeologic settings experienced a drop in the water table and hence a migration of springs downslope toward the valley environment. This was prevalent for areas within the mined panel and has been reported by Leavitt and Gibbens (1992) and others.

## Conclusions and Recommendations

Longwall coal mining disrupts the hydrogeological regime of overburden strata. However, the degree to which water wells and springs are adversely affected is difficult to discern. After several routes were tried in order to adequately analyze water level data from farm and domestic resources, a new approach in understanding water level changes in wells and spring yields resulting from mining and changing weather conditions was reached. A number of conclusions were drawn:

1. By using two U.S. Geological Survey statewide network and long-term monitoring wells as control wells, a discrete approach to filtering out the effects of precipitation on well changes and spring yields was successful.

2. Deep wells were more likely to be adversely affected by mining than shallow water wells. Wells deeper than 60 to 65 ft fall into this category under the hydrogeologic and mining settings considered for Greene and Washington Counties, PA.

3. Water levels in wells found on hilltops and hillsides were affected in all cases. Water levels in wells found in valleys were essentially unaffected.

4. Location of wells above longwall panel workings did not seem to dominate how wells responded to mining. It was important to note the effects of a typical angle of draw on wells found in a mined-around support block.

5. Overburden thickness plays a significant role as to whether springs are affected by longwall mining.

6. Affected springs exhibit brief yield spikes prior to or during undermining or mining of an adjacent panel. These spikes represent release of water from storage in the shallow aquifers most likely as a result of fracture dilation. As most springs are controlled by rock fractures combined with stratigraphic sequences, slight changes in fracture apertures can be expected to result in a substantial increase in fracture transmissivity as described by the cubic law.

7. Mining causes the local water table to be lowered. This causes a general dewatering of wells located on hilltops and portions of hillsides as well as emergence of springs downslope in valleys. Springs located higher on slopes may show above normal flow during undermining and then often become and remain dry. These findings support those of Booth (1992) and other researchers, who found that subsidence due to longwall mining causes fracturing and dilation of joints, which increases fracture permeabilities and storage coefficients.

8. A linear relationship for affected and unaffected water resources exists between overburden thickness and local topography. These results support Cifelli and Rauch's (1986) findings that given two resources at the same topographic position, the one having a greater overburden thickness will less likely be affected. The same type of reasoning can be employed given two resources with the same overburden thickness but different topographic positions. Leavitt and Gibbens (1992) showed that topographic setting of water resources was the dominant factor governing whether a resource was affected by mining. Both these conclusions are satisfied in this linear relationship for medium-to-thick overburden strata in the Appalachian coal basin. Using these results, a predictive approach could be developed to determine whether a particular water resource will be affected by longwall mining. Further data must be applied to these results to predict the impacts of mining where shallow overburden conditions are present. Also, some variations can be expected when comparing resource data for high, intermediate, and low topographic relief, and for regions with variable overburden stratigraphic sequence, e.g., thick, massive sandstone versus shale, etc.

9. Coal company water supply remediation efforts when supplies are disrupted due to mining seem to be relatively efficient. New wells are often drilled by companies rather than having to deal with cleaning out old wells. In most cases, new, deeper wells are located some 10 ft or so from the old ones and produce yields sufficient for domestic use. Fracture controlled water production allows for this relatively simple reinstallment of lost water supplies.

## Acknowledgments

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# APPLICATION OF AN ANALYTICAL GROUND WATER FLOW MODEL TO A PSEUDOKARST SETTING IN A SURFACE COAL MINE SPOIL[1]

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**Abstract:** Attempts were made to apply an analytical groundwater flow model (MINEFLO) to a reclaimed surface coal mine spoil in West Virginia, U.S.A., that exhibited characteristics of both a conventional porous medium and pseudokarst flow. The model utilizes the method of analytic elements to derive a steady-state solution to the ground water flow equation for a single aquifer. Two simulations were performed, using hydrologic features and parameter values determined from field observations, slug withdrawal tests (1989) and slug injection tests (1990). Simulated heads at base-flow conditions for 1989 were about 12% different than measured values, with a root-mean-square (RMS) error of 0.82 m. A similar simulation for 1990 produced a somewhat better match, with an average head difference of less than 10% (RMS error of 0.66 m). However, in both cases, the simulated flow rate at the toe-of-spoil seep was over 75% less than the measured rate. This suggested that the hydraulic conductivity values obtained from the slug tests may have underestimated the overall hydraulic conductivity of the aquifer materials, perhaps because of the influence of pseudokarst and/or fracture flow mechanisms. It was concluded that the model could predict (within 10%) the base-flow water levels at a spoil site where recharge is known to occur primarily from adjacent unmined strata. The use of the model's only transient feature (pumping well) at the study site is doubtful due to the large RMS error in the steady-state simulation.

**Additional Key Words:** Acid mine drainage, aquifer testing.

## Introduction

Previous studies (Hawkins and Aljoe, 1990; 1992) showed that the ground water flow system within a surface coal-mine spoil in central West Virginia, U.S.A., exhibits characteristics of both conventional porous-media flow and pseudokarst flow. Pseudokarst flow involves ground water storage and rapid movement through relatively large, open conduits; porous-media flow is characterized by slower water movement around the solid particles and through minute fractures in the medium. In mine spoil, conduit formation is facilitated by differences in spoil particle size and is caused by piping of the finer spoil material or differential settling (Groenewold and Bailey, 1979). Water levels in spoil monitoring wells suggest that porous-media flow is dominant under steady-state (low-flow) conditions, when the spoil appears to possess a single water table with gradual hydraulic gradients. Irregularities in the water table appear to be caused by permeability contrasts within the spoil and the existence of a discrete, preferred recharge area where the spoil intercepts a large natural fracture. However, indicators of pseudokarst flow are observed under transient conditions, such as during aquifer testing and single-event, high-recharge periods lasting only several days. These indicators include: (1) erratic response of observation wells to slug and pumping tests; (2) temporary changes in local ground water flow direction; (3) steepening of water table gradients; (4) artesian flow from a spoil monitoring well; and (5) rapid disappearance of channeled surface runoff into a swallet in the outslope area of the spoil and re-emergence near the toe-of-spoil seep.

The U.S. Bureau of Mines is currently conducting research to determine whether hydrologically complex spoils can be adequately characterized on a local scale by a ground water computer model. However, the most commonly-used and well-documented models, whether analytical or numerical, have been derived from porous-media flow principles. The applicability of such models to spoils with some pseudokarst characteristics is therefore uncertain. To determine the applicability of a model, it is first necessary to calibrate the model at steady-state conditions at a site where ample field data are available, use it to predict the effects of transient events or different steady-state conditions at that site, then check the predictions with field data. The study described in this paper was directed toward the first part of this procedure, i.e., steady-state calibration at a mine spoil site that is as closely monitored as could be expected in practice. The site also receives most of its recharge from the surrounding undisturbed material rather than from direct infiltration through the spoil, a characteristic that is common to many spoil aquifers. The ability of a model to accurately simulate the important aspects of the flow system at this site will help determine the general applicability of porous-media flow models in the analysis of other surface mine spoils with similar flow systems but less field data.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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## Previous Modeling Efforts

Modeling of ground water flow through and around surface mine spoils has been attempted in the past; however, none of the previous efforts have focused on lateral head variations within a single mine spoil, as the current study has attempted to do. Studies by Wilson and Hamilton (1978) and Schwartz and Crowe (1985) simulated the effects of surface mining on ground water levels in the undisturbed material surrounding a spoil in the Northern Great Plains region of North America during and after a hypothetical mining event. A later study by Schwartz and Crowe (1987) looked at the spoil on a smaller scale to examine the rates at which the spoil would resaturate following cessation of mining. In these studies, the spoil was considered to be a homogeneous mass within a large regional aquifer, and two-dimensional, finite-element numerical models were applied to vertical cross-sections through the mined area. Rogowski and Weinrich (1977) also examined spoil resaturation, using both a numerical solution of a one-dimensional moisture flow equation and a layered water-budget technique to model vertical percolation through topsoiled and non-topsoiled spoil material. The models were calibrated using data from 3-m-deep spoil profiles that had been reconstructed in the laboratory. Neither the cross-sectional modeling efforts nor the vertical infiltration studies considered lateral head variations or permeability contrasts within the spoil.

Bair and Parizek (1981) modeled the hydrologic effects of a proposed open pit mine in the anthracite coal district of eastern Pennsylvania (USA). The emphasis in this study was placed on the flow system in the undisturbed material around the mine rather than within the mine spoil. The hydrostratigraphic unit around the proposed mine was modeled as a single, unconfined, anisotropic aquifer. The pre-mining potentiometric surface in this aquifer was simulated with a finite-difference numerical model, and existing head data were used to calibrate the model. The hydrologic effects of various stages of mining were then simulated by placing constant-head nodes at the mine floor level. The potentiometric surfaces, mine inflow rates, and other simulation results could not be verified because the mine had not yet been developed.

Phelps (1984) performed a finite-element modeling study in which lateral flow through a heterogeneous spoil cross section was considered. The intent of this investigation was to analyze the flow patterns through and around a low-permeability "toxic package" placed within the saturated zone of the spoil, with a view toward minimizing the percentage of flow moving through the package. Although the water table configurations and vertical head distributions resulting from these simulations appeared to be reasonable, no attempt was made to compute areal head distributions or to relate the model to a field situation.

## Description of Computer Model (MINEFLO)

The MINEFLO computer model used in the current study was developed by R. D. Schmidt of the U. S. Bureau of Mines' Twin Cities Research Center in Minnesota[3]. MINEFLO is a two-dimensional aquifer model that formulates steady-state analytical solutions to basic ground water flow equations. The starting point of a MINEFLO simulation is a single, confined aquifer of infinite lateral extent and user-defined thickness, porosity, and permeability. The user identifies all hydrologic features impacting this idealized system, inserting them into the simulation on the basis of their spatial coordinates. These features include: (1) vertical planes of constant head, discharge, or recharge within the aquifer (termed "line sources" in the model); (2) areal zones or vertical cracks within the aquifer whose permeabilities differ from that of the base aquifer; (3) wells that can add or withdraw water from the aquifer at any point; (3) zones of continuous recharge (infiltration) or discharge (leakage) above or below the aquifer; and (4) vertical (planar) hydrologic barriers that occupy the entire aquifer thickness. A "reference point" that is beyond the area of influence of the hydrologic features must also be specified. The model can be used to simulate a wide variety of hydrologic conditions (Schmidt, 1985; 1989), but this is the first known application of the model to a surface mine spoil and its surroundings.

MINEFLO uses the method of analytic elements (Strack, 1987) to create mathematical expressions for representing each of the hydrologic features. These expressions are combined to form a large matrix that is solved by the model to yield equations for the discharge potential (head) and its complex conjugate, the stream function, at any point (x,y) in the aquifer. The Dupuit-Forchheimer assumptions are used to simulate unconfined flow near a seepage face. Flow-related characteristics of hydrologic features in the aquifer, such as heads in wells and flow rates through constant-head planes, are also computed as part of the matrix solution. The computed values can then be compared to measured field data at key points in the flow system (e.g., heads in wells, discharge from springs) to check the accuracy of the simulation. Subroutines for gridding and contouring the values of the head and stream function and for tracing streamlines are included in the MINEFLO package.

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[3]A manual describing MINEFLO is currently being prepared. For information contact R. D. Schmidt, U. S. Bureau of Mines, 5629 Minnehaha Ave. South, Minneapolis, MN 55417.

Two advantages of MINEFLO over other hydrologic models are its speed and ease of use. Since the solution technique is entirely analytical, it does not employ the discretization and iteration procedures inherent in numerical models. Solutions can thus be obtained quickly on a microcomputer. MINEFLO is menu driven, from data entry through contouring of results, thereby eliminating the need for external software; however, MINEFLO data files can be created, read, and modified via most common word processing packages. Changes in the flow simulation can be made easily, and the effects of these changes can be assessed almost immediately. In this study, a version of MINEFLO for the Macintosh[4] was used, which allowed model outputs to be exported immediately to other Macintosh applications.

The primary disadvantage of MINEFLO is its limited capability for performing transient simulations. The only transient process supported by the model is that of a pumping or injection well, in which the Theis (1935) equation for drawdown due to pumping is superimposed on a previous steady-state solution. The effects of such transient wells can be examined at specified time periods, provided that the pumping does not change the head and flow conditions of the previously-specified steady-state features. Separate simulations, involving complete analytical solutions for the entire flow system, must be run for each time period being considered. Transient recharge events cannot be simulated. Therefore, emphasis in this paper is placed on model calibration in the steady-state case.

## Description of Study Site

The study site is a 3.24 ha parcel of a 33 ha reclaimed surface mine that removed a sinuous strip of the lower and middle Kittanning coal seams along a hillside coal outcrop in Upshur County, WV (figures 1 and 2). Hawkins and Aljoe (1992) provide a description of the site geology, lithology, and mining history. The spoil was terrace backfilled and not returned to the contour of the original hillslope. As a result, most of the spoil surface slopes gently northwestward toward the buried highwall. Despite the presence of a surface drainage ditch, surface runoff becomes impounded on the spoil during and after precipitation events; however, heads in monitoring wells completed in the spoil beneath the impounded water indicated that vertical infiltration in this area was not a primary source of recharge to the spoil aquifer. Ground water inflow through fractures in the buried highwall appears to be the primary spoil recharge source (Hawkins and Aljoe, 1990). The slope of the water table appears to be controlled primarily by the slope of the original bedrock and/or land surface, which forms a low-permeability layer beneath the spoil (figure 2). The spoil outslope is steep and continues down to the interface with undisturbed ground, where a closely-spaced line of seeps is present. Flow from the seep line is channeled to a common collection point for gaging and sampling.

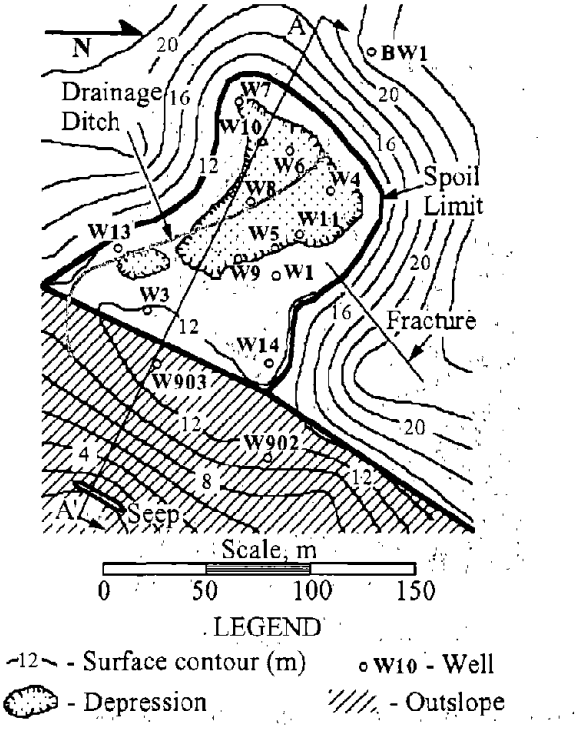


Figure 1. Surface features of mine spoil study site.

Wells 1 through 14 (figure 1) were drilled to the pit floor in the main spoil, while BW1 was drilled to a depth of about 20 m in undisturbed strata behind the final highwall. These wells were installed in the early 1980's and were used to calibrate the model. Wells 902 and 903 were drilled to the pit floor in the outslope during February 1990; these proved to be unsuitable for use in model calibration because their water elevations suggested that they penetrated localized pockets of water in the outslope rather than the continuous aquifer that comprised the main spoil wells. All wells were 5.1 cm in diameter and were finished with 3-m long slotted well screens. The study focused on the years 1989 and 1990, when monthly measurements of well water levels and discharge

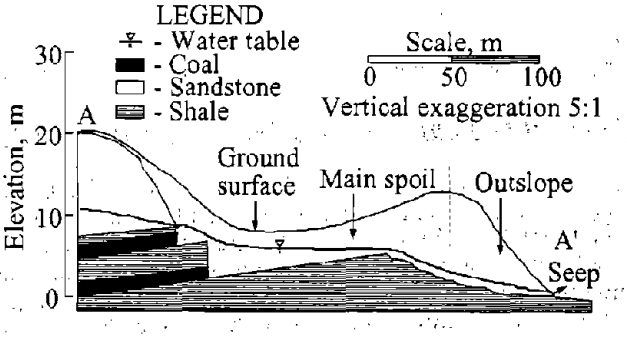


Figure 2. Cross-section of mine spoil study site.

[4]Reference to specific products does not imply endorsement by the Bureau of Mines

rates were obtained. During this period, the flow from the seep line ranged from 61 to 375 L/min with a median of 83 L/min.

## Model Calibration

In order to clarify the discussion of model calibration, it is important to define the terms hydrologic features and model parameters as employed by the MINEFLO model. As stated earlier, a MINEFLO simulation starts with a base aquifer of infinite areal extent and constant thickness, porosity, and hydraulic conductivity. A hydrologic feature is defined as a deviation from the specified base aquifer, and is characterized by the type of feature (e.g., line source, permeability zone, well, crack, barrier ) and its location (x,y) within the plane of the base aquifer. Model parameters are the numerical values of head, hydraulic conductivity, recharge, or discharge assigned to the base aquifer and each hydrologic feature, depending on the type of feature and its expected role in the field setting. The goal of model calibration is to configure the base aquifer and hydrologic features and adjust the model parameters, in a manner consistent with field observations, such that the simulated heads in monitoring wells and flow rates at discharge points match the field-measured values as closely as possible.

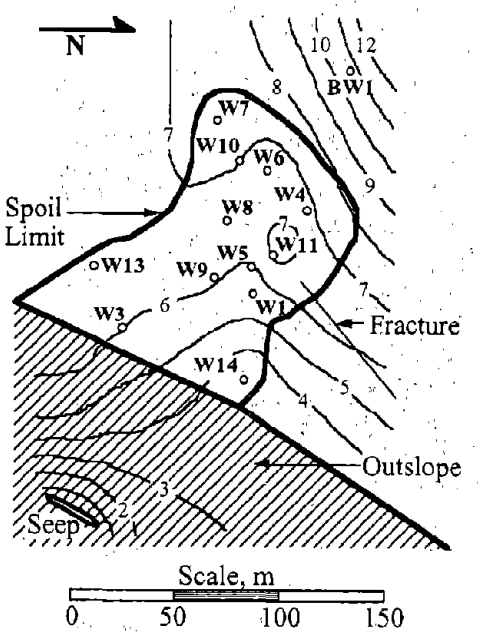


Figure 3. Water table contours (m) at steady-state (low-flow) conditions, August 29, 1990.

Calibration of a steady-state model is often performed by matching it to a low-flow period, when steady-state conditions usually prevail in the field. Figure 3 shows the water table contours for the low-flow period of 1990 (August 29); the water table map for the low-flow period of 1989 (July 26) was very similar. Flow originates mainly from the buried highwall and moves toward the seep. Irregularities in the contours in the main spoil (note the 7 m contours in figure 3) appear to be related to permeability contrasts within the spoil and/or the influence of a large natural fracture that intersects the spoil near well 11 (Hawkins and Aljoe, 1990). MINEFLO is capable of simulating these features (using line sources, permeability zones and cracks), so calibration to low-flow conditions appeared to be feasible. Although the calibration process described below would suggest that the base aquifer and hydrologic features were rigidly defined, and only the parameters were adjusted to achieve the desired match, this was far from the case. Many other sets of base aquifer conditions and hydrologic feature boundaries were examined, but only the final, "best match" configurations and parameter values are presented here. For this study, the "best match" was defined as the scenario that produced the minimum RMS (root-mean-square) error between the measured and simulated heads in monitoring wells, defined as:

$$\text{RMS error} = \sqrt{\frac{1}{n}\sum_{i=1}^{n}(h_{si} - h_{mi})^2} \qquad (1)$$

where n is the number of wells, and $h_{si}$ and $h_{mi}$ are the simulated and measured heads, respectively, in the ith well.

## Configuration of Base Aquifer and Hydrologic Features

The base aquifer defined for this study had a thickness of 12.5 m, which was slightly greater than the maximum saturated thickness of the conceptual aquifer (maximum measured head in the monitoring wells). Since the base aquifer is horizontal, it comprises both the spoil material and the undisturbed strata beneath and around the spoil. This obvious simplification in the definition of the aquifer was acceptable for the purpose of comparing steady-state heads in wells and total seep discharge, because these parameters are independent of the flow distribution within any vertical cross-section of the aquifer. Steady-state heads and flows are also independent of the aquifer porosity; therefore, although MINEFLO requires that a porosity be specified, the porosity value had no effect on the desired results. However, ground water velocities computed by the model would definitely be erroneous, since the calculations would assume uniform flow through a homogeneous vertical cross-section of the aquifer at its stated porosity. In reality, the strata

beneath the spoil are probably much less permeable than the spoil (note the location of the seep in figure 2), so most of the flow through spoil-covered cross-sections would occur through the spoil. This study therefore made no attempt to use the stream function generated by the model to analyze ground water velocities or to track water particle movement.

In order to simulate lateral (as opposed to vertical) permeability contrasts between the undisturbed material, mine spoil, and subregions within the spoil,"permeability zones" were constructed within the base aquifer. The boundaries of the permeability zones were estimated from: (1) field observations of the spoil limits; and (2) hydraulic conductivity contour maps generated from the results of two sets of slug tests conducted in the monitoring wells. Slug withdrawal tests were performed in November 1989, and slug injection tests were performed in June 1990, after a subsurface grouting project was conducted at the site. Table 1 compares the hydraulic conductivity values obtained from these tests. The differences between the hydraulic conductivity values in table 1 were determined by Hawkins, et al. (1991) to be related more closely to the test method and instrumentation used than to the effects of grouting, with the 1990 values being somewhat more accurate. However, in order to assess the sensitivity of the model to the results of the field data collection efforts, calibrations were performed for both the 1989 (slug withdrawal) and 1990 (slug injection) periods, using the low-flow heads and discharge values for the two years as calibration goals.

Table 1. Hydraulic conductivity values (m/sec) measured in slug tests.

| Well # | Well Location | Slug Withdrawal 1989 (m/sec) | Slug Injection 1990 (m/sec) |
|---|---|---|---|
| 1 | Outer spoil | 2.07E-07 | 9.37E-07 |
| 3 | Inner spoil | 8.94E-06 | 1.34E-04 |
| 4 | Outer spoil | 6.90E-06 | 6.76E-06 |
| 5 | Inner spoil | 1.93E-05 | 1.24E-05 |
| 6 | Inner spoil | 1.30E-05 | 3.14E-05 |
| 7 | Inner spoil | 5.10E-06 | 5.34E-05 |
| 8 | Inner spoil | 4.66E-06 | 8.23E-05 |
| 9 | Inner spoil | 5.69E-05 | 2.32E-05 |
| 10 | Inner spoil | 1.26E-06 | 1.01E-06 |
| 11 | Outer spoil | 9.14E-07 | 3.89E-06 |
| 13 | Outer spoil | 9.61E-07 | 8.56E-07 |
| 14 | Outer spoil | 4.93E-06 | 3.34E-07 |
| BW1 | Unmined area | 4.12E-07 | ND |
| 902 | Outslope | ND | 2.85E-08 |
| 903 | Outslope | ND | 1.73E-08 |

ND = Not Determined

In both cases, the portion of the spoil closest to the highwall (outer spoil) constituted one permeability zone, whose hydraulic conductivity was greater than that of the base aquifer (undisturbed material) but less than that of the interior spoil, which constituted a second permeability zone. The primary difference between the two aquifer configurations was the relative size of the two spoil zones, as shown in figures 4a (1989 simulation) and 4b (1990 simulation).

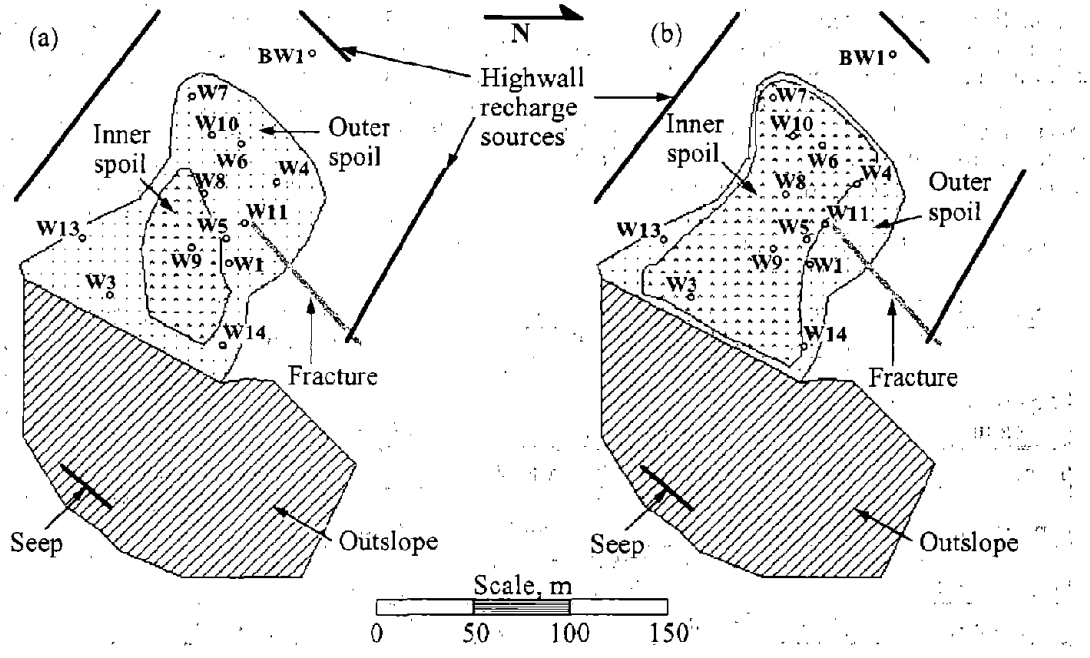


Figure 4. Hydrologic features of MINEFLO model: (a) 1989 simulation; (b) 1990 simulation.

A third important permeability zone is the outslope, which comprises the area between the spoil wells and the seep area, including the seep. The outslope was created by casting overburden material over the natural terrain downslope from the mined area . The blocky, heterogeneous nature of the outslope material lends itself easily to the development of conduits and localized, discrete, high-permeability channels; evidence of this was found during field investigations (Hawkins and Aljoe, 1990). The lateral (north-south) extent of the outslope area was not readily discernible in the field, so these boundaries were not as easily defined as those of the spoil-highwall and spoil-outslope interfaces. For consistency, the outslope boundaries were kept the same in both the 1989 and 1990 simulations (figures 4 and 5). The hydraulic conductivities measured in the two outslope wells (902 and 903 in table 1) were not used to help define the outslope boundaries because their extremely low values suggested that they were screened in relatively impermeable zones between conduits, and thus did not represent the bulk hydraulic conductivity of the outslope material. Static water levels in the two outslope wells were also inconsistent with those of the rest of the spoil, again suggesting that the spoil wells and outslope wells were not screened in the same aquifer.

A "crack" was used in both simulations to represent the effect of the large natural fracture that intersects the spoil near well 11. In MINEFLO, a crack is equivalent to a very narrow permeability zone; its location in the aquifer is defined by specifying its endpoints and its width rather than a closed areal boundary. The location of the crack in figures 4 and 5 and its width (5.0 cm) were specified on the basis of field observations.

In both simulations, three line sources of constant head were used to represent steady-state recharge to the spoil from the unmined material on the north, west, and south. The positions of these line sources corresponded to topographic highs in the undisturbed material. Since surface topography would be expected to determine ground water flow direction in the undisturbed material (unlike the bedrock-controlled flow direction in the spoil), topographic highs would generally correspond to ground water divides in the undisturbed material. A fourth line source in the outslope area was used to represent the seep; it served as the only means of removing water from the aquifer system.

## Adjustment of Model Parameters

Table 2 summarizes the hydraulic conductivity values that resulted in the "best match" between the simulated and measured heads in monitoring wells, along with the rationale used to justify the parameter values. In the 1989 simulation, the hydraulic conductivity of the base aquifer was specified at $4.12 \times 10^{-7}$ m/sec on the basis of the results of the slug withdrawal test in well BW1, the only well in undisturbed material. The boundary of the inner, high-conductivity spoil zone in the 1989 simulation and its associated hydraulic conductivity were obtained directly from a contour map of spoil hydraulic conductivity (i.e., the $2.50 \times 10^{-5}$ m/sec contour). The hydraulic conductivity of the outer spoil zone (between the inner zone and the highwall) was specified at $3.24 \times 10^{-6}$ m/sec, the geometric average of all spoil wells except well 9 (i.e., well 9 was contained within the inner zone). The outslope hydraulic conductivity in the 1989 simulation was adjusted until the best match was achieved ( at $3.00 \times 10^{-5}$ m/sec), and thus is designated as a "calibration parameter" in table 2. In the 1990 simulation, the inner spoil zone was much larger and the outer zone much smaller than in the 1989 simulation, and the hydraulic conductivities in table 2 were specified as the geometric averages of the wells contained in each zone. With this specification, the hydraulic conductivities of the base aquifer and outslope zone had to be adjusted to approximately twice their 1989 values to maintain an acceptable head match. However, these adjustments are not considered to be radical in view of the measured differences in hydraulic conductivities (table 1). The hydraulic conductivity of the crack was set at $3.00 \times 10^{-3}$ m/sec in both simulations; however, it was found that as long as the hydraulic conductivity of the crack was at least one order of magnitude greater than that of the outer spoil, further increases in crack permeability had very little effect on the simulated results.

Table 2. Hydraulic conductivity values in "best match" simulations, MINEFLO model.

| | 1989 Simulation - Slug withdrawal | | 1990 Simulation - Slug injection | |
|---|---|---|---|---|
| Hydrologic feature | Hydraulic conductivity (m/sec) | Rationale | Hydraulic conductivity (m/sec) | Rationale |
| Base aquifer | 4.12E-07 | Measured Value in BW1 | 7.62E-07 | Calibration Parameter |
| Inner spoil | 2.50E-05 | Hydraulic conductivity map; zone around well #9 | 2.45E-05 | Geometric mean, wells #3, 5, 6, 7, 8, 9, 10 |
| Outer spoil | 3.24E-06 | Geometric average of all spoil wells except #9 | 1.48E-06 | Geometric mean, wells #1, 4, 11, 13, 14 |
| Outslope | 3.00E-05 | Calibration Parameter | 5.50E-05 | Calibration Parameter |
| Fracture | 1.00E-03 | Calibration Parameter | 1.00E-03 | Calibration Parameter |

Table 3 summarizes the "best match" values of reference head for the line sources shown in figure 4. Except for the seep, whose head was set at zero by definition, all the reference heads were considered to be calibration parameters. The minor differences in reference heads between the 1989 and 1990 simulations resulted from the fact that in most of the wells, the calibration goals (measured heads) were slightly lower in 1990 than in 1989. The relative strengths of the three highwall sources were the same (south >west >north) in both simulations.

Table 3. Reference heads of line sources in "best match" simulations, MINEFLO model.

| Hydrologic feature | 1989 Simulation - Slug withdrawal | | 1990 Simulation - Slug injection | |
|---|---|---|---|---|
| | Reference head (m) | Rationale | Reference head (m) | Rationale |
| Seep | 0 | Seep location defined at base of aquifer | 0 | Seep location defined at base of aquifer |
| West highwall | 12.25 | Calibration Parameter | 11.75 | Calibration Parameter |
| South highwall | 12.50 | Calibration Parameter | 12.00 | Calibration Parameter. |
| North highwall | 11.00 | Calibration Parameter | 10.75 | Calibration Parameter |

## Discussion of Model Results

Table 4 compares the "best match" simulations to the measured values for both the 1989 and 1990 modeling exercises. Simulated heads at base-flow conditions for 1989 were about 12% different than measured values (RMS error 0.82 m); the 1990 simulation was somewhat better, with an average head difference of less than 10% (RMS error 0.66 m). This may reflect the previously-stated conclusion of Hawkins, et al. (1991) that the slug injection tests yielded somewhat more accurate values of hydraulic conductivity. In both simulations, some of the largest errors occurred in the two wells closest to the outslope (wells 3 and 14) and in well 11, which was close to the point where the spoil intercepted the large fracture. Field observations suggested that these areas would be more prone to non-Darcian flow conditions than other areas, and this may partially explain the large errors associated with these wells. Note in table 4 that the errors are approximately halved when wells 3, 11, and 14 are not included in the analysis; however, the average error without these wells was still about 5% (RMS errors 0.36 to 0.5 m).

Table 4. Comparison between measured and simulated data, MINEFLO model.

| | 1989 Simulation - Slug Withdrawal | | | | 1990 Simulation - Slug Injection | | | |
|---|---|---|---|---|---|---|---|---|
| Well # | Measured head (m) | MINEFLO heads (m) | Error (m) | % Error | Measured head (m) | MINEFLO heads (m) | Error (m) | % Error |
| 1 | 5.35 | 5.84 | 0.49 | 9.2% | 5.43 | 6.01 | 0.58 | 10.7% |
| 3 | 6.10 | 4.63 | -1.47 | -24.0% | 6.01 | 5.25 | -0.76 | -12.7% |
| 4 | 7.46 | 7.25 | -0.21 | -2.8% | 6.78 | 6.83 | 0.05 | 0.8% |
| 5 | 5.80 | 5.99 | 0.19 | 3.2% | 5.74 | 6.22 | 0.48 | 8.3% |
| 6 | 6.59 | 7.38 | 0.79 | 12.0% | 6.48 | 6.85 | 0.37 | 5.8% |
| 7 | 8.21 | 8.12 | -0.09 | -1.0% | 7.39 | 7.08 | -0.31 | -4.1% |
| 8 | 6.37 | 6.20 | -0.17 | -2.7% | 6.49 | 6.47 | -0.02 | -0.3% |
| 9 | 6.13 | 5.75 | -0.38 | -6.2% | 6.27 | 5.96 | -0.31 | -5.0% |
| 10 | 6.96 | 7.42 | 0.46 | 6.6% | 7.21 | 6.87 | -0.34 | -4.7% |
| 11 | 7.44 | 6.58 | -0.86 | -11.6% | 7.43 | 6.47 | -0.96 | -12.9% |
| 13 | 6.41 | 5.42 | -0.99 | -15.5% | 6.45 | 5.94 | -0.51 | -7.9% |
| 14 | 3.01 | 4.86 | 1.85 | 61.7% | 3.26 | 4.96 | 1.70 | 52.4% |
| BW1 | 11.34 | 11.15 | -0.19 | -1.7% | 10.61 | 10.62 | 0.01 | 0.1% |
| Head error analysis | Mean error (absolute): | All wells - | 0.63 | 12.2% | Mean error (absolute): | All wells - | 0.49 | 9.7% |
| | | w/o 3,14,11 - | 0.40 | 6.1% | | w/o 3,14,11 - | 0.30 | 4.8% |
| | RMS error: | All wells - | 0.82 | | RMS error: | All wells - | 0.66 | |
| | | w/o 3,14,11 - | 0.49 | | | w/o 3,14,11 - | 0.36 | |
| Seep discharge | Measured flow (L/min) | MINEFLO flow (L/min) | Error (L/min) | % Error | Measured flow (L/min) | MINEFLO flow (L/min) | Error (L/min) | % Error |
| | 61.80 | 11.652 | -50.15 | -81% | 83.40 | 20.82 | -62.58 | -75% |

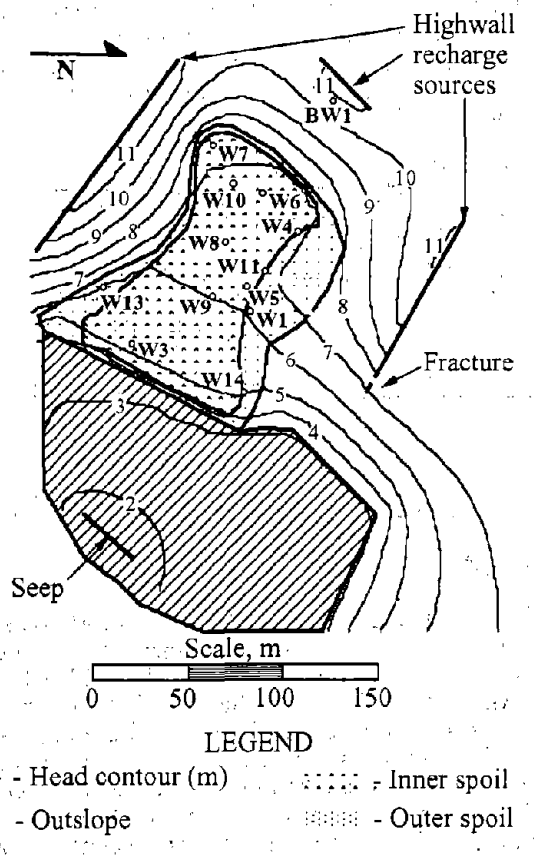


Figure 5. Head contours generated from MINEFLO model, 1990 simulation.

Figure 5 shows the water table contours generated by the model for the 1990 simulation; the map for the 1989 simulation was very similar in appearance, suggesting that the differences in the permeability zones did not have a dramatic effect on the simulation results. Note that the irregularities in the actual 7m water table contour (figure 3) did not appear in the simulated results (figure 5). Thus, field inferences and slug test results alone did not provide the information necessary to simulate the actual irregularities.

The most important problem with both simulations was that simulated seep discharge rates were 75% to 80% less than measured values (table 4). In order to make the simulated discharge rates match the measured values, while keeping the simulated heads in table 4 the same, the hydraulic conductivities of the base aquifer and all hydrologic features would have to be increased uniformly by a factor of 4 to 5. If the configurations of the base aquifer and hydrologic features are reasonably accurate, it can be inferred that the hydraulic conductivities measured in the slug tests are 4 to 5 times less than the effective hydraulic conductivity of the actual materials. This inference was supported by the fact that the recovery patterns of some of the wells during slug tests suggested the presence of discrete, highly permeable zones within the spoil (Hawkins and Aljoe, 1992). It is also possible that the fractures and bedding plane separations intercepted by well BW1 were relatively small in magnitude and extent compared to the actual fractures and bedding plane separations providing recharge to the spoil through the highwall. These conditions would cause slug test results to underestimate the overall hydraulic conductivities in both types of material.

The magnitude of the RMS errors shown in table 4, even without considering wells 3, 11, and 14, effectively precludes the practical application of the only transient capability of the MINEFLO model, that of a pumping or recharging well. Such a well can be superimposed on a steady-state solution in MINEFLO only if its inclusion does not significantly impact the previously-defined steady-state features -- in this case, the flow rates at the four constant-head line sources. This effectively places an upper bound on the stress (pumping rate and duration) that can be simulated. Furthermore, the drawdown achieved in a simulated pumping test will have practical validity only if it exceeds the RMS error between simulated and measured heads. This places a lower bound on the simulated stress. The practical difficulty in achieving these conditions can be illustrated by considering pumping simulations involving the most closely-spaced well pair (wells 6 and 10, separation 15 m) that would result in a drawdown exceeding the lowest RMS error in table 4 (0.36 m). The Theis (1935) equation that is employed by MINEFLO for transient pumping analysis can be used independently of the model; for calculation purposes, the average hydraulic conductivity of the inner spoil in the 1990 simulation (table 2) can be multiplied by the average aquifer thickness at wells 6 and 10 (6.86 m) to yield a transmissivity value of $1.68 \times 10^{-4}$ $m^2$/sec. Since the aquifer is unconfined, aquifer porosity can be used to estimate its storativity; the porosity of the spoil aquifer at this site was estimated to be 20% (Hawkins, 1993), thus storativity was estimated at 0.2. Using these parameters, pumping in well 6 or 10 at a constant rate of 4 L/min (20 % of the 1990 simulated steady-state flow rate at the seep) would have to continue for more than 100 years before the simulated drawdown exceeded 0.36 m in the other well. Conversely, a more reasonable pumping duration, 3 days, would require a pumping rate of about 28 L/min -- greater than the 1990 simulated seep flow rate -- for the drawdown to exceed 0.36 m. Clearly, a pumping test that could be simulated with validity by the model would be nearly impossible to perform.

## Conclusions

The MINEFLO model was used to predict, within 10%, the base-flow water levels in wells at a spoil sites where recharge is known to occur primarily from adjacent unmined strata. If premining ground water data on the adjacent unmined strata are available, it may be usable for the same purpose on other sites with similar recharge

patterns. Field inferences and slug test results alone are not likely to provide the information necessary to simulate water table irregularities within the spoil. If slug tests are used to derive hydraulic conductivities for use in the model, the test values may underestimate the overall hydraulic conductivity of the aquifer materials if pseudokarst and/or fracture flow mechanisms are evident. This may cause the simulated flow rates at the spoil discharge to underestimate the measured flows by as much as 80%. The practical use of the model's only transient feature (pumping well) at the study site is doubtful due to the large RMS error in the steady-state simulation. The primary advantage of MINEFLO is that simulations are quick and easy to set up, perform, and evaluate. Numerical models which are capable of modeling hydrologic features within the spoil and are less restrictive in terms of modeling transient phenomena should be investigated to determine their advantages and disadvantages compared to MINEFLO.

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# INVESTIGATION OF METAL AND NONMETAL ION MIGRATION THROUGH AN ACTIVE PHOSPHOGYPSUM STACK[1]

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**Abstract:** Phosphogypsum is a major waste byproduct of wet-process phosphate acid production and is currently regulated under the Toxic Substance Control Act (TSCA) of 1976. For each metric ton of phosphoric acid produced, approximately 5 mt of phosphogypsum is produced and must be stockpiled. The U.S. Bureau of Mines Tuscaloosa Research Center installed monitoring wells in an active 195.4-ha phosphogypsum stack. This stack is approximately 30.5 m high and has been active for 22 yr. The wells cover the entire stack perimeter and consist of five clusters, with three wells per cluster; these wells are 7.6, 12.2, and 16.7 m deep. Core drill samples were taken every 1.5 m and leached with process cooling pond water and artificial rainwater. The wells were monitored monthly, with samples being analyzed for pH and metal and nonmetal ion concentrations. In addition to site studies, pore volumes were collected from column leach tests and analyzed for metal and nonmetal ion concentrations. Initial results from the column leaching tests show that a number of metal and nonmetal ions are migrating out of the phosphogypsum. These results indicate that metal and nonmetal ions contained in the phosphogypsum have the potential for migrating into surface waters and ground water.

**Additional Key Words:** phosphogypsum, column leaching, metals contamination.

## Introduction

In the production of phosphate-based fertilizers, an apatite [$Ca_5(PO_4)_3(F,Cl,OH)$] concentrate and fossilized animal remains are leached with sulfuric acid ($H_2SO_4$) in the presence of recycled dilute solutions of phosphoric acid ($H_3PO_4$). The apatite dissolves during the leaching step, producing $H_3PO_4$. The calcium and sulfate ions subsequently combine to form gypsum, commonly referred to as phosphogypsum (May and Sweeney 1984). The $H_3PO_4$ is separated by filtration, and the phosphogypsum-water slurry, usually containing about 30 % solids, is pumped to an impoundment where it is allowed to settle. As the gypsum settles, a small dragline removes some of it for raising the height of the dikes. By this process, the gypsum settling impoundment, or "stack" as it is often called, increases in elevation. As a stack grows in height (up to 61 m) the area of the settling impoundment decreases until a point is reached where the pond capacity becomes too small and the pumping height requires too much energy. At this point (approximately 30 yr) the phosphogypsum stacks reach the end of their useful lives and are ready to be closed. During the slurry process, any $H_3PO_4$ trapped in the filtrate (2 % to 5 % of the total extracted $H_3PO_4$) and unreacted $H_2SO_4$ end up in the slurry, which also goes to the stack. Solutions associated with the slurry percolate through the stack into a cooling water pond, which supplies water to the process plant. Water from the pond is also used in various plant operations (e.g., filtering the gypsum) and is subsequently recycled to the cooling water pond (U.S. Bureau of Mines 1975).

As explained earlier, phosphogypsum is a byproduct of phosphoric acid production. For each metric ton of phosphoric acid produced, approximately 5 mt of phosphogypsum must be stockpiled. To date, more than 600 million mt has accumulated in Florida on approximately 2,035 ha, and phosphogypsum is continuing to accumulate at an estimated rate of 27 million mt/yr (FIPR 1992).

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The environmental impact of the phosphogypsum stack and cooling water pond has received considerable attention. The potential migration of metal and nonmetal ions, including anions such as sulfate, has been discussed by various regulatory agencies. The final Environmental Protection Agency (EPA) National Emissions Standards for Hazardous Air Pollutants (NESHAP) ruling on phosphogypsum was issued in June 1992 (Federal Register 1992). The ruling effectively prohibits all utilization of phosphogypsum except for research purposes that has been certified by the owner to contain less than 10 pCi/g radium 226, and requires that all phosphogypsum now stored in stacks, plus all future phosphogypsum production, remain in the stacks. The solid waste branch of EPA has also proposed regulation of phosphogypsum stacks and associated cooling pond systems. A primary concern of the solid waste division is the "pond water" associated with the phosphogypsum stack.

Another major issue currently under review is the procedure that should be used for closure of a phosphogypsum stack at the end of its use as a disposal area. To better define the migration of metal and nonmetal ions from a phosphogypsum stack, the U.S. Bureau of Mines Tuscaloosa Research Center has investigated the effect of various solutions on metal migration. This paper describes the placement of monitoring wells to study the migration of ions in an active phosphogypsum stack. Column leaching tests using synthetic rainwater and process cooling pond water were also performed to determine the amount of metal and nonmetal migration.

## Experimental

A 5.08-cm-OD by 91.44-cm-long clear plexiglass tube was used as the leaching column (American Association of State Highway and Transportation Officials 1982). A 750-g charge of phosphogypsum was used as the medium. Table 1 shows a size-screen analysis of the phosphogypsum sample. The phosphogypsum was from an active stack in the Central Florida Phosphate District. The column was filled with the media and tamped until no additional settling occurred. The synthetic rainwater used was made up based on Rosler and Lange's chemical composition of rainwater (Rosler and Lange 1972). The pH of this rainwater was 6.9.

Table 1. Particle size-screen analysis for phosphogypsum[1].

| Size, mesh | Weight, g | Pct |
|---|---|---|
| Plus 65 | 1.80 | 0.1 |
| Minus 65 plus 100 | 33.18 | 20.0 |
| Minus 100 plus 150 | 51.42 | 30.0 |
| Minus 150 plus 200 | 35.19 | 21.0 |
| Minus 200 plus 270 | 12.68 | 8.0 |
| Minus 270 plus 325 | 4.36 | 2.9 |
| Minus 325 | 30.13 | 18.0 |
| Total | 168.76 | 100.0 |

[1]Density = 1.5 g/mL.

Pore volumes were determined and the solutions were analyzed for metal and nonmetal concentrations by inductively coupled arc plasma (ICAP). One pore volume is the amount of liquid it takes to completely saturate a known quantity of material. For the experiments described in this paper, one pore volume for 750 g of phosphogypsum was 250.0 mL. Five pore volumes were taken from each leached column. It took approximately 24 h to collect one pore volume of leachate.

In January 1993, five clusters of three wells each, totaling 15 wells, were installed in a 195.4-ha active phosphogypsum stack. The cluster wells were drilled to approximately 7.6, 12.2, and 16.7 m, and all boreholes were sampled at 1.5-m intervals (fig. 1).

Figure 1 is a schematic diagram of the 195.4-ha active phosphogypsum waste stack. Cluster well positions and ground elevations are all approximate.

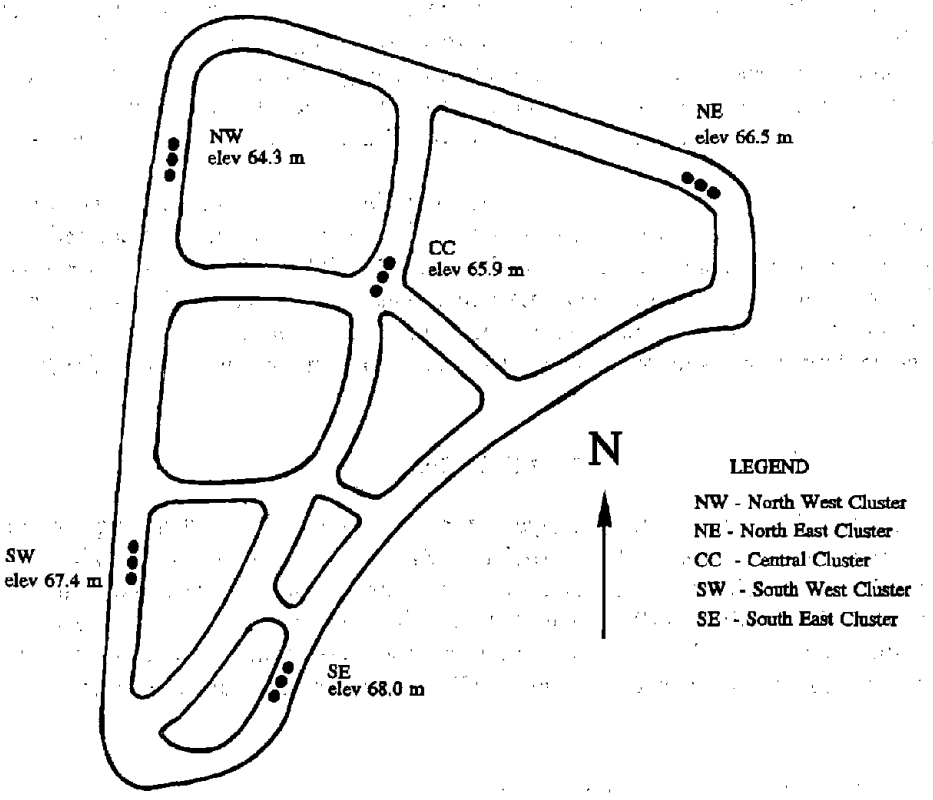


FIGURE 1.--Well locations and ground elevations for the South Pierce phosphogypsum waste stack.

The phosphogypsum drill cuttings were leached with solutions of synthetic rainwater and phosphoric acid plant process cooling pond water, thus simulating natural rainfall and active mining leaching of phosphogypsum stacks. The leachates from the ground water wells and laboratory column leaching tests were analyzed for metal and nonmetal ions. These data compare metal and nonmetal concentrations of filtrates from monitoring well fluids at the field site with filtrates from laboratory column leaching tests. In addition, these data will be used to predict the impact of leachate from phosphogypsum stacks on the ambient surface and subsurface environments.

## Results and Discussion

At an active phosphogypsum waste stack the process cooling pond water is used to transport the phosphogypsum to the top of the stack, where it settles out; the fluids return to the plant via return ditches and percolation through the stack. To monitor changes in the process cooling pond water as it percolates through the stack, wells were placed at various depths and locations. Table 2 shows an average ICAP chemical analysis of fluids collected from each of the monitoring wells from the active phosphogypsum stack for eight monthly (February-July) collections during 1993.

Table 2. ICAP analyses of monitoring wells fluids from the active phosphogypsum stack.

| Well[1] | pH | Metals, ppm | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Al | As | Ca | Cr | Cu | Fe | Hg | K | Mg | Mn |
| SW1 | 1.7 | 8.3 | 2.6 | 2,170 | 3.9 | 0.51 | 136 | 0.003 | 305 | 396.6 | 17.7 |
| SW2 | 1.7 | 4.2 | 4.3 | 2,443 | 2.6 | .50 | 89 | .003 | 311 | 342.0 | 15.2 |
| SW3 | 1.8 | 6.2 | 4.4 | 3,166 | 3.5 | .51 | 68 | .003 | 297 | 382.3 | 17.1 |
| NW1 | 1.7 | 4.7 | 2.4 | 1,716 | 2.3 | .50 | 50 | .003 | 278 | 212.7 | 10.1 |
| NW2 | 1.7 | 31.8 | 2.6 | 1,890 | 4.3 | .51 | 266 | .003 | 274 | 385.7 | 18.5 |
| NW3 | 1.7 | 5.2 | 2.5 | 1,963 | 3.2 | .51 | 46 | .003 | 290 | 285.0 | 12.5 |
| CC1 | 1.8 | 23.0 | 4.7 | 1,650 | 2.5 | .50 | 313 | .001 | 272 | 353.0 | 15.4 |
| CC2 | 2.0 | 19.9 | 4.3 | 1,810 | 3.7 | .50 | 261 | .001 | 300 | 365.0 | 16.2 |
| CC3 | 2.0 | 11.1 | 4.6 | 1,970 | 2.2 | .50 | 83 | .001 | 278 | 341.0 | 14.1 |
| NE1 | 1.7 | 44.7 | 2.5 | 1,513 | 2.8 | .51 | 134 | .003 | 298 | 302.3 | 13.1 |
| NE2 | 1.7 | 75.2 | 2.6 | 1,460 | 3.9 | .51 | 131 | .003 | 308 | 396.6 | 15.6 |
| NE3 | 1.7 | 37.5 | 2.3 | 1,873 | 3.3 | .50 | 107 | .003 | 341 | 356.0 | 14.3 |
| SE1 | 1.7 | 10.1 | 2.1 | 1,986 | 4.9 | .50 | 148 | .003 | 298 | 324.0 | 15.1 |
| SE2 | 1.8 | 5.3 | 2.6 | 2,716 | 3.9 | .50 | 101 | .003 | 304 | 348.0 | 14.5 |
| SE3 | 1.8 | 8.1 | 2.3 | 3,066 | 3.9 | .51 | 61 | .003 | 275 | 359.7 | 15.6 |

| | Metals, ppm (Cont.) | | | | | Nonmetals, ppm | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Na | Ni | Sr | Ti | Y | Se | Cl | F | P | Si | $SO_4$ |
| SW1 | 2,073 | 2.51 | 34.63 | 6.51 | 0.38 | 0.11 | 292.3 | 10,006 | 10,600 | 1,845 | 5,087 |
| SW2 | 2,110 | 1.33 | 32.66 | 6.00 | .26 | .11 | 258.3 | 7,820 | 9,540 | 1,716 | 3,310 |
| SW3 | 2,053 | 1.26 | 31.86 | 10.36 | .28 | .11 | 288.7 | 9,016 | 11,166 | 1,946 | 2,750 |
| NW1 | 2,190 | .49 | 30.86 | 6.58 | .45 | .11 | 243.3 | 8,026 | 6,366 | 1,597 | 4,193 |
| NW2 | 2,160 | 1.01 | 29.70 | 7.86 | .87 | .11 | 278.7 | 9,377 | 9,937 | 2,057 | 5,560 |
| NW3 | 2,203 | .83 | 28.36 | 9.35 | .39 | .11 | 979.3 | 7,976 | 7,876 | 1,746 | 4,233 |
| CC1 | 2,150 | .51 | 41.80 | 6.94 | .76 | .04 | 224.0 | 6,990 | 9,030 | 1,670 | 4,760 |
| CC2 | 2,100 | 3.35 | 31.80 | 7.90 | .40 | .04 | 208.0 | 6,360 | 9,350 | 1,500 | 4,170 |
| CC3 | 1,880 | .51 | 31.70 | 13.20 | .64 | .04 | 260.0 | 8,350 | 8,990 | 1,820 | 4,430 |
| NE1 | 2,260 | 2.90 | 35.83 | 9.18 | 1.25 | .11 | 270.3 | 9,050 | 8,063 | 1,926 | 5,947 |
| NE2 | 2,340 | .63 | 36.03 | 12.63 | .87 | .11 | 266.7 | 8,140 | 8,980 | 1,810 | 5,736 |
| NE3 | 2,130 | .61 | 35.80 | 7.68 | 2.42 | .11 | 268.7 | 10,633 | 9,253 | 2,323 | 5,210 |
| SE1 | 2,186 | .49 | 37.33 | 7.02 | .91 | .11 | 253.6 | 8,630 | 8,967 | 1,890 | 4,303 |
| SE2 | 2,003 | 1.06 | 38.96 | 9.12 | .31 | .11 | 284.3 | 9,140 | 9,656 | 2,090 | 3,843 |
| SE3 | 1,997 | .85 | 33.56 | 12.70 | .45 | .11 | 270.7 | 9,276 | 10,480 | 2,000 | 3,337 |

[1]All wells ending in a 1 are 16.7 m deep; all wells ending in a 2 are 12.2 m deep; all wells ending in a 3 are 7.6 m deep.

The data in table 2 show that only small changes occur to the fluids as they percolate through the active phosphogypsum waste stack. For instance, Ca concentration in the fluid from SW1 (the deepest well) was 2,170 ppm, compared with that observed in the fluid from SW3 (the shallowest well), 3,166 ppm. This trend for lower Ca concentrations at greater depth is observed for all well clusters. For the other ions, a variance is observed with no clear trend. It appears that calcium is precipitating; the exact reaction is unknown at this time. Possibilities include precipitation of CaF, chukhrovite [$(Ca_4SO_4AlSiF_{13})10H_2O$], or fluorosilicate compounds.

Table 3 shows ICAP analyses of a phosphogypsum composite sample from drill cuttings from the active phosphogypsum stack. Notice from table 3 that the concentrations of metal and nonmetal ions are considerably greater in the solids sample than in the well fluids with the exception of Mg, Cl, F, and P.

Table 3. ICAP analyses of the phosphogypsum composite sample.

| | ppm |
|---|---|
| Metals: | |
| Al | 1,300 |
| As | <300 |
| Ca | 190,000 |
| Cr | 72.5 |
| Cu | 195 |
| Fe | 940 |
| K | <1,000 |
| Mg | <100 |
| Mn | 9.2 |
| Na | 1,700 |
| Ni | 245 |
| Sr | 550 |
| Ti | 200 |
| Y | 20 |
| Nonmetals: | |
| Cl | 138 |
| F | 5,750 |
| P | 2,450 |
| Se | 0.413 |
| Si | 92,000 |
| $SO_4$ | 356,000 |

To simulate actual field conditions in the laboratory, a series of column leach tests was conducted using phosphogypsum and process cooling pond water from the active waste stack. The process cooling pond water normally has a pH range of 1.2 to 2.1 and is heavily laden with metal and nonmetal ions. In the construction of a phosphogypsum waste stack, this water is used as a transport mechanism to deposit phosphogypsum slurry to the desired cell on top of the stack. To determine any depth-dependent changes in leaching characteristics, samples were leached using material obtained at 1.5-m increments. Five apparent pore volumes of leachate were collected and analyzed for each column leach test. Table 4 shows the leach test analysis using process cooling pond water on samples collected from well CC1 located in the middle of the active phosphogypsum waste stack. Column leach tests for the other samples from the stack exhibited similar characteristics to those presented in table 4. The first pore volume shows a slight increase in pH from the 1.17 value to approximately 1.3. The last pore volume shows that the final pH is below the initial 1.17, showing that some acidic components are being leached. For the ions As, Cr, Cu, and Se, it shows that very small amounts of these ions are being leached by the pond water. The data also show that for Hg and Mn, there is essentially no leaching. For Ca and Fe, there is a reduction in concentration, showing that some precipitation is occurring. When looking at the anions, there is a large variance for Cl, Si, and $SO_4$. For fluoride and P (especially at depth), there is a reduction in concentration. Therefore, there is a high probability that compounds containing Fe, Ca, F, and P are precipitated.

To simulate field conditions existing after the abandonment of an active phosphogypsum waste stack, a series of column leach tests similar to the previous test series was conducted using synthetic rainwater with a pH of 6.5. This test series was designed to determine the effect of rainfall events on the leaching of contaminants from a newly deactivated phosphogypsum waste stack. Table 5 shows the leach test analyses for samples from well CC1 using synthetic rainwater. Leach tests using synthetic rainwater on other samples from the stack showed similar results. In each case, the initial pore volume collected contained the highest level of metals and anions. Each succeeding pore volume showed a decrease in cations and anions present. After the third pore volume, all cation and anion concentrations were very low. The results of this test series can be explained by assuming that the bulk of the migrating species moving through the columns is a function of residual process cooling pond waters trapped in the pore spaces of the phosphogypsum. The presence of the residual process cooling pond water makes it difficult to determine the net effects of rainwater percolating through the stack. Another test series to determine the long-term leaching effects is underway. In this study, the point at which the process cooling pond waters have been completely rinsed from the phosphogypsum will be determined. Also, the point at which compounds now precipitated in the phosphogypsum begin to leach will be determined. An attempt to identify these compounds will be made using a variety of techniques.

From table 4, moderate leaching of metal and nonmetal ions is occurring. These data coincide with previous phosphogypsum column leaching tests using mixed acid ($H_2SO_4$ and $H_3PO_4$) solutions to mimic process cooling pond water as the leaching solution (Carter 1992). The large concentration of ions that is found in the pore volumes is due to the metals and nonmetals in the cooling pond water. At almost every depth of the core material the contaminants are percolating through the phosphogypsum. For Cr and Fe, there is a slight increase in concentrations, and for the nonmetals Cl and P, there is a significant increase. In-depth analysis of the data from monitoring wells and leach column tests is continuing.

## Conclusions

Column leaching tests using phosphogypsum as the medium showed metal and nonmetal ions were mobile when either synthetic rainwater or process cooling pond water was used as the leaching solution. The majority of these ions are migrating out of the phosphogypsum in the first three pore volumes. This observation could have a great impact on phosphogypsum stack closure scenarios. A possible closure technique would be to rinse the entire stack with a nonpotable "clean" water, collect the leachate, and treat it.

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Table 4. ICAP analyses of process cooling pond water and pore volumes from well CC1 using a phosphate plant cooling pond water as the leachate.

| Hole depths and pore volume number | Metals, ppm | | | | | | | | Nonmetals, ppm | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | pH | As | Ca | Cr | Cu | Fe | Hg | Mn | Se | Cl | F | P | Si | $SO_4$ |
| Cooling pond water | 1.17 | 0.01 | 1,957 | 1.61 | 0.50 | 240 | 0.001 | 11.7 | 0.04 | 293 | 13,133 | 7,506 | 2,787 | 5,806 |
| 0-5 ft: | | | | | | | | | | | | | | |
| 1 | 1.32 | 3.6 | 1,710 | 1.30 | .57 | 155 | .001 | 12.5 | .1 | 202 | 10,100 | 7,810 | 2,400 | 4,490 |
| 2 | 1.23 | 3.4 | 1,900 | 1.75 | .56 | 213 | .001 | 12.5 | .1 | 352 | 13,500 | 7,860 | 3,100 | 6,610 |
| 3 | 1.18 | 3.5 | 1,750 | 3.85 | .56 | 218 | .001 | 11.5 | .1 | 284 | 12,200 | 7,410 | 2,910 | 5,780 |
| 4 | 1.07 | 3.7 | 1,870 | 1.30 | .57 | 214 | .001 | 12.4 | .1 | 336 | 12,500 | 7,590 | 2,990 | 6,030 |
| 5 | 1.06 | 3.6 | 1,860 | 3.84 | .56 | 204 | .001 | 11.9 | .1 | 366 | 13,600 | 7,660 | 2,980 | 6,580 |
| 5-10 ft: | | | | | | | | | | | | | | |
| 1 | 1.31 | 3.9 | 1,960 | 3.90 | .57 | 152 | .001 | 14.7 | .1 | 312 | 10,000 | 9,500 | 2,500 | 4,030 |
| 2 | 1.16 | 3.7 | 1,800 | 1.67 | .56 | 192 | .001 | 12.2 | .1 | 338 | 12,700 | 7,930 | 2,810 | 5,670 |
| 3 | 1.07 | 3.5 | 1,870 | 2.68 | .56 | 218 | .001 | 12.2 | .1 | 316 | 12,600 | 7,740 | 3,020 | 6,180 |
| 4 | 1.09 | 3.4 | 1,880 | 2.58 | .57 | 210 | .001 | 12.3 | .1 | 354 | 13,000 | 7,890 | 3,050 | 6,320 |
| 5 | 1.06 | 3.3 | 1,580 | 3.25 | .57 | 174 | .001 | 11.9 | .1 | 201 | 10,300 | 7,180 | 2,440 | 5,770 |
| 10-15 ft: | | | | | | | | | | | | | | |
| 1 | 1.32 | 3.7 | 1,630 | 2.65 | .57 | 167 | .001 | 14.6 | .1 | 221 | 9,370 | 9,000 | 2,260 | 5,120 |
| 2 | 1.17 | 3.4 | 1,620 | 3.29 | .57 | 166 | .001 | 12.1 | .1 | 203 | 10,700 | 7,430 | 2,660 | 5,670 |
| 3 | 1.11 | 3.3 | 1,840 | 3.40 | .56 | 230 | .001 | 12.6 | .1 | 347 | 13,300 | 7,840 | 3,010 | 6,230 |
| 4 | 1.06 | 3.4 | 1,910 | 2.29 | .56 | 222 | .001 | 12.2 | .1 | 305 | 13,100 | 7,890 | 3,070 | 6,580 |
| 5 | 1.04 | 3.6 | 1,740 | 4.25 | .57 | 186 | .001 | 11.8 | .1 | 313 | 13,300 | 7,560 | 2,690 | 6,610 |
| 15-20 ft: | | | | | | | | | | | | | | |
| 1 | 1.38 | 3.1 | 1,500 | 4.48 | .57 | 156 | .001 | 16.5 | .1 | 218 | 8,750 | 1,950 | 1,950 | 5,220 |
| 2 | 1.26 | 3.3 | 1,480 | 2.71 | .57 | 171 | .001 | 13.8 | .1 | 223 | 9,320 | 2,220 | 2,220 | 6,030 |
| 3 | 1.09 | 3.5 | 1,810 | 1.62 | .56 | 215 | .001 | 12.6 | .1 | 315 | 12,500 | 2,920 | 2,920 | 6,630 |
| 4 | 1.09 | 3.5 | 1,920 | 2.63 | .56 | 185 | .001 | 12.9 | .1 | 342 | 12,800 | 3,000 | 3,000 | 6,580 |
| 5 | 1.07 | 3.6 | 1,920 | 1.69 | .56 | 173 | .001 | 12.6 | .1 | 465 | 11,700 | 3,020 | 3,020 | 5,930 |

Table 4. Continued.

| Hole depths and pore volume number[1] | Metals, ppm | | | | | | | | Nonmetals, ppm | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | pH | As | Ca | Cr | Cu | Fe | Hg | Mn | Se | Cl | F | P | Si | $SO_4$ |
| Cooling pond water | 1.17 | 0.01 | 1,957 | 1.61 | 0.50 | 240 | 0.001 | 11.7 | 0.04 | 293 | 13,133 | 7,506 | 2,787 | 5,806 |
| 35-40 ft: | | | | | | | | | | | | | | |
| 1 | 1.32 | 3.8 | 1,440 | 4.62 | .62 | 194 | .001 | 17.4 | .1 | 243 | 9,090 | 2,100 | 2,100 | 5,950 |
| 2 | 1.27 | 3.4 | 1,480 | 4.32 | .61 | 196 | .001 | 13.5 | .1 | 269 | 10,400 | 2,340 | 2,340 | 6,210 |
| 3 | 1.14 | 3.3 | 1,720 | 5.78 | .61 | 205 | .001 | 11.7 | .1 | 342 | 12,300 | 2,790 | 2,790 | 6,430 |
| 4 | 1.12 | 3.5 | 1,770 | 5.56 | .60 | 194 | .001 | 11.6 | .1 | 335 | 12,400 | 2,850 | 2,850 | 6,470 |
| 5 | 1.06 | 3.4 | 1,820 | 2.70 | .60 | 193 | .001 | 11.4 | .1 | 356 | 12,700 | 2,940 | 2,940 | 6,480 |
| 40-45 ft: | | | | | | | | | | | | | | |
| 1 | 1.89 | 4.2 | 1,500 | 6.73 | .47 | 173 | .001 | 18.2 | .1 | 362 | 9,310 | 2,010 | 2,010 | 5,990 |
| 2 | 1.30 | 3.7 | 1,450 | 3.12 | .46 | 174 | .001 | 14.6 | .1 | 349 | 9,900 | 2,120 | 2,120 | 6,200 |
| 3 | 1.21 | 3.5 | 1,660 | 2.22 | .45 | 219 | .001 | 11.6 | .1 | 379 | 12,200 | 2,610 | 2,610 | 6,450 |
| 4 | 1.15 | 3.5 | 1,710 | 3.51 | .45 | 232 | .001 | 12.2 | .1 | 397 | 12,700 | 2,700 | 2,700 | 6,500 |
| 5 | 1.14 | 3.5 | 1,750 | 2.44 | .45 | 235 | .001 | 12.4 | .1 | 412 | 13,000 | 2,780 | 2,780 | 6,630 |
| 45-50 ft: | | | | | | | | | | | | | | |
| 1 | 1.47 | 3.9 | 1,510 | 4.60 | .46 | 196 | .001 | 17.3 | .1 | 376 | 9,620 | 2,130 | 2,130 | 6,410 |
| 2 | 1.31 | 3.4 | 1,550 | 2.76 | .46 | 198 | .001 | 15.1 | .1 | 358 | 9,910 | 2,300 | 2,300 | 6,280 |
| 3 | 1.20 | 3.7 | 1,730 | 2.45 | .45 | 232 | .001 | 12.3 | .1 | 419 | 12,100 | 2,730 | 2,730 | 6,240 |
| 4 | 1.21 | 3.5 | 1,690 | 1.20 | .45 | 234 | .001 | 12.3 | .1 | 374 | 12,200 | 2,710 | 2,710 | 6,260 |
| 5 | 1.17 | 3.5 | 1,720 | 1.88 | .45 | 240 | .001 | 11.8 | .1 | 378 | 12,600 | 2,800 | 2,800 | 6,430 |
| 50-55 ft: | | | | | | | | | | | | | | |
| 1 | 1.32 | 3.8 | 1,610 | 6.27 | .47 | 205 | .001 | 18.2 | .1 | 355 | 9,520 | 2,300 | 2,300 | 6,080 |
| 2 | 1.31 | 3.6 | 1,560 | 3.14 | .46 | 202 | .001 | 15.8 | .1 | 353 | 10,200 | 2,340 | 2,340 | 6,320 |
| 3 | 1.18 | 3.4 | 1,740 | 1.20 | .45 | 235 | .001 | 12.6 | .1 | 383 | 12,000 | 2,750 | 2,750 | 6,360 |
| 4 | 1.22 | 3.4 | 1,690 | 2.84 | .45 | 234 | .001 | 11.8 | .1 | 417 | 12,300 | 2,720 | 2,720 | 6,380 |
| 5 | 1.17 | 3.9 | 1,760 | 2.00 | .45 | 245 | .001 | 12.3 | .1 | 399 | 12,400 | 2,800 | 2,800 | 6,380 |

[1]Data for depths 20-35 ft have been omitted due to page limitations.

Table 5. ICAP analyses of synthetic rainwater and pore volumes from well CC1 using synthetic rainwater as the leachate.

| Hole depths and pore volume number | Metals, ppm | | | | | | | | Nonmetals, ppm | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | pH | As | Ca | Cr | Cu | Fe | Hg | Mn | Se | Cl | F | P | Si | $SO_4$ |
| Synthetic rainwater | 6.5 | 0.02 | 0.24 | 0.03 | 0.02 | 0.004 | 0.001 | 0.001 | 0.001 | 1.51 | 0.1 | 0.21 | 0.77 | 1.0 |
| 0-5 ft: | | | | | | | | | | | | | | |
| 1 | 2.1 | 2.02 | 1,370 | 1.30 | 0.56 | 5.79 | 0.001 | 5.06 | 0.10 | 156 | 3,880 | 3,180 | 1,020 | 2,290 |
| 2 | 2.3 | .39 | 735 | 1.30 | .56 | 4.28 | .001 | 1.84 | .10 | 25.5 | 490 | 1,140 | 118 | 2,160 |
| 3 | 2.6 | .20 | 753 | 1.30 | .56 | 2.22 | .001 | .48 | .10 | 17.8 | 467 | 408 | 153 | 1,700 |
| 4 | 3.1 | .02 | 650 | 1.30 | .56 | 1.62 | .001 | .05 | .01 | 16.4 | 47.4 | 77 | 22 | 1,480 |
| 5 | 3.3 | 2.36 | 676 | 1.30 | .56 | .59 | .001 | .05 | .01 | 10.0 | 40.1 | 55 | 3 | 1,480 |
| 5-10 ft: | | | | | | | | | | | | | | |
| 1 | 1.9 | 1.77 | 1,480 | 1.30 | .74 | 15.20 | .001 | 6.82 | .10 | 221 | 4,590 | 4,480 | 1,210 | 2,560 |
| 2 | 2.0 | .8 | 1,110 | 1.30 | .56 | 8.63 | .001 | 3.21 | .10 | 51.9 | 2,490 | 2,350 | 650 | 2,210 |
| 3 | 2.6 | .06 | 693 | 1.30 | .56 | 4.84 | .001 | .05 | .01 | 17.5 | 162 | 224 | 53 | 1,680 |
| 4 | 2.9 | .07 | 675 | 1.30 | .56 | 4.31 | .001 | .05 | .01 | 10.0 | 77.3 | 111 | 21 | 1,620 |
| 5 | 3.1 | .02 | 675 | 1.20 | .56 | 2.99 | .001 | .05 | .01 | 16.9 | 61.9 | 70.5 | 13 | 1,580 |
| 45-50 ft: | | | | | | | | | | | | | | |
| 1 | 1.8 | 1.53 | 1,000 | 1.2 | .45 | 29.1 | .001 | 9.1 | .10 | 260 | 2,160 | 4,060 | 495 | 3,840 |
| 2 | 1.9 | .86 | 929 | 1.2 | .45 | 17.0 | .001 | 6.57 | .01 | 262 | 1,600 | 3,000 | 362 | 3,350 |
| 3 | 2.4 | .18 | 733 | 1.2 | .45 | 4.5 | .001 | .36 | .01 | 27.8 | 210 | 463 | 59 | 1,920 |
| 4 | 2.9 | .05 | 679 | 1.1 | .46 | 4.7 | .001 | .05 | .01 | 24.4 | 51 | 445 | 8 | 1,600 |
| 5 | 3.2 | .02 | 673 | 1.2 | .46 | 1.8 | .001 | .05 | .01 | 23.8 | 36 | 67 | 2 | 1,530 |
| 50-55 ft: | | | | | | | | | | | | | | |
| 1 | 1.9 | 1.40 | 1,170 | 1.2 | .46 | 29.4 | .001 | 8.95 | .10 | 250 | 4,450 | 3,720 | 977 | 3,800 |
| 2 | 2.0 | .89 | 1,040 | 1.2 | .45 | 16.4 | .001 | 6.38 | .01 | 150 | 3,350 | 2,760 | 753 | 3,370 |
| 3 | 2.4 | .08 | 752 | 1.2 | .45 | 2.9 | .001 | .46 | .01 | 28 | 253 | 419 | 86 | 1,910 |
| 4 | 2.8 | .05 | 715 | 1.1 | .45 | 3.1 | .001 | .05 | .01 | 25 | 70 | 172 | 34 | 1,680 |
| 5 | 3.1 | .03 | 693 | 1.2 | .45 | 2.0 | .001 | .05 | .01 | 24 | 44 | 73 | 12 | 1,590 |

Double line indicates data between depths 10-45 ft have been omitted due to page limitations.

# GEOPHYSICAL METHODS: REMOTE TECHNIQUES APPLIED TO MINING-RELATED ENVIRONMENTAL AND ENGINEERING PROBLEMS[1]

T. E. Ackman and K. K. Cohen[2]

**Abstract:** Acid mine drainage (AMD) and subsidence are two of the serious environmental and engineering problems associated with active, abandoned, and recently active mine lands. Geophysical methods are available to simply and inexpensively determine the location and extent of the problem area, to monitor the progress of remediation work, and to assess the long-term impact of mining-related environmental and engineering problems. Various techniques can be used to locate hydrologic flows through buried spoil, to study combustion areas in coal waste bank fires, and to detect the subsurface conditions that lead to the occurrence of mine subsidence. The utility of two of these geophysical tools has been demonstrated in specific case studies: the use of terrain conductivity to facilitate stream sealing and the application of seismic techniques to characterize longwall mine overburden conditions. Other geophysical methods are discussed in terms of their potential application to mining-related problems.

## Introduction

The engineer implementing remediation programs for recently active or long abandoned mines is confronted with a wide spectrum of geotechnical problems. The solutions to many of these problems require detailed information on subsurface conditions. Geophysical tools for determining subsurface conditions include electromagnetics, electrical resistivity, magnetics, seismic refraction and reflection, gravity, and borehole geophysics (table 1). Several of these methods and their application to mining-related environmental problems are discussed in this paper. For a more comprehensive description of geophysical surveying methods, the following references are recommended: Kearey and Brooks (1991); Dobrin and Savit (1988); Telford et al. (1983); and Parasnis (1973). Ward (1992) is recommended for a review of the application of geophysical tools to a myriad of different near-surface environmental and engineering problems.

## Theory and Application of Geophysical Techniques

### Electromagnetics

Many different electromagnetic (EM) techniques exist, but all are based upon measuring the response of the earth to electromagnetic fields propagating through the earth. The electromagnetic field can be naturally occurring or manmade. Examples include ground conductivity surveying, the very low frequency (VLF) method, ground penetrating radar (GPR), magnetotellurics, time-domain electromagnetics (TDEM), and audio-frequency magnetics (AFMAG).

**Ground Conductivity.** Ground conductivity profiling (McNeil 1992) is an induction technique. No ground contact is necessary. The equipment for this system generates an electromagnetic field that induces a secondary field in the ground; the ratio of the secondary field to the primary field is proportional to bulk ground conductivity. Transmitter and receiver coils are the basic components of these systems. The earth volume sampled is a function of coil separation, operating frequency, and apparent conductivity of the ground. A number of different systems are commercially available, allowing conductivity mapping from shallow depths of

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| Table 1. Summary of selected geophysical techniques applicable to mine land reclamation. | | |
|---|---|---|
| Technique | Physical basis | Application |
| Electromagnetics | | |
| Ground conductivity | Measures induced EM field in ground to determine bulk terrain conductivity. | Stream sealing (Ackman et al. 1989). |
| VLF | Uses induced EM produced by low frequency radio transmitters located throughout the world and measures secondary field produced in conducting body to identify its location. | Water-filled fractures (Schueck et al. 1993). |
| GPR | Utilizes a radar pulse directed into the earth to image the earth in terms of boundaries of varying electrical properties. | Metal mine shafts (Friedel et al. 1990). |
| Resistivity | Measures the voltage across electrodes in contact with the ground while applying current to two other electrodes, to determine apparent resistivity of the earth. | Coal mine shafts (Gallagher et al. 1978). |
| Magnetics | Measures perturbations in the earth's magnetic field to determine anomalies due to the presence or absence of magnetic material in the subsurface. | Metal mine workings (Cohen et al. 1992); fires, AMD (Schueck 1993). |
| Seismic reflection or refraction | Measures the propagation of seismic waves in the subsurface to image the subsurface in terms of layers with varying velocities or layers with contrasting velocity and/or densities. | Underground coal mine workings (Steeples and Miller 1992). |
| Gravity | Measures perturbations in the earth's gravitational field to determine anomalies due to subsurface bodies with density contrasts. | Underground iron ore mine workings (Ghatge and Waldner 1991). |
| Borehole Geophysics | Uses various borehole tools to determine the physical properties of material adjacent to borehole; crosshole measurements determine the distribution of physical properties between two or more boreholes. | Hydrogeology of reclaimed strip mines (Diodato and Parizek 1992) |

3 m (10 ft) or less, to over 45 m (150 ft). Readings are made at specified stations, and the data are usually presented in profile or contour map form. Advantages of these systems include their portability, the rapidity of surveying, the relatively low cost of the equipment, and the good representation of bulk ground conductivity obtained in conductive earth. Disadvantages include a limited depth of sounding (though adequate for many mining-related problems), unreliable results in highly resistive ground, and interference from manmade objects.

**VLF.** The VLF (very low frequency) method (McNeil 1992) is a tilt-angle technique in which the VLF receivers utilize low-frequency electromagnetic energy (15 to 25 kHz) generated by radio transmitters for long-range communications and navigational systems. Contact with the ground is not necessary. At large distances from the transmitter, the electromagnetic waves are planar and induce ground current and a secondary electromagnetic field in conductive earth. The VLF unit is a receiver that tunes into this frequency energy. To complete a survey, traverses are made with the VLF unit in a direction perpendicular to the strike of the conductive body, and at right angles to the transmitted electromagnetic waves. Measurements are made of the in-phase (tilt-angle) and quadrature (ellipticity) portion of the vertical magnetic secondary field. Traversing a conductive body with vertical dip, the tilt angle of the field over the body will vary from negative (out of the ground) to positive (down into the ground). Zero tilt of the field lies over the conductive body. Thus, the position of a conductor can be located. Numerous VLF transmitting stations in the necessary frequency band are positioned around the world. These signals can be received thousands of kilometers from the station. Both ground and aerial surveys are possible. Station readings are usually plotted in a profile for the study area; measurements can also be filtered so that zero tilt values result in peaks, which can be plotted on a map and

contoured by standard procedures. The technique is useful in detecting faults and conductive fracture zones, conductive clay zones, and conductive ore bodies. The equipment is portable and relatively inexpensive, and surveying can be completed rapidly in a simple operation. Other advantages are a good depth of exploration in nonconductive ground and good response to small conductive bodies producing anomalies. However, the VLF method is largely qualitative, and highly conductive ground limits depth of exploration. Also, in some locations, an appropriate VLF transmitter station may not be available.

**GPR**. In a GPR system (Alsup and Simmons 1984), an antenna emits a radar pulse in the frequency band of 10 to 1000 mHz that is directed into the ground. The pulse propagates through the ground and is reflected, refracted, diffracted, or scattered back to the surface, where it is detected by a receiver antenna. Two-way traveltime of the returning energy is used to image the subsurface in terms of boundaries with contrasting electrical properties. Dielectric properties and conductivity of earth material control the propagation of this energy. Extremely high resolution can be achieved with GPR because of the high frequency of energy used. However, depth of penetration may be severely limited when water-saturated zones, clay zones, or highly conductive material such as salt or manmade metallic objects are encountered; these zones either attenuate or reflect most of the energy back to the surface. Depending on conditions, the depth of imaging may be as little as a few meters or as great as 300 m (1,000 ft). Surveying is usually done with the GPR unit towed over an area to develop a scan of the subsurface. These scans are analogous to seismic reflection records in many ways. The two-way traveltime for the energy returns are recorded and displayed as a profile scan in real time; traveltimes can be converted to depth. The technique is useful in detecting shallow soil inhomogeneities and buried manmade objects such as drums and pipelines. Advantages of the GPR technique are that it can produce high-resolution imaging of the shallow subsurface under optimum conditions and that it operates in real time. However, GPR equipment is expensive, subsurface conditions may not be conducive to imaging or may limit the depth of investigation, and manmade factors (buried pipes, powerlines) may cause interference.

**Magnetics**. In magnetometry (Hinze 1992), various elements of the earth's magnetic field are measured using a magnetometer. The earth's main magnetic field is approximately dipolar and is perturbed by near-surface rocks and material, which are magnetic to variable degrees. From these perturbations or anomalies in the expected magnetic field, subsurface conditions can be deduced. Because the field is a vector quantity, measured parameters include total field intensity, direction (declination and inclination), and field gradient. Gradient is obtained by measurement of the field at two locations, simultaneously, using two sensors separated by a fixed distance. Contact with the ground is not necessary, and both ground and aerial surveys are possible. Magnetic surveying commonly consists of measuring total field intensity values at stations along a survey line or at an array of stations that cover an area. Total field data must be corrected for daily changes in the earth's magnetic field, and depending upon application, separation of anomalies due to different sources may be necessary. The data can be used to determine shallow subsurface conditions of 3 m (10 ft) or less, or deep crustal conditions. Data can be displayed as profiles or as contour maps. The information can be used qualitatively to determine where magnetic sources are located. Quantitative models can be derived to determine dimensions, geometries, and depths of subsurface magnetic bodies. The magnetic surveying technique can be used to detect subsurface voids, to map bedrock, to map contacts between bodies of varying magnetic properties, to map fracture zones, and to locate metallic manmade objects such as well casings, drums, and pipelines. The advantages of the technique are that the equipment is inexpensive and portable, large areas can be rapidly surveyed, and data processing is relatively simple. However, cultural interference can generate noise, and a data set may produce nonunique solutions.

### Seismic Techniques

Seismic surveying (Steeples and Miller 1992; Pullan and Hunter 1992; and Lankston 1992) includes a number of different techniques based upon the propagation of seismic energy in the earth. Regardless of the methodology, seismic energy generated at or near the surface propagates through the earth, where it is refracted, reflected, and diffracted. Traveltimes of the returning energy are used to determine subsurface

conditions. Equipment consists of an energy source (sledge hammer, explosives, firearms, airgun, or others), a recorder, seismic detectors (usually geophones), and cable. In seismic refraction, returning arrivals refracted along subsurface boundaries can be used to determine a velocity distribution of the subsurface from which a model of subsurface conditions can be inferred. Seismic velocity must increase with depth to utilize the refraction technique. In seismic reflection, energy is reflected at subsurface boundaries at which there is a significant contrast in seismic velocity and/or density. Energy that returns to the surface is displayed on records in terms of two-way traveltimes, and the subsurface can be imaged as a function of reflecting boundaries. The resolution of the techniques and depth of exploration depend on such factors as energy source and recording parameters. Voids in coal seams as shallow as 3 m (10 ft) have been imaged using the seismic reflection technique (Steeples and Miller 1992); in contrast, research consortiums have imaged to depths over 30 km (48 miles) and the crust-mantle boundary (Brewer and Oliver 1980). Under optimum conditions, seismic techniques can provide a rather detailed image of the subsurface. However, seismic equipment is relatively expensive, surveys can be time and labor intensive, and processing techniques can be complex, expensive, and time consuming. Culturally developed areas may be difficult to work in, and the techniques may not work well when subsurface boundaries are moderately or steeply dipping. Both vertical and horizontal resolution are limited.

## Environmental Problems - Geophysical Technology

Although problems differ significantly from one site to another, the questions asked are often similar. Due to site-specific conditions of mine sites, it is frequently difficult to recommend one geophysical technique over another. Table 2 lists common questions often encountered in remediation work and offers techniques that may potentially answer them. This evaluation is based on existing case studies; however, site-specific conditions may allow other applications not listed in this table. AMD and subsidence rank high among abandoned mine land problems and are discussed in more detail below.

AMD problems raise questions relative to hydrologic flow path through pyritic spoil or coal refuse piles, location(s) of ground water entering or leaving a disturbed area (mine site), depth to ground water, locations of mine voids and features such as water-filled fractures, and extent of inundation. Geophysical techniques can provide qualitative and quantitative answers to some of these questions and only qualitative answers to others. In addition, creative approaches incorporating a combination of several techniques are currently being investigated.

Subsidence of mine lands usually involves two basic issues: (1) risk assessment and prediction, i.e., which mined areas are susceptible and where and when subsidence is imminent, and (2) remediation of the affected area. Specific information that could be acquired using geophysics includes general areal extent of workings; thickness and other physical properties of the overburden; depth and dimensions of mine voids and pillars left for support; and whether mine voids are open or filled with air, water, or rubble.

## Case Studies

### Repairing Stream Channels to Reduce Water Loss into Underground Mines

Multiple zones of infiltration (natural and induced fractures) normally exist in stream channels overlying shallow (approximately 60 m (200 ft) or less) mined areas. These may cause water-handling and/or acid drainage problems in underground mines, as well as loss of stream flow. Total or partial water loss can occur within a stream channel. Partial losses are often undetectable by visual observation, making it difficult to accurately locate loss points. Therefore, conventional methods of stream repair, such as plastic membranes or clay and rock rip-rapping, are inefficiently applied to both damaged and undamaged portions of the stream channel.

Table 2. Common questions and potential geophysical techniques.

| | Electromagnetic techniques | Resistivity | Magnetics | Seismic techniques | Gravity |
|---|---|---|---|---|---|
| What is the areal extent of mine workings? | X | X | | X | |
| Where are the mine voids located? | X | X | | X | X |
| What is the depth to the mine workings? | | X | | X | |
| What is the extent (location) of the mine pool? | X | | | | |
| Where are the resources that were left in place (pillars, highwall)? | | | | X | |
| What is the depth/nature of the ground water table? | X | X | | X | |
| What is the character of the overburden? | X | X | | X | |
| What is the character of the mine workings (collapsed, water-filled, burnt)? | | | X | X | |
| Where are the fractures in the overburden? | X | X | | | |
| What is the character of the spoil? | X | | X | | |

The approach initially developed by the U.S. Bureau of Mines (USBM) accurately locates loss zones by using electromagnetic terrain conductivity surveys and stream gaging. Ground water is transported to and from a stream through fractures intersecting the stream bottom; consequently, the ground beneath these fracture zones exhibits increased conductivity due to saturation. Conductivity is measured at fixed spacings, typically 7.6 to 15 m (25 to 50 ft), depending on the instrument and observation depth while the surveyor is walking the stream channel. The zones of high or anomalous conductivity are targeted as potential stream loss zones. Because of natural (e.g., presence of clays) or cultural (e.g., buried metal) interferences that can affect terrain conductivity data, stream-gaging stations are then established around the targeted stream loss zones to confirm the results of the geophysical measurements.

At one site where this approach was used, a stream was in the path of an advancing longwall panel. The direction of the stream channel was at an oblique angle to the advancing longwall face (fig. 1), which would pass beneath a segment of the stream at a depth of approximately 60 m (200 ft). Terrain conductivity stations were established on 10-m (33 ft) intervals within the stream channel and overlapped the headgates and tailgates (supported sides) of the panel. Surveys were completed before, during, and after passage of the panel, using a terrain conductivity instrument with effective depths of about 3 and 6 m (10 and 20 ft) (fig. 2).

A positive conductivity anomaly was identified between stations 26 and 32 prior to mining (fig. 2A). A land survey parallel to the stream also depicted high conductivity readings adjacent to the anomalous stream zone, thus indicating that this was a natural fracture zone (fig. 2A). Shortly after the longwall panel passed beneath this anomalous zone (fig. 1), a 100% flow loss occurred between stations 30 and 32. Polyurethane grout was injected approximately 1 m (3 ft) beneath the surface of the streambed at the loss zone to seal

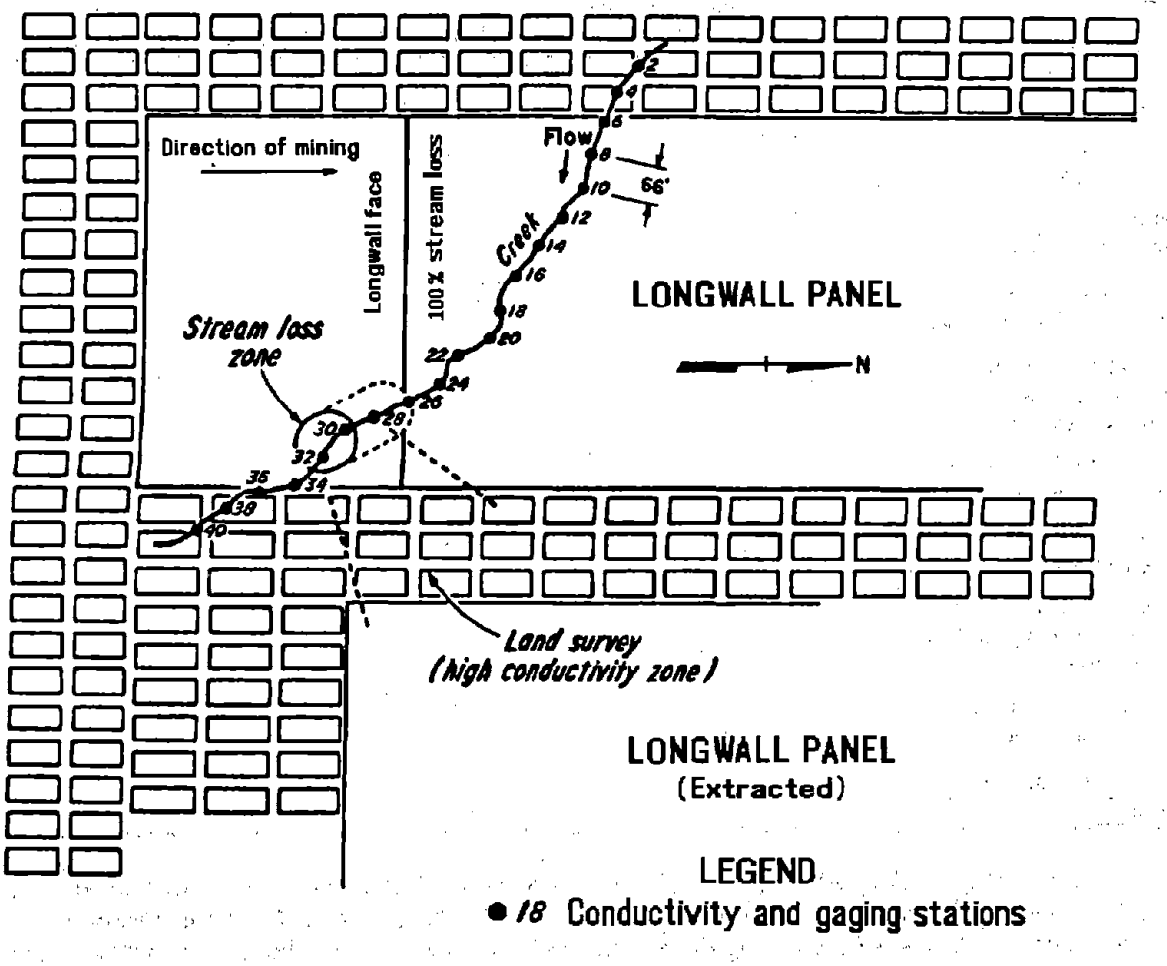


Figure 1. Map of longwall mining operation, stream position, and ground conductivity stations.

fractures. Surveys subsequent to grouting did not detect high or anomalous conductivity readings (fig. 2B), thus indicating that the fracture zone had been sealed beneath the stream channel. Gaging data also showed a 100% recovery across the damaged stream zone (Ackman et al. 1989).

## Seismic Refraction To Characterize Longwall Panel Overburden

Subsidence above longwall panels can produce adverse engineering and environmental effects such as damage to infrastructure and changes in the hydrologic regime. Subsidence is generally predictable, with the greatest percentage of ground disruption taking place shortly after mining. Identifying and quantifying this ground disruption is important from both an environmental and engineering perspective. The USBM recently designed a seismic refraction study to characterize overburden conditions above a longwall panel in Maryland (ICF Technology Inc./SRW 1992). This study identified and quantified changes that took place in the overburden shortly after mining. The study demonstrates that premining and postmining conditions in the overburden can be monitored through remote, surface geophysical techniques such as those discussed in this paper. When this geophysical information is combined with borehole data, it can result in an improved assessment of overburden conditions.

The study site was located in Garrett County, western Maryland, and was underlain by Pennsylvanian-aged strata of the Conemaugh and Allegheny Formations. The coalbed mined was the Upper Freeport. The coal panel being mined was about 2.5 m (8 ft) thick, 158.5 m (520 ft) wide and 1015 m (3,330 ft) long. Prior to undermining, a surface seismic refraction survey (line A-A', fig. 3A) was conducted along the centerline of the coal panel (approximate depth of 180 m (600 ft)). The survey was repeated after the longwall face had progressed to about 200 m (660 ft) beyond the line surveyed (fig. 3A).

The seismic line consisted of four spreads, with each spread about 67 m (220 ft) in length (fig. 3A). Each spread consisted of twelve geophones spaced at 6 m (20 ft) intervals. Seismic energy was generated by detonating small explosive charges (0.45 kg (1 lb) dynamite) in shot holes; the shot holes had nominal depths of 1.8 m (6 ft). Shots were positioned at the ends and in the middle of the spreads. A seismograph was used to record seismic wave traveltimes. First arrivals on the seismic records were identified and their traveltimes were used to generate a velocity distribution for the subsurface.

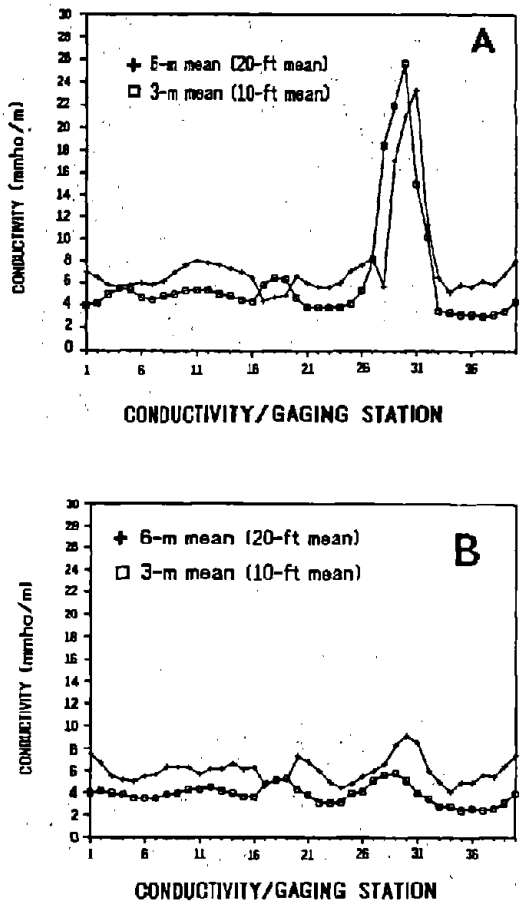


Figure 2. Ground conductivity survey before mining (A) and after mining and grout procedure (B).

From the initial premining survey, a two-layer subsurface model was derived in terms of velocities (fig. 3B). The uppermost layer with lower seismic velocities was interpreted as representing unconsolidated material and/or weathered bedrock. The second layer, with significantly higher velocities, was likely bedrock, known to be predominantly interbedded shale and sandstone at this location. As observed on figure 3B, depth to bedrock was determined as part of the seismic refraction model. Line A-A' was resurveyed (postmining survey) by seismic refraction techniques about 1 month after the initial survey and after the longwall face had passed the survey line. As before, a two-layer model was derived, but resultant velocities in both layers were lower (fig. 3B).

The velocity changes observed between the premining and postmining surveys were a result of changes that occurred in the overburden. Prior to the postmining survey, subsidence had occurred in the area; the greatest vertical change was a decrease of about 1.5 m (5 ft). The velocity changes observed in the overburden were likely induced by mining. Subsidence increases overburden fracture density, changes fluid saturation in overburden, and can change the density of the overburden rockmass. These factors, in part, determine elastic constants of the overburden. Elastic constants (Young's modulus and others) determine seismic wave velocity. An increase in fracture density, a decrease in fluid saturation (lowering of water table), or decrease in density of the overburden rock mass would each serve to lower seismic velocity values. One or all of these factors can explain the decrease in seismic velocities along the centerline of the longwall panel less than 1 month after undermining. Thus, one interpretation of these seismic data is that the lower velocities measured in the postmining survey were due to an increase in fracture density of the rockmass overburden. Consequently, an

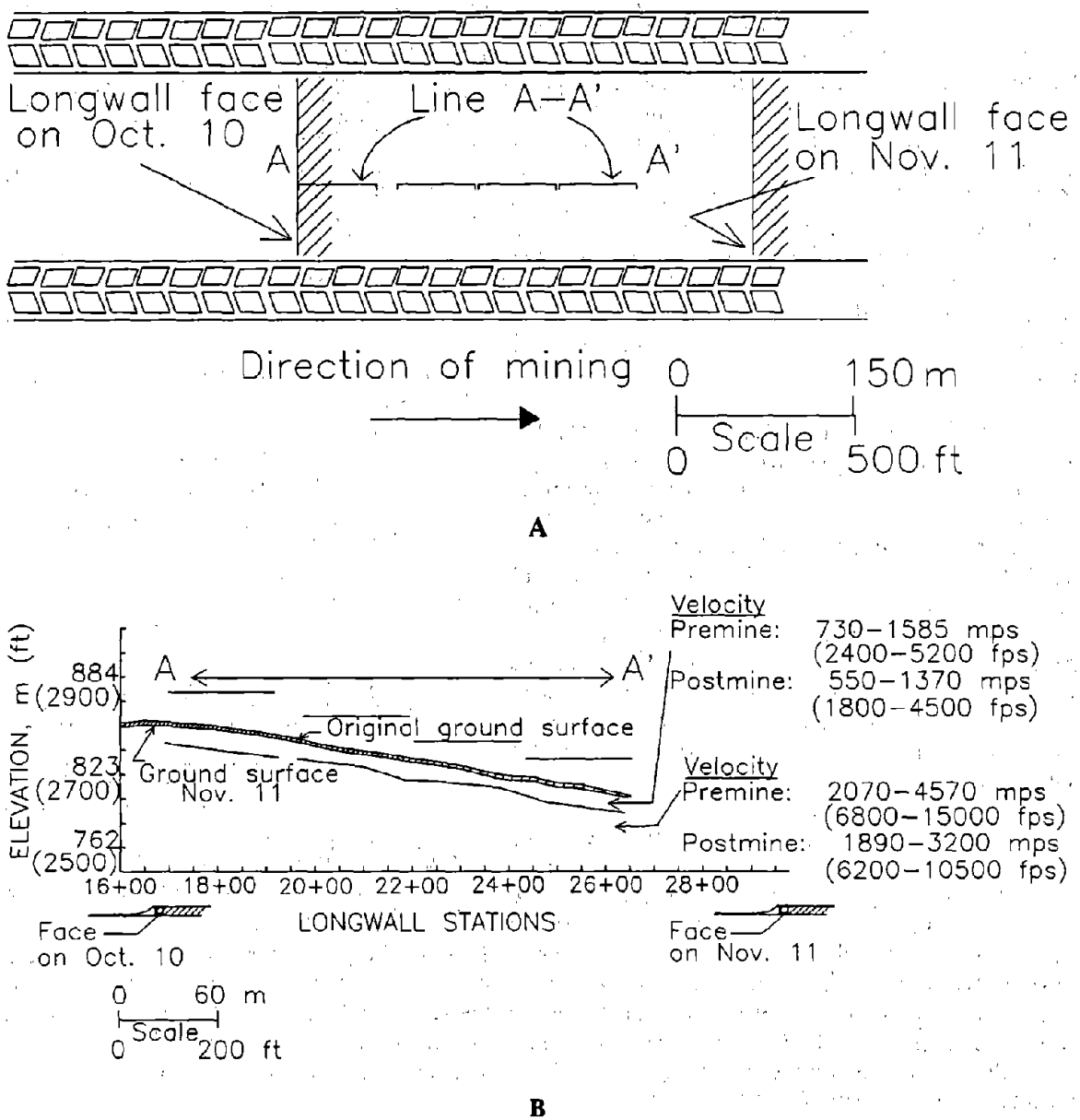


Figure 3. Map of and modeled profile through longwall mining operation where seismic refraction study was conducted. (Modified from ICF Technology Inc./ SRW 1992.) A. Map with surface seismic line (A-A') and subsurface coal mining operations (depth about 180 m (600 ft)). Changing position of longwall face indicated. B. Two-layer modeled profile through surveyed area with pre- and postmining seismic velocities indicated. MPS=meter per second and FPS=feet per second.

increase in fracture density could result in a lowering of the water table. Although not available for this study, borehole data could be used to substantiate these interpretations.

From these data, the distribution of overburden rockmass properties and changes in these properties were remotely measured over a large area using geophysical techniques. When combined with other information such as borehole data, they provide an improved assessment of premining and postmining overburden conditions.

## Summary

A variety of geophysical techniques are currently available for application to mine-related environmental problems. Examples of case studies are described in published literature (table 1), and several were discussed in this paper. With continuous improvements in geophysical hardware and software, the industry will see increased use of this technology for mine-related environmental problems. In addition, the integration of multiple techniques has value and will see increased use. For example, in the problem area of AMD, magnetometry has already been coupled with terrain conductivity to successfully locate buried, acid-producing refuse in reclaimed surface mines (Schueck 1993). Future coupling of conductivity with VLF to locate water-filled fractures may eliminate the need for stream gaging to identify stream loss zones and will permit a more rapid stream remediation. The remediation and restoration of mine lands will continue for years to come; geophysical techniques utilized by experienced personnel may provide a powerful tool to answer many of the remediation questions in a cost-effective, expeditious way. However, creative applications and/or integration of geophysical techniques will be required.

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# ACID MINE DRAINAGE ABATEMENT USING FLUIDIZED BED COMBUSTION ASH GROUT AFTER GEOPHYSICAL SITE CHARACTERIZATION[1]

Joseph Schueck,[2] Terry Ackman,[3] and Barry Scheetz[4]

**Abstract:** Pyritic coal refuse and pit cleanings buried in a 15-ha (37-acre) surface mine produce severe acid mine drainage (AMD). The pyritic material had been buried in discrete piles or pods in the backfill. The pods and the resulting contaminant plumes were initially defined using geophysical techniques and were confirmed by drilling. Fluidized bed combustion (FBC) ash, mixed with water to form a grout, was used in different ways to isolate the pyritic material from water and oxygen. In the first approach, grout was pressure injected directly into the buried pods to fill the void spaces within the pods and to coat the pyritic materials with a cementitious layer. A second approach used the grout to divert water from specific areas. Pods which did not accept grout because of a clay matrix were isolated from percolating water with a cap and trench seal of the grout. The grout was also used in certain areas to blanket the clay pit floor since clays are believed to be a primary source of aluminum at this site. In certain areas, the AMD migrates downward through fractures in the pit floor to the groundwater table. Grout was injected along the fractures in some of these areas to seal them. This would inhibit further AMD migration toward one of the receiving streams. The initial postgrouting water quality data have been encouraging.

**Additional Key Words:** AMD, magnetometry, electromagnetic terrain conductivity, VLF, grouting.

## Introduction

Effective AMD abatement first requires locating the acid production source(s). For example, pyritic material, such as coal refuse or pit cleanings, was often placed in discrete piles and buried at surface mine sites. Improper isolation of the piles from infiltrating precipitation or groundwater often results in severe localized AMD production. These pods and the resultant AMD plume can be detected using geophysical mapping techniques (Schueck 1988, 1990). Geophysical detection of the sources would be less successful at sites where the pyritic material was disseminated throughout the backfill.

Once the acid-producing materials buried within the backfill are delineated, abatement technology can be applied. Permanent isolation of pyrite from water or oxygen will stop AMD production. This paper describes the use of FBC ash as a grout to potentially isolate pyrite from oxygen and water. Geophysical applications are discussed as well.

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## Site Description

The project site is located in north-central Pennsylvania in the Sproul State Forest, Clinton County. This site was mined by mountaintop removal between 1974 and 1977. The Lower Kittanning coal seam was present in two splits separated by 3 to 6-m (10 to 20-ft) of clay. Only the upper split was mined, leaving the thick underclay as pavement. The coal was overlain by black shale capped by a sandstone unit. The black shale is suspected of being pyritic and potentially acid producing. The only source of groundwater is infiltrating precipitation. Acidic discharges developed soon after reclamation and were first noted when they resulted in a fish kill in 1978. One discharge is a toe-of-spoil seep which flows to Camp Run. Three other discharges flow into Rock Run as groundwater base flow. Total flows from the site are estimated to average 2.2 lps (35 gpm). The discharges completely destroyed 8 km (5 miles) of native trout streams. Water treatment was initiated by the operator, who subsequently went bankrupt; $9,400 in bonds were forfeited.

## Technical Approach

The primary objective of this AMD abatement approach was to isolate pyritic material from water and oxygen. Geophysical investigations were used to locate the pods of acid-producing materials, evaluate local hydrology, and monitor grout propagation. Monitoring wells were used to evaluate water quality within and adjacent to the site. Water quality was also monitored at the toe-of-spoil seep and in the receiving streams.

FBC ash was the material selected to be used in this isolation approach. FBC ash was chosen because it was available at a low cost, is alkaline, and is pozzolanic (or cementitious). A low-viscosity FBC ash grout was pressure injected into the geophysically targeted zones (pods) as a means of encapsulating the pyritic materials with a cementitious coating. In addition, diversion grouting was used to create zones of low permeability and redirect groundwater flow, so as to further reduce contact of the pyrite with water.

## Pregrouting Investigations

### Geophysical Investigation

Several geophysical mapping techniques were used for site characterization. Electromagnetic terrain conductivity (EM) measures ground conductivity at various depths and can be used to determine the presence and flow path of AMD within the spoil. Mine spoil is a poor electrical conductor, whereas AMD, because of its high ionic content, is an excellent conductor. The contrast readily provides for the detection of AMD within the spoil. Static water elevations across the site confirmed the flow path and direction. The depth of measurement depends on equipment and the configuration used. An EM-31 was used at this site to measure conductivity at depths of approximately 3 and 6 m (10 and 20 ft). Measurements were taken on a 7.6-m (25-ft) grid spacing across the 15-ha (37-acre) site. The values were contoured and an isoconductivity contour map was prepared (fig. 1). Only values greater than 8 mmho/m are shown in figure 1, depicting the flow path of the AMD. The groundwater drains from the northern (updip) portion of this mountaintop site to the southeast then splits into the east and south lobes (downdip portions of the mine site). The high conductivity values on both lobes indicate where pit water has pooled within the spoil. The EM mapping also indicated discharge plumes from the site at three locations where toe-of-spoil seeps are not present. Correlation of mining events, VLF mapping, drilling, and monitoring data provide evidence that the nonsurfacing plumes represent AMD draining

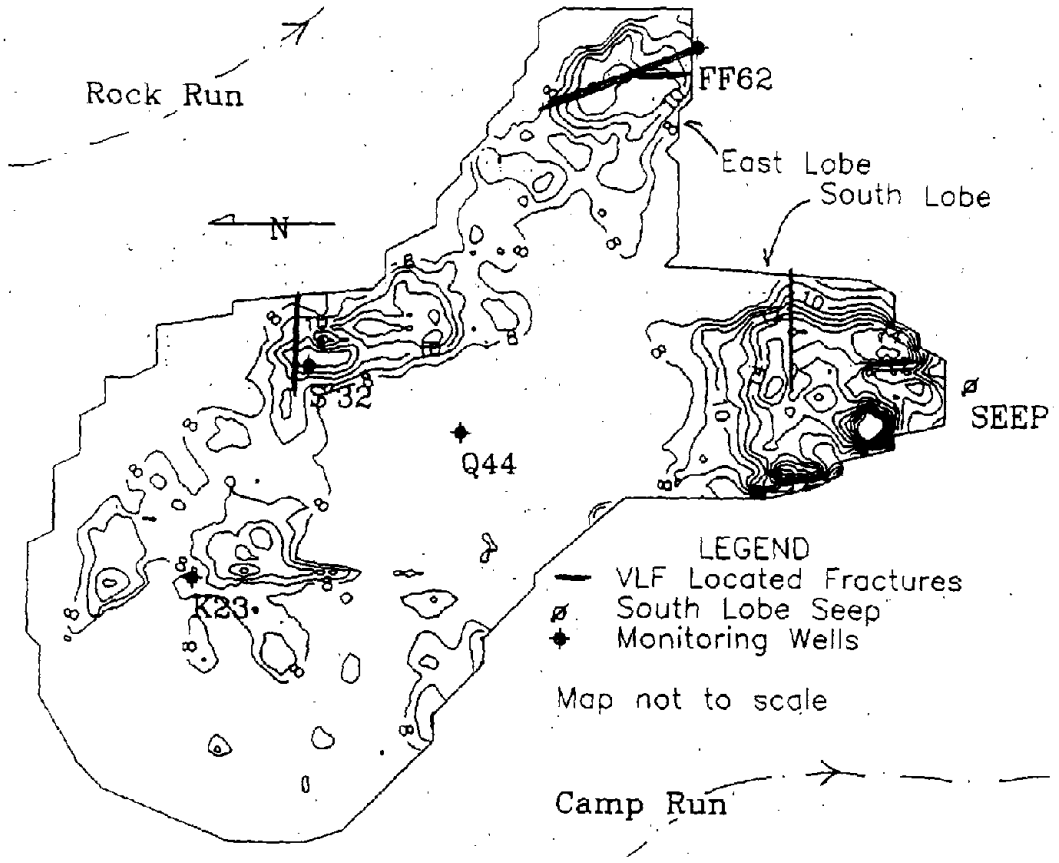


Figure 1. Isoconductivity contour map of site. Contour interval = 1 mmho/m.

through the pit floor via fractures and discharging as base flow into Rock Run.

A very low frequency (VLF) electromagnetic technique was used for fracture delineation. VLF utilizes the magnetic components of the electromagnetic field generated by military radio transmitters on the VLF frequency band (15 to 30 kHz). When the electromagnetic field from a VLF transmitter passes through a conductive body within the earth, secondary currents are induced. These secondary currents in turn generate a magnetic field, which is sensed by the instrument. However, only steeply dipping conductive bodies (e.g., water-filled fractures) are sensed. VLF surveys were performed on and off site as a means of locating water-filled fractures in the pit floor and in bedrock adjacent to the site. The onsite VLF mapping was completed on 3-m (10-ft) centers in the areas of the suspected discharge plumes. The mapping indicated fractures in the pit floor at locations that corresponded to the discharge plumes indicated by EM (fig. 1). Based on the available evidence, these fractures channel the AMD from the mine spoil to a

major joint system beneath the site. The joint system then conveys the AMD as base flow to Rock Run, some 76-m (250-ft) lower in elevation. The evidence includes timing of mining events, monitoring and grout well drilling and sample analyses, and discharge locations and water quality in Rock Run.

A "total field" proton precession magnetometer was also used in this study. The premise for the magnetic investigations is that the oxidation of pyrite (which is paramagnetic) produces a mineral phase change with magnetic properties that locally affects the earth's total field. A magnetometer or gradiometer detects total field and local changes, respectively, hence locating zones of pyrite oxidation. Magnetometer mapping was also done on the same 7.6-m (25-ft) grid spacing to determine the locations and configurations of concentrated pods or piles of pyritic materials such as coal refuse. Similar to the EM, the readings were plotted as a contour map (fig. 2). Only the high polarity portions of the anomalies are shown. In this case, the magnetic anomalies (closed contours) represent the locations of the pods of pyritic material buried beneath the surface. Each of these anomalies was later mapped on 3-m (10-ft) centers in the gradiometer mode to provide for better definition of the pile configuration. During drilling of the grout wells, samples were visually identified and magnetic susceptibilities were measured. This information, along with the gradiometer mapping, was used to

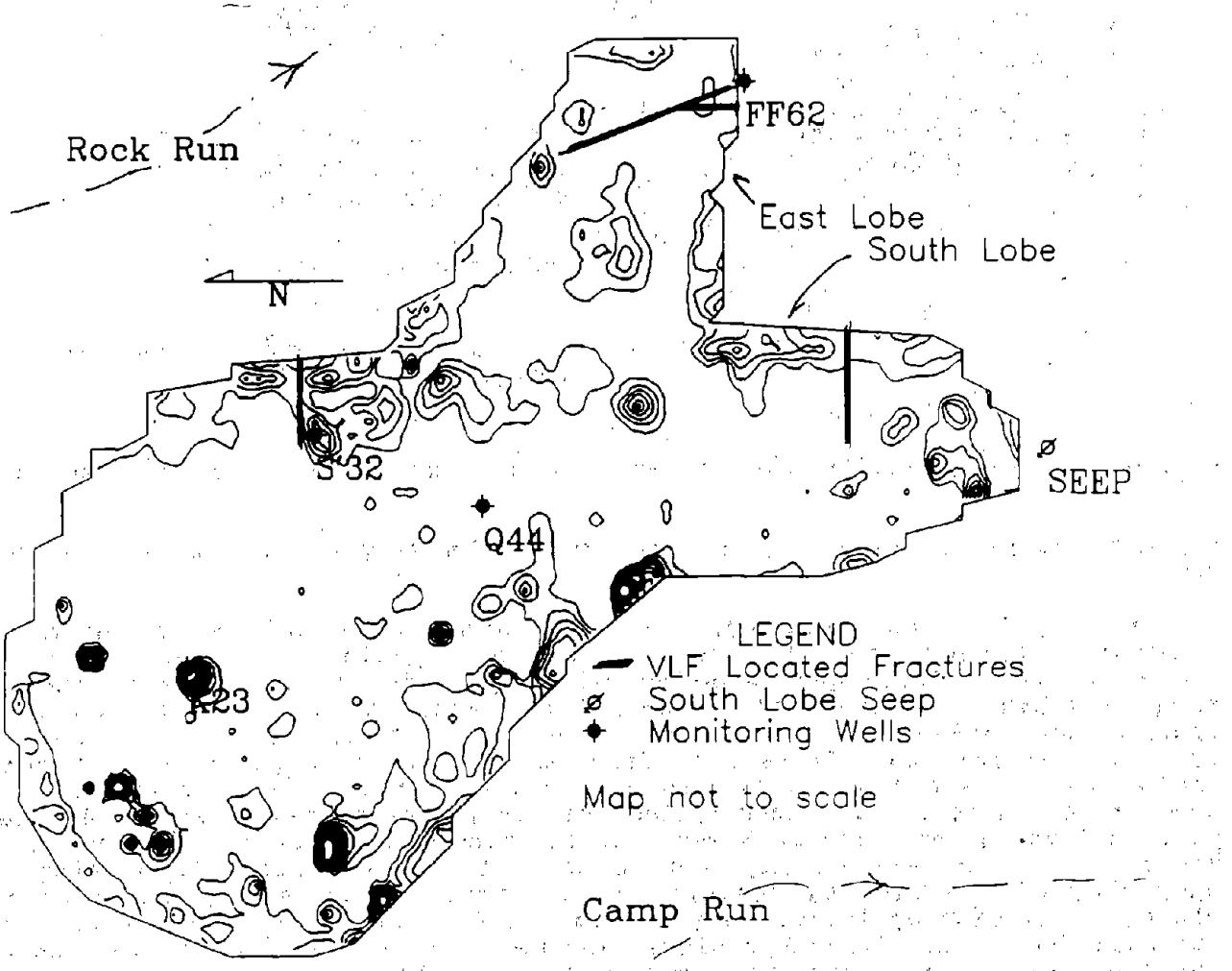


Figure 2. Magnetic anomaly map of site. Contour interval = 1 gamma.

delineate the depths and lateral extent of the oxidized pyritic materials.

When the magnetic anomaly map was overlain with the terrain conductivity map and compared to water quality data, it became apparent that the pods of refuse and pit cleanings were the sources of severe AMD production. The observed conductivity values were highest at or adjacent to the magnetic anomalies. The conductivity values gradually decreased in the direction of flow. This observation is consistent with severe AMD production within the buried pod of pyritic material followed by dilution as the AMD migrates further away from the pod. Water quality data confirmed that the most severe AMD was located within the pods and less severe AMD was found at distances downgradient from the pods.

## Water Sampling

Thirty-six monitoring wells were drilled in April 1990. These wells are located on and adjacent to the site. Monitoring wells located adjacent to the site were drilled to the bottom of the lower split of the Kittanning coal seam. The wells located onsite were drilled through spoil to the pit floor, with depths ranging from approximately 3 to 12 m ( 10 to 40 ft). Several additional monitoring wells were installed during and after the grouting operation. Monitoring well sampling was done on approximately a monthly schedule after their installation (weather permitting). Sampling included purging and bailing of the wells. Temperature, static water level, pH and conductivity measurements were completed in the field. The samples were then analyzed in the laboratory for 20 parameters.

Water sampling was also performed at the only surface discharge, which is a toe-of-spoil discharge located 60 m (200 ft) beyond the south lobe. The discharge rate varied from 0.13 to 1.3 lps (2 to 20 gpm). This discharge flows into Camp Run. The receiving streams, Camp Run, Rock Run, and Cooks Run, were also sampled on a monthly basis.

## Water Quality

Table 1 presents the average, high, and low pH values and selected metal concentrations for the toe-of-spoil seep and selected monitoring wells considered to be representative of the site. These data are based on 8 to 15 sampling events. Results from the monitoring wells show that pods of refuse or pit cleanings were indeed sources of severe AMD production. Concentrations

Table 1. Pregrouting water quality

| Range | pH | Acidity,mg/L | $SO_4$,mg/L | $Fe_{tot}$,mg/L | Al,mg/L | Cu,ug/L | Cr,ug/L | Cd,ug/L |
|---|---|---|---|---|---|---|---|---|
| Seep: | | | | | | | | |
| High | 4.00 | 3,520 | 4,368 | 466 | 304 | 958 | 269 | 74 |
| Low | 2.40 | 576 | 1,245 | 93 | 242 | 741 | 191 | 18 |
| Avg. | 2.60 | 2,830 | 2,642 | 271 | 290 | 855 | 232 | 59 |
| Monitoring Well FF62: | | | | | | | | |
| High | 2.60 | 6,740 | 5,773 | 1,500 | 562 | 989 | 301 | 167 |
| Low | 2.20 | 1,940 | 1,272 | 386 | 114 | 611 | 142 | 29 |
| Avg. | 2.32 | 4,088 | 3,458 | 877 | 255 | 730 | 221 | 98 |
| Monitoring Well K23: | | | | | | | | |
| High | 2.50 | 23,900 | 25,110 | 5,000 | 2,240 | 6,950 | 1,060 | 1,190 |
| Low | 2.00 | 9,420 | 3,560 | 1,120 | 758 | 6,710 | 443 | 270 |
| Ave. | 2.32 | 14,657 | 12,705 | 2,955 | 1,247 | 6,830 | 904 | 841 |
| Monitoring Well Q44: | | | | | | | | |
| High | 5.30 | 134 | 175 | 16 | 5 | 28 | 6 | 2 |
| Low | 3.20 | 10 | 62 | 1 | 1 | 22 | 4 | 1 |
| Avg. | 4.06 | 42 | 101 | 7 | 3 | 25 | 4 | 2 |

of the mine drainage parameters in the water in or adjacent to these piles are often several times greater than the concentration of the same parameters in the discharge or elsewhere on the site.

The toe-of-spoil seep and well FF62 (table 1) represent site discharge quality. Well FF62 is located within the discharge plume (as identified by EM survey) and is located offsite and adjacent to the east lobe. In addition, well FF62 is located in the lower split of coal, thus demonstrating fracture communication between the pit floor and lower coal seam.

The water quality of monitoring wells K23 and Q44 (table 1, fig. 1) illustrate the effect that pods of concentrated pyritic material can have on AMD production. Well K23 penetrates a pod of coal refuse that was located using magnetometry. Well Q44 is located near the center of the 15-ha (37-acre) site, but is located some distance from any of the mapped piles of pyritic material. The water quality differences of these two wells vary by orders of magnitude. This implies that the final discharge quality should improve considerably if the AMD production from the pyritic material at K23 and similar locations can be reduced.

The concentrations of ferric iron generated from these pods of pyritic material must also be considered. Garrels and Thompson (1960) have shown that pyrite is rapidly oxidized by ferric iron in the absence of oxygen and at low pH values:

$$FeS_{2(s)} + 14Fe^{3+} + 8H_2O \rightarrow 15Fe^{2+} + 2SO_4^{2-} + 16H^+. \quad (1)$$

In addition to total dissolved iron, concentrations of ferrous and ferric iron were determined in the lab. According to the monitoring well data, ferric iron concentrations from these pods commonly exceed 1,000 mg/L, with pH values close to 2.0. Once the water exits from the pod, such as the one at the K23 location, it must migrate through 450 m (1,500 ft) of spoil before discharging from the site. This ferric iron is available to rapidly oxidize any pyrite located in its flow path. A reduction in ferric iron formation should thus result in reduced pyrite oxidation.

Suspected sources of Al at this site include clay within the coal refuse and spoil as well as the 3 to 6-m (10 to 20-ft) thick clay unit that splits the coal seam and constitutes the pit floor. Clays are the most common of the sedimentary aluminum-enriched minerals, and the $Al^{+3}$ cation predominates in many solutions in which pH is less than 4.0 (Hem 1992). Concentrations of Al greater than 0.1 mg/L can be deleterious to aquatic life (EPA 1973). Table 1 shows that the concentrations of Al leaving the site in groundwater are three orders of magnitude greater than the 0.1-mg/L limit for aquatic life. The concentration of Al almost 3.2 km (2 miles) from the site in both receiving streams ranged from 4 to 8 mg/L.

Literature shows that Al solubility becomes limited at a pH above 4.5 and will precipitate out of solution (Snoeyink and Jenkins 1980, Stumm and Morgan 1981, Hem 1992). Consequently, this suggests that if the pH of the ground water within the spoil can be raised to a pH 4.5 or greater, the instream hazard from dissolved aluminum can be reduced.

## FBC Ash Characterization

In order to use high-sulfur fuels in power plants, atmospheric FBC systems have been developed. Coal and/or coal refuse is burned at 815 to 870 C in a limestone fluidized bed combustion system, limestone is calcined, and most of the sulfur oxides produced are adsorbed by the calcium oxide. The ash

from this system is different from traditional fly ashes generated from pulverized coal combustion. The formation of glassy phases in ash particles is retarded because of the lower combustion temperatures. The ash also exhibits high reactivity with water and has cementitious properties due to a high content of calcium and sulfate. The ash used for this project contained 12.51% $Al_2O_3$, 38.03% CaO, 23.91% $SiO_3$, and 16.02% $SO_2$ (Zhao 1993). Mixed with only water, FBC ash forms a low-strength cement. After 20 days at a water-to-solids ratio of 0.5, the ash develops a strength of 135 kg/cm$^2$ (1,920 psi). The compressive strength continues to develop slowly, up to slightly over 140 kg/cm$^2$ (2,000 psi) in about 90 days (Zhao 1993).

## Grout Injection

Only those pods of pyritic material identified by magnetometry were targeted for grouting. Each of the pods was mapped on 3-m (10-ft) centers using the magnetometer in the gradiometer mode for better pile definition. Grout well locations were based on the gradiometer maps. The wells were installed in August 1992 on 3-m (10-ft) centers using an air track drill rig with a 10.2-cm (4-in) diameter rock bit.

A total of 650 holes were drilled, with depths ranging from 3 to 9 m (10 to 30 ft). Almost 3,940 m (13,000 ft) of 6.3-cm (2-1/2 in) perforated schedule 40 PVC casing was used for this project.

Grouting operations began September 1, 1992, and continued through the end of October 1992. During that time period, 215 of the 650 grout wells were injected with grout. The rest of the wells were grouted when operations resumed in June 1993. The project was completed in August 1993.

The FBC ash was transported to the site in bulk from Fort Drum, NY. A mix ratio of 4.8 m$^3$ (6.25 yd$^3$) of FBC ash for every 3,780 L (1,000 gal) water was used. A conventional cement truck was used for mixing and transporting the grout to the injection wells. A portable grout pump, capable of producing pressures up to 42 kg/cm$^2$ (600 psi), was used for well injection. Each well was pumped until refusal. The amount of grout accepted by the wells ranged from less than 0.2 to 63.5 m$^3$ (0.3 to 83 yd$^3$).

Originally, only pressure injection directly into the pods was planned for the isolation of the pyritic materials. However, soon after the grouting operations began, it became evident that this method would not work for all the pods within the site. Several of the piles that refused to accept grout were identified as coal refuse with a clay matrix. It was decided to form a grout cap over these piles to ward off the infiltrating precipitation. The ground above the piles was excavated and the perimeter was trenched. The excavation was then pooled with fly ash grout. After hardening of the grout, these areas were backfilled and regraded.

Paving areas of the pit floor with a grout slurry along the AMD flow path was another diversion technique applied at this site. Chemical analysis of the water within the spoil showed Al concentrations in excess of 1,000 mg/L at various locations. The pit floor is clay and is considered to be a primary source of Al. Success in this approach depends upon the permeability of the spoil materials covering the pit floor.

A third method of diversion involved the use of the grout in the east lobe to seal targeted fractures in the pit floor. These were located geophysically using the VLF technique. Sealing these fractures would cause water within the spoil to flow to the south and eventually discharge into Camp Run instead of Rock Run.

## Postgrouting Investigations

### Postgrouting Water Quality

Since grouting continued until August 1993, a complete evaluation of the impacts the FBC ash grout had on AMD production from the entire site will not be possible until at least 1994. However, it is possible to comment on some local water quality changes associated with pods grouted in 1992.

Water quality improved in several monitoring wells located downgradient of the pods grouted in 1992. Abundant snowfall and heavy spring precipitation provided an opportunity for the site to be well flushed following the fall grouting effort. Concentration decreases of 50 to 90% in mine drainage parameters were noted in several of these downgradient wells. For example, Table 2 compares 11 pregrouting and 6 postgrouting samples of well S'32 (fig. 1) located within a pod grouted during September 1992. This well is also in the primary flow path of mine drainage produced further to the north and the postgrouting water quality is influenced by that drainage. The water quality improvements observed in this and other wells are encouraging. The reduction in sulfate levels suggests that at least part of the improvement is due to decreased AMD production rather than just neutralization. Other parameters commonly associated with AMD also show reduced concentrations. Water quality in wells not located downgradient of the 1992 grouting remained within observed pregrouting ranges. The postgrouting monitoring period includes all of 1993.

Table 2. Monitoring well S'32 water quality data

| Range | pH | Acidity, mg/L | $SO_4$, mg/L | $Fe_{tot}$, mg/L | Al, mg/L | Cu, ug/L | Cr, ug/L | Cd, ug/L |
|---|---|---|---|---|---|---|---|---|
| Pregrouting: | | | | | | | | |
| High | 3.60 | 6,300 | 6,380 | 1,860 | 455 | 1,410 | 415 | 323 |
| Low | 2.40 | 1,220 | 1,125 | 184 | 121 | 1,030 | 257 | 161 |
| Avg. | 2.91 | 3,701 | 3,195 | 934 | 307 | 1,220 | 348 | 223 |
| Postgrouting: | | | | | | | | |
| High | 3.60 | 2,900 | 3,761 | 934 | 166 | 435 | 109 | 38 |
| Low | 2.60 | 1,040 | 760 | 116 | 57 | 16 | 20 | 2 |
| Avg. | 2.96 | 1,486 | 1,610 | 452 | 112 | 281 | 78 | 11 |

Another encouraging sign is well FF62 (fig. 1) which is representative of the quality of water leaving the site from the east lobe and eventually discharging to Rock Run. Sample analyses show a 25 to 30% decrease in concentrations of mine drainage parameters in 1993 when compared to pregrouting data. Grouting upgradient of the toe-of-spoil seep from the south lobe was not completed until August 1993.

### Geophysical Monitoring of Grout Propagation

The feasibility of monitoring grout propagation with EM is being evaluated. EM surveys of a 75- by 50-m (250- by 150-ft) section of the east lobe were completed in June 1993 immediately before and after grouting. Grouting activities took a week to complete, and no rainfall occurred between the two mapping periods. Figure 3 was compiled by comparing the pregrouting and postgrouting data; it shows only the observed changes in ground conductivity. Only wells that accepted more than 0.7 $m^3$ (1 $yd^3$) of grout are shown in the figure. It is believed that conductivity changes show the extent of grout propagation within the spoil. Future ground-proofing activities (core drilling and excavated pits) are planned to confirm this observation. The observed conductivity increase is believed to be due to spoil saturation resulting from grouting (excess water associated with the thin grout mixture). This area was resurveyed in August 1993, 2 months after grouting was

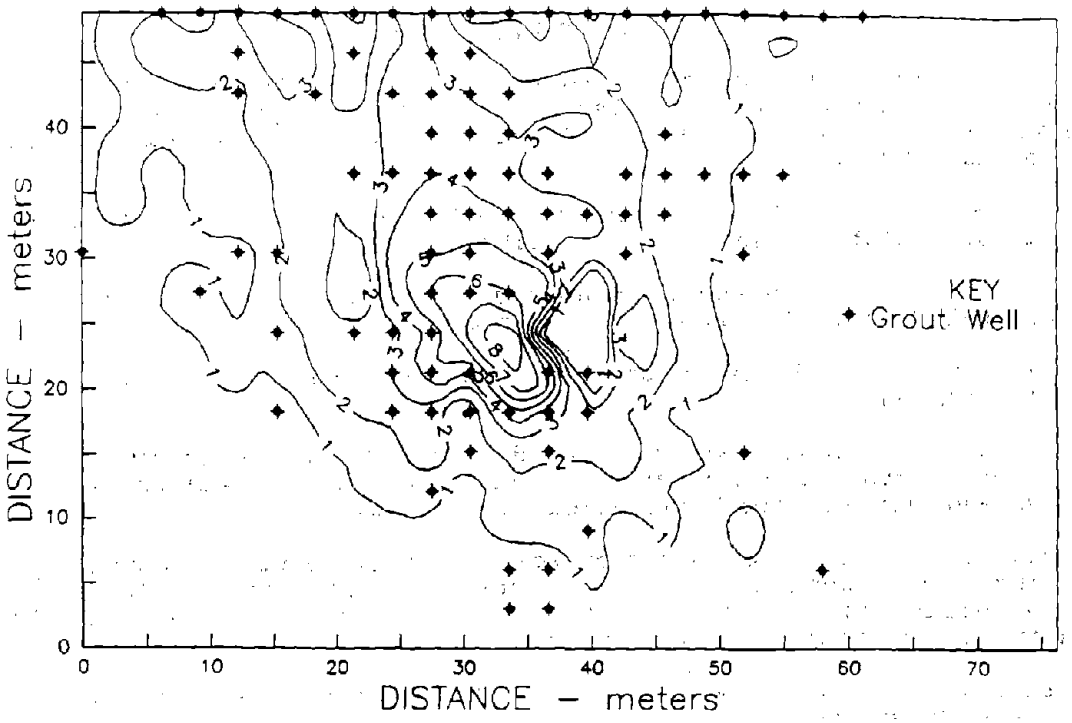


Figure 3. Grout propagation determination using EM. Contour Interval=1 mmho/m.

completed. When compared to the initial postgrouting survey, a general decrease in ground conductivity was observed in this area. It is believed that these observed changes result from the excess water draining away as well as hardening of the grout.

## Conclusions

Geophysical site characterization was critical to the development of the abatement plan. In the pregrouting evaluation, confirmation by drilling demonstrated the value of magnetometry for locating buried acid-producing materials. Magnetometry and EM data also showed a correlation between the pyritic pods and resultant AMD plumes, which was confirmed by drilling and water quality monitoring. EM and VLF data were also used to identify fractures beneath the pit floor. Data suggest that there is a potential for geophysical monitoring of grout propagation as well.

The diversion technique of excavating and placing a grout cap and trench seal is a positive, direct approach with the operator having total control over the application. However, in the pressure injection approaches, spoil permeabilities may influence grout placement. Low permeability targets are likely to accept only limited grout quantities.

In addition to continued groundwater and stream monitoring, remapping the site with EM is planned. This mapping will be compared with pregrouting mapping to determine if any large-scale changes in conductivity patterns have occurred. Such changes will then be correlated with abatement of AMD from the pyritic materials and changes in the flow patterns through the spoil. Although complete evaluation of the project will not be possible until at least 1994, the current trend in the monitoring data in the postgrouting evaluation phase is encouraging.

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# DISPOSING OF COAL COMBUSTION RESIDUES IN INACTIVE SURFACE MINES: EFFECTS ON WATER QUALITY[1]

Ann G. Kim and Terry E. Ackman[2]

**Abstract**: The disposal of coal combustion residues (CCR) in surface and underground coal mines can provide a stable, low-maintenance alternative to landfills, benefitting the mining and electric power industries. The material may be able to improve water quality at acid generating abandoned or reclaimed coal mine sites. Most combustion residues are alkaline, and their addition to the subsurface environment could raise the pH, limiting the propagation of pyrite oxidizing bacteria and reducing the rate of acid generation. Many of these CCR are also pozzolanic, capable of forming cementitious grouts. Grouts injected into the buried spoil may decrease its permeability and porosity, diverting water away from the pyritic material. Both mechanisms, alkaline addition and water diversion, are capable of reducing the amount of acid produced at the disposal site. The U.S. Bureau of Mines is cooperating in a test of subsurface injection of CCR into a reclaimed surface mine. Initially, a mixture of fly ash, lime, and acid mine drainage (AMD) sludge was injected. Lime was the source of calcium for the formation of the pozzolanic grout. Changes in water quality parameters (pH, acidity, anions, and trace metals) in water samples from wells and seeps indicate a small but significant improvement after CCR injection. Changes in the concentration of heavy metals in the water flowing across the site were apparently influenced by the presence of flyash.

## Introduction

Approximately 80% of the coal produced in the United States is used for electric power generation, and 10% to 15% of this amount is recovered as coal combustion residues (CCR). Since 1988, CCR have been produced at the rate of more than 100 million tons per year, including 60% fly ash, 16% bottom ash, 4.5% boiler slag, and 18% flue gas desulfurization sludge. The increased consumption of low-sulfur coal and the use of clean coal technologies are expected to increase the amount of CCR produced to 250 million tons by the year 2000 (Chugh 1992). Over the next 20 years, while coal use in some markets is expected to decline, demand for coal for electricity is projected to increase by more than 50% (Rose 1991, Alpert 1991).

CCR are generally classified as residual wastes to be placed in landfills. Several States currently permit or are developing regulations for disposal in surface or underground coal mines as a beneficial use. Fly ash is an alkaline material that has pozzalanic properties (Pozzolan: a finely divided silicious material that in the presence of moisture will chemically react with alkali and alkaline earth hydroxides at ordinary temperatures to form cementitious compounds). If it is placed with acid-producing mine spoil, either during or after reclamation, alkaline addition and water diversion may reduce the rate of acid formation at these coal mine sites. The U.S. Bureau of Mines is evaluating the effect of CCR injection on the discharge of acid mine drainage (AMD) at a reclaimed surface mine. In this study, a pozzolanic grout prepared from fly ash, AMD sludge, lime, and treated water was injected over a 12-month period into an experimental section of a reclaimed surface mine that continues to produce AMD. The alkaline grout if injected directly into areas of acid-producing mine spoil may reduce the permeability and porosity of the backfill, diverting ground water away from acid producing material. The mixture can also encapsulate acid-producing material, reducing potential exposure to air and water. Adding alkaline material to the subsurface environment may neutralize acidic groundwater. Both mechanisms, water diversion and alkaline addition, should decrease the rate of acid-producing reactions and inhibit bacterial activity.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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There are several types of CCR's. Fly ash is the type that was used at this site. It is generated from the burning of powdered coal and is noncombustible material swept through the furnace with hot flue gases. It ranges in size from 5 to 100 microns in diameter. Its physical properties are similar to those of volcanic ash; its chemical properties are similar to clay. Fly ash is produced from conventional power plants and from fluidized-bed combustion (FBC) plants. Fly ash from conventional U.S. power plants typically has a pH between 6.5 and 9.0 and contains less than 1% calcium (Paul et al. 1992). The FBC plants burn a mixture of low-Btu coal or coal refuse and limestone, producing an ash that is high in calcium oxide (>50%). Both materials are considered non-hazardous under the standards of the Toxicity Characteristic Leaching Procedure (TCLP), as defined by the U.S. Environmental Protection Agency (Federal Register 1990).

## Site Description

The experimental site is a 14.5-ha (36-acre) reclaimed strip mine in Greene County, PA. Mining ceased in 1981 after partial removal of the Waynesburg coal seam (Ackman, et al, 1993). A buried highwall was left along the eastern boundary of the site. The overburden associated with this coal seam consisted of a sandstone and thin carbonaceous shale, both pyritic. Acid-base accounting at 13 postmining monitoring wells indicated a net acid-producing potential (NAP) of 4 to 44 mt of calcium carbonate ($CaCO_3$) equivalency per 908 mt of material. Drill cuttings from seven wells across the experimental portion of the site had an average NAP of 20 mt of $CaCO_3$ per 908 mt of material. The buried spoil ranges between 5 and 15 m, averaging about 10 m, under 2 m or less of vegetated cover.

After reclamation, AMD discharged at an initial rate of 0.19 L/s; the rate has increased to 1.9 L/s (30 gpm) (Harshbarger, 1991). Currently, water treatment includes seepage collection, lime neutralization, aeration, and a series of sludge settling and polishing ponds. Ground water flows from east to west across the site. The flow path is inferred from the relative locations of the buried highwall and seep, geophysical investigations, and drilling.

Sixty-three wells were drilled on the reclaimed mine; 34 of these are located on the 1.2-ha (3-acre) experimental section. Twelve are monitoring wells, and the remainder are injection wells (fig. 1). Well A (1010)[3] is located at the edge of the site outside the mined area to the east of the experimental section. It is considered indicative of background water quality. Well C (1030) is near the buried highwall, where ground water enters the previously mined area. The primary discharge point, Seep A (1001), is also outside the experimental area and drains most of the reclaimed mine, including sections that were unaffected by grout injection.

## Procedures

Fly ash was loaded into a 3.8-$m^3$ (5-$yd^3$) steel bin using a small front-end loader, then fed into a 5.1-$m^3$ (1,350 gal) steel mixing tank. Water from the AMD treatment system and AMD sludge and/or lime waste (which consists of about 30% unused lime) were pumped into the mixing tank. Turbulence created by the feed streams and a 20-hp submersible slurry pump mixed the grout. The grout was pumped through a flexible 7.62-cm hose to the cased injection wells. The maximum distance was about 150 m across the surface. Grout was pumped into the injection wells until refusal, at a maximum pumping pressure of 3.5 kg/$cm^2$ (50 psi).

Grout was injected intermittently between July 1992, and June 1993, (table 1). Fly ash (50 $m^3$) from a conventional power plant was injected into 12 holes to the west of the experimental section between July and October 1992. Between October 1992 and May 1993, 54 $m^3$ of fly ash from another conventional source was injected into 12 holes on the western half of the experimental section. In May and June 1993, 67 $m^3$ of ash from a third conventional power plant was injected into 7 holes around Well 11 (1111). On a volume basis, fly ash constituted an average 5.6% of the injected grout.

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[3]Four digit numbers in parenthesis refer to location codes on figure 1.

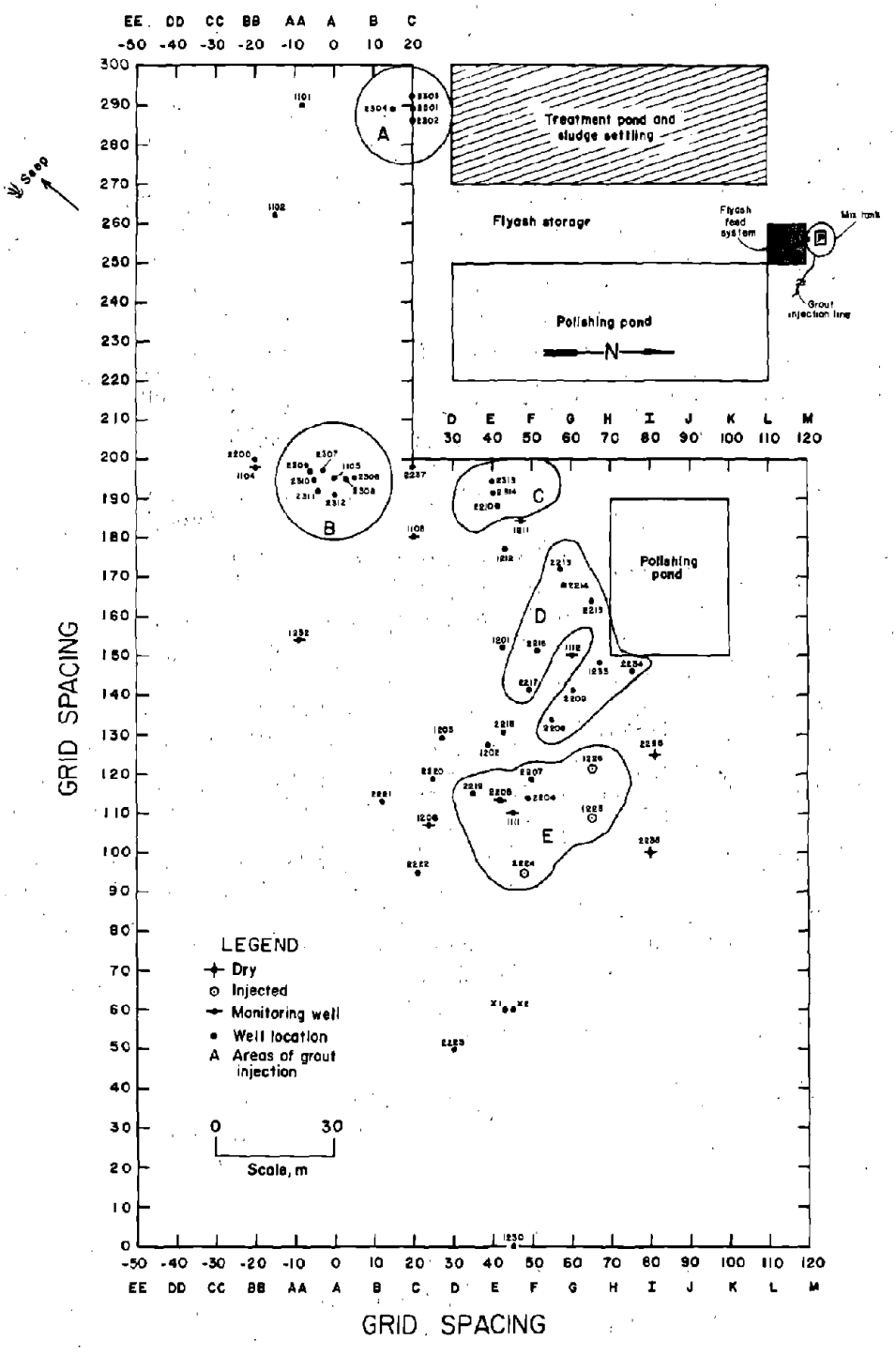


Figure 1. Well locations on map of experimental section of reclaimed strip mine.

Table 1. Fly ash injection at Bognanni Site.

| Injection area[1] | Date | Number of wells | Fly ash, $m^3$ | Fly ash source | Number of injection days |
|---|---|---|---|---|---|
| A | Aug 1992 | 1 | 13.6 | F | 4 |
| A | Sept 1992 | 3 | 11.9 | F | 2 |
| B | Sept 1992 | 6 | 22.2 | F | 2 |
| B | Oct 1992 | 2 | 2.5 | H | 1 |
| C | Oct 1992 | 3 | 24.9 | H | 2 |
| D | Nov 1992 | 3 | 22.2 | H | 2 |
| D | May 1993 | 7 | 5.7 | H | 2 |
| E | May 1993 | 6 | 36.3 | G | 4 |
| E | June 1993 | 2 | 31.1 | G | 2 |

[1]Refer to figure 1.

## Results

Water samples were collected during a 1-year period prior to injection from several wells and from the central seep (1001). Because of their location across the site, the data for Well C (1030), Well 8 (1108), Well 11 (1111), and Well 12 (1112) and for Seep A (1001) are considered representative of changes in water quality. Table 2 lists water quality data for Well A (1010), which is located outside the mined area and is considered indicative of the ground water entering the site. Prior to grouting, the pH of the water decreased across the site (fig. 2). After grouting, the average pH at monitoring wells increased. During the active period the pH at Well C (1030) was decreasing (fig. 3). Although the average pH at the seep showed little variation, the data points were slightly higher after grouting. After injection of the fly ash grout, the total acidity decreased at the monitoring wells and at the seep. Similar trends were observed for ferrous iron, total iron, aluminum and sulfate (table 3).

Based on a comparison of parametric and nonparametric tests of significance, on standard normal scores and the frequency distribution, almost all values for the pregrouting and postgrouting water quality data for this site are found to be normally distributed. To test if there is a significant difference between water quality parameters measured before and after grouting, a one-tailed t-test was computed for the variables listed in table 2. The test evaluates the differences between two data sets (before and after fly ash injection) based on their respective means and their standard deviations (Zar 1984). The level of acceptance or rejection was established at $p <= 0.05$ (table 4). The decrease in total acidity and total iron was significant at Well 8, Well 11, and Well 12 and at Seep A. Although the sulfate concentration decreased at these monitoring points, the change was significant only at Well 11, Well 12, and Seep A. The increase in the concentration of calcium and magnesium, significant at Well 8, Well 11 and Seep A, is probably due to the presence of these elements in the grout.

An analysis for trace metals indicated that barium ($Ba^{++}$) was present in the incoming water but decreased across the site and was not detected at the seep (fig. 4). Lead ($Pb^{++}$), chromium ($Cr^{++}$) and copper ($Cu^{++}$) were present in samples from the monitoring wells, but were not detected in water at the seep. Cobalt ($Co^{++}$) and zinc ($Zn^{++}$) (fig. 5) were detected at the monitoring wells and at the seep, but the concentrations at the seep did not exceed drinking water standards.

Table 2. Water quality data - Well A (1010)

| Water quality parameters | Sample date: June, 1991 | Sample date: June, 1993 |
|---|---|---|
| pH | 6.93 | 7.50 |
| Analysis, mg/L | | |
| Alkalinity | 169 | 167 |
| Ferrous iron | ND | ND |
| Total iron | 2.1 | .5 |
| Calcium | 53 | 57 |
| Magnesium | 12 | 13 |
| Aluminum | 1.8 | .4 |
| Manganese | .5 | .5 |
| Sulfate | 35 | 35 |
| Arsenic | NA | <0.10 |
| Barium | NA | .36 |
| Beryllium | NA | <0.01 |
| Cadmium | NA | <0.01 |
| Cobalt | NA | 0.01 |
| Chromium | NA | 0.01 |
| Copper | NA | 0.01 |
| Nickel | NA | 0.03 |
| Lead | NA | <0.10 |
| Antimony | NA | <0.10 |
| Selenium | NA | <0.50 |
| Zinc | NA | <0.01 |

NA Not analyzed.
ND Not detected.

Table 3. Mean water quality values (mg/L) for incoming water Well C (1030), across site (Well 8, Well 11, and Well 12) and at outflow (Seep A) for periods before (B) and after (A) grouting.

| | Well C | | Well 8 | | Well 11 | | Well 12 | | Seep A | |
|---|---|---|---|---|---|---|---|---|---|---|
| | B | A | B | A | B | A | B | A | B | A |
| Acidity | -18 | 22 | 1407 | 641 | 1525 | 796 | 1977 | 614 | 1735 | 1200 |
| Ferrous iron | 6 | 4 | 294 | 128 | 332 | 218 | 131 | 50 | 369 | 241 |
| Total iron | 9 | 7 | 302 | 207 | 345 | 254 | 149 | 77 | 382 | 251 |
| Calcium | 44 | 62 | 349 | 566 | 396 | 525 | 443 | 544 | 399 | 460 |
| Magnesium | 16 | 25 | 292 | 387 | 467 | 383 | 320 | 325 | 455 | 319 |
| Aluminum | 6 | 7 | 134 | 186 | 142 | 137 | 254 | 112 | 177 | 127 |
| Manganese | 6 | 4 | 251 | 199 | 164 | 151 | 397 | 226 | 249 | 161 |
| Sulfate | 9 | 285 | 3766 | 4029 | 4653 | 3979 | 5202 | 3335 | 4965 | 3987 |

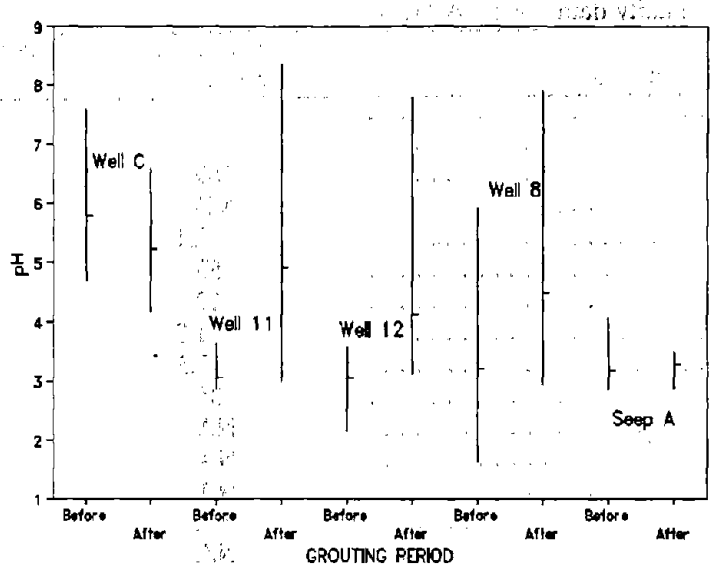


Figure 2. Range (|) and mean (–) values for incoming water (Well C), across site (Well 8, Well 11, Well 12), and at outflow (Seep A) before and after grouting.

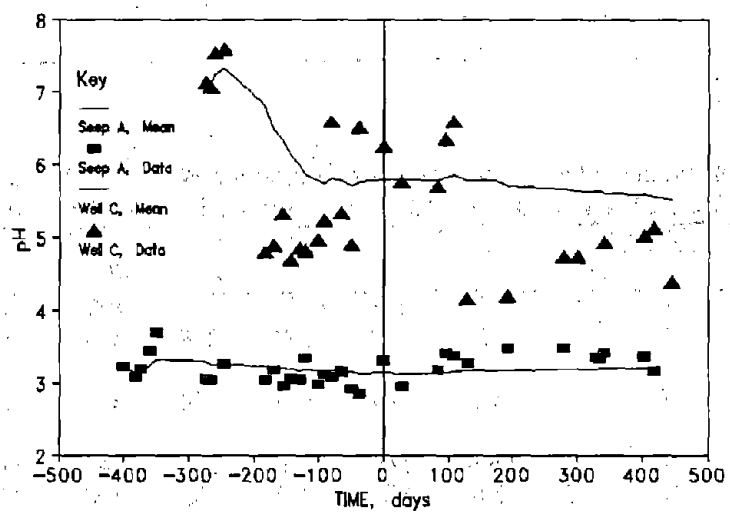


Figure 3. Variation in pH with time at highwall (Well C) and at outflow (Seep A) with time.

Table 4. t-test of significance[1] for water quality variables before and after grouting.

| Well | pH | Acidity | Ferrous iron | Total iron | Calcium | Magnesium | Aluminum | Manganese | Sulfate |
|---|---|---|---|---|---|---|---|---|---|
| Well 8 | **0.0031** | **0.032** | **0.000** | **0.015** | **0.001** | **0.007** | 0.118 | 0.410 | 0.110 |
| Well 11 | **.017** | **.003** | **.002** | **.02** | **.047** | **.024** | .289 | **.002** | **.010** |
| Well 12 | .065 | **.000** | .065 | **.015** | .180 | .390 | **.000** | .387 | **.000** |
| Seep A | .080 | **.002** | **.000** | **.000** | **.005** | **.000** | **.002** | **.000** | **.000** |

Values < =.05 indicate significant difference between samples collected before and after grouting.

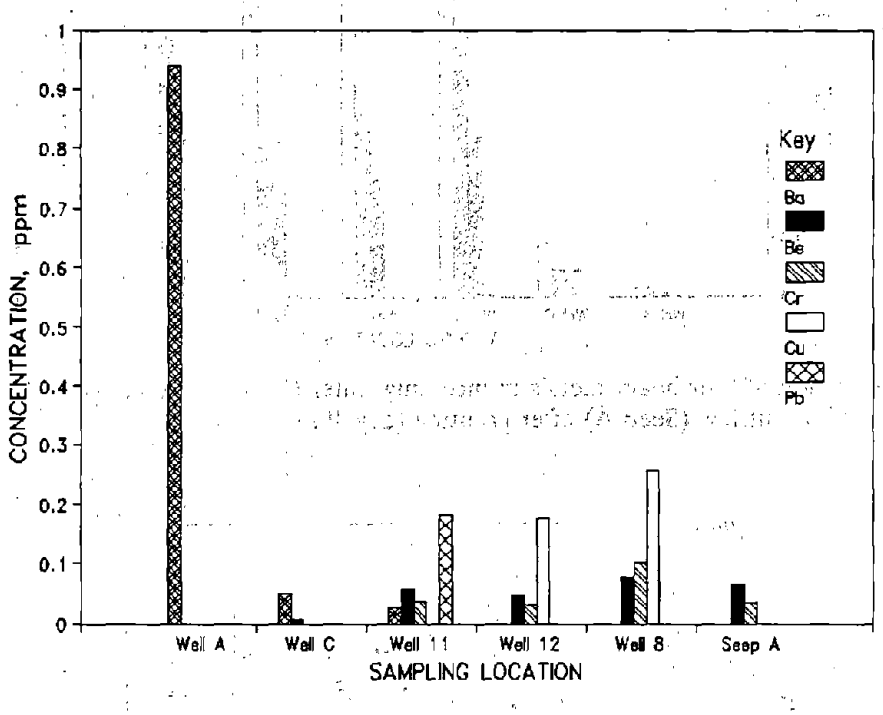


Figure 4. Concentration of five heavy metals in incoming water (Well C), across site (Well 8, Well 11, Well 12), and at outflow (Seep A) after grouting (day 402).

## Summary

Although the amount of fly ash injected was small and the volume of subsurface area it could have affected is a relatively small portion of the 14.5 ha site, the fly ash grout produced a measurable change in the acidity at the seep and the monitoring wells. Statistically significant changes in the concentration of metal ions and sulfate were also were observed at the seep. There was no significant change in the pH at the seep, and the observed reductions in Fe and Mn were not sufficient to meet discharge standards. The release of heavy metals from the fly ash can be inferred from the concentrations in the monitoring wells. It should be noted, however, that the concentrations observed to date do not represent a hazard to surface or ground water.

Some improvement in water quality may be transient, due to the highly alkaline water (pH>10) associated with the grout. The alkaline water could have created a plume of alkalinity that passed through the site as the solid portion of the grout was deposited in the spoil. If it is assumed that the grout followed the path of least resistance, its effect should coincide with the ground water movement. Increases in alkalinity observed at several of the wells may be related to this effect (fig. 6). Observed decreases in pollution levels

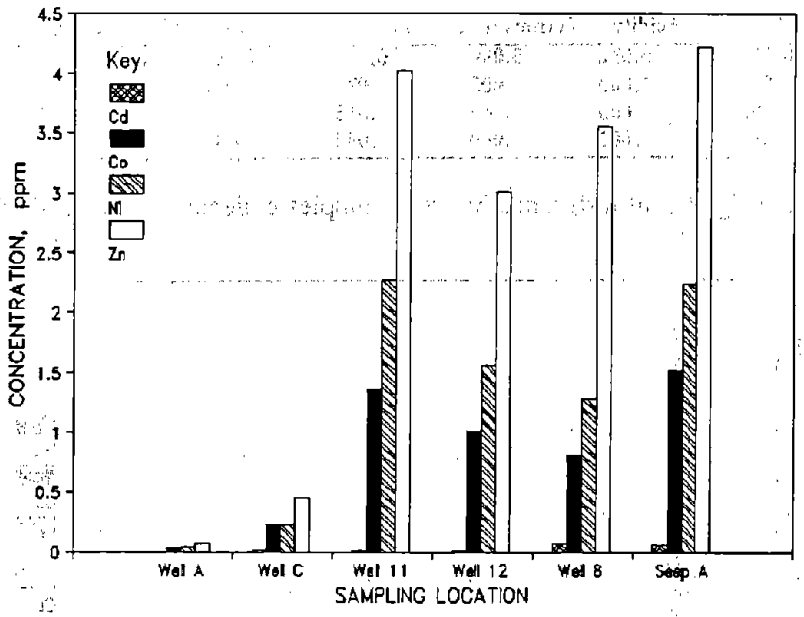


Figure 5. Concentration of four heavy metals in incoming water (Well C), across site (Well 8, Well 11, Well 12), and at outflow (Seep A) after grouting (day 402).

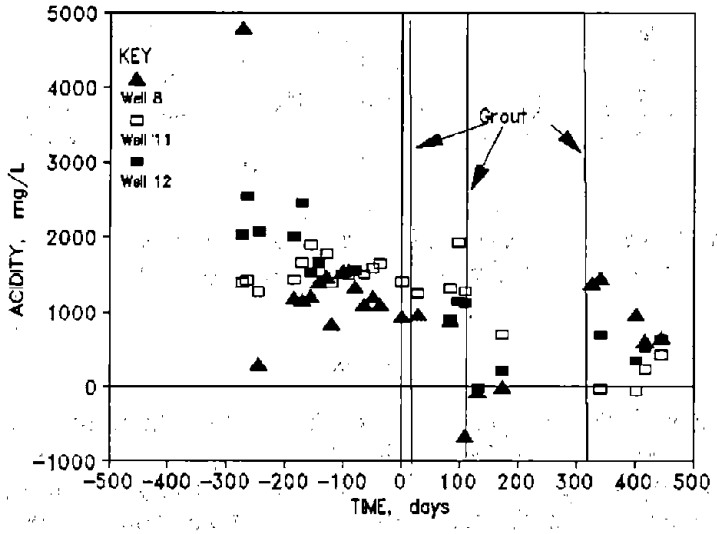


Figure 6. Variation in acidity at monitoring wells for pre and postgrouting periods.

may be due to encapsulation or armoring of pyritic material and/or to reduced permeability and porosity of the backfill material. Continued monitoring (> 1 year) at the site is planned to determine if CCR injection produces long-term improvement in water quality.

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# DETERMINATION OF METAL ADSORPTION CAPACITY OF SOILS FOR DISPOSAL OF MINING PROCESS SOLUTIONS BY LAND APPLICATION[1]

Edward Spotts and William M. Schafer[2]

**Abstract:** During mining and processing of precious metals it is often necessary to dispose of treated process solutions to manage the on-site water balance. The solutions typically have elevated levels of trace metals and possibly cyanide. Land application, where solution is applied to soils at a controlled rate, can be an effective means of treatment and disposal. Treatment of the solution occurs as metals are attenuated by soils. Important mechanisms of removal include adsorption by clays, organic matter, and hydroxide and oxide minerals.

Adsorption batch tests using synthetic treated mine process solution were performed on a 0 to 6 inch loam soil from a proposed land application area at a western U.S. precious metals mine to characterize the adsorption behavior of silver, cadmium and copper. Results of testing were used to construct adsorption isotherms to predict the maximum adsorptive potential of, and metal loads to, the soil. The results indicate that the soils were effective at reducing silver, cadmium and copper levels in the process solution and that soil had ample adsorption capacity for these metals, suggesting periodic land application of treated mining process solutions can be a safe, effective management tool.

## Introduction

This paper addresses the determination of cadmium (Cd), copper (Cu) and silver (Ag) adsorption capacity of soils collected from a proposed land application area at a precious metal mine located in an alpine environment in the western United States. A discussion of the principles of land application and modeling of the mobility and attenuation of metals in natural soils precedes the discussion of experimental results.

## Principles of Land Application

During mining and processing of precious metals it is often necessary to periodically dispose of treated process solutions to manage the on-site water balance. The solutions typically have elevated levels of trace metals and possibly cyanide. Land application, where solution is applied to soils at a controlled rate, can be an effective means of treatment and disposal. Process solutions are typically treated with strong oxidants (hydrogen peroxide, calcium hypochlorite, or sodium hypochlorite) to reduce cyanide to acceptable levels. Treatment may occur in lined ponds or in sophisticated treatment plants, which may utilize other means (e.g. lime addition) to lower metal levels prior to land application. Treated solutions are typically applied to gentle to moderate slopes using a sprinkler, mister or drip emitter distribution system. Areas chosen are typically grass and shrub meadows, although forested areas can also be utilized.

The purpose of land application is to utilize the ability of natural soils to attenuate the migration of metals and other deleterious substances in wastewater. Important mechanisms of removal include adsorption by clays, organic matter, and hydroxide and oxide minerals. Soil characteristics important to the use of an area for land application treatment include hydraulic conductivity, soil thickness, heterogeneity and metal adsorption capacity of

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[1]Paper presented at the International land Reclamation and Mine Drainage Conference ant the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994

[2]Edward Spotts, Senior Soil Scientist, and William M. Schafer, President, Schafer and Associates, Bozeman, MT, USA.

soils, and the uniformity of these features. Additional factors considered in the siting of a land application area include site physiography, hydrology, vegetation and proximity to mine facilities. Important concepts integral to the use of land application treatment systems are:

- The rate of solution application and total load of metals are controlled to prevent runoff and migration of metals at unacceptable levels;
- Long-term uses of land application area soils are not unduly restricted.

**Mobility and Attenuation of Metals in Natural Soils**

The soil system is a complex, dynamic three-phase system composed of a skeleton of solids with pore space filled with liquids and gases. Chemical equilibria of soil systems differ from those of lakes and streams in that the reactive surface area of soil solids in contact with soil solution is many times greater than in aqueous systems (Dragun 1988). The mobility of many dissolved ions is strongly affected by interaction with the surfaces of solids in soils.

Although the mobility of metals in soils is controlled by many factors, including dissolution and precipitation reactions, chemical diffusion, and microbial activity, the primary mechanism governing the behavior of metals in many soils is the surface reaction (adsorption) of ions onto the surfaces of clay minerals, metal hydroxyoxides, and organic matter (Lindsay 1979, Bohn et al 1985).

Most metals are similar in that they exist as positively-charged ions that can be adsorbed by negatively-charged clays and minerals in soils and that their solubility generally increases as solution pH decreases. Notable exceptions to this trend are arsenic (As), mercury (Hg), molybdenum (Mo) and selenium (Se), which show an increase in solubility at circumneutral pH values. The solubility of some metals has been ascribed to the formation of sparingly soluble secondary minerals (Lindsay 1979). While this is almost certainly the case for iron, aluminum, and manganese, most researchers agree that sorption of many trace metal ions on the surfaces of iron, aluminum, and manganese oxides, organic matter, and clay minerals governs their solubility in soils (Salomons and Forstner 1988).

Metal ions in soils also form a number of ion pairs, especially at higher pH levels (8 to 11) typical of many treated mining process solutions. Ion-pairing is especially prevalent with hydroxyl ($OH^-$), sulfate ($SO_4^{-2}$), and chloride ($Cl^-$) ions (Stumm and Morgan 1979), which are often present in elevated concentrations in treated process solutions. Metal ions can also "complex" with soluble organic ligands, in a process similar to ion-pairing (Stevenson 1982). Thus, as the abundance of soluble hydroxide (i.e. high pH), sulfate, chloride, and organic carbon increases, metal ion solubility also tends to increase. This fact underscores the utility of site specific testing of soils in a proposed land application area.

The most "reactive" component of the soil system is that portion less than 2 microns in size. The clay-sized fraction is composed mostly of clay minerals, metal hydroxyoxide minerals, and amorphous organic material or humus. Each of these compounds have surface functional groups, primarily consisting of exposed hydroxyl groups or oxygen along surfaces. The large surface area of the clay-sized material and the presence of negatively-charged surface functional groups allows clay-sized particles to react with soluble positively-charged metals and remove them from solution. Surface complexation refers to the process of a surface functional group reacting with a soluble ion to form a covalent bond. This process is known as *specific* adsorption and the resulting complex is often referred to as an "inner-sphere" complex (Sposito 1984). This is the primary reaction affecting sorption of metals in soil. When the bond is electrostatic attraction, an "outer-sphere" complex is formed. Metals are not bound as strongly in an outer-sphere complex as in an inner-sphere complex. Outer-sphere complexation is readily reversible and is also referred to as ion exchange. This is the predominant binding mechanism affecting major cations like calcium, magnesium, and sodium.

## Adsorption Modeling

Adsorption of metals in soils has been intensively studied to provide a means of predicting metal behavior in soil systems. The capacity of soils to adsorb most metals appears to be inversely related to pH and to be affected by competition between ions (Salomons and Forstner 1988). For example, the adsorption of cadmium is reduced by addition of calcium. Metal ions also differ in their affinity for specific adsorption surfaces (Sposito 1984).

Adsorption isotherms are used to describe the adsorption of metals by soils. An adsorption isotherm plots the amount of metal adsorbed per unit mass of adsorbate on the y-axis versus the equilibrium solute concentration on the x-axis. The shape of the curve of this plot provides information regarding the adsorptive behavior of the adsorbate (metal), and several mathematical models have been developed to explain this behavior. The most commonly employed are the Freundlich and the Langmuir. The Freundlich equation employs two empirical constants and is plotted on a log-log scale. It is thus very robust. However, it does not allow for the calculation of an adsorption maximum for a given metal. For this reason, it is often useful to employ the Langmuir equation. The Langmuir equation assumes (1) a constant energy of adsorption independent of surface coverage (i.e. a homogeneous surface), (2) adsorption on specific sites with no interaction between adsorbate molecules, and (3) a maximum adsorptive capacity determined by a complete monolayer of molecules on all adsorbent surfaces. Although constant energy of adsorption is rare in natural systems, there typically is significant interaction with molecules already adsorbed. Thus the two factors tend to compensate one another and a relatively constant energy of adsorption is yielded. Given the theoretical limitations, the Langmuir isotherm is best viewed as an empirical description of actual adsorption processes (Bohn et al 1985).

The common form of the Langmuir equation is given as

$$\frac{x}{m} = \frac{KCb}{1 + KC} \qquad (1)$$

where C is the equilibrium concentration of adsorbate in question, x/m is the mass of adsorbate per unit mass of adsorbent, K is a constant related to the binding strength, and b is the maximum amount of adsorbate that can be adsorbed on a single complete monomolecular layer. If adsorption conforms to the Langmuir model, the equation defines a limit to adsorption where the amount of material adsorbed fails to increase despite increasing solution concentration, yielding a logarithmic curve with a strong inflection at the point of maximum adsorption.

Equation [1] is often rearranged to the linear form

$$\frac{C}{x/m} = \frac{1}{Kb} + \frac{C}{b} \qquad (2)$$

By plotting C/x/m versus C, a straight line is yielded if the data conform to the Langmuir equation. This line has a slope of 1/b and intercept of 1/Kb. The value for K is the quotient of the slope and the intercept.

Thus, it can be seen that the Langmuir equation allows identification of a maximum adsorption capacity by soils for metals of concern.

This paper discusses the results of adsorption testing for Ag, Cd and Cu in a 0 to 6 inch soil sample collected from a proposed land application area at a precious metal mine located in an alpine environment in the western United States. The soil was exposed to a laboratory solution synthesized to represent the chemistry of a

treated process solution. Several additional metals and various soil depth increments were evaluated during the study but only the previously mentioned data are discussed owing to the large scope of the study. These data are sufficient to illustrate the principles of soil adsorption of metals and land application area evaluation.

## Materials and Methods

### Soil Collection

A series of adsorption batch tests were conducted on a representative sample collected from the A horizon at the 0 to 6 inch depth of a loamy-skeletal, mixed Dystric Cryochrept from the proposed land application area. The sample was retained at near-freezing conditions at field moisture content to prevent biological or physical changes that might affect chemical adsorption.

The soil was a loam with 16 %, 47% and 37 % clay, silt and sand, respectively; 4.5 % organic matter and a cation exchange capacity of 33.50 meq/100 g. Paste pH and conductivity (SC) values were 4.5 and 0.27 mmhos/cm, respectively. The soil was classified as a loamy-skeletal, mixed Dystric Cryochrept.

### Adsorption Testing

Two batch solutions (stock and metal) were formulated to closely simulate the chemistry of treated process solution. The stock solution consisted of major ions added at concentrations similar to actual solution values including sulfate (1800 mg/l), calcium (2500 mg/l), sodium (450 mg/l), chloride (3500 mg/l), TDS (9400 mg/l), and specific conductance (11500 µmhos/cm). A second solution (metal) with identical major ion concentrations had trace metals added at roughly 10 times the levels in treated process solution including copper (9.43 mg/l), cadmium (0.122 mg/l), mercury (0.028 mg/l), lead (0.129 mg/l), selenium (0.101 mg/l), silver (0.240 mg/l), and zinc (4.91 mg/l). Metals were added using inductively coupled plasma (ICP) standards to insure accurate results.

The two solutions were mixed at stock:metal ratios of 0:1, 1:1, 2:1, 5:1, 10:1, 20:1, and 100:1. Forty gram subsamples of soil were added to 800 ml of each stock:metal dilution and agitated for 24 h in a rotary extraction apparatus. The extract was then filtered through 0.7 µm glass fiber filters. Silver and cadmium were analyzed using graphite furnace-atomic absorption methods at 0.0005 and 0.002 mg/l instrument detection levels. Copper was analyzed using the ICP method at a 0.15 mg/l instrument detection level. Table 1 shows the initial concentrations of silver, cadmium and copper for each dilution.

Table 1. Initial concentrations of metals in stock:metal solutions.

| Metal | Stock:Metal Solution | | | | | | |
|---|---|---|---|---|---|---|---|
| | **0:1** | **1:1** | **2:1** | **5:1** | **10:1** | **20:1** | **100:1** |
| Ag | 0.33 | 0.17 | 0.11 | 0.06 | 0.03 | 0.015 | 0.001 |
| Cd | 0.12 | 0.06 | 0.04 | 0.02 | 0.01 | 0.005 | 0.001 |
| Cu | 9.43 | 4.72 | 3.14 | 1.57 | 0.86 | 0.45 | 0.09 |

## Results and Discussion

Results of adsorption testing are presented in Table 2. Values for x/m are calculated by subtracting the value for a metal at a given dilution ratio in Table 2 from the corresponding value in Table 1. The amount of metal adsorbed by soil was plotted against the amount remaining in the extracted solution for each dilution. Cadmium adsorption (Figure 1) did not appear to fit a Langmuir curve, which may be in part because the equilibrium

Table 2. Equilibrium metal concentrations in soil extracts.

| Metal | Stock:Metal Solution | | | | | | |
|---|---|---|---|---|---|---|---|
| | 0:1 | 1:1 | 2:1 | 5:1 | 10:1 | 20:1 | 100:1 |
| Ag | 0.18 | 0.08 | 0.04 | 0.01 | 0.006 | 0.002 | <0.0005 |
| Cd | 0.10 | 0.05 | 0.04 | 0.02 | 0.01 | 0.009 | 0.005 |
| Cu | 2.20 | 0.64 | 0.30 | <0.15 | <0.15 | <0.15 | <0.15 |

concentrations of cadmium were far below the soil adsorption capacity. Hence, all soil cadmium adsorption data may lie along the lower linear portion of a Langmuir curve. Nonetheless, a minimum estimate of the soil adsorption capacity can be developed. At high cadmium loads, the soil appears to be able to retain a minimum of 1 mg Cd per kilogram soil, but maximum adsorption capacity may be much higher. At very low solution cadmium concentrations, the soil appears to desorb cadmium into solution, perhaps due to competition for adsorption sites between cadmium and more abundant ions like copper. The net effect is for soils to maintain a solution cadmium concentration of 0.02 mg/l even at cadmium loads far below 0.02 mg/l. The levels of cadmium in the treated solution (actual and synthetic at high dilution) appear to be very close to natural soil equilibrium values, hence little change in the soil cadmium system would be expected due to land application. Copper was added to batch solutions at higher concentrations than other constituents because it was more abundant in the treated process solution.

The linear isotherm for copper (Figure 2) indicates that it fits the Langmuir curve closely (Figure 3) with the exception of some data points that were below analytical detection (which were reported at the detection level).

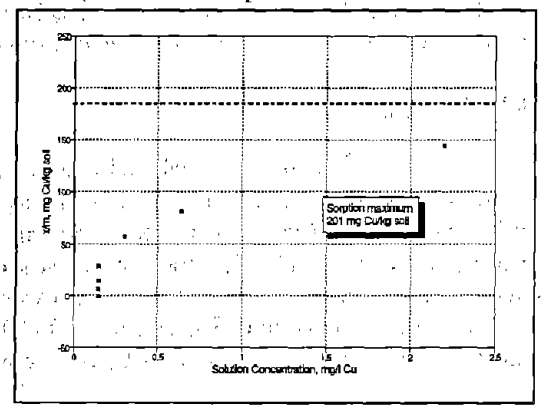


Figure 2. Linear adsorption isotherm for copper exhibiting a Langmuir-type curve of logarithmically decreasing adsorption of copper by soil with increasing solution concentration. Note effect of instrument detection limits on the curve at low C values.

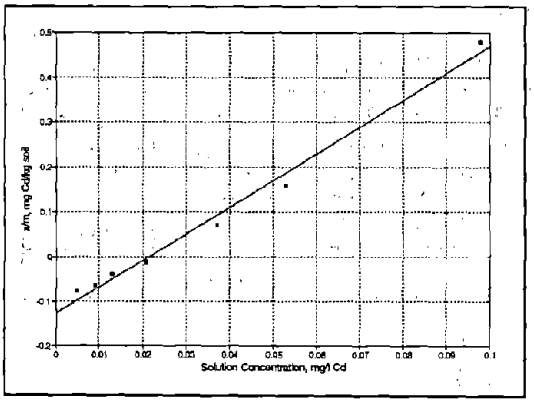


Figure 1. Linear adsorption isotherm for cadmium showing linear adsorption behavior of cadmium by soil with increasing solution concentration.

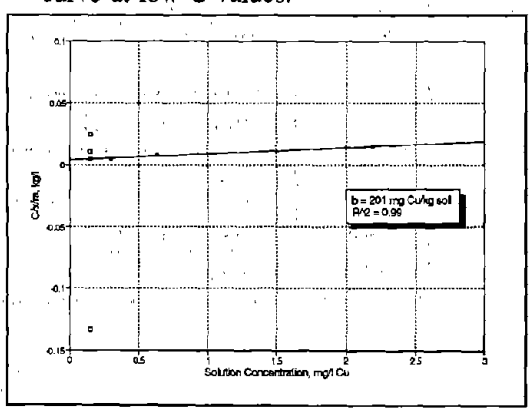


Figure 3. Langmuir isotherm for copper showing maximum adsorption of copper by soil at 201 mg Cu/kg soil.

Using all data points above the detection level, the value of b, maximum adsorption capacity, in the Langmuir equation was 201 mg Cu per kilogram soil. Copper is known to have a strong affinity for soil organic matter and is typically retained strongly in surface soils.

Silver, like copper, has a very strong affinity for soil organic matter and also like copper appeared to follow a Langmuir curve (Figure 4) very closely. The linearized plot of silver on the Langmuir isotherm (Figure 5) indicates an adsorption maximum of 3.29 mg Ag per kilogram soil.

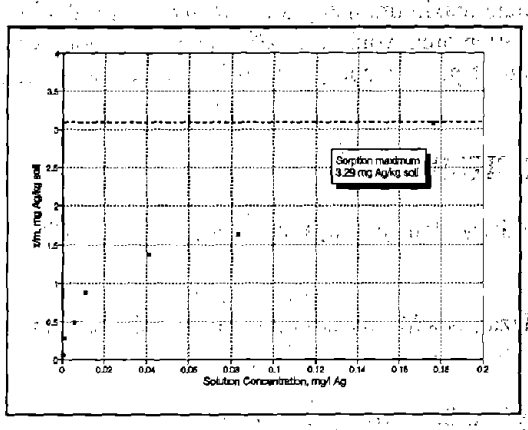


Figure 4. Linear adsorption isotherm for silver exhibiting a classic Langmuir-type curve of logarithmically decreasing adsorption of silver by soil with increasing solution concentration.

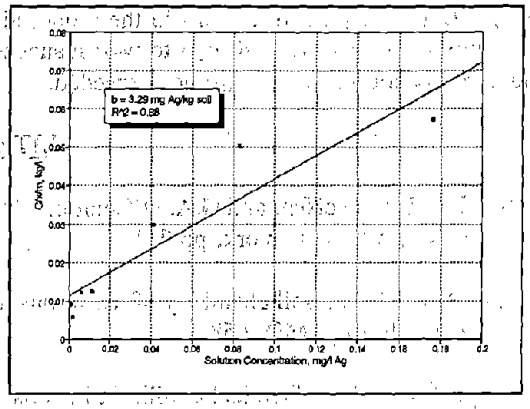


Figure 5. Langmuir isotherm for silver showing maximum adsorption of silver by soil at 3.29 mg Ag/kg soil.

## Determination of Maximum Metal and Solution Loading

The maximum solution load can be calculated by dividing the maximum metal load (shown in kilograms metal per hectare of soil for convenience) by the average solution concentration for a given metal. These values are shown in Table 3.

Table 3. Maximum metal and solution loading to soilsbased on adsorption batch test data.

| Metal | Maximum Metal Load[1] (kg/ha) | Average Metal Concentration of Treated Solution (mg/l) | Maximum Solution Load | |
|---|---|---|---|---|
| | | | (m/ha) | (l/ha) |
| Silver | 12.83 | 0.15 | 8.55 | $8.55 \times 10^7$ |
| Cadmium | 3.90 | 0.02 | 19.50 | $1.95 \times 10^8$ |
| Copper | 783.90 | 4.0 | 19.60 | $1.96 \times 10^8$ |

[1] - Assuming a 30 cm treatment zone thickness, a bulk density of 1.3 g/cm$^3$, and uniform solution application.

Of the three metals shown in Table 3 it can be seen that silver would limit the amount of solution which could be applied per hectare ($8.55 \times 10^7$ l). Values are also shown in meters of solution per hectare for convenience. In practice, the *rate* at which a solution is applied to a soil is limited by the hydraulic conductivity of the soil, the depth to groundwater and the evapotranspiration demand. Characterization and management of these

factors ensures that solution application rates are low enough to prevent ponding of solution, runoff, or significant migration to groundwater.

## CONCLUSIONS

As part of a land application study for a precious metals mine, an evaluation of soil adsorption capacity was performed for a proposed land application area which would receive treated process solution. Adsorption isotherms were developed to characterize metal adsorption and to calculate maximum metal loads to soils. Results indicated that the 0 to 6 inch depth of soils in the proposed land application area would provide ample adsorption of silver, cadmium and copper if subjected to well-managed application of process solutions and provided the loading rate and capacity for adsorption are not exceeded.

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# AERIAL AND AQUEOUS DISPERSION OF INORGANIC CONTAMINANTS - SIGNIFICANCE FOR THE CORNISH FARMING INDUSTRY[1]

Caroline P. Waller[2], Richard P. Edwards[2] and Carolyn Wilkins[3].

**Abstract:** Cornwall has numerous derelict Cu-Sn mine sites scattered throughout the county - a legacy of past intensive mining activity. Due to the toxic concentrations of heavy metals many sites are poorly vegetated which leads to continued dispersion of toxic metals into the surrounding farming environment.

The aim of this research is to establish the extent of contamination, to determine its form, and to ascertain how much dispersion is currently occurring. A case study of the Tresavean Mine near Redruth, west Cornwall, UK, has shown that the distribution of As and Cu from the mine extends up to 150 m. An extensive soil survey of the farmland surrounding the mine revealed two distinct distribution patterns for As and Cu. Arsenic distribution appears to correlate with the prevailing wind direction at the site, whereas the Cu distribution follows the down slope drainage pattern.

Sequential extractions were made on soil samples taken at intervals along a transect from the mine to determine which soil fractions Cu and As are held in; i.e. water-soluble, exchangeable, adsorbed, organic, Fe-Mn oxides or residual. These sequential extractions indicate the Cu is held mainly in the residual and organic fractions and As is held mainly in the residual and iron-organic fractions. Although the absolute concentrations of As and Cu decrease with distance from the mine site, as expected, the percentage of As and Cu in each fraction does not follow this pattern in every case.

**Additional Key words:** Copper, arsenic, mine waste, soil, sequential extractions.

## Introduction

During the heyday of Cornish mining from 1750 to 1900 numerous Cu-Sn mines were operating throughout the county. The "boom-bust" activity of these mines has left a legacy of derelict mine sites scattered across south-west England. Reclamation was virtually unheard of during the height of Cornish mining, and the closed mines were left in a state of dereliction with no remedial action taken. This left high concentrations of heavy metals at the surface which has deterred natural revegetation of some sites. This is how they have remained to the present day. The lack of vegetative cover exacerbates the continued dispersion of metals into the surrounding farmland.

Previous work carried out in south-west England has highlighted the problem of As-Cu contamination in this region (Abrahams and Thornton 1987, Colbourn et al. 1975, Gaukroger 1986). Copper and As are potentially toxic to animals and man if present in large amounts (Thornton 1980). Copper can also be phytotoxic (Kabata-Pendias and Pendias 1991, Lepp 1981) and therefore needs to be monitored in the farming environment. In Cornwall, where background values of both of these elements are naturally high (Abrahams and Thornton 1987), guidelines need to be followed for land which is subject to a change of use, for example, returning derelict mine sites to farmland (ICRCL 1990) or where planning permission is required (ICRCL 1983). However, "trigger values" ("trigger value" is defined as "that below which the soil is considered to be safe, i.e. the concentration present should not give rise to phytotoxicity or zootoxicity effects" (ICRCL 1990)) given in these ICRCL

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guidelines are difficult to use in Cornwall because the county's naturally elevated background concentrations are very close to the "trigger values". The local farming community (which farms predominantly livestock) has experienced the problems of high soil-As concentrations. Animals grazed in high soil-As areas endure a period of poor health while becoming accustomed to the higher As levels. As these problems of As and Cu contamination are inherent in Cornwall, this paper will concentrate on these two elements.

The study area for this research was the Tresavean Mine, Lanner, near Redruth, west Cornwall, United Kingdom (fig.1). Tresavean is an old, disused, underground mine with records dating back to 1745. It was a rich mine with relatively high quality copper ore. During its history it has been closed and reopened a number of times due to fluctuating markets, but finally closed in 1928. Dines (1956) states that Tresavean produced over 3,000 tons of black tin (tin concentrate for smelting), 13 tons of arsenic and 228,000 tons of copper ore.

Tresavean straddles the contact of the Carnmenellis Granite (part of the Cornubian granite batholith) and metasedimentary country rock. The contact strikes ESE and dips 35°-45° to the NE with metasediments overlying the granite.

There were four principal lodes at Tresavean (fig. 2) containing copper sulphides and cassiterite as the main ore minerals. Impurities in Cornish lode ore, such as arsenopyrite and pyrite, interfere with the smelting process and were removed during the ore-dressing stage by calcining. This is the process of roasting heaps of ore to decompose the arsenopyrite and pyrite by atmospheric oxidation to $As_2O_3$ and $SO_2$, which were removed in the fumes.

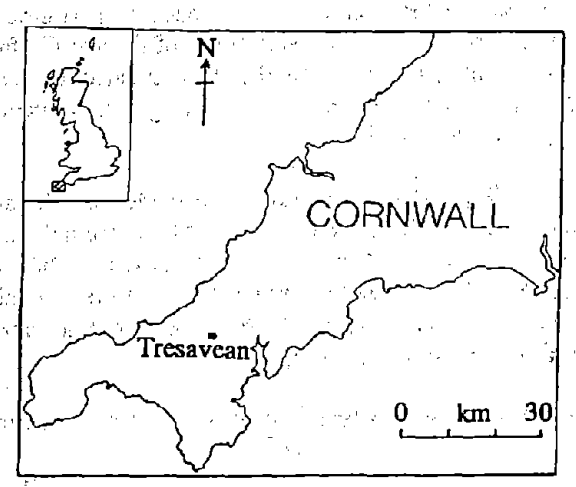

Figure 1. Location map.

## Soil Metal Distribution

The initial soil survey at Tresavean was designed to cover all the adjacent farm fields. Surface (0 to 20cm) soil samples were taken on a regular grid pattern (50 by 100 m) over the site, using a soil auger and combining three sub-samples at the sample site. Table 1 summarises the results obtained for the soil analysis of total elements. The results for Cu and As are also presented as element distribution maps in figures 3 and 4. Values for isolines were determined from population graphs of the data. The threshold concentration of Cu in soils at Tresavean was found to be 300 µg/g; anything above this is considered to come from an anomalous population. The data also suggest a further population split at 550 µg/g. For As the threshold value is 220 µg/g, with a further population split at 450 µg/g.

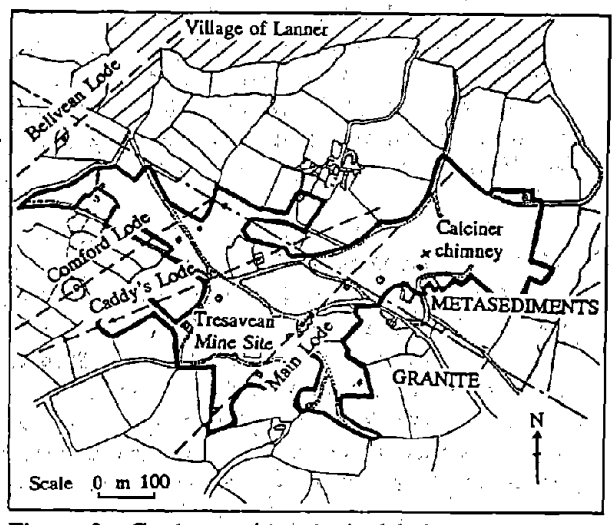

Figure 2. Geology with principal lodes at Tresavean Mine.

The 300 µg/g isoline for Cu largely follows the boundary of the mine site on the western side, but on the eastern side the isoline encroaches much further into the fields, by up to 150 m (fig. 3). The pattern is similar for As; however, there is more encroachment of the isoline into the fields in the south-east and slightly less into the fields in the north-east (fig. 4). The results show more intense As contamination to the south-east compared with Cu, and greater Cu contamination to the north-east.

Table 1. Results obtained from surface soil samples at Tresavean.

| | Total element concentration, (μg/g)[1] | | | | | | |
|---|---|---|---|---|---|---|---|
| | Cu | As | Pb | Zn | Ni | Mn | Fe |
| Maximum | 2,750 | 1,410 | 580 | 1,680 | 48 | 1,350 | 47,800 |
| Minimum | 60 | 30 | 30 | 80 | 0 | 100 | 16,200 |
| Mean | 250 | 180 | 130 | 360 | 16 | 380 | 26,100 |
| Std. dev. | 270 | 150 | 80 | 200 | 9 | 240 | 5,900 |
| n = | 147 | 147 | 147 | 147 | 147 | 147 | 147 |

[1]Samples were air-dried at 30°C, sieved to obtain the minus 2 mm fraction, and digested by a nitric-perchloric acid mixture (Hesse 1971).

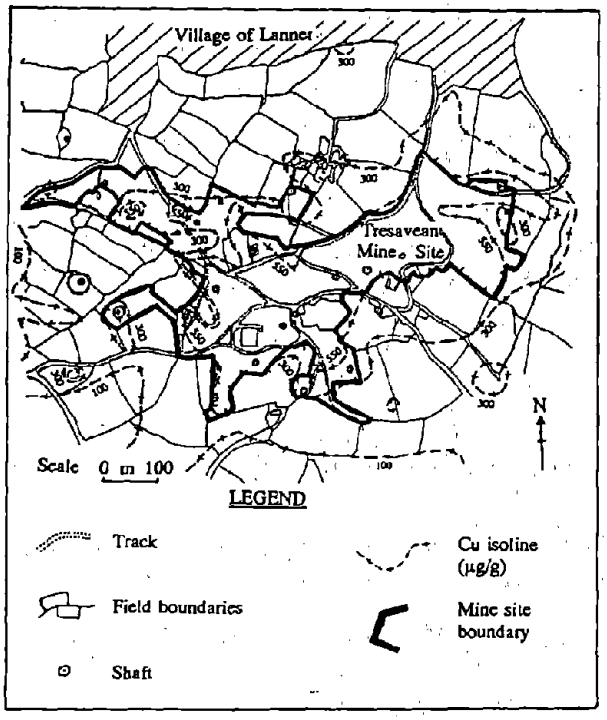


Figure 3. Distribution map of Cu in soil at Tresavean.

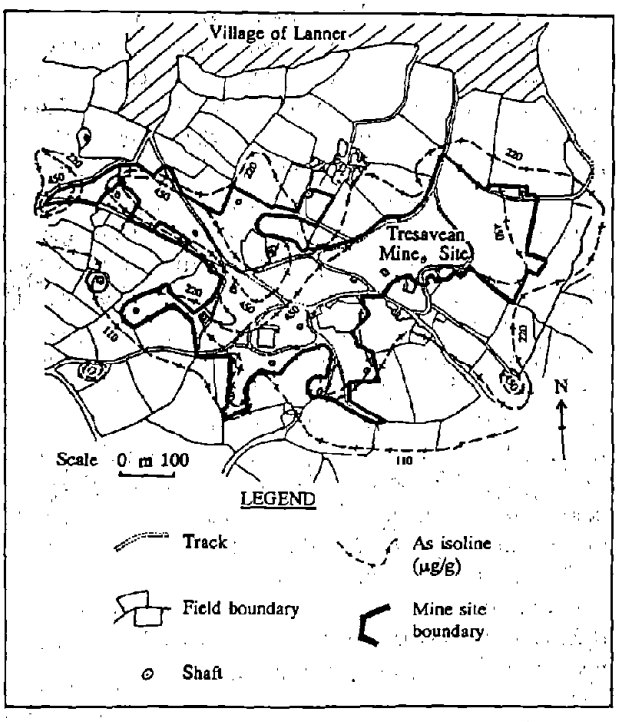


Figure 4. Distribution map of As in soil at Tresavean.

## Sequential Soil Extractions

Sequential extractions are a useful way of providing information on the partitioning of heavy metals into certain soil fractions. This in turn provides information about the origin, mode of occurrence, biological and physico-chemical availability, mobilisation and transport of heavy metals (Tessier 1979). For example, metals which are water soluble are likely to be more mobile and potentially phytotoxic compared with metals which are sequestered on to iron oxides (hydroxylamine hydrochloride extract). These metals, solubilised by the latter extractant, are in a more stable condition within the soil (hence the need for a stronger extractant). A number of authors have used sequential extraction procedures for specific purposes, modifying them to suit their own needs (Tessier 1979, Rose and Suhr 1971, Gatehouse et al. 1977, Hoffman and Fletcher 1979, Kersten and Förstner 1986). Consequently there is no standard single sequence of extractants. Some of the common extractants are reviewed by Beckett (1989).

The sequence starts with weak extractants, which remove those metals most loosely bound, gradually increasing in strength to the final extractant, which effectively removes any remaining metals within the crystal lattices of the resistate minerals. The extracts are not completely specific; therefore, all the metals removed by a particular extractant are not solely related to one phase. However, the majority of the element extracted in one part of the sequence can be attributed to that soil fraction.

Using sub-samples of the soils analysed for total metal content, a series of sequential extractions was undertaken to determine the soil fractions which held Cu and As. Due to the time-consuming nature of these experiments, analysis was restricted to a transect across an area at Tresavean which appeared most interesting from the total element analysis (fig. 5). It is necessary to use two different sequential extraction methods because cations and anions behave differently in soils. As a metal, Cu tends to form cations, whereas As is a metalloid and as such usually forms anions in soils. Two methods were used for the sequential extractions: one to determine the Cu cation distribution and one to determine the As anion distribution. The methods used for these extractions are given in figures 6 and 7.

The sequential extraction technique used for heavy metals divides the soil into seven fractions: water soluble, exchangeable, adsorbed, organic, sulphides, iron and manganese oxides and residual. These fractions are based on Viets' (1962) five pools of micronutrient cations in soil: (1) water soluble, (2) cations exchangeable by a weak exchanger like $NH_4^+$, (3) adsorbed, chelated, or complexed ions exchangeable by other cations possessing high affinities for exchange sites or extractable with stronger chelating agents, (4) micronutrient cations in secondary clay minerals and insoluble metal oxides, (5) cations held in primary minerals. Pool 3 has here been sub-divided into adsorbed and organic fractions as it was considered important to discriminate between these two fractions in the soil. An "extra" extract purely for sulphide determination has been included because Cu is present in the mine waste as sulphide. The extractants used for each of these fractions are, respectively; water, ammonium acetate, sodium acetate, sodium pyrophosphate, sodium hypochlorite, hydroxylamine hydrochloride and nitric-perchloric acid.

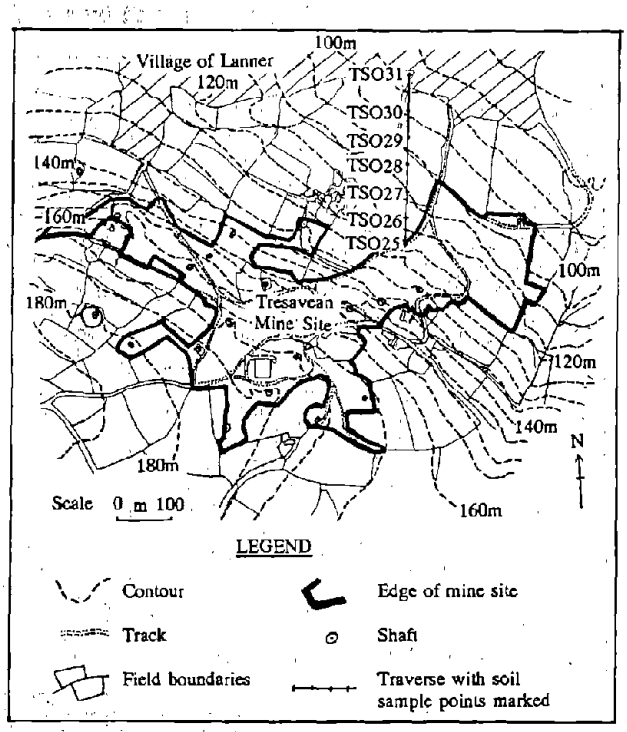


Figure 5. Topographical map of Tresavean with traverse and sample sites marked.

The series of extractants for As divides the soil into the following fractions: water soluble, Al-associated, Fe-associated, Ca-associated, occluded Al-associated (i.e. that which has been incorporated into the Al oxide or hydroxide crystal structure) and residual. The extractants used are, respectively; water, ammonium fluoride, sodium hydroxide, sulphuric acid, ammonium fluoride and nitric-perchloric acid, based on the sequential extraction technique of Woolson et al. (1973). Very little work on As extractions has been reported in the literature; the work of Woolson et al. (1973) is based on a modified Jackson procedure for soil phosphorus (Petersen and Corey 1966). However, this technique addresses the inorganic forms of soil P extraction and results may be open to misinterpretation if organic-anion compounds are likely. The sequential extraction is shown in figure 7 with the inorganic forms of As expected to be extracted. However, it must be borne in mind that sodium hydroxide may extract As related to Fe compounds, organic-Fe compounds and/or organics in the farmland soils.

To determine the residual fraction for both methods, the summation of the extracted element was subtracted from the total content previously determined by the nitric-perchloric digestion. Results from these two sequential

extractions are presented in figures 8-11.

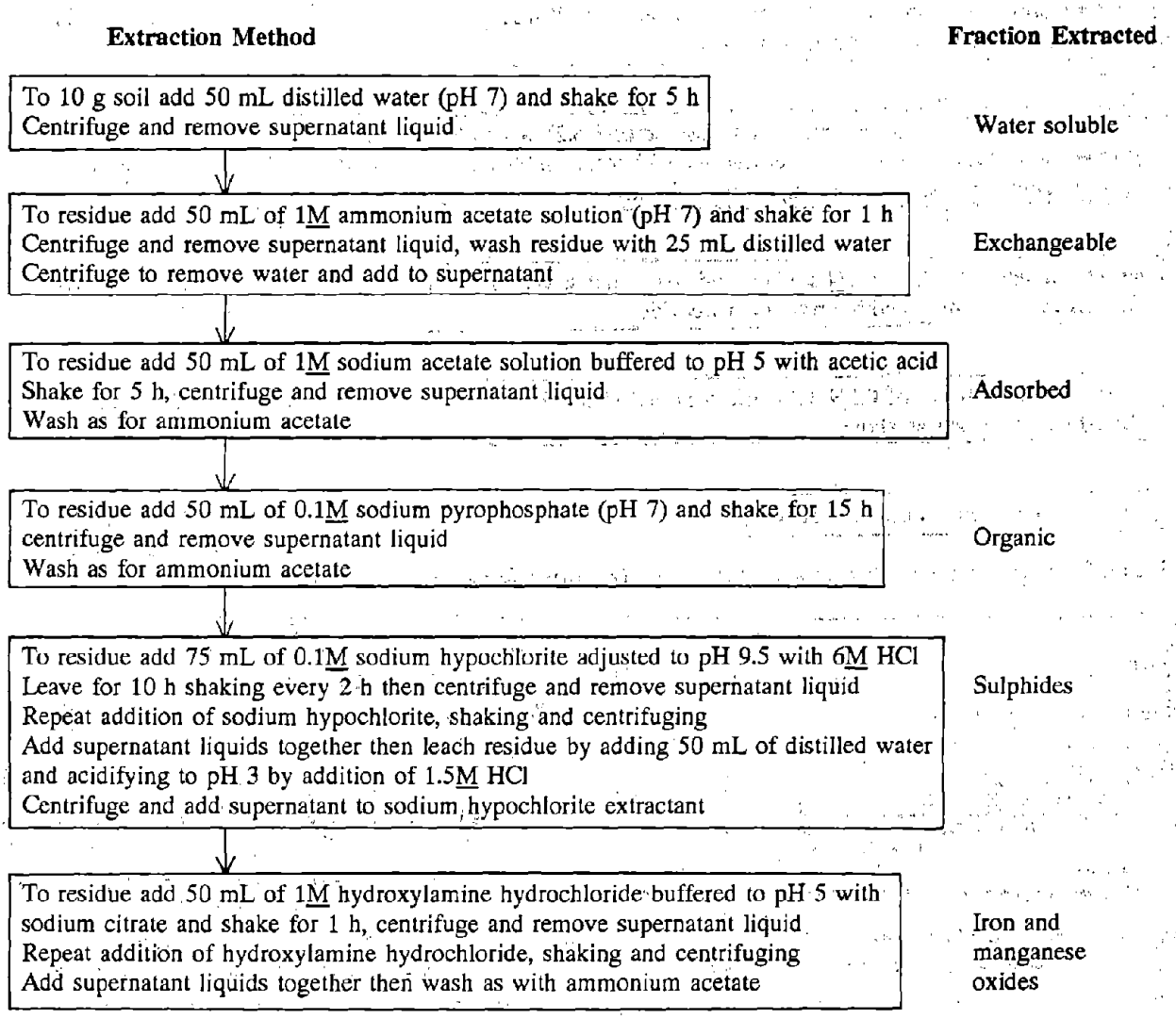


Figure 6. Sequential extraction method for copper (modified from Tessier et al. 1979).

## Results of Copper Fractionation in Soil

The Cu concentration in each extract is shown in figure 8. As a whole, the concentrations for each extractant decrease with distance from the mine site and, as such, mimic the total Cu content of the soils. However, with regard to Cu content in each fraction as a percentage of total Cu, all bar the residual and organic fractions behave similarly and appear to be constant. In each soil sample the majority of the Cu (fig. 9) is held in the residual phase. This ranges between approximately 45% and 60% of the total Cu content and shows a slight increase with distance from the mine. The second highest percentage of the total Cu, in every case, is the organic fraction, extracted by sodium pyrophosphate. This organic fraction, unlike the others, shows a decrease with distance from the mine site. It decreases from 33% to 21% over the 300 m traverse. The actual Cu concentrations in the organic fraction drop from 195 µg/g to about 60 µg/g from TSO25 to TSO31. By the end of the traverse the concentration appears to have levelled off (fig. 8).

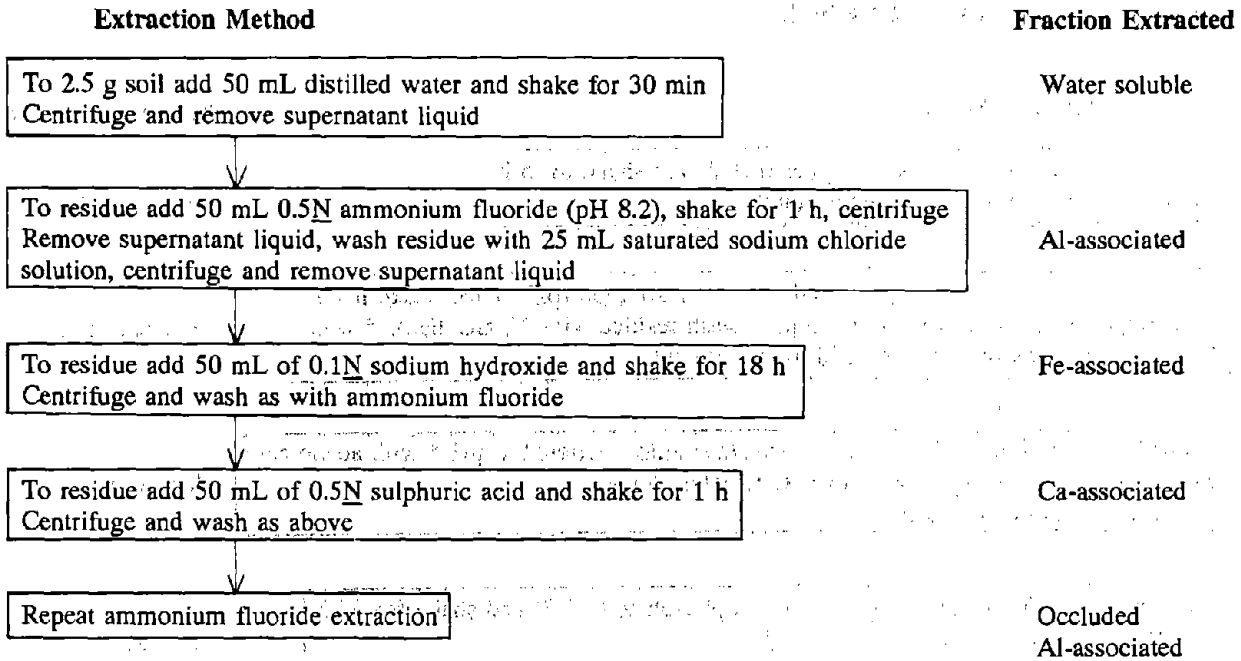


Figure 7. Sequential extraction method for arsenic (modified from Woolson et al. 1973).

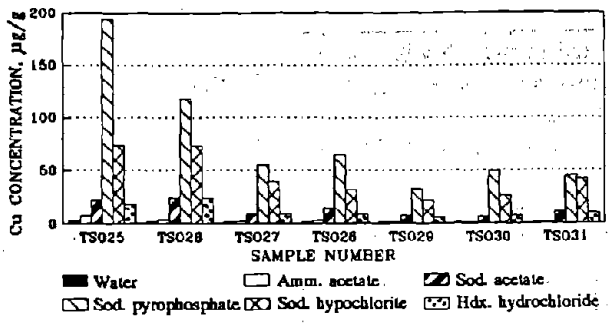


Figure 8. Sequential soil analysis for extractable Cu (µg/g) for soil traverse at Tresavean.

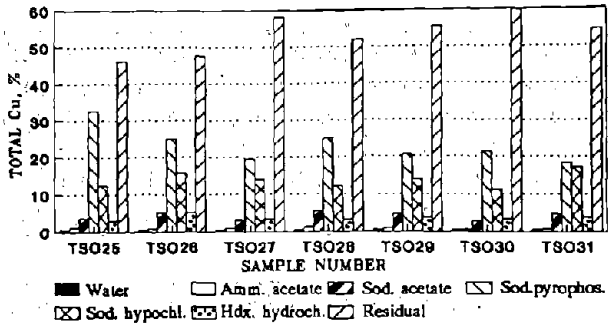


Figure 9. Sequential soil extraction for Cu (expressed as % of total Cu in each sample).

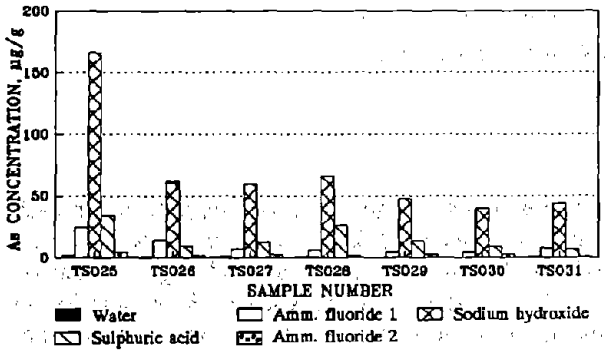


Figure 10. Sequential soil analysis for extractable As (µg/g) for soil traverse at Tresavean.

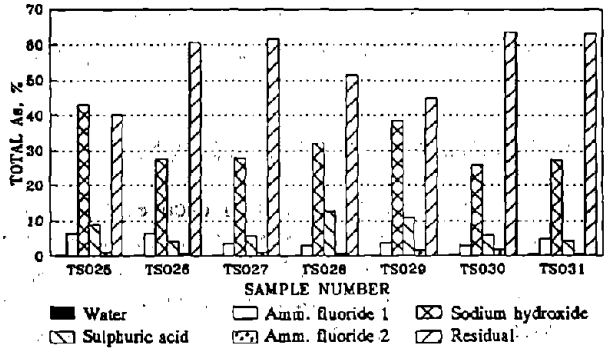


Figure 11. Sequential soil extraction for As (expressed as % of total As in each sample).

The remaining extractants are listed in order of decreasing importance with regard to Cu concentration: sodium hypochlorite (sulphides) > hydroxylamine hydrochloride fraction (Fe-Mn oxides) ≥ sodium acetate fraction

(adsorbed) > ammonium acetate fraction (exchangeable) ≥ water soluble fraction.

## Results of As Fractionation in Soil

A general decrease of As content in each soil fraction occurs with increasing distance from the mine (fig. 10). When presented as a percentage of the total As in each sample, the pattern is not so clear (fig. 11). Apart from TSO25, the residual fraction contains the most As in each case. For TSO25, which is the sample nearest the mine, the residual fraction is only slightly less than in the sodium hydroxide extract (40% compared with 43% respectively). For all the other samples the sodium hydroxide extract (Fe-organic As) contains the second highest percentage of As. The remaining extractants are listed in order of decreasing importance with regard to As concentration: sulphuric acid (Ca-associated) > the first ammonium fluoride extract (Al-associated) > the second ammonium fluoride extract (occluded Al-associated) > water. The first ammonium fluoride extract of TSO31 contains more As than the sulphuric acid extract (4.8% and 4.3% respectively), but this is only marginally greater.

The large drop in concentration between samples TSO25 and TSO26 in each soil fraction is obvious, but is most notable in the sodium hydroxide extract (that relating to Fe-organic As), which drops from 166 µg/g to 61 µg/g (fig. 10). The sodium hydroxide extract shows a further decrease in As content with distance from TSO26 to TSO31, but at a much reduced rate.

## Comparison of Sequential Extraction Results for Mine Waste and Soil

The distribution of Cu is very different in mine waste when compared with soil (fig. 13). The most striking differences are in the proportions found in the sodium acetate fraction and the residual fraction. For soils the residual fraction at 52% is by far the largest, but for mine waste this fraction only accounts for approximately 20%

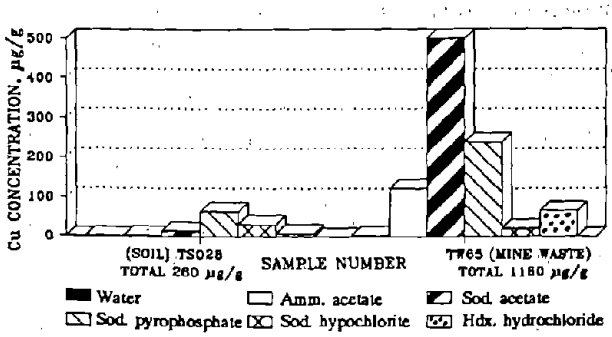


Figure 12. Comparison of sequential analysis of soil and mine waste for extractable Cu.

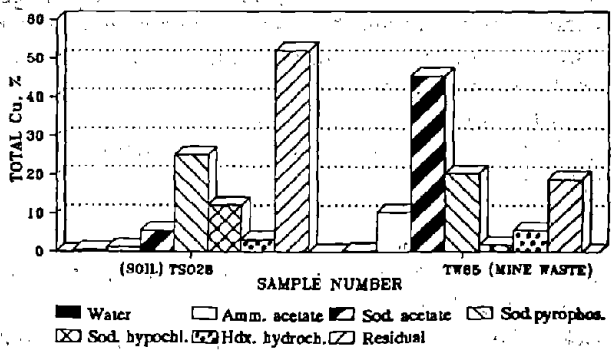


Figure 13. Comparison of sequential analysis for Cu (expressed as % of total Cu in each sample).

of the total Cu. In mine waste, Cu appears to be mostly held in the sodium acetate extractable fraction, i.e. that equivalent to adsorbed Cu. The organic Cu fraction of the two samples appears to be similar at around 20% of the total Cu concentration. This fraction in both cases is the second most important in terms of Cu concentration, but for the mine waste sample it is only just greater than the residual Cu concentration. The mine waste again differs from the soil in the amount of Cu found in the ammonium acetate extractable fraction. For mine waste this is 10% of the total Cu, but for the soil it is only 1%. The hydroxylamine hydrochloride fraction appears to be similar for both soil and mine waste at around 5% of the total Cu concentration. The sodium hypochlorite fractions (sulphide content) also differ. Surprisingly, soil contains more Cu in this fraction than does the mine waste (12% compared with 2%), but absolute concentrations are about the same (between 20 µg/g and 30 µg/g). Water soluble Cu is approximately the same in terms of percentage of total Cu, but in absolute values for mine waste it is at least twice the soil concentration.

Results from the sequential analysis of soil and mine waste for As are given in figures 14 and 15. The

results in figure 14 are the actual concentrations found in each fraction; as expected, the mine waste contains more As in each fraction than is found in the soil. Figure 15 shows that the distribution of As in soil and mine waste is extremely similar. It is important to note that the sodium hydroxide fraction (Fe-organic) is the most important extract, removing approximately 30% of the total As present in both soil and mine waste.

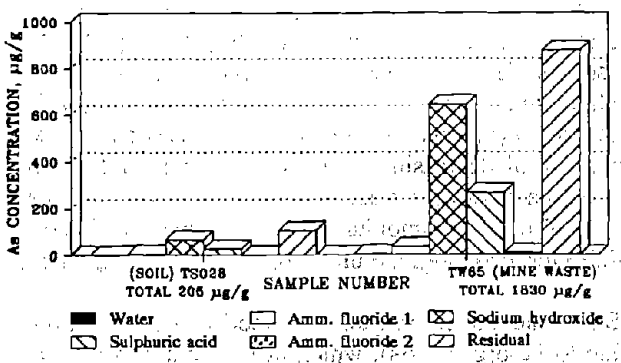


Figure 14. Comparison of sequential analysis for soil and mine waste for extractable As.

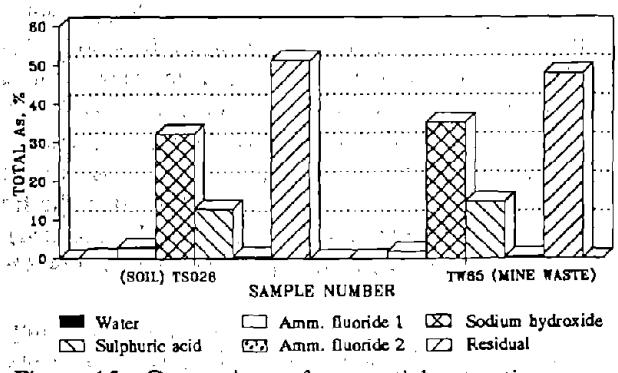


Figure 15. Comparison of sequential extractions (expressed as % of total As in each sample).

## Discussion

### Metal Distribution

The Cu distribution map (fig. 3) shows that the more mobile Cu has tended to migrate down slope away from the mine in the direction of greatest slope, which is to the north and east (fig. 5). However, As (fig. 4) is more likely to have been dispersed by wind as it is generally not soluble in the low pH conditions found in the mine waste at Tresavean. Arsenic-rich dust will have been blown from west to east with the prevailing wind, as will any fumes from the calciner plant. Hence As would be expected to predominate on the eastern side with less on the northern side (as found in the distribution map, fig. 4). This, of course, does not preclude some of the Cu from being dispersed in this fashion. There is evidence of some Cu contamination to the east, but it is not as marked as the As. It is possible that much of the As dispersion is relict and occurred when the mine was in operation.

The distribution maps for As and Cu (figs. 3 and 4) also show that despite high concentrations of As and Cu in Tresavean mine waste (up to 1.9% As and 2.8% Cu), dispersion is apparently of limited extent (approximately 150 m); however, in samples directly adjacent to the mine, it is quite severe. This may suggest a lack of extensive wind dispersion, which may be attributable to Cornwall's relatively wet climate reducing airborne dust. The high rainfall will, however, leach the available Cu from the mine waste.

### Metal Fractionation in Soil

It is important to note that the organic fraction is the most significant fraction of the partially extractable metals. Therefore any organic matter added to the fields surrounding the mine site (e.g. spreading of animal waste) may preferentially adsorb heavy metals dispersed from the mine. The organic fraction becomes less significant with distance from the mine, whereas the residual fraction appears to increase in percentage of total Cu.

From figure 11 it is apparent that residual As and Fe-organic associated As are by far the most important soil fractions, accounting for up to 90% of the total As. This concurs with the general association of As with Fe oxides and hydroxides (Woolson 1983, National Academy of Sciences 1977) and reiterates the importance of the organic fraction in soil (as mentioned above). Evidence from the literature indicates that As is strongly attracted to Fe oxides and hydroxides (Dudas 1987, Johnston and Barnard 1979) but not to soil organic matter (Johnson and Hiltbold 1969). Other authors report As may be bound to soil organic matter (National Academy of Sciences 1977, Woolson 1983).

## Comparison of Metal Content in Mine Waste and Soil

In the mine waste, about 75% of the Cu is present in the ammonium acetate, sodium acetate and sodium pyrophosphate fractions; that is, it is present in exchangeable, adsorbed and organic form. This compares with only 32% for these fractions in soil. Secondary minerals formed during the weathering of mine waste may partly account for this high percentage of easily extractable Cu. Any acid mine drainage would therefore be capable of removing large amounts of Cu from the mine waste. Fortunately there are no streams running through Tresavean, but surface drainage is likely to be a problem and has the potential to be a Cu dispersal mechanism.

As the sodium pyrophosphate fraction in the mine waste is a major Cu-containing fraction, the amount of organic matter found in a mine waste sample will also be very important. Generally the organic content of the mine waste is low (< 3% total organic carbon) and this may result in the saturation of the organic matter with Cu because of the very high Cu content of the spoil. The surprisingly high Cu concentration in the sodium hypochlorite fraction of soil compared with this fraction in mine waste may be due to inefficient extraction of organic Cu by the previous sodium pyrophosphate extraction. It seems very unlikely that soil will contain more Cu sulphides than the mine waste. Beckett (1989) has reported some research to have shown that sodium pyrophosphate can dissolve Cu sulphides. Therefore it is possible the sodium pyrophosphate fraction of the mine waste not only reports organic Cu but also some sulphidic Cu. This will in turn reduce the amount of Cu extracted by sodium hypochlorite (designed to remove sulphides) as it follows the sodium pyrophosphate extraction.

The As fractionation in the mine waste is very similar to that in the soil and appears to be associated mainly with the residual and Fe-organic fractions, with the Ca-associated fraction also being important. As mentioned earlier, $As_2O_3$ is present in the mine waste at Tresavean; as this is soluble in alkalis, both the ammonium fluoride and sodium hydroxide extractions may report it. The results show that the majority of the As in the mine waste is not held in the easily extractable fractions (i.e. the water and first ammonium fluoride extracts), which only account for approximately 2% of the total As. Hence any dispersion of As would have to be of a physical nature. For As to be dispersed any distance it would therefore be as wind-blown dust.

## Conclusions

Element distribution maps produced from the extensive soil survey of farmland surrounding the Tresavean Mine have revealed two distinct patterns of Cu and As distribution. Arsenic distribution extends in an easterly direction and appears to correlate with the prevailing wind direction. Cu distribution extends more to the north and east and follows the down slope drainage pattern. Considering the high Cu-As content of the mine waste, the extent of dispersion is less than expected. This may be due to Cornwall's high rainfall reducing dust-blow.

Sequential extractions were used to determine the fractionation of Cu and As in soils at Tresavean. It was found that Cu is held mainly in the residual and organic fractions, whereas As is held mainly in the residual and iron-organic fractions. Although the absolute concentrations of As and Cu decrease with distance from the mine site as expected, the relative percentage of As and Cu in each fraction tends to be independent of distance from the mine. The sequential extractions performed on the mine waste show Cu to be present largely in the easily extractable fractions; As is mostly present in the less easily extractable fractions. Copper is therefore more likely to be subject to leaching and hydromorphic dispersion (e.g. by drainage), whereas As is more prone to mechanical dispersion (e.g. as wind-blown dust).

## Acknowledgements

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# ASSESSMENT OF AN INACTIVE CYANIDE HEAP LEACH STACK LOCATED IN THE SOUTHEASTERN UNITED STATES[1]

James R. Boyle, Jr., and Carl W. Smith[2]

**Abstract:** The U.S. Bureau of Mines has established a research program to gather and evaluate data on cyanide (CN) degradation from a Southeastern U.S. gold mining and processing operation. As part of this program, the Bureau installed a sampling network of boreholes to monitor the CN loss from an inactive heap leach stack. The stack was made up of two 9.2-m lifts comprising an estimated 1.1 million mt of cyanide leached ore solids and covered an area of approximately 8 ha. The sampling network consisted of 15 spent ore solids retrieval boreholes placed in 5 clusters of 3 boreholes each; the distance between the boreholes within a cluster was approximately 4.5 m, and the borehole depths in each cluster were placed at 5.5, 8.5, and 11.5 m. Samples were collected using a hand sampling device. Sampling began in November 1991 and ended in May 1993. For this period of approximately 570 days, eight sets of samples were collected for use in determining the total and weak acid dissociable (WAD) cyanide of the spent ore and associated entrained leach solution, and seven sets were collected for metals analysis by inductively coupled plasma (ICP). Five composite samples were collected and subjected to the Environmental Protection Agency's (EPA) Toxic Characteristic Leachate Procedure (TCLP) to determine toxicity with respect to metals. X-ray diffraction was used to identify mineralogical composition, and ion chromatography was used to identify CN species.

**Additional Key Words:** cyanide, heap leaching, precious metals.

## Introduction

The U.S. Bureau of Mines (USBM) Environmental Research Program to evaluate cyanide degradation in humid conditions led to a research project in the gold mining region of the Southeastern United States to characterize the effects of rinsing on a recently deactivated heap leach stack. A Memorandum of Agreement (MOA) was negotiated with officials of the then Piedmont Mining Co., Inc., and the USBM to allow Bureau personnel access to the active operations. In November 1991, a network of rinse water recovery wells and spent ore retrieval boreholes was established at Piedmont Mining Co.'s Chase Hill heap leach stack. After network placement, the company began to releach the stack with a low-CN-level solution in an effort to recover residual gold values. A significant drop in the pH of the stack effluent led company officials to suspect that acid generation, due to the oxidation of the pyrite within the stack, was overcoming the natural buffering characteristics of the ore and leach solution, and this caused the company to idle the stack to fully evaluate final closure plans. In an effort to document CN loss within the spent ore and associated entrained solution, the USBM continued sampling the spent ore until May 1993, when the wells and boreholes were decommissioned. The spent ore with the entrained solution was analyzed for total CN, WAD CN, and metals content. A composite ore solids sample from each sampling station was evaluated for toxicity with respect to metals using the EPA TCLP (U.S. Code of Federal Regulations, 1992). The data in this study along with similar studies being conducted by the USBM (Church and Boyle 1990, Comba et al. 1992), will aid in determining the environmental characteristics of heap leaching as affected by variations in the ore body and climatic conditions.

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## Mechanisms of Cyanide Degradation and Neutralization

Because of its high reactivity, cyanide is often considered a transient pollutant. Previous research has identified many of the more prevalent mechanisms of CN degradation and neutralization (Davis et al. 1991, Engelhardt 1984, Franson 1985, Fuller 1984, Howe 1984, Huiatt 1984, Stanton et al. 1986, Stevens 1984). The following are the major degradation processes associated with the gold processing industry; they can occur individually or in combination when environmental conditions favor such reactions.

### Volatilization

Volatilization is an important mechanism in the degradation of cyanides. Simple cyanides and certain weak metal cyanide complexes are dissociated into free cyanide ions and hydrogen ions. The dissociation reaction of HCN is

$$HCN = H^+ + CN^- \quad (1)$$

At 20° C, the dissociation constant (Ka) of HCN is $10^{-9.36}$; then

$$Ka = [H^+][CN^-] / [HCN] = 10^{-9.36}; \quad (2)$$

$$[CN^-] / [HCN] = 10^{-9.36} / [H^+]. \quad (3)$$

At a pH of 9.36, the hydrogen ion $[H^+]$ concentration is $10^{-9.36}$ so that [CN]=[HCN], or half the CN will be present as HCN. As the pH approaches 7, the majority of the free CN will be present as HCN. Certain soluble, alkali-metallic, complex cyanides will dissociate to the point of liberating $CN^-$ with the consequent formation of HCN. The HCN may then be liberated from solution as a gas, depending on the vapor pressure, which is dependent upon the temperature. At 25.9° C the vapor pressure of HCN is 760 mm Hg.

### Oxidation

Chemical oxidation of cyanide (chlorine, $SO_2$ in air, peroxidation) is currently in use by the industry to detoxify CN solutions. The reaction of cyanide to form cyanate, $CNO^-$, is very rapid at a pH of 11.5 or higher; the subsequent breakdown of cyanate to carbon dioxide and nitrogen is slower. Natural oxidation of cyanide in process solutions has been suggested, but the process is not fully understood.

### Biodegradation

The use of bacterial species is a relatively new method for the treatment of cyanide waste water solutions. Some bacteria have been shown to degrade cyanide under aerobic and anaerobic conditions over a wide range of pH. Aerobic biodegradation occurs in an oxidized environment by the following reaction:

$$HCN + 1/2CO_2 + \text{enzyme} \rightleftharpoons HCNO. \quad (4)$$

Anaerobic biodegradation is usually restricted to the moderate to strong reducing environments and occurs only where $HS^-$ or $H_2S$ is present, depending on the pH of the sulfur species. When the pH is less than 7, then $CN^- + H_2S$ (aqueous) $\rightarrow HCNS + H^+$. When the pH is greater than 7, then $HCN + HS^- \rightarrow HCNS + H^+$. The thiocyanate can then be hydrolyzed.

## Complexing

Cyanide will combine with certain heavy metals (ferrous and ferri iron, cadmium, cobalt, copper, nickel, silver, zinc, and others) to form complex cyanides. The ferrocyanide complex, which is commonly found in gold processing effluents, is considered nonhazardous owing to its stability even in low-pH solutions.

## Photodecomposition

The stable iron complexes, ferrocyanide and ferricyanide ($Fe(CN)_6^{4-}$ and $Fe(CN)_6^{3-}$), have shown susceptibility to decompose when subjected to ultraviolet radiation. The decomposition of these complexes has generated some concern for the potential to form free cyanide. Since only the surface of the spent ore would be exposed to any ultraviolet light, this is not a major factor contributing to cyanide decomposition. There is an environmental concern with these complexes decomposing in effluents. Research has indicated that this may be negligible in deep, turbid waters. Solutions of ferrocyanide and ferricyanide have proved more toxic to trout when tested under light than in total darkness, although that research indicated that the free cyanide formed by the decomposition of iron cyanide complexes was not the only toxic agent under these conditions.

## Thiocyanate Reactions

Thiocyanate ($SCN^-$) can be formed by reactions between cyanide and ores containing sulfur and its compounds. Thiocyanate is substantially less toxic than cyanide, but is highly dependent upon the Eh-pH conditions of the soil, and the mineralogical state and sulfur content of the ore and host rock.

## Study Site

Construction of the Chase Hill leach pad began in the latter part of 1988. The pad was constructed using 45 cm of compacted clay with a 60-mm high-density polyethylene (HDPE) primary liner and a 40- to 60-mm HDPE secondary liner. The area of the study is shown by the existing pad in figure 1; it covers 6 ha. The pad extension covers an additional 2 ha. The leach stack that is the focus of the study is made up of 1.1 million mt of ore that was placed on the leach pad by a radial stacker in 9.2-m-high lifts. The ore was crushed to minus 5 cm and agglomerated with an average of 6.8 kg/mt cement. The first lift was stacked and leached by late 1989, and the second lift was stacked through August 1991 and leached through August 1992. Sporadic attempts at heap spraying with no additional CN being added to the circuit continued on this stack until October 1992. Leaching of the pad extension began in August 1992 and continued through February 1993.

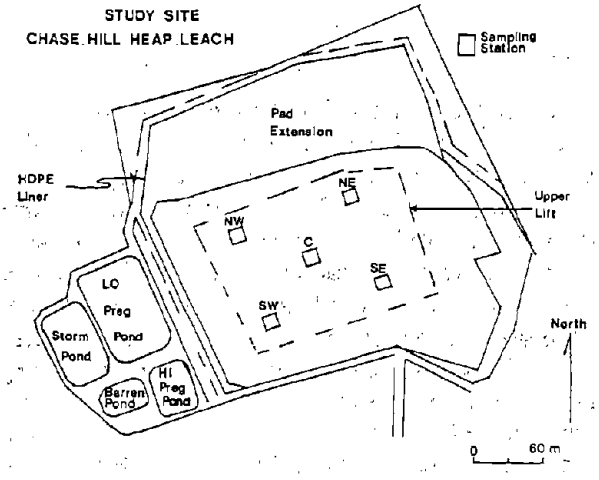


Figure 1. Study site.

The ore was leached with a 500-ppm free-CN solution at the start of the leach cycle, and the cyanide concentration was gradually decreased until the solution was approximately 100 ppm as gold values began to decrease as leaching progressed. The solution was distributed on the ore by sprays at the rate of 950 L/min. The study site was leached until August 1991. At the time of the sampling network installation in November 1991, the study site had remained idle while the extension section of the pad was leached. In an attempt to recover residual gold values, leaching with a low-level-CN solution was begun in April 1992. Problems with acid generation were encountered in the following month, when it was discovered that the pH of the effluent had decreased to below 7. Sporadic attempts at heap

spraying were made, using additions of sodium hydroxide to limit acid production, without additional CN being added to the circuit. The company determined that attempting to minimize acid generation, with the associated risk of increased metals migration, while continuing to spray was uneconomical, and spraying was suspended in October 1992. Studies by the company are currently ongoing to determine the optimum permanent closure scenario.

## Network Design and Installation

The network was designed to enable the systematic sampling of spent ore at various depths and locations within the leach stack without having to utilize a drill rig for each sampling period. This was accomplished by dividing the upper lift into quadrants and placing a sampling station in each quadrant and at the quadrants' intersection for a total of five sampling stations. Each station was made up of three spent ore retrieval boreholes. The stations are designated NW-northwest, NE-northeast, SW-southwest, SE-southeast, and C-central (fig. 1). The depths of the boreholes were varied at 5.5, 8.5, and 11.5 m. By varying the depths (fig. 2) within each station, the network would profile the stack and identify anomalies occurring at depth.

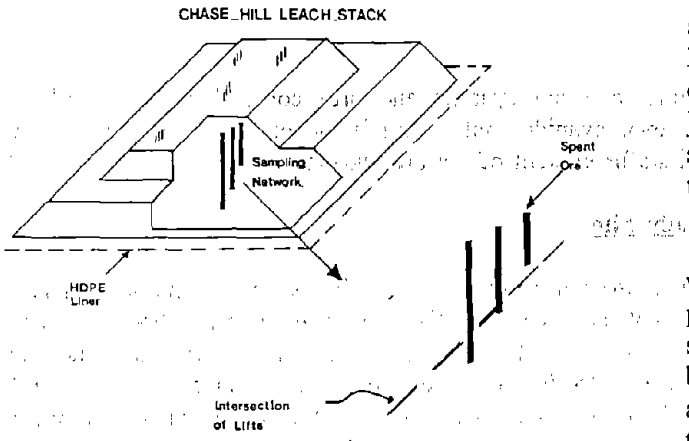


Figure 2. Sampling network.

The installation of the boreholes was accomplished using a track mounted rig drilling with hollow-stem augers. The augers have an outer diameter of 19 cm and an inner diameter of 8 cm. When the installation depth was reached, the center steel was removed, and the casing was lowered through the center of the hollow stem augers.

The spent ore retrieval boreholes were cased with 6.4-cm flush-threaded casing that allowed the passage of a hand auger for the retrieval of the solids sample. Each borehole has approximately 0.8 m of bentonite placed at the base of the casing that acted as a seal to prevent the cement grout from intruding into the casing. The boreholes were grouted with neat cement from the upper level of the bentonite to the surface collar. The collar prevented channeling of rain water into the borehole. The borehole casing was capped and sealed with an O-ring.

## Sample Collection

Sample collection of the spent ore began during drilling for the network installation. The systematic sampling at 3-m intervals of each deep borehole, at each station, was accomplished using a split spoon sampler. The split spoon was a 7.6-cm-diam steel tube, 60 cm in length. The spoon was lowered through the augers to the depth to be sampled and driven into the borehole bottom using a falling hammer. The tube recovered a sample of approximately 2,000 g, which was halved lengthwise; one half was preserved for cyanide analysis, and the other was retained for metals analysis. Subsequent samples were collected using a hand sampling device. The sampler was a 5-cm-diam by 45-cm-long stainless steel open-ended auger. The general sampling procedure was to lower the sampler into the casing using 1.5-m steel rods and to twist the sampler into the ore solids until approximately 25 cm of solids was forced up into the sampler. The top 5 cm of the sample was discarded, and the remaining 20 cm of approximately 500 g was divided in the same manner as the split spoon samples for preparation for analysis. Owing to the oversized nature of some of the solids encountered, the sampler would have to be lowered in and out of the borehole several times until an adequate sample was retrieved. From November 1991 to May 1993 (approximately 570 days) eight sets of samples were collected for total and WAD CN analyses, and seven sets were used for metals analyses.

## Sample Preparation

Previous work by the Bureau (Comba et al. 1992) has shown that extreme care must be taken when preparing solid samples for CN analysis. Each sample collected from the stack was double-bagged in plastic zip-lock-type sample bags to minimize contact with air and placed on ice for transport to the laboratory. Once received at the laboratory, the samples were frozen; while frozen, they were crushed to minus 20 mesh by a roll crusher. The sample was stored frozen until CN analysis could be conducted. These measures were necessary to prevent any loss of CN.

Samples to be analyzed for metals content were mixed and split to yield a representative 75- to 100-g fraction. This fraction was ground to minus 100 mesh and analyzed for a suite of 31 metals by ICP. Fractions were also split out of selected samples and ground to minus 325 mesh for mineralogical determination by X-ray diffraction. Five as-received composite samples (one from each station) were subjected to EPA's TCLP to determine toxicity with respect to metals.

## Metals Analyses

The average values by station for a selected group of metals, which occurred in sufficient concentrations for reliable ICP analysis, are shown in table 1. From this table it can be seen that despite some minor variations, these metal values are reasonably consistent when taking into account the variability of the ore. The Fe values for the NW, SW, and C stations indicate a variability by the relatively high standard deviation. Table 2 further breaks down the Fe data by depth. The data listed by depth are more consistent and reveal that the NW, SW, and C stations have higher Fe concentrations in the lower levels of the stack. The Mn content also shows a high variability that appears even when the data are grouped by depth. This variability appears to be caused by the nature of the ore.

Table 1. Average values of selected metals in spent ore samples by station.

| Metal | Station | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | NW | | NE | | SW | | SE | | C | |
| | Avg | StD | Avg | StD | Avg | StD | Avg | StD | Avg | StD |
| Al . . . . . % | 4.7 | 0.6 | 4.9 | 0.4 | 4.5 | 0.7 | 4.0 | 0.6 | 6.0 | 1.3 |
| Ca . . . . . % | .6 | .2 | .6 | .2 | .7 | .4 | .5 | .2 | .5 | .1 |
| Fe . . . . . % | 3.3 | 1.3 | 1.6 | .4 | 3.8 | 1.8 | 1.2 | .2 | 2.6 | 1.9 |
| K . . . . . % | 2.2 | .4 | 2.1 | .5 | 1.9 | .4 | 1.6 | .3 | 2.4 | .4 |
| Mg . . . . . % | .2 | .03 | .2 | .05 | .19 | .04 | .16 | .02 | .16 | .02 |
| Ba . . . . . ppm | 270 | 30 | 282 | 34 | 211 | 39 | 247 | 33 | 320 | 47 |
| Mn . . . . . ppm | 64.7 | 22.7 | 47.8 | 12.2 | 72.5 | 22.7 | 59.8 | 18.3 | 55.7 | 15.2 |

StD Standard deviation.

X-ray analyses identified the major mineral constituent as quartz with minor amounts of mica (lepidolite) and kaolinite and trace to minor amounts of pyrite. Semiquantitative analyses of the X-rayed samples indicated a total sulfur of between 0.3 and 1.0 wt %. The majority is most probably associated with the pyrite.

Results of the EPA TCLP tests on the five composite samples are shown in tables 3 and 4. Table 3 lists the metals that are considered toxic by the EPA. The data listed in this table show that no metal exceeded the levels set by the EPA; therefore, the ore would not be considered hazardous with respect to metals. Table 4 lists additional metals of interest to the study. Under the physical and chemical conditions that the TCLP attempts to duplicate, a certain amount of these metals would be expected to leach out of the spent ore. The Cu, Fe, and Zn would be available to complex with any available CN under these conditions.

Table 2. Average values of Fe in the spent ore at depth, percent.

| Depth, m | Station | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | NW | | NE | | SW | | SE | | C | |
| | Avg | StD | Avg | StD | Avg | StD | Avg | StD | Avg | StD |
| 5.5 | 1.6 | 0.8 | 1.4 | 0.3 | 1.4 | 0.3 | 1.3 | 0.1 | 0.8 | 0.1 |
| 8.5 | ND | ND | 1.5 | .2 | 5.2 | 1.1 | 1.3 | .2 | ND | ND |
| 11.5 | 3.9 | .5 | ND | ND | 4.4 | .4 | 1.0 | .1 | 4.7 | .3 |

ND Not determined.
StD Standard deviation.

Table 3. Results of EPA's TCLP on composite spent ore samples, ppm.

| Station | As | Ba | Cd | Cr | Pb | Hg | Se | Ag |
|---|---|---|---|---|---|---|---|---|
| NW | 0.54 | 0.29 | <0.005 | <0.04 | <0.09 | <0.0014 | <0.3 | <0.01 |
| NE | <.06 | .64 | <.005 | <.04 | <.09 | <.0014 | <.3 | <.01 |
| SW | <.06 | .17 | <.005 | <.04 | <.09 | <.0014 | <.3 | <.01 |
| SE | <.06 | .17 | <.005 | <.04 | <.09 | <.0014 | <.3 | <.01 |
| C | <.06 | .16 | <.005 | <.04 | <.09 | <.0014 | <.3 | .02 |
| EPA TCLP levels | 5.0 | 100.0 | 1.0 | 5.0 | 5.0 | 0.2 | 1.0 | 5.0 |

Table 4. Results of EPA's TCLP on composite spent ore samples for selected metals, ppm.

| Station | Cu | Fe | Mg | Mn | Zn |
|---|---|---|---|---|---|
| NW | 0.05 | 0.07 | 2.3 | 0.12 | 0.14 |
| NE | .07 | .13 | 2.8 | .15 | <.04 |
| SW | .03 | .14 | 1.5 | .12 | 1.7 |
| SE | .21 | .12 | 1.8 | .07 | 2.2 |
| C | .05 | .03 | 1.8 | .07 | .12 |

## Cyanide Analyses

The spent ore and associated entrained solution were analyzed for total and WAD CN by the distillation method recommended by the Standard Methods for the Examination of Water and Wastewater (Franson 1985). The analyses for total CN include the complex iron cyanides, the WAD CN, free cyanide, and most of the inorganic CN complexes with the exception of gold, cobalt, and some platinum complexes (Huiatt 1984). The analyses for WAD CN include the cyanide ion, hydrogen cyanide, and the copper, nickel, silver, zinc, and cadmium complexes. No CN recovery from the iron complexes and cobalt is reported. The distillation method is similar to the one used for total CN except that the pH of the distillation solution is only slightly acidified (approximate pH of 4.5) (Huiatt 1984).

Tables 5 and 6 list the average amount of total CN and WAD CN by depth. The standard deviations of the averages indicate the variability that can be encountered when analyzing for CN in solid samples (Comba et al. 1992). These data, though variable, do indicate an unmistakable decline in the CN content for the stack. The relationship between the moisture content and the CN content over time is illustrated in figures 3-5. From the figures it is evident that the CN content at all depths decreased significantly, from the peak values in April 1992 (day 180) to the values reported in May 1992 (day 210), yet the moisture contents of the two sample sets remained virtually the same. The peak values reported for April correspond to the company's attempt to recover residual

gold values by respraying with a low level CN solution. The sharp decline in the CN values corresponds to finding that the pH of the effluent had declined to below 7. The CN values then gradually trend downward with the moisture content, at the 5.5-m depth, to slightly below the initial values; the final values at the 8.5- and 11.5-m levels are virtually the same as the initial values.

Table 5. Average values of total CN in spent ore by sample date and depth, ppm.

| Date (days) | Depth, m | | | | | |
|---|---|---|---|---|---|---|
| | 5.5 | | 8.5 | | 11.5 | |
| | CN, ppm | StD | CN, ppm | StD | CN, ppm | StD |
| 11-91 (30) | 7.6 | 2.3 | 4.5 | 2.0 | 6.9 | 3.1 |
| 04-92 (180) | 16.1 | 3.9 | 14.2 | 3.5 | 16.6 | 3.1 |
| 05-92 (210) | 7.4 | 5.3 | 7.3 | 3.6 | 10.4 | 5.8 |
| 08-92 (300) | 8.2 | 3.9 | 4.6 | 1.5 | 10.8 | 4.6 |
| 10-92 (360) | 7.0 | 3.2 | 4.1 | 1.5 | 8.2 | 2.7 |
| 11-92 (390) | 5.5 | 3.4 | 5.9 | 5.1 | 10.1 | 2.4 |
| 03-93 (510) | 5.0 | 3.7 | ND | ND | 6.7 | 1.4 |
| 05-93 (570) | ND | ND | 4.2 | 1.4 | 6.4 | 2.7 |

ND Not determined.
StD Standard deviation.

Table 6. Average of WAD CN for spent ore by date at depth, ppm.

| Date (days) | Depth, m | | | | | |
|---|---|---|---|---|---|---|
| | 5.5 | | 8.5 | | 11.5 | |
| | CN, ppm | StD | CN, ppm | StD | CN, ppm | StD |
| 11-91 (30) | 2.2 | 1.0 | 2.0 | 1.8 | 1.6 | 0.5 |
| 04-92 (180) | 10.4 | 2.7 | 8.6 | 2.0 | 10.3 | 3.9 |
| 05-92 (210) | 4.4 | 3.7 | 4.9 | 3.5 | 5.7 | 3.2 |
| 08-92 (300) | 3.6 | 2.0 | 2.9 | 2.3 | 5.6 | 2.6 |
| 10-92 (360) | 3.2 | 1.6 | 2.0 | 1.4 | 4.6 | 2.2 |
| 11-92 (390) | 1.8 | .8 | 2.7 | 2.6 | 2.4 | .4 |
| 03-93 (510) | 1.3 | .8 | ND | ND | 2.4 | 1.7 |
| 05-93 (570) | ND | ND | 2.0 | 1.6 | 2.3 | 1.0 |

ND Not determined.
StD Standard deviation.

In an effort to determine the CN species within the samples, 50-g splits from the samples collected in November 1992 were hand-stirred at 15-min intervals for approximately 6 h in a 0.1-$\mu$ solution of sodium hydroxide. The samples were then covered to prevent evaporation and light exposure and left overnight. The leachate was then analyzed by ion chromatograph to determine CN species that may be present. The results are shown in table 7. The methodology used for sample preparation and the use of the ion chromatograph are research approaches; this methodology may not be applicable for the routine determination of cyanide.

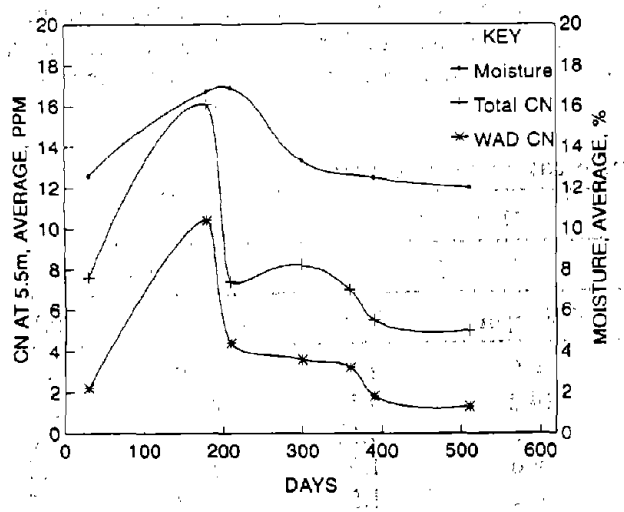


Figure 3. CN and moisture content as a function of time at 5.5 m.

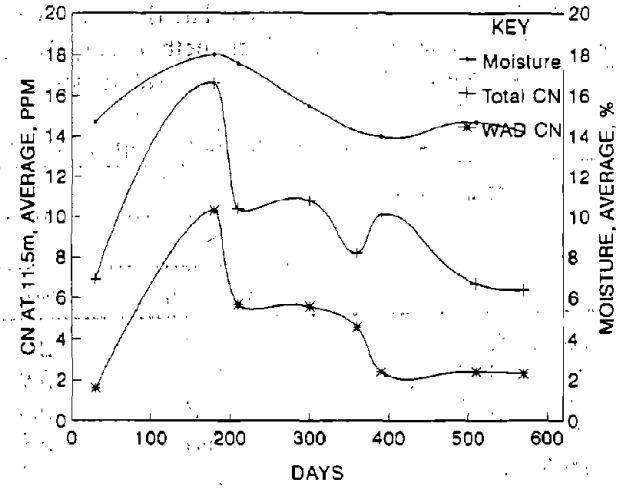


Figure 5. CN and moisture content as a function of time at 11.5 m.

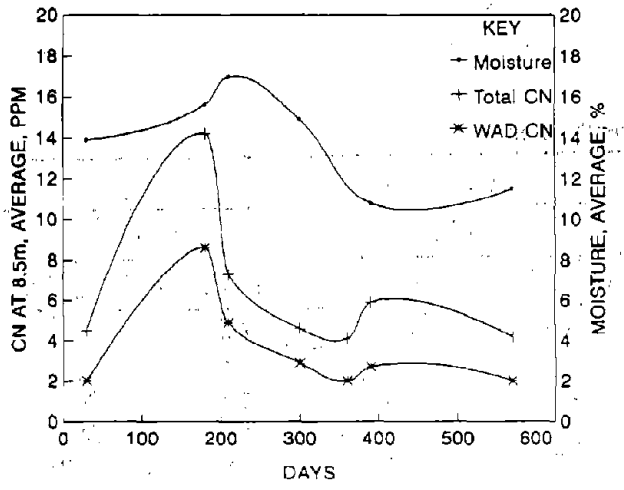


Figure 4. CN and moisture content as a function of time at 8.5 m.

Table 7. CN species identified in spent ore samples, ppm.

| Depth, m | Station | Free CN | $CNO^-$ | $SCN^-$ | $Fe(CN_6^{4-})$ |
|---|---|---|---|---|---|
| 5.5 | NE | 0.7 | 6.7 | 9.7 | 11.5 |
| | SW | <.01 | 3.5 | 7.1 | 3.3 |
| | SE | <.01 | 4.3 | .4 | 1.7 |
| | C | .03 | 6.7 | .5 | 7.8 |
| 8.5 | NE | 1.3 | 13.2 | 12.3 | 6.3 |
| | SW | <.01 | 3.3 | 2.9 | 1.8 |
| | SE | <.01 | 5.1 | .4 | 3.2 |
| 11.5 | NW | .8 | 6.6 | 3.5 | 4.9 |
| | SW | <.01 | 5.1 | 7.7 | 8.1 |
| | SE | .5 | 10.9 | 1.3 | 7.7 |
| | C | .4 | 8.9 | 15.9 | 3.3 |

## Discussion

The CN values in the initial samples were low compared with those of similar studies (Comba et al. 1992; Engelhardt 1984), where the stack would have been considered neutral in at least one Western State (Nevada). The initial low values could be attributed to the decrease in the amount of CN added to the leach solution as the gold values began to decline and to subsequent evaporation and drainage from the stack as evidenced by the low moisture contents of the initial samples. The attempt to releach the stack added cyanide to the ore, the significant portion as solution entrained within the pore spaces of the ore. The steep decline in CN corresponds with the decline in pH of the stack effluent. Since there was no corresponding decline in the moisture content of the samples, it is probable that a certain amount of the CN loss was due to the volatilization of the free CN and weaker metal complexes that make up the WAD CN values. The CN values then gradually decreased with the moisture content, indicating a CN loss as the solution percolated through the stack and drained from the base.

The company attempted to control acid generation in order to prolong spraying by adding additional sodium hydroxide to the leach spray. This proved to be uneconomical, and the spraying was discontinued; the stack was idled, and closure plans are now being formulated. In recent discussions, company officials have reported a further decline in the pH of the effluent to 3, indicating significant acid generation, which is attributed to the oxidation of the pyrite in the stack.

The results from the TCLP indicate that the ore solids are not hazardous with respect to metal leachates, but do indicate a certain mobility for Fe under laboratory conditions. These results indicate that the TCLP would not be an acceptable indicator of acid rock drainage (ARD).

Speciation efforts using ion chromatography indicate the presence of cyanate ($CNO^-$), thiocyanate ($SCN^-$), and ferrocyanide complexes in the samples. The presence of these species is relatively common where cyanide is used in processing (Huiatt 1984). The formation of the SCN can occur from the reaction of free CN with various forms of sulfur associated with sulfide minerals (Huiatt 1984), in this case pyrite and any pyrrhotite available. The formation of $Fe(CN)_6^{4-}$ would be due to any available iron in solution that would react with the available free CN. Both $SCN^-$ and $Fe(CN)_6^{4-}$ are considered to be relatively nontoxic. The presence of cyanate is in all probability due to the oxidation of a certain amount of the aqueous cyanide. This reaction normally is very slow without the presence of a catalyst or strong oxidizer such as hypochlorite or hydrogen peroxide (Wang 1990).

The placement of the sampling system was governed to a large extent by economics. Since it was not possible to mobilize the drill and crew to sample the stack when needed, a compromise was made by installing the cased ore solids retrieval boreholes. The 6.4-cm casing was the largest that would pass through the available hollow stem augers. This, to a large extent, dictated the sample that could be collected. The data presented in this report correlate well with similar data collected by the company in its own sampling of the stack.

## Conclusions

The ore solids in the stack attained very low levels of CN while the stack was idle. This is attributed to evaporation and drainage of the entrained solution. Attempts to releach the stack added CN to the ore solids via additional entrained solution. The additional CN was in part volatilized by the decrease in pH of the entrained solution as a result of the oxidation of pyrite within the stack. Subsequent evaporation and drainage accounted for the additional CN losses within the ore solids, and the final CN values correspond closely with the values of the initial samples. By evaporation and drainage alone, the stack reached total CN values that would be considered neutral (10 ppm) in at least one Western State (Nevada). South Carolina has no set limits; allowable concentrations are determined on a case-by-case basis. Company officials anticipate that the required values for closure will be lower than 10 ppm. The formation of the species $SCN^-$ and $Fe(CN)_6^{4-}$ indicates that some of the free CN has been neutralized by complexing with sulfur and iron. The ore solids are not considered toxic with respect to metals as defined by the EPA's TCLP.

The major concern from this stack is the oxidation of the pyrite present in the ore solids that is causing acid rock drainage. The addition of water for the purpose of rinsing would only aggravate the situation because of the increased possibility of solubilizing metals. This in turn would necessitate the treatment of the waste water by pH adjustment, causing additional problems with precipitated metal hydroxides.

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# THE BUREAU OF LAND MANAGEMENT ACID ROCK DRAINAGE POLICY AN EVOLUTION IN ENVIRONMENTAL PROTECTION[1]

R. David Williams [2]

**Abstract:** In the late 1980's the Bureau of Land Management came under scrutiny from the environmental community and our Nation's Congress for lax enforcement of operations conducted under the Nation's Mining Laws. This attention often focused on operations utilizing cyanide in the processing of gold ores. In response to this, the Bureau developed a Cyanide Management Policy and Cyanide Advisory Committee to provide guidance and technical assistance to field offices and advise management on related issues. The Bureau's Cyanide Management Policy was released in August 1990. Later the role of the Committee was expanded to include other technical issues related to hard-rock mineral development. During this time period it became apparent to the Bureau that a policy on acid rock drainage was as important as the cyanide policy. The Committee identified the need to develop a Bureau policy for mines with the potential to develop acid rock drainage. Several mines on Bureau managed public lands are extracting mineral deposits that have the potential for developing acid rock drainage.

The Bureau Acid Rock Drainage Policy has several major points:

1. All proposals for surface disturbance must be evaluated for acid rock drainage.
2. Mitigating measures addressing acid rock drainage must be applied.
3. Operations that propose active water treatment of acid rock drainage discharge require financial guarantees for water treatment facilities.
4. All portions of the mine facilities that may contain acid-generating materials are to be bonded for 100 percent of the cost to implement the approved reclamation measures.

**Additional Key Words:** acid rock drainage, mine wastes, Bureau of Land Management.

## Introduction

Major open pit mines can have severe impacts on the surrounding landscape and pose threats to air and water quality. For these reasons, strict requirements are placed on companies which propose large scale development of mineral resources.

In the United States, the prospective operator must apply for permits to construct or expand a mining operation. The permitting process is multifaceted and can involve numerous political jurisdictions and agencies. Clean air and water laws must be complied with as well as a host of local, State and Federal laws which protect other resource and natural values.

If the proposed operation is on land administered by the Federal Government, an Environmental Analysis (EA) or Environmental Impact Statement (EIS) must be prepared. Some states also require some form of Environmental Analysis for mining proposals regardless of land ownership.

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Federal lands are most likely under the jurisdiction of either the U.S. Forest Service (Department of Agriculture) or the Bureau of Land Management (Department of Interior).
Both of these agencies are responsible for managing Federal lands located mostly in the Western United States. For larger operations the requirements for an operating permit are similar for both agencies: The operator submits a plan of operations describing all aspects of theproposed mining operation for the agency's review. Following completion of the necessary environmental review, the agencies issue a formal approval. The agencies rely on inspection and enforcement programs to assure the conditions of the permit are complied with.

It is important to note the agencies ordinarily do not have the authority to reject a proposal submitted under the General Mining Law of 1872, which has served as the cornerstone of Federal Mineral Policy for 121 years. Both houses of congress passed revisions to the Mining Law in 1993, but it remains unclear what the ultimate mining law reform legislation will look like.

The General Mining Law of 1872, prior to its numerous amendments, was designed to encourage the settlement of the West and development of the Nation's mineral resources. As the West was settled and developed, the environmental effects of mining activity began to become issues. Some of these issues were surprisingly early on. Farmers in the Sacramento River Valley of California were among the first to look to the legal system for protection from the effects of unrestricted placer mining in the upper reaches of the Sacramento River drainage. A decision in a U.S. Circuit court in 1884 that acted to protect prime agricultural land essentially ended the era of unrestricted hydraulic placer mining in California. Other court decisions throughout the West generally established some minimum level of environmental protection, based primarily on impacts to other land owners and users downstream or downwind of mines or mineral processing facilities. In the 1960's and 1970's new Federal legislation on clean air and clean water began to impact mining development, and several State laws governing mines in the Western States also date to this period.

It was not until 1974, when the U.S. Forest Service developed the 36 CFR 228 Surface Management Regulations, that Federal regulations specific to operations on mining claims came about. These regulations were followed in late 1980 by the Bureau of Land Management's 43 CFR 3809 regulations. These regulations differed from the Forest Service regulations in one important aspect: operations involving less than 5 acres of surface disturbance do not normally need formal approval from the Bureau prior to beginning operations.

## Policy Development

The Bureau's surface management regulations coincided approximately with what has been called "the new American gold rush". Increased gold prices (due to the US Government's abandonment of fixed gold prices), new processing technologies using dilute cyanide solutions, and a better understanding of disseminated gold occurrences combined to create renewed interest in mineral properties throughout the West. Much of this initial activity was centered along the Carlin trend of north-central Nevada, but other properties in other Western states were not far behind.

Problems and perceived problems related to the use of cyanide quickly followed some of these early operations. Problems often centered on leaks of solution from the leach pads and solution ponds and on waterfowl mortalities as migrating waterfowl set down on process ponds containing cyanide solutions. These and other unrelated problems on mining claims, typically occupancy trespass and unreclaimed disturbances were presented as evidence in favor of reforming the 1872 mining law by its many critics. The criticism and support from reformers in the U.S. Congress led to a series of reports by both the General Accounting Office (GAO)(1986,1989,1990), the investigative branch of Congress and the Inspector General (IG)(1992), an investigative branch within the U.S. Department of the Interior. The reports were generally quite critical of the Bureau's efforts to enforce the surface management regulations. Internal Bureau reviews also revealed some inconsistencies in inspection and enforcement policies in the various Western States.

These problems forced the Bureau to revaluate its approach to permitting mines using cyanide. The Bureau developed a Cyanide Management Policy which was released in August 1990. This policy addressed (1) minimum acceptable design criteria, (2) coordination among State and Federal agencies, (3) bonds to ensure reclamation of disturbances, (4) mandatory reporting of wildlife mortality and cyanide discharges, (5) regular BLM inspections, (6) mandatory training for Bureau employees, and (7) procedures for closure and reclamation. This policy required State-specific cyanide management plans. In November 1990, the Bureau established a Cyanide Advisory Committee composed of Bureau staff familiar with technical aspects of cyanide recovery technology. Later the role of the Committee was expanded to provide guidance and technical assistance on other issues related to hard-rock mineral development.

As the early heap leach operations have begun to mine out oxidized portions of their ore bodies, operations may be proposed to continue mining into unoxidized sulfide bearing ore at depth. Many of the staff dealing with mine permitting within the Bureau realized that a more pervasive long term environmental threat than cyanide is posed by the potential for acid rock drainage (ARD) to develop from these types of sites. The prospect of several mining companies proposing operations with the potential to generate ARD spurred the Bureau's Cyanide Advisory Committee to develop a policy specific to the long-term threat posed by ARD. This policy was developed in early 1993 and following review by the members of the Cyanide Advisory Committee was released in draft form in late June 1993 (BLM 1993).

The Acid Rock Drainage Policy has several major points highlighted in table 1 and discussed below:

Table 1. Major points in the Bureau of Land Management acid rock drainage policy

| |
|---|
| • All proposals evaluated for acid rock drainage.<br>• All mining proposals require material characterization.<br>• Acid rock drainage must be mitigated.<br>• Long term treatment proposals require financial guarantees.<br>• Portions of the mine containing acid-generating materials bonded at 100% of the reclamation costs. |

-All proposals for surface disturbance must be evaluated for acid rock drainage during review by the Bureau.
-All mining proposals must include waste characterization and appropriate waste handling considerations. These must based on the physical and geochemical properties of the waste and the site specific environmental conditions.
-Mitigating measures must be applied to prevent unnecessary or undue degradation by acid rock drainage.
-Proposed operations which include reclamation measures relying on long term water treatment are not encouraged and will be closely reviewed.
-Proposed operations which include reclamation measures relying on active water treatment of ARD discharges in perpetuity will require bonding or financial guarantee mechanisms sufficient to construct, operate and maintain treatment facilities.
-All portions of the mine facilities that may contain acid-generating materials are to be bonded for 100 percent of the cost to implement the approved reclamation measures.

The Bureau policy makes it clear that waste characterization must be fully integrated with operational procedures throughout the mine life, perhaps even beginning with initial characterization work during the advanced exploration stage. Waste characterization is required for all operations. Assertions by the operator not supported by technical data that the materials are not acid generating or annual precipitation is too low for ARD development will not be adequate. The policy discusses what types of preliminary waste characterization work might be appropriate as well as more advanced testing. More advanced predictive work might include static, kinetic, or some combination of both types of tests.

When initial characterization reveals the potential for an ARD problem, the plan of operations or notice must include measures that prevent or control ARD. The Bureau policy encourages applied research, including the use of test plots at individual mine sites to field test reclamation measures prior to mine closure. The treatment of ARD effluent is almost always a last resort, and the policy makes it clear that long term proposals to treat ARD effluent will be closely reviewed, and that any proposal for active treatment of ARD effluent in perpetuity will require bonding or financial guarantee mechanisms sufficient to construct, operate and maintain treatment facilities.

Other important features of the policy include-

-Use of the Environmental Review process to evaluate the impacts of proposed activities. The analysis will address all the expected impacts anticipated during all phases of the proposed operations.

-Inspections of operations with the potential for the development of ARD will be made no fewer than four times a year or as identified in a site specific plan. Inspections should be scheduled when any potential ARD effluent is most likely, e.g. after major precipitation events or during spring runoff.

-Training is required for BLM personnel responsible for inspecting operations and should include safety, mining and milling processes, and ARD evaluation, monitoring, and mitigation.

-Operator monitoring plans for ground and surface waters throughout the mine life are required for all mine facilities with the potential to produce ARD. The monitoring results are to be reported to the Bureau, which will coordinate with State agencies to assure environmental protection.

-Quality assurance of engineering designs for proposed facilities must be included in plans of operations. The Bureau may require independent verification by a registered professional engineer that facilities are constructed according to the approved design plan and specifications.

-Interagency coordination with other State and Federal agencies that may have responsibility in permitting mines is important in order to avoid duplication and increase efficiency in the regulation of permitted operations.

A 1991 GAO Report, 'Increased Attention Being Given to Cyanide Operations' (GAO/RCED-91-145) concluded, "The cyanide management policy that BLM has adopted appears to be an appropriate response to the potential hazards that cyanide operations present to wildlife and the environment."

## Conclusion

By early adoption and refinement of the Acid Rock Drainage Policy the Bureau can hopefully avoid a lengthy and divisive series of reports such as those resulting from the difficulties with early cyanide operations. The Acid Rock Drainage Policy may be even more crucial than the Cyanide Management Policy, Considering the long- term environmental problems that can be associated with acid rock drainage.

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# THE ENVIRONMENTAL MANAGEMENT PROGRAMME ROUTE TO INTEGRATED ENVIRONMENTAL MANAGEMENT: A SOUTH AFRICAN MINING INDUSTRY PERSPECTIVE[1]

Johan C. Greeff[2]

**Abstract:** The Minerals Act, No. 50 of 1991, stipulated that all prospecting or mining operations submit rehabilitation programs for approval. Members of the Chamber of Mines of South Africa concluded that this requirement concentrates more on restoring the surface than on the management of environmental impacts. This is especially important as the major opencast coal mines are located in the Eastern Transvaal Highveld, which contains some of South Africa's high potential agricultural land. Industry's view was that it had to implement integrated environmental management to resolve this potential conflict. The concept of an environmental management program report (EMPR) was consequently developed. The report includes an environmental impact assessment and an environmental management program. The EMPR concept has been approved by the four regulatory authorities concerned. Joint consultation between the various stakeholders gave each ownership of the process, fostered understanding of needs, and clarified views on sometimes differing requirements. Implementation of EMPR's on mines was facilitated by the compilation of an Aide-Memoire for the preparation of environmental management program reports for prospecting and mining. Work involved in the preparation of the Aide-Memoire became a joint venture between industry, the regulatory authorities, and interested and affected parties. Development to fruition of the EMPR concept in South Africa places the mining industry as an equal partner in a tripartite government-interested affected parties - proponent scenario.

## Introduction

South Africa is a major producer of gold, coal, platinum and manganese. The mines, both working and defunct, have significant pollution potential. The need for effective environmental management to control this potential gained momentum during the seventies and eighties, so much so that the nineties could be described as the "Decade of the Environment".

The mining industry in South Africa is a prime mover in initiating effective environmental management. While the approach used for the development of the management of impacts applies to all mining operations, it was the advent of major opencast coal mines that spurred activity.

## The Chamber of Mines

The Chamber of Mines of South Africa is a voluntary association of mining companies and mining finance houses, or groups, as they are known in industry, which produce over 85% of South Africa's minerals. The Chamber's prime objective is to protect, advance and promote the interests of its members, but not to fund and control their activities. Chamber coal mines employ 31,675; Chamber gold mines employ 467,273. Because so many mines are members, the Chamber, in effect, serves almost the whole mining industry, and in many cases is regarded as its spokesperson.

The broad mining perspective of the Chamber includes collection and dissemination of accurate information to *inspire* investor confidence, addressing authorities and government of the day, opposing and

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proposing new laws, encouraging legislative revision when industry's interests are affected and supporting and coordinating joint mining industry operational functions. Primary input to initiate this joint action is provided by a number of specialist technical committees on which delegates from groups serve. One of these specialist technical committees is the Environmental Management Committee, which deals with all environmental matters. Their recommendations on policy issues are submitted to the Chamber's Executive Committee.

## The Mines and Works Act: Scratching the Surface

Major opencast or strip mining operations in South Africa commenced in the mid 1970's. The impetus for this mining method was twofold--

- To provide reasonable-cost coal for use by a new generation of power utilities, supplying 3,600 MW into the national grid.
- To assist South Africa to become a major coal exporter.

These mines were designed to produce 8 - 10 million mt per year from depths of up to 60 m. Legislation of the time directed at rehabilitation and minimization of environmental impacts was woefully inadequate. The Mines and Works Act contained general provisions which, among other things--

- Required broken ground that had finally subsided to be filled in.
- Prohibited the release of water containing injurious matter.
- Stipulated that dumps be covered with soil or sludge or otherwise dealt with to prevent the dissemination of any form of pollution.

The majority of these strip mines are located in the Eastern Highveld of the Transvall. The mineralization covers an area of 4,770,000 ha, of which 409,500 (8.6%) are expected to be affected by high-recovery coal mining in the long term. On the whole, the landscape is gently undulating without prominent hills. The central and eastern parts are characterized by the presence of pans and lakes. The area is classified as a summer rainfall area with an average annual rainfall that increases from 700 mm in the west to 800 mm in the east. In comparison with the general situation in South Africa, the area is rich in water and has a high agricultural potential. Maize, which is a basic food throughout southern Africa, is one of the major crops in this area.

The differing land use requirements posed by mining and agriculture had the potential for conflict, and the Chamber of Mines accordingly commissioned or developed sets of guidelines to provide for rehabilitation of these strip mines:

- Volume 1/1983 - The Design, Operation and Closure of Metalliferous and Coal Residue Deposits.
- Volume 2/1979 - The Vegetation of Residue Deposits against Wind and Water Erosion.
- Volume 3/1981 - The Rehabilitation of Land Disturbed by Surface Coal Mining in South Africa.
- Volume 6/1985 - Pollution Problems and Hydrological Disturbances Resulting from Increased Underground Extraction of Coal.
- Volumes 4 and 5 have been discontinued.

An additional motivation was, in the absence of suitable legislation, to discharge the operator's responsibility toward the environment. Development of the guidelines was coordinated by the Environmental Management Committee. These guidelines were subsequently accepted by the Government Mining Engineer as a standard against which all rehabilitation was evaluated, and were promulgated as amendments to the Mines and Works Act in 1980.

The main directives in these amendments were aimed at the submission of a layout plan and rehabilitation program and restoration of the surface. The amendments applied to opencast mines only.

Approval of the layout plan and the rehabilitation program was not a prerequisite to either commencing or continuing mining operations. The manager's obligation was thus to a certain extent discharged by the mere submission of the layout plan and rehabilitation program. However, the statutory route, originally directed at restoration of the surface rather than the management of environmental impacts, increasingly veered to encompass the latter in preparation of submissions. The legislation was also deficient in that it did not make provision for statutory consultation with other involved government departments (Water Affairs and Forestry, Environmental Affairs and Agriculture) in the approval process.

A decade of experience was amassed as environmental managers learned to cope with the challenges and demands posed by the restoration of ground disturbed by opencast mining. Expansion of experience was not limited to coal mines. More than a century of gold mining operations had left some 7,500 hectares of residues, about which environmental managers had to expand their knowledge base. Crucial to this was the role of the Chamber in optimizing technical input and decisions, and commissioning research, reports and guidelines as well as disseminating information and formulating industry policy.

## The Minerals Act: Turning a New Leaf

The Minerals Act, No. 50 of 1991, replaced the Mines and Works Act. One of the objectives of this Act is: "...to regulate the orderly utilization and the rehabilitation of the surface of land during and after prospecting and mining operations...".

Statutory directives to attain this include--

- Prior approval of the layout plan and rehabilitation program.
- Consultation with other Departments before approval.
- Extension to include both opencast and underground mining operations.

Chamber members, on assessing the requirements, concluded that--

1. The rehabilitation plan concentrates more on repairing damage than on holistic environmental management. There was thus a need to provide for statutory environmental management. Furthermore, there was an increasing amount of pressure on industry to demonstrate its environmental performance.
2. The principles of integrated environmental management were already accepted by the mining industry.
3. The industry recognized that, under the original procedures, all mines would be subject to an investigation requiring an environmental impact assessment and, therefore, there was no need for a screening procedure.
4. The commonly accepted ideas of an environmental impact assessment (EIA) contained no mention of an environmental management plan during the implementation stage. This was only introduced as a condition of approval after the assessment stage.
5. Potentially at least four environmental documents were needed for a mining project - a rehabilitation plan (by the Department of Mineral and Energy Affairs), an EIA as well as an air pollution control plan (the Department of Environment Affairs), and a water pollution control plan (the Department of Water Affairs and Forestry). In the interests of employee effort as well as time and money, the industry wanted only one document.

6. A lead agent to mining projects, the Department of Mineral and Energy Affairs, was clearly identified in the Minerals Act. The act stated unequivocally that while the other Departments had to be consulted in the approval process, Mineral and Energy Affairs was the approval authority.

## Amendment of Regulations: Another Branch

Simultaneously with the promulgation of the Minerals Act, the Government Mining Engineer submitted to the Chamber for comment proposals for amendment of environmental regulations for promulgation in terms of the Minerals Act. Evaluation of the environmental requirements determined in the Minerals Act and of the proposed amendments to the regulations necessitated a review of the environmental policy thrust by Chamber members. This led to a decision to recommend that the Minerals Act be further amended to embrace the concept of integrated environmental management, rather than to concentrate on rehabilitation concepts only.

## Integrated Environmental Management

Integrated Environmental Management (IEM) is designed to ensure that the environmental consequences of development proposals are understood and adequately considered in the planning process. The concept was proposed to the Minister of Environment Affairs by the Council for the Environment during 1989. The view of the Department of Environment Affairs is that the IEM procedure should be implemented in such a way that it complements, rather than duplicates, existing planning and other procedures. Where appropriate, IEM should be used to supplement existing requirements, rather than to replace them. The basic principles underpinning IEM are that there be--

- Informed decision making.
- Accountability for information on which decisions are taken.
- Accountability for decision taken.
- An open, participatory approach in the planning of proposals.
- Consultation with interested and affected parties.
- Due consideration of alternative options.
- An attempt to mitigate negative impacts and enhance positive aspects of proposals.
- An attempt to ensure that the social costs of development proposals be outweighed by the social benefits.
- Democratic regard for individual rights and obligations.
- Compliance with these principles during all stages of the planning, implementation and decommissioning of operations.
- The opportunity for public and specialists to have input in the decision-making process.

Central to the concept of IEM is that its underlying principles should direct the planning of proposals, rather than being considerations to be addressed once a proposal has been planned.

## The Environmental Management Programme Report

The principles of integrated environmental management were thus recommended for implementation in the mining industry through an Environmental Management Programme Report (EMPR). The EMPR is a document with the following overall objectives--

1. To meet the environmental requirements and directives as contained in the Minerals Act, No. 50 of 1991, and its regulations.
2. To provide a single document that will satisfy the various authorities concerned with the regulation of the environmental impact of mining.

3. To give reasons for the need for and for the overall benefits of the proposed project.
4. To describe the relevant base line in environmental conditions at and around the proposed site.
5. To describe briefly the prospecting or mining method and associated activities so that an assessment can be made of the significant impacts that the project is likely to have on the environment during and after mining.
6. To describe how the negative environmental impact will be managed and how the positive impacts will be maximized.
7. To set out the environmental management criteria that will be used during the life of the project so that the stated and agreed land capability and closure objectives can be achieved and a closure certificate can be issued.
8. To indicate that resources will be made available to implement the environmental management program set out in part 6 of the document.

The EMPR document is not intended to be an exhaustive description of the project. Rather, it is a document containing sufficient information to make the reader aware of the overall character of the site and the surroundings, the mining method, the likely impacts and how these are to be managed. The document should be simple, yet as comprehensive as possible, in order to accommodate a prospecting or mining operation of any size or complexity; it should be flexible so that the environmental management program can be tailored to the site-specific mine and environmental conditions; and finally it should be adaptable, so that with judicious pruning it can be used for the smallest and simplest prospecting or mining operation.

An EMPR is prepared on the strength of the facts pertaining at the time of planning the operation. It must, however, be seen as a dynamic document that may require updating during the life of the project. The preparation of the EMPR should not be mechanical. It is important that the information needs are fully understood.

Involvement in the process places a great responsibility on the proponent to commit himself to the effective implementation and management of the EMP until closure of the mine is granted. Implementation and management of the program in effect comprise an environmental contract between the proponent and the approval authority. It is thus crucial that the proponent fully understand the consequences and risks involved. The proponent has to ensure that--

- The EMP is an integral element of the overall management plan of the mine.
- The information needs required under management strategies proposed are fully developed.
- The development and/or compilation of the EMPR and especially the EMP does not constitute a paper exercise.

## The Aide-Memoire

Industry recommended, and the other interested and affected parties accepted, the compilation of a Aide-Memoire to assist the preparation of the EMPR. A Steering Committee (representative of the Chamber of Mines of South Africa, the Aggregate and Sand Producers' Association of South Africa, the South African Agricultural Union, and the Departments of Water Affairs and Forestry, of Agriculture, of Environment Affairs, of National Health and Population Development, and of Finance), chaired by a representative of the Government Mining Engineer's Office, was formed to collaborate the drafting of the Aide-Memoire. The final version was agreed to on August 25, 1992 and was formally handed to the industry by the Director-General, Department of Mineral and Energy Affairs, on January 7,1993. It must be borne in mind that the Aide-Memoire is only a guideline for the preparation of an EMPR. The Aide-Memoire attempts to reflect the information needs of the various authorities as fully as possible at present. However, the aspects to be considered, as outlined in that document, are open to a wide range of interpretation, and the requirements of

the authorities are not as clear as they should be. For example, in the process of developing an EMPR, it is of specific interest to the Department of Water Affairs and Forestry to --

- Demonstrate that the "best practice" based on the best available technology not entailing excessive cost (BATNEEC) will be implemented by the mine irrespective of whether impacts on the water environment are expected or not.
- Demonstrate that the impact of the mine's intended actions on the water resource is understood and that satisfactory mitigation measures have been proposed in order to meet the required water quality objectives.
- Demonstrate that the above two aspects will be effected during the full life cycle of the mining venture.

In the evaluation process, the Department of Water Affairs and Forestry will essentially establish whether these requirements have been fulfilled. Notwithstanding the level of detail of the EMPR, approval will not be granted if the mentioned requirements have not been met. Thus, it is of utmost importance for the proponent to establish the information needs and evaluation criteria of the various authorities and to approach the development of the EMPR from that perspective. The EMPR process is the first in the country that gives expression to how IEM principles can be applied in practice. There are differences though--

- It was agreed during the EMPR negotiations that all mines will follow the "impact assessment" route; therefore, for mines, there is no need for the "classification of proposal" stage, the list of activities, the list of environments, or the summary list of environmental characteristics.
- The EMPR process does not include a "record of decision". Protagonists of the EWMPR have indicated that the record of decision is a matter for the authorities.
- The onus is on the proponent of the EMPR to prepare it, rather than on consultants, as intimated throughout the IEM process. This was done deliberately to ensure that the proponent has full ownership of the proposal, including its environmental management program, since it is the proponent, not the consultant, who must face the consequences if something goes wrong. It is also only the proponent who understands the full impact that various environmental commitments will have on his business operations. However, the proponent is free to use consultants to assist in compiling the EMPR, and frequently does so.

The Minerals Act has been amended to provide for inclusion of the EMPR as recommended and agreed to rather than the layout plan and rehabilitation program originally promulgated.

## Conclusion

The mining industry has grasped the significance of using IEM principles in achieving very essential developments in South Africa without unnecessarily impacting on the environment. In the process, it has demonstrated its willingness to apply self-regulation. Every mine in South Africa is currently preparing, or has already prepared, an EMPR. The result has been a quantum leap in focusing employee effort, time, and money on environmental issues at the mines. It is imperative that the process be allowed to come to maturity before being amended in any significant manner.

# REVIEW OF CANADIAN LEGISLATION RELEVANT TO DECOMMISSIONING ACID MINE DRAINAGE SITES[1]

**Ingrid Reichenbach**[2]

**Abstract:** Regulatory strategies specifically geared towards decommissioning sites with acid mine drainage are in the development stage. In most jurisdictions, there are provisions for proposed mines to complete an environmental impact statement. The complexity of the statement varies depending on jurisdiction, discretion granted the Minister and size of the proposed mine. Several provinces have guidelines in place that require predictive testing to determine the acid generation potential of rock units. Nova Scotia has gone a step further and requires monitoring, and sometimes predictive testing, at any development on pyrite-rich shale of the Halifax Formation. In Canada, non-enforceable water quality guidelines are the norm. Site-specific enforceable parameters, based on water quality guidelines may be incorporated in mine operation permits. Provinces use the federal guidelines or develop their own approach and criteria. For example, with time, surface water bodies in Manitoba will be classified according to present and potential use and water quality criteria set accordingly. British Columbia sets water quality objectives on a site-specific basis, taking into account background water quality and water uses. This approach potentially has the flexibility to address the anomalous background water chemistry sometimes encountered at mine sites. With respect to effluent limitations, most provinces use the federal regulations, although some provinces have developed their own.

## Introduction

The Mine Environment Neutral Drainage (MEND) program sponsored this review of Canadian legislation relevant to decommissioning acid mine drainage (AMD) sites. The MEND program is a co-operative program financed and administered by the Canadian mining industry, the Canadian government, and the governments of British Columbia, Saskatchewan, Manitoba, Ontario, Quebec and New Brunswick. The purpose of the review is to provide government agencies, industry representatives and other organizations and individuals with an overview of the existing Canadian regulatory framework relevant to decommissioning AMD sites, and to summarize the criteria used by Canadian regulatory agencies to assess the performance of decommissioning an AMD site. This paper is a summary document based on MEND Report 4.2.1; Review of Canadian and United States Legislation Relevant to Decommissioning Acid Mine Drainage Sites, 1993.

## Canadian Federal Legislation

Federal jurisdiction over decommissioning tends to be more restricted than provincial jurisdiction. The Constitution Act of 1867 grants specific federal power over coastal and inland fisheries, navigable interprovincial rivers and migratory birds. However, recent federal environmental legislation such as the Canadian Environment Protection Act and the Canadian Environmental Assessment Act represents an increasing federal government role in the environmental regulatory field previously dominated by the provinces. Federal legislation and guidelines relevant to decommissioning AMD sites are discussed in the following paragraphs.

The intent of the Fisheries Act of 1985 is to protect fish, marine mammals, and their habitats. This responsibility is shared between the Ministers of Environment and Fisheries and Oceans. The Act prohibits the deposit of a "deleterious substance" directly into water frequented by fish or in a place where the substance may enter such water. Therefore, this Act would apply to most mining operations that discharge effluent into rivers,

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lakes or the ocean. The Act also prohibits any work that results in the harmful alteration or destruction of fish habitat. The Metal Mining Liquid Effluent Regulations and Guidelines (MMLER) of 1977, under authority of this Act, contain enforceable effluent regulations for new, expanded and reopened mines and non-enforceable effluent guidelines for existing mines (Table 1). The guidelines have the same numerical values as the regulations. The guidelines provide flexibility for mine operators and the Minister to negotiate a compliance schedule. Limits are "specified for those parameters which are known to occur commonly in base metal mining effluent in sufficient amounts to be deleterious to fish and for which demonstrated practicable technology exists to reduce these substances to low levels". The Arctic Waters Pollution Prevention Act regulates effluent disposal in arctic waters.

Table 1. Metal Mining Liquid Effluent Regulations (Environment Canada, 1977).

| Parameters[1] | Maximum Monthly Arithmetic Mean Value | Maximum Value in a Composite Sample | Maximum Value in a Grab Sample |
|---|---|---|---|
| pH (unitless) | 6.0 | 5.5 | 5.0 |
| Total Suspended Matter | 25.0 | 37.5 | 50.0 |
| Arsenic | 0.5 | 0.75 | 1.0 |
| Copper | 0.3 | 0.45 | 0.6 |
| Lead | 0.2 | 0.3 | 0.1 |
| Nickel | 0.5 | 0.75 | 1.0 |
| Zinc | 0.5 | 0.75 | 1.0 |
| Radium 226 (pCi/L) | 10.0 | 20.0 | 30.0 |

[1]The concentrations are given as total values with the exception of Radium 226 which is a dissolved value after filtration of the sample through a 3 micron filter. All units are mg/$\ell$ unless otherwise noted.

The Canadian Environmental Protection Act (CEPA) of 1988 is administered by the Department of Environment. The Minister of Health provides advice on human health aspects. CEPA is a prevention-oriented statute which outlines a comprehensive scheme for the control and regulation of toxic chemicals. Heavy metals found in AMD could potentially be regulated under CEPA. However, it is more probable that AMD will continue to be regulated under the Fisheries Act which has adequate legal authority. One aim of CEPA is to establish nationally consistent levels of environmental quality. CEPA provides that federal regulations with respect to toxic substances will not apply in provinces where equivalent provisions are already in force.

The Canadian Environmental Assessment Act of 1992 administered by Environment Canada, ensures that the environmental consequences of all federal projects and activities are assessed before final decisions are made. An Environmental Assessment (EA) is triggered when the Federal Government: is the proponent of a project, makes federal funds available for a project, makes federal lands available for a project, or evokes a statutory or regulatory provision that allows a project to proceed. When a federal authority is required to conduct an EA, screening is one of several procedural options to follow. Projects requiring a more intensive level of assessment will undergo a comprehensive study. The types of projects likely to require a comprehensive study based on their likelihood to cause significant adverse environmental effects are on the draft Comprehensive Study List of 1991. This list may be established by regulation at a later date by authority of the Act. Of interest to the issue of AMD is the inclusion on the list of "a metal mine with a mill capacity greater than 10,000 tonnes/day". This will cover most major ferrous metal and major new non-ferrous metal mines. The comprehensive study will be required to show that no significant adverse effects are expected, or that such effects can be mitigated, for the project to proceed. The question of how to handle small-scale, potentially high-impact mines (e.g. gold mines), and coal and industrial mineral mines, remains unresolved.

The Canadian Council of Ministers of the Environment (CCME) initiated the National Contaminated Sites Remediation Program for remediation of high priority contaminated sites in Canada. To promote consistency in the assessment and remediation of these sites, CCME adopted the Interim Canadian Environmental Quality Criteria for Contaminated Sites in 1991 from existing guidelines and criteria currently in use in various jurisdictions across Canada. Many of the criteria do not have a complete set of supporting rationale and, therefore, are considered "interim". Criteria are assessed and modified as required to reflect current knowledge of the environmental and human health effects of contaminants. The criteria do not constitute values for uniform environmental quality at all contaminated sites, and their use is site-specific. Although they are non-enforceable, they may serve as a basis for site-specific enforceable parameters incorporated in mine operation permits or licenses.

## Territorial Legislation

The federal government owns and controls most of the lands and resources in the territories and, therefore, most mining activities require federal permits and licenses. The territorial governments have the authority to enact legislation; however, territorial legislation is subject to federal legislation and may be limited in scope where there is a contradiction. Territorial environmental legislation commonly mirrors federal legislation and may be based on federal environmental controls. The Territorial Lands Act (TLA), administered by the Department of Indian Affairs and Northern Development (DIAND), addresses concerns with respect to the abandonment and decommissioning of mines in the territories. The TLA is administered under two sets of regulations governing surface use: the Territorial Land Use Regulations (TLUR) and the Territorial Land Regulations (TLR). Typically, a prospective mining company will be issued a permit under the TLUR and will register a mining claim under the Canada Mining Regulations. The latter regulations grant authority to the Minister of DIAND to order the owner of a mining operation that is discharging a "harmful" substance to treat the substance prior to discharge, limit the discharge, or cease the operations causing the discharge. The relevant control mechanisms for abandonment and decommissioning of mines are the TLR and the leases issued pursuant to these regulations.

The federal Northern Inland Waters Act has recently been replaced by the Northwest Territories (NWT) Waters Act and the Yukon Waters Act, both promulgated in 1993. The NWT Waters Act, administered by the NWT Water Board and DIAND, regulates the use of water and the disposal of waste into waters of the NWT. A mining company that wishes to operate a mine must first obtain a water license by satisfying the following conditions: any waste produced by the mining operation will be treated and disposed of in the appropriate manner under prescribed effluent standards, and the financial responsibility of the applicant is adequate for completion of the undertaking as well as any required mitigation, maintenance or restoration. If present restoration technology will not satisfactorily prevent long term degradation of the environment, a water license may not be issued. The Water Board may require an applicant to furnish and maintain a security with the Minister to compensate users who are entitled to compensation or to recover the Crown's costs incurred in counteracting, mitigating or remedying adverse effects. Industrial licences generally contain conditions requiring abandonment and restoration plans and their implementation. If the approved abandonment and restoration plan is not satisfactorily implemented, the security deposit or a portion of it may be withheld by the Water Board. In 1990, the NWT Water Board published: Guidelines for Abandonment and Restoration Planning for Mines in the NWT. The guidelines include a discussion of design features (e.g. tailings and containment structures, cover treatments, waste rock piles), plan development (e.g.abandonment scenarios) and monitoring. The maximum acceptable concentrations for selected substances in discharges from abandoned tailings areas are from the MMLER, but lower values may be considered for environmentally sensitive areas.

The Water Resources Division of Northern Affairs administers and enforces the Yukon Waters Act, and is responsible for licensing. The Water Board processes and issues the licenses. The Act is similar to the NWT Waters Act. A mining company needs a license to operate. Conditions included in the license include provisions for monitoring, collection and treatment and closure. The Minister has the authority to take the necessary action to mitigate or remedy adverse effects caused by the operation and a security deposit to cover this eventuality is incorporated into the license.

## Northwest Territories

The mineral industry in the NWT accounts for a major portion of the territorial economy. Almost one-quarter of Canada's lead and zinc production originates in the NWT, which also produces gold, cadmium and silver. There is no legislation in the NWT directly affecting prospecting, exploration and mining. Environmental legislation includes the territorial Environmental Protection Act and the Environmental Rights Act, both administered by the NWT Department of Renewable Resources. The former provides general protection from contamination by hazardous substances. For example, it prohibits the discharge of contaminants to the environment and holds the party responsible for a spill liable for clean-up and damages. The latter is primarily concerned with the release of contaminants and gives residents of the NWT the right to take certain actions to protect the environment. The NWT Department of Energy, Mines and Petroleum Resources has prepared and keeps updated A Guide to Legislation Affecting Exploration and Mining in the NWT.

## Yukon

The Yukon mining industry comprises mineral exploration, hardrock mining and placer mining. Zinc, lead and gold are the most important minerals. Mining in the Yukon is regulated under the Yukon Quartz Mining Act of 1924. The Act does not have any specific regulations governing land use activities on mining claims. An amended Act and regulations have been proposed by the Yukon Mining Advisory Committee, which includes representatives from government and industry. It is expected to encompass exploration, development, production, mine closure, financial security and penalties. A production license will be required for any producing mine. The license application would trigger review under the Canadian Environmental Assessment Act.

# Provincial Legislation

Most of the legislative power relating to decommissioning AMD sites falls within provincial jurisdiction because the provinces exercise proprietary rights over vast areas of Crown land and minerals; have legislative authority for the development, conservation and management of mineral resources; and have powers over property and civil rights that provide a basis for the regulation of land use and mining activities, including reclamation work. Provincial jurisdiction in these areas is expressed in provincial legislation related to environmental assessment and licensing, land-use planning, water rights and mineral tenure and mining, all which directly affect decommissioning activities. Provincial legislation and guidelines relevant to decommissioning AMD sites are discussed in the following sections.

## British Columbia

Mineral products constitute over one-fifth of the province's total exports. Coal and copper are British Columbia's most important minerals, followed by zinc, gold and silver. The Mine Development Assessment Act of 1990 provides the legal framework for comprehensive environmental assessments of proposed mine developments. It is intended to dovetail with the Canadian Environmental Assessment Act. Mine developments subject to the Act are termed "reviewable" and include new coal or mineral mines capable of producing 10,000 tonnes per year, or other mines designated by the Chief Inspector of Mines where potential impacts warrant an integrated review. An application for a mine development certificate must contain an environmental protection plan approved by the Minister of Energy, Mines and Petroleum Resources with the concurrence of the Minister of the Environment. A certificate must be obtained before a reviewable mine development can be constructed. As part of the approval process, the government requires testing to identify potentially acid producing materials and assurance that all sources of potential acid generation have been identified and that prevention and control measures have been incorporated into the plan, if appropriate. The Ministers may refer the application to an assessment panel or accept, modify or reject an application. The assessment panel may recommend amendments to the certificate, which can overrule existing permits, even if those permits were obtained in compliance with applicable legislation.

The Mines Act of 1989 governs all mining activities including exploratory drilling, excavation, processing, concentrating, waste disposal and site reclamation. Before commencing work on a mine, proponents are required to outline the proposed work plan and a program for the protection and reclamation of the land and watercourses affected by a mine, and to obtain a Reclamation Permit. Disturbed land and water resources must be reclaimed to a level of productivity not less than that which existed previously, and water released from the minesite must meet long term water quality standards. A reclamation security deposit may be required. The inspectors appointed under the Act may do work around a closed or abandoned mine to abate pollution and a charge may be registered against the mineral title.

In 1992, the Mines Act also established the Health, Safety and Reclamation Code for Mines in British Columbia. The code is subject to an annual review. Compliance with the code includes worker health and safety, mechanical/electrical and reclamation requirements. Proposed mines must submit reclamation plans if a surface disturbance is likely to result. The Chief Inspector of Mines decides when exploration work is likely to cause a significant surface disturbance and can exercise broad discretion with respect to requiring and approving reclamation programs. Reclamation plans should include: 1) location and extent of the mine, 2) present land uses, 3) nature of the mine, with special reference to "prediction of acid generation for all strata and deposits, including static and, if necessary, kinetic tests", 4) reclamation plan for construction and operational phases of mining operation, and 5) final reclamation plan. Mine reclamation standards are also specified in the code and include returning the land and watercourses to a productive land use, ensuring that impoundment structures and waste rock dumps are stable over the long-term, and ensuring that water quality released from a mine site is of an acceptable standard. With respect to acid generating material, the mine reclamation standards specify that "all potential acid generating material shall be placed in a manner which minimizes the production and release of AMD to a level that assures protection of environmental quality".

The Pollution Control Objectives for the Mining, Smelting and Related Industries, developed by BC Environment in 1979 for discharges to air, water and land, provide for a wide range of discharge concentrations depending on the needs of particular receiving environments. The more stringent values apply to sensitive environmental situations, the less stringent where it can be shown that unacceptably deleterious changes will not follow. The objectives are intended to apply province-wide, but in special circumstances the Director of Pollution Control may apply more or less stringent requirements.

British Columbia does not have any province-wide water quality objectives. BC Environment develops objectives on a site-specific basis using scientific guidelines or criteria. For example, if an area has very high copper, objectives for copper would be set for this area. Each objective and criteria is published in a separate report. There are presently reports on 33 objectives and 15 criteria. Objectives may serve as a guide for mine permitting and may help to assess the Ministry's performance in protecting water uses. Neither criteria nor objectives are based on any legislation, so they are non-enforceable. However, these guidelines are used in the review of a mine development certificate and failure to comply may result in significant delays. Furthermore, each mine site has site specific parameter concentrations specified in their waste management permits.

The Victoria-based Reclamation Advisory Committee has developed a series of working policies and technical initiatives to deal with AMD. These have recently been released as an "Interim Policy" for public comment. The interim policy reflects the committee's current philosophy of preventing AMD generation through prediction and design, wherever possible avoiding long-term treatment. The policy contains sections dealing with prediction, prevention, collection and treatment, reclamation permitting, bonding, monitoring, historic sites, existing mines, commercial leaching and exploration. Many of these issues are discussed in detail in: Draft Acid Rock Drainage Technical Guide, developed by BC AMD Task Force in 1989, that provides guidance and recommendations in the application of state-of-the-art technology in prediction, control and monitoring of AMD.

Of the 16 metal mines currently operating in BC, six are presently producing AMD and several more have the potential to do so. These six mines are currently collecting and treating all acidic drainage. Of the eight

operating coal mines, none are presently generating AMD. Although one coal mine has acid generating potential, it has been designed to prevent AMD. For proposed new mines, Ministry of Energy, Mines and Petroleum Resources policy is to approve only those mines that develop plans to prevent or control AMD. Existing mines with AMD that commenced operation prior to present legislation are required to submit detailed closure plans. For example, the Equity Silver Mine in the central interior of British Columbia has scheduled closure and developed detailed closure plans. At this time, the water quality objectives are not yet available. British Columbia also has at least six historic mine sites with AMD and new policies pertaining to them are currently under development. One such site is the Mt. Washington Mine on Vancouver Island, which is undergoing reclamation. However, water quality objectives are in the early draft stage and not yet available.

## Manitoba

Next to agriculture, mining is the leading primary resource industry in Manitoba. Nickel, copper and zinc are the major metallic minerals produced. There are some serious AMD sites in Manitoba, but not many compared to other provinces participating in MEND. There are no readily available case studies of AMD sites that have been decommissioned. Manitoba passed a new Mines and Minerals Act in 1991, which contains two sections relevant to decommissioning AMD sites: (1) new and existing mining operations must file closure plans that address the "protection of the environment during the life of the project" and rehabilitation of the site upon mine closure; and (2) mining operations must put up security for performance of rehabilitation work. However, the practical details of the closure and rehabilitation requirements are still to be worked out and developed into regulations.

Prior to commencing operation of a mine, an environmental impact statement must be completed to comply with the Environment Act. In Manitoba, the Clean Environment Commission (CEC) holds hearings and gathers evidence from any "concerned individuals" including the public, the government Environmental Department, and the proponent. Following this, the CEC issues a report with recommendations, which the Director of the Environment Department uses in setting terms and conditions of an Environmental License. The Director is not bound by the CEC recommendations, but must indicate why any such recommendations are rejected. Any issued License is subject to an appeal process. The mine license includes limitations on effluent. Federal effluent limitations are sometimes used, but in general, criteria are set on a site-specific basis using the following approach.

Surface waters in Manitoba are classified from 1 to 6. Manitoba Environment assesses the potential impact area of the proposed mine and, if the surface waters in the vicinity have not been previously classified, makes an assumption as to water classification. As can be expected, the class of surface waters for the entire province has not been determined yet. Public hearings are required to set the classification, therefore, during mine licensing only an initial assumption is made. Manitoba Environment then selects the most stringent surface water quality objectives applicable, based on the assumed classification, and sets effluent limitations such that the loading to the surface water can be accommodated without exceeding the water quality objectives. At times, this approach results in criteria more stringent than the federal mine effluent limitations. Details on surface water classification and water quality objectives are contained in the following three Department of Environment documents: Surface Water Quality Objectives, The Development and Use of Water Quality Objectives and Watershed Classifications. Groundwater standards are still in the development stage.

## New Brunswick

Mining is New Brunswick's second-largest industry. The most valuable minerals are zinc, potash, lead, silver, coal, antimony and peat, with zinc accounting for over 50 percent of the total value of production. AMD is an overwhelming problem in New Brunswick, but very few (if any) AMD sites have been decommissioned at this time. The Mining Act of 1985 provides the authority for the imposition of reclamation requirements on mining operations. As part of the mine approval process, an application must be made to the Minister of Natural Resources and Energy. Requirements for closure and reclamation are regulated under the General Regulation of

1986, which specifies that a reclamation program for the protection, reclamation and rehabilitation of the environment be submitted as part of the application for a mining lease. In addition, mining lease applicants are required to provide security for the costs of reclamation work conducted both during and after mining operations. A mining lease will not be granted until the Minister has approved the reclamation program.

The approvals process for a new mine may commence with an environmental impact assessment (EIA) if, in the Minister's opinion, the proposed mining operation will result in a significant environmental impact. New Brunswick has a standing committee consisting of members from the Department of Natural Resources and Ministry of the Environment who oversee the EIA process by authority of the 1987 Environmental Impact Assessment Regulation. This regulation does not specifically deal with mine decommissioning, but the feasibility of mine reclamation would likely be addressed in the EIA.

The 1982 Water Quality Regulation, by authority of the Clean Environment Act, prohibits anyone, without approval, from permitting a source to discharge a contaminant such that it may cause water pollution. The Clean Water Act of 1989 gives authority to the Minister of Health and Community Services to prescribe the maximum concentration of any contaminant or waste that is permissible in potable water and to the Minister of the Environment for water that is not potable. However, there are no compliance numbers in either the Clean Environment Act or its regulations, or in the Clean Water Act. Criteria for water quality and effluent are usually taken from federal sources such as the MMLER or the interim criteria adopted by the CCME, but criteria are negotiable and site-specific.

**Newfoundland**

Iron ore accounts for over 80 percent of the provincial mineral industry; the other two leading minerals are zinc and asbestos. Newfoundland does not have a significant AMD problem. The Environmental Assessment Act of 1980 governs the approval of new mines in Newfoundland and provides the authority to impose reclamation conditions on these operations. It does not apply to mines already in operation when the Act was passed. Mining operations are regulated under The Department of Mines and Energy Act of 1989. In addition, the Environment and Lands Act specifies that any building or construction requires a discharge permit from the Minister of Environment. The Minister issues a discharge permit to mines before they can start and effluent limits are set at this point. Newfoundland utilizes the federal MMLER and the interim criteria adopted by the CCME, and any other regulations or criteria on a site-specific basis. In addition, the pH of any effluent can be set under the Water and Sewage Regulations.

There are no regulations or policies specific to AMD in Newfoundland. The Newfoundland Department of Mines and Energy deals with AMD problems at the point when the mining company wants to terminate its lease and return the land back to the province. The mining company must show that the mine site will not pose any environmental liability. Site-specific effluent data is reviewed by relevant provincial and federal agencies and a decision is made on whether the mine poses a problem or not. One mine returned to the province does have an AMD problem. As part of the remedial assessment, the province is reviewing stream geochemical data collected previously by the Newfoundland Geological Survey for prospecting purposes. The province plans to utilize the data to establish background concentrations of parameters of concern.

Nova Scotia

Coal and industrial minerals constitute the bulk of mining in Nova Scotia. Coal is Nova Scotia's most important mineral resource, followed by gypsum. Canadian primary tin production came from one open-pit mine in East Kemptville until 1991. Nova Scotia has a significant AMD problem due to the fact that pyrite-rich Halifax Formation slate underlies many areas of the province. Many aspects of mining operations are regulated under the new Mineral Resources Act of 1991, administered by the Nova Scotia Department of Natural Resources. It deals

primarily with land management issues such as mining leases, mining permits, reclamation and bonding. The department is currently developing guidelines for reclamation.

Mining operations are also affected by the Environmental Protection Act of 1973, which requires industry to apply for an industrial discharge permit at the commencement of operations. Effluent criteria are determined on a case-by-case basis as part of the permitting. As a result, the Department of Natural Resources, who have jurisdiction over permitting, works closely with the Department of Environment, who has jurisdiction over effluent quality. Nova Scotia does not have their own water quality objectives or effluent guidelines. They use objectives from other jurisdictions including the federal Metal Mining Effluent Regulations. The Environmental Protection Act also requires industry to propose close-out plans and put-up security at the same time. The close-out plans tend to be very general, for example: "a reclamation plan will be available one year prior to closure".

Mine decommissioning may be influenced in some cases by the Environmental Assessment Act of 1988. This Act requires proponents to register proposed mining activities with the Minister of the Environment. The Minister then decides if the activity has the potential to have a significant environmental impact, in which case an environmental assessment report would be required. Based on the report and input from government agencies and the public, the Minister has the authority to grant approval for the proposed operation and to place conditions on the approval. Specific authority is provided in the Act for conditions requiring reclamation research and rehabilitation to a level acceptable to the Minister. The Act also requires that owners register abandonment of particular operations. Decisions on whether a complete environmental assessment is required are based on input from government agencies and the public.

Of relevance to more than just the mining industry are the Guidelines for Development on Slates in Nova Scotia, jointly prepared by the Nova Scotia Department of Environment and Environment Canada in 1990. The objective of the guidelines is to protect aquatic habitat and water resources from acid runoff from disturbed Halifax Formation slate. The guidelines apply to all developments on, or disposal of, mineralized slates where a total volume greater than 1000 $m^3$ is to be disturbed. The guidelines require monitoring effluents and groundwater at all development sites for pH, arsenic, iron, sulphate, copper, aluminum, total acidity, alkalinity and conductivity. No compliance criteria are given; they are determined on a site-specific basis. The guidelines also include protocols for the sampling and evaluation of bedrock and overburden with respect to acid generation potential.

Recently, Nova Scotia initiated a review and overhaul of the province's environmental laws. The process will culminate in a consolidation of 16 pieces of environmental legislation into a single act, titled the Nova Scotia Environment Act. Among legislation included in the review will be the Environmental Protection and Environmental Assessment Acts.

**Ontario**

Ontario accounts for one-third of total Canadian mineral production: two-thirds of Canada's nickel and salt, almost half its gold, a third of its sand and gravel, cement, copper and uranium, and over a quarter of its silver and zinc. Four minerals: gold, nickel, copper and uranium, constitute over half the value of Ontario's production. Activities specific to mining operations are regulated by authority of the Mining Act which was amended in 1991 to include Part VII. Part VII and its accompanying Regulation 114/91 addresses mine closure and rehabilitation. Any mining company that plans advanced exploration, mine production, expansion or alteration of a project, or temporary suspension or closing out of a project must follow the rehabilitation requirements outlined in an accepted closure plan or the rehabilitation standards set out in the Regulation. Part VII also gives the Director of Mines Rehabilitation the ability to demand closure plans for properties already abandoned when the Act came into force, if the owner can be found.

To assist proponents in dealing with the requirements of Part VII pertaining to closure, the Ontario Ministry of Northern Development and Mines issued: Rehabilitation of Mines - Guidelines for Proponents, that include

details on: closure plans, the Regulations, closure technology, closure components, monitoring, costing and financial assurance. For example, the closure plans should provide data on ore mineralogy and on the acid generating potential of the ore and host rock. Data should also be provided to allow the Ministry to assess potential water quality impacts adjacent to and downstream from the site. Predicted downstream water quality should be compared with Provincial Water Quality Objectives. The closure plan should also provide details on the chemical monitoring program to be carried out during closure including the location of the monitoring points, the parameters to be measured and sampling frequency. Parameters to be measured will be determined on a site-specific basis and will depend on the mineralogy of the particular site, results of the predicted downstream water quality, and relevant provincial and/or federal effluent and water quality criteria.

Policies and implementation procedures regarding the management of surface and groundwater quality and quantity are contained in Water Management - Goals, Objectives, Policies and Implementation Procedures of the Ministry of the Environment (MOE). The document sets Provincial Water Quality Objectives, which represent a desirable level of water quality that the MOE strives to maintain in Ontario's surface waters (Table 4). The document also contains objectives for groundwater quality including Ontario's Drinking Water Objectives and water quality criteria for agricultural use.

Ontario is currently in the midst of implementing the Municipal Industrial Strategy for Abatement (MISA) which is a regulatory program initiated by the MOE to tighten standards that must be met by mines and other industries that discharge an effluent into surface waters. Initially, each of nine major industrial sectors have been subject to a monitoring regulation that requires them to identify, measure and report concentrations of toxic chemicals in the effluents that they discharge into surface waters. After one year of monitoring, the MOE reviews the results and develops regulations that specify the concentrations of toxic pollutants permitted in the effluent of each operation. Effluent limits will generally be established on a sector-by-sector basis, but more stringent limits may be set for a particular operation in order to protect sensitive waterbodies. MOEE will set effluent limits that are attainable by using the "best available technology economically achievable". The draft regulations for the metal mining sector were recently released for public review.

MISA's legislative authority is based on the Environmental Protection Act (EPA) of 1980. The purpose of the Act is to protect and conserve the air, land and water of Ontario. It prohibits discharges that contaminate the environment, and enables the MOE to require that certain discharges be stopped or controlled, preventive measures taken and damage repaired. The Ontario Water Resources Act (OWRA) of 1980 also prohibits discharges of materials that may impair water quality, and enables the MOE to require measures to be taken to prevent or reduce water quality impairment. The general purpose of the OWRA is to protect and conserve the lakes, rivers, streams and groundwater of Ontario. There is some overlap between the EPA and OWRA, but the Acts complement each other and are both administered by the MOE.

Ontario also has an Environmental Assessment Act. Proposed mining operations in Ontario have generally not been subject to the Environmental Assessment process, although MOE is considering requiring that all proposed private sector mining undertakings undergo some form of environmental assessment and approval process.

**Quebec**

Over half the value of Quebec's mineral production is derived from gold, iron ore, asbestos and copper. Mining has traditionally been a major source of employment in the northeastern, north-central and Gaspe regions of the province. Quebec has a significant AMD problem. The Mining Act of 1992 sets out requirements for exploration licenses and for operators involved in mining activities and, due to recent amendments, ensures the rehabilitation and restoration of land affected by mining activities. The Ministry of Energy and Resources is responsible for developing mineral resources in the public domain and has control over the restoration of mining sites. Mine operators are required to submit a rehabilitation and restoration plan for approval before exploration

or mining activities begin (for new mines), or within a set period of time (for existing mines), meet the requirements of the plan and furnish a financial guarantee. The content of restoration plans will be site-specific. Acid generation tests can be requested to determine if there is a danger of AMD, and restoration plans will be stricter when acid tailings are present. Rehabilitation and restoration plans are subject to consultation with the Ministry of the Environment. In cases of non-compliance, the Minister has authority to order the work done at the operator's expense. The amendments also give the Minister authority to order an operator that has ceased mining to perform rehabilitation and restoration work due to the presence of tailings, even if the operation predates the amendments.

Mining activities are subject to the Environmental Quality Act of 1977, which requires that certificates of authorization be obtained from the Ministry of the Environment prior to the commencement of any mining activity which may change the environment. A certificate is also necessary for any modifications. Procedures and requirements for obtaining the certificate of authorization are outlined in Directive 019, a guidance document developed by the Ministry of the Environment in 1989, which addresses aspects of the Environmental Quality Act applicable to the mining industry. For example, it contains criteria for final effluent from mines, derived largely from the federal MMLER. The directive applies to both new and existing mines.

The Drinking Water Regulation contains enforceable drinking water quality criteria. Quebec does not have any regulations containing surface water quality criteria.

**Saskatchewan**

About 40 percent of world exports of potash are produced in Saskatchewan. The province also contains some of the highest-grade reserves of uranium known in the western world. Other metals and minerals produced in Saskatchewan include coal, sodium sulphate, special clays, copper, zinc, gold, silver, cadmium, selenium and tellurium. Newer uranium ore bodies, which are located in the sulphide-bearing Athabasca Sandstone, tend to have a greater AMD problem than older ones located in sulphide-poor basement rocks. Saskatchewan has a fairly arid climate and AMD can take a while to manifest itself. Prior to licensing, new mines in Saskatchewan must go through the Environmental Impact Assessment (EIA) process by authority of the Environmental Assessment Act of 1979. Once the Minister of the Environment is satisfied that the proponent has met the requirements of the Act, he will decide whether to approve the development. The Minister may impose any terms and conditions on the approval that he considers necessary or advisable. For example, in the case of major coal mine operations, conditions have been placed on the approvals requiring compliance with the reclamation program described in the Environmental Impact Statement (part of the EIA process) or with area specific reclamation guidelines.

The Mines Pollution Control Branch of the Department of Environment is responsible for licensing mines by authority of the Environmental Management and Protection Act of 1983, which the Branch administers. The Branch provides input during the EIA process, but its primary role lies in approving and ensuring compliance with the site specific reclamation plans required as part of the EIA process. Security deposits are generally not required. Abandonment of a mine site requires approval by the Minister.

Effluent criteria for mines are site-specific. Some criteria are contained in the Mineral Industry Environmental Protection Regulations of 1991, but these are very broad-based and apply to all types of mining. Effluent criteria are generally determined by considering the Surface Water Quality Objectives of 1988, which are incorporated into the EIA prior to approval.

In 1992, Saskatchewan Environment and Public Safety, with input from a committee of industry and regulatory representatives, published: Mine Rock Guidelines - Design and Control of Drainage Water Quality. The guidelines are to provide a summary of the processes which control effluent quality and to provide guidance and recommendations for the state of the art in prediction, control and monitoring of AMD and metal leaching for the uranium mining industry in Saskatchewan, in particular.

# ARD POLICY FOR MINE SITES IN BRITISH COLUMBIA[1]

**William A. Price and John C. Errington[2]**

**Abstract:** From previous project reviews and research, the British Columbia Reclamation Advisory Committee (RAC) has developed a set of policies, rules, and procedures that guide its assessment of projects with issues involving acid rock drainage (ARD). These have been compiled into a comprehensive working policy, which has been released both for public information, and industry discussion and comment.

In attempting to cover the entire spectrum of ARD issues, the ARD Policy contains sections dealing with exploration, proposed mine developments, prediction, prevention, collection and treatment, commercial leaching, permitting, bonding, monitoring, historic sites, and existing mines. Excerpts of the Policy set forth in this paper govern the information required from a prediction program, the type of materials used for construction purposes, preventive techniques such as underwater disposal, blending and segregation, and inventory and monitoring requirements associated with waste handling.

Through ongoing research and monitoring, and as ARD is addressed at more and more projects, the ability for informed decision making is continually being improved. Based on the ministry's recent experience in reviewing mine projects and comments received from the public and industry, the Policy is presently being revised. As better methods are developed, the RAC will endeavor to continually incorporate them in the provincial ARD policy.

## Introduction

The British Columbia government regards mining as a temporary use of the land. This principle is enforced by current mining legislation, which requires that disturbed land and water resources be reclaimed to a level of productivity not less than that which existed previously, and that water released from the mine site meet long-term water quality standards.

As part of the regulation of the mining industry, all mine proposals and closure plans are examined by interagency Regional Mine Development Review Committees (RMDRC). The Victoria-based Reclamation Advisory Committee (RAC) reviews the decisions of the RMDRC and issues the Reclamation Permits, which are required before mining can proceed. A major factor in many mine reviews is the consideration of acid rock drainage (ARD).

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To enable the British Columbia mining industry to comply with the present comprehensive reclamation requirements, the provincial government is participating in a program of research aimed at developing cost-effective solutions to major environmental problems. A major part of this program focuses on acid rock drainage. As part of this research, the provincial government coordinates and provides funding for the British Columbia Acid Mine Drainage Task Force and is a vital participant in activities of the national acid rock drainage program (Mine Environmental Neutral Drainage Program - MEND). In 1992, research expenditures for the British Columbia Acid Mine Drainage Task Force amounted to $1.5 million, with funding shared by industry and the provincial and federal governments.

From previous project reviews and research, the RAC has developed a series of working policies and technical initiatives to deal with acid rock drainage. These procedures, rules, and guidelines were first compiled as a working document in 1992. Much of the text of this paper has been drawn from a draft discussion document released in July, 1993. This paper incorporates some changes made as a result of comments received during the public and industry review and from the Policy's use in mine reviews. The RAC will continue to refine its ARD guidelines as new information and technology becomes available.

The ARD Policy set forth herein reflects the RAC's current philosophy of preventing ARD generation through prediction and design, avoiding long-term treatment wherever possible. In attempting to cover the entire spectrum of ARD issues, the complete ARD Policy contains sections dealing with proposed mine developments, prediction, prevention, collection and treatment, permitting, bonding, monitoring, historic sites, existing mines, commercial leaching, and exploration. This paper presents excerpts of that document.

## General Principles

All mining and exploration activity will be regulated in a manner that protects the environment and minimizes the economic risks to the province.

All potential environmental and social impacts of acid drainage for a proposed mine will be assessed during the Mine Development Assessment Process, prior to issuance of a Mine Development Certificate.

Owing to the significant cost and site-specific needs of each mining project, a proponent is advised to discuss the ARD prediction or remediation program with regulatory agencies prior to its implementation.

Because some uncertainty is associated with predicting the occurrence and extent of acid rock drainage, treatment and control contingencies may be part of some mine approvals. Prediction is generally more difficult for waste rock dumps, open pits, and underground workings than for tailings impoundments. While the technology is available to prevent the release of acid rock drainage from operating mine sites, it is not yet possible to completely abandon acid-generating mines without requiring long-term programs to collect and treat acid rock drainage or to maintain prevention and control measures.

Security will be required as a condition of a reclamation permit to ensure that sufficient funding is available to cover all outstanding reclamation obligations, including long-term costs associated with monitoring, maintenance of preventive structures, and treatment of acid rock drainage following mine closure.

Through research, the Province will continue to assist the development of improved acid rock drainage technology.

The Province will revise the ARD Policy and regulatory framework to reflect changes in acid rock drainage technology. Modifications and additional requirements for individual mines will be based upon site specific conditions.

## Prediction Of Acid Rock Drainage

The proponent is required to predict the acid rock drainage potential for each overburden and rock type mined or exposed, and for any waste material, such as waste rock or tailings, created by the operation.

The prediction of acid rock drainage typically will require an iterative process of sampling, analysis, and classification, similar to that used to determine other geological characteristics such as ore reserves. Based on the results of the first round of sampling and analysis, it may be necessary to further subdivide some of the original rock types or to do additional analysis. Conversely, it may be possible to reduce the number of different materials when determining management units.

### Sampling and Statistical Requirements

Perhaps the most important phase of the prediction program is sampling. The objective in sampling should be to show the type of materials that will be exposed and to indicate which ones are likely to cause water quality problems. As much of the variability will result from differences in geology, sufficient samples should be taken to accurately characterize the variability and central tendency for critical parameters in each significant rock type, based on accepted statistical procedures.

The number of samples taken from a particular material will depend on its variability and the questions being asked. Where no additional guidance is available, the suggested *minimum* number of samples collected from *each* rock and/or overburden type during *initial* sampling is as follows:

| Mass of Each Separate Rock Type (tonnes) | Minimum Number of Samples |
|---|---|
| < 10,000 | 3 |
| < 100,000 | 8 |
| < 1,000,000 | 26 |
| 10,000,000 | 80 |

Owing to the significant cost and specific requirements of each mining project, a proponent is advised to discuss the sampling program with regulatory agencies prior to its implementation.

Sample size should be based on the geology, the variability of the materials and the mode of extraction, exposure and deposition. Compositing of samples should be avoided unless either the geological information indicates it is advisable (e.g., homogeneously disseminated sulfides and carbonate) or the materials will be composited during mining (e.g., over bench heights).

**Requirements of the Prediction Program**

The prediction program should incorporate the following procedure:

**Step 1. Separate and Describe Different Materials** Based on particle size, petrology, mineralogy and chemical composition, separate significantly different overburden and rock types and describe their distribution, lithologic unit, and identifying properties.

For each distinct rock or overburden type, determine the quantity, spatial distribution, and eventual form of exposure (e.g., pit wall, tailings, or waste rock).

**Step 2. Initial Screening** Conduct initial screening of materials based on the sulfide content, available metal content, and pH value. Materials with a paste pH less than 3.5 will be considered a potential source of acid drainage. All materials with either more than 0.3% sulfide-S or a pH less than 5.0 should be tested further.

The form and amount of readily soluble constituents should be measured in materials that are already oxidized to identify situations where there is a potential for the dissolution of residual weathering products. The method for determining availability should be selected according to site-specific criteria.

**Step 3. Acid Base Accounting and Other Static Tests** The next phase of the ARD test work should include static tests to measure the maximum acid generation potential (AP) and maximum neutralizing potential (NP) of each rock and overburden type, and traditional acid base accounting procedures to calculate the net neutralization potential (NNP) and neutralization potential ratio (NPR).

The criteria used to evaluate materials based on acid base accounting tests are:

- Materials with sulfide minerals whose net neutralizing potential (NNP = NP-AP) is negative are likely to be a potential acid drainage source. Exceptions are possible if the sulfide content is very low and/or there are significant slow release, non-carbonate sources of alkalinity.
- The acid drainage potential will be considered uncertain if materials have a ratio of neutralization potential to potential acidity (NP:AP) of less than 4:1.

The ratio of 4:1 represents the worst case to date and was selected to ensure the detection of sites where there is an unfavorable balance between long-term acid generation and neutralization reactions or where the composition of a waste rock's fine-sized fraction varies significantly from that of the whole rock. It is recognized that a 4:1 ratio is conservative and will be much higher than the actual composition of acid drainage generating materials at most sites.

Where the acid rock drainage potential of a waste material or geological unit is uncertain, it will be considered acid drainage generating unless the proponent can show through further testing (step 4) that the acid and neutralization reaction rates are favorably balanced.

It is cautioned that acid base accounting rules can only be used for generally assessing whether there is a potential for acid drainage. More refined site and material-specific interpretation or testing should be used if less conservative, more accurate information is required.

**Step 4. Prediction of Reaction Rates and Water Quality**. To more accurately predict the ARD potential, the proponent will be required to develop site and material-specific criteria based on the relative rates of acid generation and neutralization reactions, and the availability of each fraction to react. Depending on the materials and site, it may also be necessary to determine the neutralizing potential of non-titratable, slow release minerals, or the size and rate of additions from biotic and climatic sources.

The data used to predict reaction rates and water quality may be obtained from a variety of sources, including a more detailed characterization of the materials, comparisons with similar materials at other sites, weathering and leachate quality observed in soils and outcrops, the quality of seepage from existing waste materials, kinetic laboratory tests, and field trials.

Kinetic tests are often used to estimate relative acidity-neutralization reaction rates and to predict metal release and loading. The value of kinetic tests as a predictive tool will be greatly enhanced where it is shown that these tests simulate either the real field release rates or the actual balance of important processes, such as oxidation, dissolution, and entrainment.

Where there is a potential for the generation of acid drainage or metal release through weathering or dissolution, the proponent should determine the range, variability, and central tendencies for the following properties:

- Elemental composition
- Mineralogy
- Readily soluble constituents
- Sulfide types (amount, reactivity, and spatial distribution)
- Carbonate types (amount, reactivity, and spatial distribution)
- Mineralogical and rock fabric characteristics that will influence weathering.

The properties listed above may be modified to match the specific conditions of each site, material and disposal option.

## Prevention of Acid Rock Drainage

All mines with a potential for acid drainage must provide operational reclamation and abandonment plans showing how they will reduce the generation of acid drainage to levels not considered deleterious to the environment, both in the short and long term. In cases where the success of prevention measures cannot be assured, an assessment of the potential for failure and of the possible impacts will be required. Detailed contingency plans to reduce the risk to the environment will also be needed. This will usually consist of a collection and treatment system.

Until it is determined that there is no potential for acid drainage or other water quality problems, material with one or more of the following properties shall not be used for roads, dams or other construction purposes:

- A total sulfide level greater than 0.3%
- A paste pH less than 5
- A potential for significant metal loss from residual weathering products.

### Underwater Disposal

Secure underwater disposal of tailings or waste rock in manmade structures is currently an acceptable form of acid rock drainage prevention. Underwater disposal in natural water bodies will only be considered where there is shown to be no significant impact on water quality, fisheries, recreation, or downstream flow, and where the water bodies are shown to be the most environmentally suitable disposal site.

Further research is required to assess the impact of the underwater disposal of tailings or waste rock in biologically or water-quality-sensitive lakes before this disposal alternative will be considered.

In cases where underwater disposal of tailings or waste rock is proposed, the proponent must show that:

- The mine wastes do not contain significant readily soluble deleterious substances
- The water balance ensures that all potentially acid-generating wastes will be continuously covered by water
- There will be no significant impact as a result of wave action, ice, avalanches, flooding, earthquakes, thermal overturn, and other relevant natural factors.
- The mode of deposition, the water depth and other design features satisfy the requirement for long-term prevention of acid drainage.

Manmade impoundments shall be designed and maintained for long-term geotechnical stability, taking into account the possible effects of biological activity.

A water cover is currently an acceptable form of acid rock drainage prevention for underground workings or open pits.

The timing and inflow rate requirements in flooding open pits and underground workings will be based on the hydrologic conditions, the relative reaction rates of acid generation versus neutralization, and the potential release of acid products. Proponents must demonstrate that any water released to the environment will be of acceptable quality. Hydraulic bulkheads or material barriers in underground workings shall be designed to allow ongoing verification of the water level and of geotechnical stability.

**Blending and Segregation**

Blending of acid generating and acid consuming materials may be an acceptable acid rock drainage prevention strategy.

The proponent will be required to show that there is sufficient information about the materials to ensure that blending or segregation can be carried out safely. Given the potential for incomplete mixing, more conservative NNP and NPR criteria will likely be required than those set for single homogeneous materials (see earlier discussion of acid rock drainage prediction).

The proponent will also be required to demonstrate that the blending plan is compatible with the mine plan and that it is possible to achieve the required control in materials handling. This should include any interim prevention measures for the protection of exposed material stockpiled prior to final disposal.

## Collection And Treatment Of Acid Rock Drainage

Although prevention will be the primary method of acid rock drainage control for new mines, collection and treatment may be accepted as a mitigation method if the proponent can demonstrate that -

- All preventative methods have been examined and are not economically feasible with current technology
- The risk (likelihood of occurrence and consequences) to the environment caused by a failure to the system is acceptable
- The quantity of acid rock drainage is readily manageable
- Treatment sludge can be disposed of safely.

In addition, the proponent must provide adequate security to finance long-term collection and treatment.

Where collection and treatment is proposed as the principal means of protection, the proponent must show that the system can be maintained in perpetuity. The supportive evidence should include detailed engineering and economic analysis, including consideration of relevant biological factors, and a

comprehensive risk analysis to show that environmental values will not be jeopardized. Any proposed collection and treatment technology must be proven to be effective, with adequate resources set aside to support its operation.

While treatment technology is generally effective, long-term collection and treatment is among the most expensive remediation options. Because of the long time frame and uncertainties about future costs, government's bonding requirements are likely to make collection and treatment an unattractive option for new mines. Since final site reclamation is not possible and there are ongoing sludge disposal problems, proponents must recognize that the use of collection and treatment as a primary means of control will only be approved in exceptional circumstances. Approval of such programs will be at the discretion of review agencies.

During the mine operation all collection ditches, dikes, impoundments, and pumping systems must be designed to handle a one-in-200-year flood during the mine operation. For abandonment, the minimum design criteria for structures whose failure would jeopardize the effectiveness of the acid rock drainage prevention plan should be based on the consequence of failure. Where the consequences of failure are high, the minimum design criteria should be the probable maximum flood (PMF) and the maximum credible earthquake (MCE).

## Reclamation Permitting

Following a detailed review under the Mine Development Assessment Process, mines are regulated under mine plan approvals, reclamation permits, and waste and water management approvals, permits, and licenses. All of these approvals, permits, and licenses are issued subject to monitoring and confirmation of predictions, and with conditions to prevent the generation and dispersal of acid rock drainage.

## Bonding

As a condition of a reclamation permit, a security will be required to ensure that sufficient funding is available to cover all outstanding reclamation obligations, including long-term costs associated with the necessary monitoring, maintenance, collection, and treatment of acid rock drainage following mine closure.

Bonding requirements will be addressed at each stage of the Mine Development Assessment Process. The Regional Mine Development Review Committee, following a detailed technical review, will recommend an appropriate level of security to be required as a condition of the reclamation permit. The level of security will increase as reclamation obligations increase during mine operation. The security will be set at a level that will, in a self-perpetuating manner, pay for -

- the long-term inspection, monitoring, and maintenance, and if necessary the operation of collection, containment, and treatment plant facilities
- the long-term inspection, monitoring, and maintenance of preventive and control structures.

The security will take into account the high degree of uncertainty in predicting these costs. Securities attached to existing mines will be increased should acid rock drainage be discovered. Should remediation efforts reduce the extent of acid generation or the risk of future acid generation, security levels will be reduced.

## Monitoring

Monitoring programs designed for environmental protection and for the early detection of acid rock drainage will be required as a condition of permits issued by the Ministries of Energy, Mines and Petroleum Resources and Environment, Lands and Parks. Intermittently, QA/QC programs should be carried out to verify the accuracy and precision of the data.

During mine operation, companies will be required to maintain a detailed inventory of all waste or exposed materials. This information should be continually updated and readily accessible in a complete package. It should include analysis results for drill core, operational sampling and analysis of waste materials and exposed surfaces, and monitoring results for the water draining from various wastes and off site. The ARD information should include the rock or overburden type, acid/base accounting data, elemental analysis, storage location, and date of emplacement. Paste pH and readily soluble constituents should be measured where some weathering has already occurred.

Following mine closure, the permittee will be required to undertake all long-term requirements including monitoring, inspecting, and reporting on, and maintaining the mine site and ancillary ARD facilities. Monitoring programs will be designed to ensure that operational procedures are being carried out and to determine if prevention techniques have been successful.

## Annual Reports

Annual reports summarizing the materials, methods, and results of the acid rock drainage monitoring program will be required as a condition of all reclamation permits. The annual reports should also summarize the previous year's mining and waste disposal, the materials mined and any anticipated changes to the mine plan. Cumulative detailed ARD monitoring records should be appended.

# VEGETATION PRODUCTIVITY EQUATION COMPATIBILITY WITH SELECTED STATE ENVIRONMENTAL RECLAMATION LAWS AND REGULATIONS[1]

Jon Bryan Burley[2]

**Abstract.** Federal and State reclamation laws and regulations in the United States of America are becoming more precise and specific concerning evidence to support reclamation confirmation and subsequent bond release. Scientific methods to assess reclamation success are in their formative stages, with some investigators pursuing the development of efficient and effective assessment techniques. One analytic tool, the vegetation productivity equation, has been recently developed that may play an important role in future reclamation assessment activities. This paper examines current reclamation laws in seven States (North Dakota, Montana, Wyoming, South Dakota, Minnesota, Indiana, and Michigan) to determine their compatibility with the concept of vegetation productivity equations. Of the States examined for this study, five would require amendments to their reclamation regulations before the vegetation productivity equation would become a sanctioned assessment technique, and in one state empirical assessment techniques are not required. Only in South Dakota would vegetation productivity equations be compatible with existing regulations, providing review agencies accept the assessment technique as valid. This incompatibility with current reclamation regulations does not mean that vegetation productivity equations are intentionally being excluded from assessment methods; rather, vegetation productivity equations are still a new technique that has not yet been incorporated into the reclamation regulation fabric. While vegetation productivity methods may be eventually adopted in several States, reclamation regulations will require amendment before they can be fully employed.

**Additional Key Words:** natural resource policy, environmental law, landscape planning, ecological modeling.

## Introduction

Reclamation laws and regulations in North Dakota require surface mine coal operators to reclaim the postmining landscape to a condition able to support vegetation growth that is equal to or better than the premining landscape situation. Operators must quantitatively demonstrate that the postmining landscape has been effectively reclaimed. The quantitative procedures employed to substantiate reclamation effectiveness are evolving and being improved through a collaborative effort between governmental regulators and policy makers, surface mine operators, the scientific community, and the public. This paper examines the current quantitative approaches, primarily the Cropland Reference Area Standard and the Soil Conservation Service (SCS) Cropland Technical Standard, in comparison to the prospects presented by the Vegetation Productivity Equation Method, a new quantitative approach that may be a candidate for eventual inclusion as a standard evaluation technique.

Federal involvement in surface mine reclamation is a relatively recent activity, starting with proposed legislation in the 1940s and culminating in significant legislation in 1977. Workman (1987) states that national legislation was necessary because many States were hesitant to develop a regulatory system that might discourage industrial activity within the State and encourage industry to seek production in other States. Without a regulatory system, surface-mined lands were often abandoned and not reclaimed. Thus, Federal legislation was paramount to have an economically equitable situation between the states and to enforce reclamation activity.

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Simpson (1985) reviews the chronology of applicable environmental movements and Federal legislation leading to the Surface Mining Control and Reclamation Act (SMCRA) of 1977 (Public Law 95-87), signed into law by President Carter. Dunlap (1976) reviews the legislative history of this Act, noting that 1975 legislative activity formed the precedent to the 1977 SMCRA. Scheter (1980) reviewed the historical background and effects of the 1977 act. Munshower and Judy (1988) state that the SMCRA of 1977 defined the expectations associated with the reclamation of surface mined coal landscapes. They note, "... legislation did not intend to prevent surface mining for coal, but rather attempted to convey society's recognition that reclamation must be an integral part of any coal surface mining activity. In effect, the U.S. Congress was reflecting this desire for protection from impacts of such development and the need for reclamation." As required by the National Environmental Policy Act of 1969, the Federal Government prepared an Environmental Impact Statement (EIS) of the policies and regulations concerning the 1977 SMCRA, with the Office of Surface Mining (OSM) as the lead agency conducting the EIS. The OSM (1979) EIS also identifies the purposes of the Act. The SMCRA of 1977 gave each State the primary responsibility for regulating surface mining and reclamation. This regulatory approach is called "State Primacy" (Workman 1987). Workman (1987) remarks that Congress recognized the need for State administration to address a diversity of terrain, climate, biology, geochemistry, and other physical attributes of coal mining areas. Note the emphasis upon coal and not other surface mines such as sand and gravel or kaolinite mines. The State primacy approach is not without criticism. Lucas (1987) ascertains that there is much confusion in a Federal oversight situation where surface mine operators have been subjected to the confusion of a dual agency regulation system (State and Federal). Cardwell (1987) maintains that except for lawyers, everyone (land owners, operators, State regulators, Federal regulators, and environmentalists) has been a loser. Despite problems, Federal legislation exists that attempts to maintain a set of minimum reclamation standards for surface-mined coal lands across the United States. This legislation entrusts the development and enforcement of regulations to each State. Dernbach (1986), Udall (1979), and Vestal (1988) also provide insight into the effectiveness of and retrospective perspectives to, the 1977 SMCRA.

## Discussion

### North Dakota Rules and Regulations

In response to the SMCRA of 1977, the State of North Dakota charged the Public Service Commission (PSC) with regulation and enforcement of surface mining and reclamation within the State. Hagen (1974) presents a pre-SMCRA of 1977 perspective concerning North Dakota reclamation law. The laws examined in this study governing surface mining and reclamation operations were published by the PSC in 1989. These laws include three statutes: Surface Mining and Reclamation Operations, Surface Owner Protection Act, and the Exploration Data. Essentially, the Surface Mining and Reclamation Operations statute addresses the permitting and inspection process. The intent of this statute (PSC 1989) includes recognizing the impact of surface mining, acknowledging the importance of coal mining, the importance of regulation, the protection of the public tax base, and the importance of inspections. The statute addresses the importance of restoring the landscape so that it is capable of supporting premining land-uses, including industrial, commercial, agricultural, residential, recreational, or public facilities. Also, the statute describes the protection of prime farmland (including soil horizon protection), backfill characteristics, topographical requirements, erosion control obligations, and hydrological requirements. In addition, on page 38 of the published statute (section 38-14.1-24,(17)), the PSC (1989) stresses the importance of maintaining soil productivity for vegetation growth. This philosphic position in the statute and the background history of the SMCRA of 1977 should provide insight into the origins and requirements for quantitative reclamation approaches in North Dakota. Complementing the statutes, the State of North Dakota has written "Rules Governing the Reclamation of Surface-Mined Lands" (PSC 1990). These rules describe in greater detail the requirements governed by the law, including performance standards for suitable plant growth, the issuance of performance bonds, backfilling and grading, revegetation, prime farmland requirements, and even the development of fish and wildlife habitat. Finally, the PSC (1988) has published "Standards for Evaluation of Revegetation Success and Recommended Procedures for Pre- and Postmining Vegetation Assessments." This document describes in great detail the allowable methods and techniques that are permissible to demonstrate that the reclamation work has been in compliance with the law so that performance bonds can be released. The document states--

> According to NDAC 69-05.2-22-07(4)(b), "crop production from the permit area shall be equal to or greater than that of the approved reference area or standard with ninety percent statistical confidence for the last two consecutive growing seasons of the responsibility period." For prime farmlands, North Dakota rules require equal or greater productivity in each of the last three consecutive growing seasons of the responsibility period.

Notice that the document mentions a desired confidence level associated with the determination. In other words, to determine the soil productivity of the reclaimed landscape, inferential statistical procedures are mandated. Essentially there are two allowable techniques to determine the 90 % confidence level: the Cropland Reference Area Standard and the Soil Conservation Service (SCS) Technical Reference Standard.

**Cropland Reference Area Standard.** This approach employs an undisturbed parcel of land containing the same soil series as the premining soil profile. Crops are grown on the postmining soil and on the reference area site for 2 years (or 3 years for prime farmland). If the premining site contained more than one soil profile type, each disturbed profile must contain a corresponding reference profile from the same series. The PSC (1988) describes the mathematical procedures to employ to generate a weighted average yield from a site with multiple premining soil series.

**SCS Technical Reference Standard.** The second approach utilizes SCS soil survey data to generate the unadjusted premining crop yield standard to which the postmining soils are compared. The unadjusted yield is based upon climatic averages and can be adjusted using correction methods described by the PSC (1988).

For both techniques, the PSC (1988) describes the sampling procedures to conduct the statistical tests. The procedures detail the formula to determine the number of samples required in order to collect the observations. In addition, the PSC prescribes the allowable methods to measure vegetation productivity. The PSC (1988) also prescribes the type of statistical analysis to be employed in the comparison. The statistical method is the one-tailed Student t-test. Consequently, the State of North Dakota has a very descriptive and prescriptive set of laws, standards, methods, and statistical procedures to evaluate the capability of the landscape in a reclaimed condition. However, these approaches are not without criticism.

**Concerns About Existing Approches**

Until recently, the only method to determine the agricultural suitability of reclaimed surface mine soils (findings of reclaimability) was to grow crops on the reclaimed land and compare plant growth with that in reference areas or with existing SCS productivity data. The method has been in existence and employed for only a decade, so it is still a relatively novel approach and may be considered still in its formative stages. This approach has some severe limitations (Doll and Wollenhaupt 1985), especially concerning the extended length of time required to conduct the postmining reclamation evaluation and the inability of the approach to assist in the premine planning and design process. The method is considered relatively costly. Another problem with the approach is the effect upon the variation of the measured productivity values of uncontrollable environmental factors such as rainfall, hail damage, and insect damage. Reference areas may receive a key burst of rainfall from a thunderstorm that increases plant productivity while only a few hundred yards away no rainfall is recorded; conversely, a hail storm damaging the crops on the reclaimed site may not damage a nearby reference area. While the SCS Technical Standard approach can account for yearly variations in rainfall, there are presently no correction factors for hail or insect damage. Thus the method is not infallible and may require further field trials before an operator successfully demonstrates reclamation achievements. In addition, the cropland reference area and technical reference area approaches are *post hoc* evaluative methods, meaning that they offer little premining reclamation assessment and no corrective solutions. Once a soil has been placed in its planned reclaimed area configuration, changing or correcting a soil profile can be extremely costly. The reference approaches only confirm an outcome; they do not assist in providing information that may lead to successful reclamation results. The shortcomings of the reference area approach have resulted in the call for new quantitative measures (Walsh 1985).

**Vegetation Productivity Equation Method**

Essentially, reclamation research teams and individuals in the United States have attempted to build a reclamation equation that would predict average plant growth for a variety of vegetative crops based upon soil characteristics. Neill (1979) and Doll and Wollenhaupt (1985) were notable investigators who first suggested the mathematical composition of these models. The implied future advantages of these models include the application of the equation during the premining planning stages to predict optimum soil profile configurations for maximum plant growth at minimal reclamation costs, numerical prediction of the reclaimability of the landscape using soil parameters and the potential to apply reclamation equations to other forms of landscape disturbance such as construction activities. Several factors inhibited the instant development of such equations. First, to develop these models would require extensive data, meaning the collection of soil parameters (pH, bulk density, hydraulic conductivity, etc.) for a wide variety of soils and growing several types of crops (grains, vegetables, trees, and grasses) on these soils through diverse environmental conditions (drought, normal, and wet years). Second, to develop the model, one had to be versed in advanced statistical methods (multivariate procedures such as Principal Component Analysis). Burley (1988) and Burley, *et al.* (1989) published the first highly predictive reclamation model (Equation 1) by combining the fieldwork of the Soil Conservation Service and applying advanced statistical methods. This model was developed for seven agronomic crops in Clay County, Minnesota. In addition, Burley and Thomsen (1987) published the methodology to produce reclamation productivity models. Since that time, Burley (1991) has developed a productivity model applicable to both woody plant and agronomic crops, a productivity model for sugarbeets (Burley 1990), a paper describing the application of these procedures for reclamation specialists (Burley and Thomsen 1990), the development of two equations for a Florida case study (Burley and Bauer 1993), and a recent overview of issues concerning the equations (Burley 1992).

$$\begin{aligned} PLANTS = {} & .6206 + (-1.1805 * ((HC - 3.9296)/4.0030)) + (-0.3575 * (((SL - 3.0000)/4.6810)^{**}2)) \\ & + (-1.9375 * ((BD - 1.3584)/0.2644)((FR - 0.9075/3.4929)) + (-2.3420 * ((EC - 2.526)/1.0947)((FR - 0.9075)/3.4929)) \\ & + (1.2424 * ((OM - 3.9512)/0.6638)((EC - 2.5269)/1.0947)) \end{aligned} \quad (1)$$

where: $R^2 = 0.7399$, PLANTS = predicted productivity score, HC = hydraulic conductivity, SL = % slope, BD = bulk density, FR = % rock fragments, EC = electrical conductivity, OM = % organic matter

While Burley and colleagues have demonstrated that it is possible to build highly predictive reclamation models, the model presented in Equation 1 and similar models developed by Burley are applicable only to Clay County, Minnesota. However, as data concerning soil productivity are accumulated, future models applicable to large regions of the United States and to large regions in other countries may be created (although this process may take several generations of investigators to develop these models). Productivity equations have one similarity to the SCS Technical Reference Area Approach. The equations are generated from SCS data published in SCS County Soil Surveys, meaning that they are generated from a source acceptable to the PSC. The productivity equations are applicable to statistical sampling procedures approved by the PSC. Postmining soils can be sampled in accordance with the sample size determination techniques recommended by the PSC for crops but applied to soil sample sizes. Based upon the variables in a given productivity equation, the measured soils can generate a prediction of soil productivity. In addition, the soil samples can be compared using the Student t-test as recommended by the PSC (1988). The productivity equations are congruous to the soil weighting procedures described by the PSC (1988).

While the above paragraphs describe the suitability of productivity equations in evaluating the reclamation effectiveness of disturbed lands, the equations may also have some incompatibility problems. The first problem is that presently no productivity equation has been generated or compared to known yields on reclaimed soils. The models have been generated with strictly undisturbed soils. Before any reclamation productivity equation is accepted by any regulating authority, the agency needs to have sufficient evidence that these models predict the productivity of disturbed soils. Second, the equations are generated without the introduction of variables encountered in disturbed

soils but not encountered in existing undisturbed soils. For example, the models do not consider soils with toxic materials such as high levels of selenium or boron (Williams and Schuman 1987). Therefore, to apply these equations, the reclaimed soils cannot contain toxic materials beyond the boundaries of the soil profiles studied.

**Other State Laws**

The following discussion describes selected State laws and environmental issues pertinent to vegetation productivity reclamation. Montana, Minnesota, and South Dakota were selected because of their adjacency to North Dakota. In addition, Wyoming, Indiana, and Michigan were selected for examination as a second tier of States radiating from North Dakota. No other States or Canadian provinces were selected.

**Montana**. The State of Montana contains numerous coal deposits on the Northern Great Plains suitable for surface mine extraction. As in North Dakota, the State laws of Montana indicate a concern for restoring agricultural productivity after surface mine operations. The Montana State Code (State of Montana, 1985:Title 82, Minerals, Oil, and Gas. Chapter 4. Reclamation Minerals- Coal & Uranium: Part 2, 82-4-232-6-ii-B) declares no reclamation bond will be released, "... before soil productivity for prime farm lands to which the release would be applicable has returned to equivalent levels of yield as nonmined land of the same soil type in the surrounding area under equivalent management practices, as determined from the soil survey." This statement indicates the importance of determining premining soil productivity through such documents as soil surveys and then demonstrating that the postmining landscape can produce similar productivity values. To develop the rules concerning the assessment of postmining agricultural productivity, the State of Montana has mandated that the Board of Land Commissioners shall define the rules governing reclamation assessment (State of Montana 1985:82-4-233). This board shall define the rules for segregation of soil horizons, grading, and revegetation (State of Montana 1985:82-4-232-3). Part 3 (State of Montana 1985:82-4) describes the laws governing surface mines extracting metallic ores. Part 4 (State of Montana 1985:82-4) describes the laws pertaining to the reclamation of lands mined for bentonite, clay, scoria, phosphate rock, and gravel. The code acknowledges that acceptable postmining landuses include forestry, pasture, orchard, cropland, residence, recreation, industry, and wildlife habitat. Reclamation plans are subject to annual review and modification by the Board of Land Commissions to reclaim the land to a productive use. The State of Montana has developed considerable administrative rules governing coal reclamation, including the necessity for baseline vegetation productivity information and special application requirements for prime farmlands (Montana 1990:26-468 to 26-473, 26-488 to 26-490). According to the State of Montana (1990:26-597 to 26-599), there are several allowable monitoring methods to demonstrate revegetation success. These methods include the reference area comparison and the technical standard approach derived from historical data (similar to North Dakota's technical standard approach). For revegetation parcels less than 100 acres in size, U.S. Department of Agriculture or U.S. Department of the Interior technical guides may be employed. On prime farmlands, the crop with the greatest rooting depth must be one of the reference crops employed for reference area comparisons and monitoring (Montana 1990:26-642), with "at least at the 10% level of significance using statistically appropriate sampling techniques approved by the department in consultation with the Montana state office of the U.S. soil conservation service." (Montana 1990:26-643). For trees and shrubs, density and mortality data are employed to evaluate reclamation success (Montana 1990:26-602). While the State of Montana does not specify the mathematical methods required to demonstrate vegetation reclamation success, it appears that Montana's reclamation rules and regulations are similar in intent to North Dakota's rules and regulations. Montana's reclamation rules and regulations may accommodate productivity equations as a method to evaluate reclamation success, provided that the State of Montana and the Montana State Office of the U.S. Soil Conservation Service approve the methodology in advance of its application.

**South Dakota**. The State of South Dakota has developed the South Dakota Mined Land Reclamation Act (South Dakota 1983, 13A:418-446). South Dakota legislators have decided that the minerals covered by this Act include all organic and inorganic minerals except water, oil, gas, sand, gravel or rock to be crushed and used in construction, pegmatite minerals, and limestone, sand, gypsum, shale, or iron used in the process of making cement (South Dakota 1990b). Essentially, operators must submit a plan, and the plan must be approved before mining proceeds. According to the laws, reclamation plans that propose installation of vegetation related to a specific postmining

landuse requiring vegetation must select suitable tree species and appropriate seeds pertinent to the chosen postmining landuse (South Dakota 1983, 13A, title 45-6B-45). However, defining appropriate or suitable material can be an ambiguous task open to debate and varying opinion. Therefore, the State of South Dakota (1990:13A:title 45-6B-81) has authorized the Board of Minerals and Environment to promulgate requirements for surface mining and reclamation. ARSD 74:29:07:06 (State of South Dakota 1988) addresses reclamation standards. While these standards may not be as specific as North Dakota's rules governing revegetation, South Dakota's standards state "The applicant must develop methods and procedures for revegetation which incorporate reference areas, baseline data comparisons, or other procedures to determine postreclamation revegetation success," (State of South Dakota 1988:37), meaning that productivity equations are not necessarily excluded from consideration as a method to demonstrate reclamation success. For postmining landscapes that will contain cropland, "reclaimed land must have the capability of meeting or exceeding the premining crop production of the affected land or of the reference area, if used," (State of South Dakota 1988:44), another requirement facilitating the use of productivity equations. In addition to successfully implementing the development of desired vegetation, the State of South Dakota is concerned with the prevention of noxious weed establishment. The State of South Dakota recognizes a broad range of postmining landuses including forested lands, rangeland, agricultural land, wildlife habitat, recreational lands, industrial use, housing, and future mineral exploration on the site. Besides the landuses that predominantly contain vegetation, other postmining landuses may include the establishment of vegetation and could employ productivity equations. The State of South Dakota has a unique natural resource, the Black Hills, which are mined for their gold and silver. This area is an important visual, aesthetic and recreational resource. To protect the resource, the State of South Dakota governs the activities of large gold and silver mining operations. Vegetation is related to surface mining in the Black Hills of South Dakota because the visual experience within the Black Hills includes vegetation. Therefore the establishment of woody vegetation and native herbaceous plants is an important consideration when preparing a reclamation plan for silver and gold mining operations. The State of South Dakota also requires the reclamation of drill sites employed in mineral exploration (South Dakota 1990c:9-10), and in sand, gravel, and construction aggregate mining (South Dakota 1990d:2). Both activities could employ the use of productivity equations. While the State of South Dakota's reclamation rules and regulations may not be as stringent as those of North Dakota, the intent is similar in both States. South Dakota appears to have addressed a broad range of landscape disturbance activities that may require revegetation, giving the operator flexibility under some circumstances in selecting postmining landuses and in demonstrating reclamation success. The key to South Dakota's landscape reclamation success appears to be identified with the permitting process and the requirement that reclamation must meet the revegetation needs of the postmining land owner, often a South Dakota citizen.

**Minnesota.** Unlike North Dakota, Minnesota contains little coal, and none of it is being mined. Instead, Minnesota has iron ore and taconite mines, sand and gravel mines, quarries, kaolin clay mines, peat mines, and a great potential for nonferrous metallic mineral mines. The metallic mineral mines are governed by Minnesota State Law 1969 C. 774-1, effective 28 May 1969, which was amended by legislation in 1983, C. 270-1 through 4, effective 7 June 1983, to address the reclamation of land subjected to the mining of metallic minerals and peat (Minnesota 1977, 1991). The Minnesota Department of Natural Resources is given authority to write rules concerning the reclamation of metallic-mined and peat-mined lands. Quantitative vegetation productivity equation methods for demonstrating reclamation effectiveness are not required by the State of Minnesota; however, Minnesota does require vegetation standards, of 90 % ground cover on all iron ore and taconite lands that have not been disturbed for 3 years (Minnesota Department of Natural Resources 1980:5703). The Minnesota Department of Natural Resources (1993) has adopted rules for nonferrous metallic minerals. These rules contain reclamation revegetation requirements similar to those for iron ore and taconite mines and facilities. On peatland, the Minnesota Department of Natural Resources (1985:6) states, "During the fourth and fifth year following initiation of revegetation, a vegetated reclaimed area shall have a 75 percent live vegetative cover comprised of wetland or typical peatland species that are either planted or naturally occurring. The vegetation shall be self-sustaining, and either regenerating or in a stage of natural succession. Cover estimates of revegetation shall be measured by standard procedures approved by the commissioner for assessing ground cover and productivity." Revegetation activities on peatlands are required to be compatible with the postmining landuse and surrounding landuses. In Minnesota, local governments (municipalities, townships, and counties) can develop zoning and permitting processes that require bonding and evidence of reclamation success.

While these local governmental agencies have sometimes addressed the permitting process associated with surface mining in their planning and zoning codes, there appears to be no strong concern for stipulating quantitative evidence for demonstrated reclamation success. It appears as though Minnesota's revegetation rules and regulations are not as strenuous and exacting as North Dakota's revegetation rules and regulations; yet satisfactory reclamation has been achieved under Minnesota's system. While quantitative vegetation productivity assessment (other than percent cover) may not be required in Minnesota, vegetation productivity equations could potentially be employed to develop reclamation plans.

**Wyoming.** Wyoming contains substantial coal deposits suitable for surface mining. During the formative development of Wyoming's current reclamation regulations, Munshower and Judy (1988) note that in 1988 Wyoming was in the process of developing criteria to establish overburden suitability. These guidelines are intended to further clarify regulations. The guidelines are summarized in a table presented by Munshower and Judy (1988). The table lists overburden soil parameters that can affect revegetation success and development including: pH, electrical conductivity, percent saturation, texture, sodium absorption ratio, selenium, boron, nitrate and nitrogen, molybdenum, copper, lime, acid-base potential, lead, and arsenic. This approach is based upon the checklist method of analysis where one assumes that as long as each parameter is within an acceptable range, the suitability of the overburden is achieved. This proposed approach (a checklist approach) is quite different from the approach taken by the State of North Dakota. Eventually, a more empirical assessment approach, similar to that of North Dakota, was adopted. The problem with any checklist approach is that there will be no direct physical evidence of the probable suitability of the soil. Nevertheless, this approach is cost effective, requiring relatively simple sampling of the soil to determine the soil's ability to support vegetation. For the State of Wyoming, soil properties, appropriate for a checklist, are described in "Chapter IV: Environmental Protection Performance Standards" (Wyoming Department of Environmental Quality 1989) but are not as exacting as those checklist properties described by Munshower and Judy (1988). In contrast to the checklst method, appendix A of Wyoming's Department of Environmental Quality's document describes vegetation sampling methods and reclamation success standards for surface coal mining operations that are similar in detail (specific equations, reference area approaches) to North Dakota's regulations. It appears as though the State of Wyoming adopted a more quantitative approach, rather than following the proposed checklist approach. Finally, in an inspection of Wyoming's reclamation regulations, vegetation productivity equations are not yet part of the accepted methods for assessing reclamation success.

**Indiana.** The State of Indiana has extensive coal deposits suitable for surface mining. Indiana has enacted legislation to govern the reclamation of coal, clay, shale, and oil shale (Indiana 1990). The Director of Soils and Minerals must approve revegetation plans. Legislation requires that the A and B horizons of prime farmland must be preserved and replaced on the postmining landscape. In addition, the surface of the land must be graded in a uniform manner (Indiana 1990). Hodel vs Indiana, 1981, 101 S.Ct. 2376, 452 U.S. 314, 69 L.Ed. 2d40 (Indiana 1990) is a pertinent court case concerning prime farmland reclamation. The ruling decreed that "Prime farmland provisions of Surface Mining Control and Reclamation Act did not on their face, deprive property owner of economically beneficial use of his property without just compensation in violation of the Fifth amendment. Id. 'Prime farmland' and approximate original contour provision of Surface Mining Control and Reclamation Act did not violate the equal protection or due process guarantees of Fifth Amendment. Id." (Indiana 1990). While the State of Indiana has extensive coal resources and surface mine operations, specific revegetation monitoring programs and assessment methods are not stipulated. Instead, a practical approach is utilized, where topsoil horizons are replaced in land that was formerly on prime farmland.

**Michigan.** While the State of Michigan does not have extensive surface-mined coal lands, the State does address the reclamation of these lands in P.A. 1970 No. 92 sections 425.181 et seq. (Michigan 1978). The surface mine operator must submit plans for approval for coal, gypsum, stone, and metallic ore surface mines (excluding clay, gravel, marl, peat, or sand surface mines). In 1982, the State of Michigan created the Michigan Surface and Underground Mine Reclamation Act, P.A. 1982, No. 303, effective 12 October 1982 (Michigan 1990a), "An Act to regulate the mining of coal..." (Michigan 1990a:187-224) including the reclamation of abandoned mined lands. Article 3 of the Act addresses the permitting process with specific concern for an agricultural impact statement describing the boundaries,

area affected in acres, anticipated future effect, and anticipated time to restore the land (Michigan 1990a). Article 4 of the Act describes performance standards. These standards specifically address the management and postmining reconstruction of the A and B soil horizons and soil toxicity. However, no quantitative procedures addressing vegetation productivity are described in the performance standards. Michigan has an extensive sand dune system, and the State has enacted legislation to protect this resource, the Sand Dune Protection and Management Act, P.A. 1976 No. 222, effective 31 March 1977 (Michigan 1990b). This legislation was amended by P.A. 1989, No. 146-1-Chapter 2 Sand Dune Mining. Essentially, these laws are intended to protect Michigan's scenic sand dune resource; however, portions of these laws are slated for automatic sunset legislation, meaning they are to be repealed in 1995 (Michigan 1990b). In 1995 the State of Michigan will have to decide whether to continue the laws, allow the laws to be repealed, or amend the laws with new legislation.

## Summary and Conclusions

Each State examined in this study has a slightly different perspective concerning the reclamation of surface mined lands. Minnesota, South Dakota, and Michigan all have special landscape environments that each State has decided require special legislation. While North Dakota is specifically concerned about preserving the vegetation productivity of the land, some States are less concerned about demonstrating quantitative vegetation productivity. These States may list performance standards for reconstructing the soil profiles. They may also use checklist procedures and surface mine regulation strictly through the permitting and review process by citizens and experts. Productivity equations appear to be compatible with the SMCRA of 1977, the statutes of the State of North Dakota, and the regulations prepared by the PSC. However, the PSC (1988) document "Standards for Evaluation of Revegetation Success and Recommended Procedures for Pre- and Postmining Vegetation Assessments" will require amendment if these equations are to become an approved method to assess postmining soil productivity. The amendment process would begin by the scientific reclamation community and the PSC technical staff favoring the adoption of such an approach. Productivity equations may not yet be pertinent to the reclamation laws and regulations in other States. However, reclamation regulations may be still undergoing formative development, meaning that the use of productivity equations may eventually be widespread and incorporated into the rules and regulations governed by State reclamation boards.

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# THE DEVELOPMENT OF A FIELD METHOD FOR EVALUATING THE SUCCESS OF RECLAMATION EFFORTS ON ABANDONED MINE LANDS[1]

Ms. Elizabeth L. Hunsberger and Dr. Louise H. Michaud[2]

**Abstract:** Abandoned Mine Lands (AML) are prevalent throughout Pennsylvania and in other areas of the US. Reclamation of these sites has been an ongoing concern of the Pennsylvania Department of Environmental Resources (PA DER) for over 20 yr. As the state of the technology improves, a variety of techniques have been utilized to rehabilitate AML. These reclamation efforts have resulted in vast improvements in the conditions of the sites, especially in water quality, erosion control, and aesthetic beauty. However, little work has been done to evaluate and document the success of individual reclamation techniques.

Working with the Bureau of Abandoned Mine Reclamation, PA DER, a study was conducted at The Pennsylvania State University to address this need. The main goal of the project was to develop an evaluation system that could be easily carried out in the field by one person. The result of this study was the development of the Reclamation Success Evaluation System (RSES). The system utilizes three main parameters to evaluate reclamation success: (1) Surface Water Quality, (2) Extent of Erosion, and (3) Success of the Vegetative Cover. A series of guidelines and recommendations was developed for each of these evaluation parameters.

The RSES was tested under field conditions by applying it to a watershed that contains both reclaimed and unreclaimed AML sites. This test proved that the RSES is an easily implemented and effective tool for evaluating the success of AML reclamation efforts. The system facilitates the comparison of reclamation efforts at different sites, it can be conducted by one person, and the results are easily interpreted.

**Additional Keywords:** acid mine dainage, revegetation, remediation.

## Introduction

Abandoned Mine Lands (AML) pose a serious threat to the health and safety of humans and to the environment. Because of the direct hazards, the PA DER, Bureau of Abandoned Mine Reclamation (BAMR) has been working to reclaim abandoned mines lands for the last 20 yr. These reclamation efforts have resulted in vast improvements in the conditions of AML sites, especially in areas such as water quality, erosion control, and aesthetic beauty. However, no system exists that can be used to evaluate the long-term success of these reclamation efforts. Such a system would allow the BAMR to evaluate the effectiveness of its past reclamation efforts and to modify its reclamation techniques in order to improve the efficiency and success of its operations.

Working with the BAMR, The Pennsylvania State University developed a system to evaluate the success of AML reclamation efforts. This evaluation system is entitled the Reclamation Success Evaluation System or RSES. The development of the RSES is discussed in this paper. The field testing of the system and the results of the application will be the subject of a future paper.

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## Development of the Reclamation Success Evaluation System

### Guidelines for Developing the Evaluation System

To develop an effective evaluation system, several important guidelines were set forth. It was determined that the RSES must be easily performed by one person in the field. This requirement is necessary owing to the limited availability of BAMR staff to conduct site evaluations. Also, the varied educational backgrounds of the potential investigators must be taken into consideration. Secondly, because of the nature of AML and reclaimed areas, any necessary equipment must be easily transportable over potentially rough terrain. Thirdly, the RSES should not require a lengthy period of time to acquire the data, the results must be reliable and statistically valid, and the data should not require extensive mathematical manipulations. Overall, simplicity and reliability are the keys to an efficient evaluation system.

**Selecting the Evaluation Criteria for the RSES.** After carrying out an evaluation of the common disturbances that result both directly and indirectly from mining activities, it was determined that several possible criteria could be used to evaluate the success of a reclamation program. These include surface and ground water quality, changes in surface and ground water quantity, stream sedimentation, suspended solids concentration, surface erosion, soil quality, and the characteristics of the vegetative cover.

After evaluating each of the potential disturbances and assessing the requirements of an effective evaluation system, three evaluation criteria were chosen for the RSES. These are named: (1) Surface Water Quality, (2) Extent of Erosion, and (3) Success of the Vegetative Cover.

The first criterion, Surface Water Quality, is an important key to the success of reclamation and gives indirect clues to the state of the other two parameters. The second criterion, the Extent of Erosion, has broad implications with regards to soil and slope stability, stream pollution, and potential post mining land productivity. As with surface water quality, minimizing erosion is regarded as a crucial goal of surface mine reclamation efforts (Dollhopf *et al.* 1977, Hodder 1975, Toy 1989, Toy and Hadley 1987).

The final parameter, Success of the Vegetative Cover, is linked to the other parameters through the site hydrology and soil stability. Good vegetative cover results in reducing the volume of surface runoff, increases soil and slope stability, and leads to the formation of an organic layer. Also, a lush vegetative growth is visually correlated with successful reclamation. Although questions have been raised concerning the use of the vegetative cover as an effective measurement of reclamation success (Lusby and Toy 1977), this parameter was selected for inclusion because it provides a simple and effective measure of productivity.

Several other parameters could potentially be as effective, or more so, for evaluating the success of reclamation than the three that were ultimately selected for the RSES. Among others, these include soil microbial population, vegetative species, land use, and ground water chemistry. However, the purpose of the RSES is to provide a rapid method of evaluating the success of the reclamation efforts. Most often, these alternate parameters cannot be easily evaluated within the guidelines of this system. They require extensive time, cost and/or resources for their implementation, measurement and evaluation. The RSES can be used as an initial indicator of potential problem and, where required, a further, more detailed evaluation of a property can then be carried out.

### Development of the Surface Water Quality Evaluation System

To examine the relationship between reclamation and surface water quality, it was necessary to decide which chemical analyses are required and to establish a detailed field sampling methadology. Because this project was conducted under the auspices of the DER, their laboratory facilities were used to perform all water quality testing. Three major series of DER water sample tests are relevant to AMD pollutants. These are the Basic AMD Analysis (pH, acidity, alkalinity, sulfate, total iron, and ferrous iron), the Basic AMD Analysis plus

Selected Metals (all of the above plus aluminum, manganese, and hardness as total $CaCO_3$), and the Drinking Water Analysis (all of the above plus arsenic, barium, cadmium, chromium, mercury, $NO_2$ total, $NO_3$ total, selenium, lead, copper, zinc, and chloride). The specific laboratory procedures used to determine these parameters can be found in "Bureau of Laboratories Methods Manual, Volume I" (Department of Environmental Resources 1992).

To collect the maximum amount of data related to the water quality, the Drinking Water Analysis was chosen for this study, although this degree of thoroughness may not be necessary for all projects. Most projects will probably only require the Basic AMD plus Selected Metals Analysis.

**Description of the Surface Water Quality Evaluation System.** The surface water quality of a reclaimed area can be evaluated by several different methods. First, it can be compared to standards set by State and/or Federal governments. This could include the US Environmental Protection Agency's (EPA) Drinking Water Standards, the DER's Mine Drainage Discharge Standards, or the Pennsylvania Fish and Game Commission's Stream Classification Criteria. Second, a comparison can be made between historical pre-reclamation and post-reclamation water quality. Finally, a comparison can be made between the water quality upstream and downstream of the project and of specific sites within the project. The optimum method or combination of methods that should be used to compare water quality data for a specific project is dependent on the specific project's field conditions and availability of data.

The first method of evaluation, comparing post-reclamation samples to standards, could be useful when reclamation efforts were initiated to solve a specific water quality problem. The second method involves comparing pre-reclamation and post-reclamation water qualities. Where pre-mine data are available, this method provides a simple and clear analysis of any changes in the water quality over time. Locating historic water quality information and assessing its reliability can, however, be time consuming. It is therefore important to decide what information is necessary for a specific project and how far back in time the data must be searched prior to completing the search.

Possible sources of historic water quality information include National Abandoned Land Inventory System-NALIS (computer database administered by the DER for Pennsylvania); Pennsylvania's Fish and Game Commission; mining permit files, Mining and Reclamation Regional Offices, DER; office files, BAMR, DER (regional offices often have more thorough water quality data than can be found on NALIS); the US Geological Survey (USGS), and NAWDEX (computer data administered by the USGS).

Additional concerns for comparing pre-mining and post-mining water quality include the facts that this method ignores any external impacts on the system, does not take into account the long term or seasonal variations in precipitation or ground water quality, and does not account for the potential impact of new mining activities in the area. These concerns notwithstanding, where historical water quality is available, this method of comparison offers a good picture of how water quality has changed in response to mining and reclamation activities.

The third method involves comparing water samples from points upstream and downstream of the individual sites in a project. This comparison isolates the water chemistry changes that can be attributed to conditions at each site, including any mining, industrial, or human activities that may influence the surface water quality. The upstream and downstream water quality data can also be compared with any reliable historical data.

**Application of the Surface Water Quality Evaluation System**

The Surface Water Quality Evaluation System for the RSES is conducted by carrying out the following steps:

1. Delineate and map the watershed boundaries (USGS watershed boundaries).

2. Locate and map all known mining areas (or sites): abandoned, active, and reclaimed. These data can be compiled from USGS maps, DER AML Inventory Overlays, the DER Bureau of Mining and Reclamation's Mining Permit Maps, and any other available sources.
3. Search government and mine documents for historical water quality data. These data are used to identify potential sampling point locations. 4. Carry out a field investigation of the project area to confirm the data. Map any undocumented waterways, seeps, and discharges. Other sources of water contamination sources (raw sewage, illegal dumps) should also be recorded.
5. Determine field sampling point locations. Samples should be collected at the headwaters and mouth of the project's main streams, upstream and downstream of each distrubed sites, and from all impoundments, seeps and discharges. Historic sampling point locations should be used wherever possible. In areas that are heavily mined on both sides of the stream, it may be extremely difficult to establish sampling points that will isolate and evaluate the effects of mining operations on one side of the stream. All sampling point locations should be clearly marked in the field and noted on the project map.
6. Collect water samples from all established sample point locations. The water samples should be analyzed for the specific analysis series necessary for the particular project.
7. Analyze the surface water quality data with respect to one or more of the three evaluation methods.

**Development of the Extent of Erosion Evaluation System**

Evaluating the extent of erosion is a common method of determining the success of reclamation on numerous types of disturbed lands (De Boodt and Gabriels 1978, Schaller and Sutton 1978). In fact, the Surface Mining Control and Reclamation Act of 1977 requires that coal mining companies prevent erosion on their sites and that the site be stable (erosion free) prior to the release of the company's bonds.

The dominant erosional force at abandoned and recently reclaimed mining sites in the Eastern US is water, resulting in sheetwash, rills, gullies, and slumping. Sheetwash is the even removal of a thin layer of soil from a fairly broad area of land by a "sheet" of water that does not create channels. This form of erosion occurs in areas that lack ground cover and quickly leads to the formation of rills and gullies. Rills are very small channels where soil material has been removed due to the concentrated flow of water. Gullies are larger erosional channels where soil and larger material has been eroded away. Slumping is an erosional feature that occurs on a slope due to saturated, unstable soil conditions. Technically, it is the downward movement of a soil or rock mass, usually with a backward rotation on a horizontal axis, parallel to the slope from which it descends (Bates and Jackson 1984). Other, more serious erosional features that are less common at mine sites, but more dangerous, include mud flows, landslides, and debris flows.

Several techniques exist to measure the extent of erosion on a site, including: (1) soil erosion mapping, (2) hill slope profile mapping, and (3) the evaluation of sites with the aid of a visual rating system.

**Soil Erosion Mapping.** Soil erosion mapping, a method commonly used in agriculture, involves mapping either the actual erosional features or the potential soil erosion hazards. Data collection for this technique utilizes satellite photography, aerial photography, or field surveys. This form of evaluation provides a good visual representation of the erosional features and is adaptable to any scale. However, the method is time consuming and expensive, and its effectiveness has not yet been determined with respect to coal mining and reclamation. Also, the method does not allow an easy comparison of different project and sites, since the results are purely visual. For these reasons, soil erosion mapping did not meet the guidelines of the RSES.

**Hill Slope Profile Mapping.** Hill slope profile mapping, which looks at cross-sectional views of a hill slope, have been used to assess erosional problems in numerous mining studies (Elliot 1990, Haigh 1980, Haigh and Wallace 1982, Toy 1989). It is an excellent method of examining the relationship between the extent and severity of erosion and hill slope features such as slope grade and concavity. This method requires several surveys of the slope profile over time in order to measure any changes. Owing to extensive time and work requirements, this method was ruled out for use in the RSES.

**Visual Rating System Evaluation.** The final method, a visual rating system, uses a scale to classify the extent and/or type of erosional features found on a specific site. Although numerous articles are available that detail the use of a rating system to identify potential soil erosion hazards or site erosivity (Bollinne *et al.* 1980, Hallsworth 1987, Morgan 1980), few extend this method further by using it to measure present erosional features (Schwing and Vogt 1980). The method assigns a rating, or classification, to each site or sub-site in a project. A simple example of the classifications would be to place each site into one of four classes: (1) no erosion, (2) limited erosion, (3) moderate erosion, and (4) extensive erosion. Each of the classifications is then assigned a color and mapped onto a project map for visual interpretation. An example of this type of map can be seen in Figure 1 (symbols are utilized instead of colors on this map). Depending upon the detail required for the project, the ratings can be assigned by carrying out a field evaluation of the sites or by the analysis of aerial photographs.

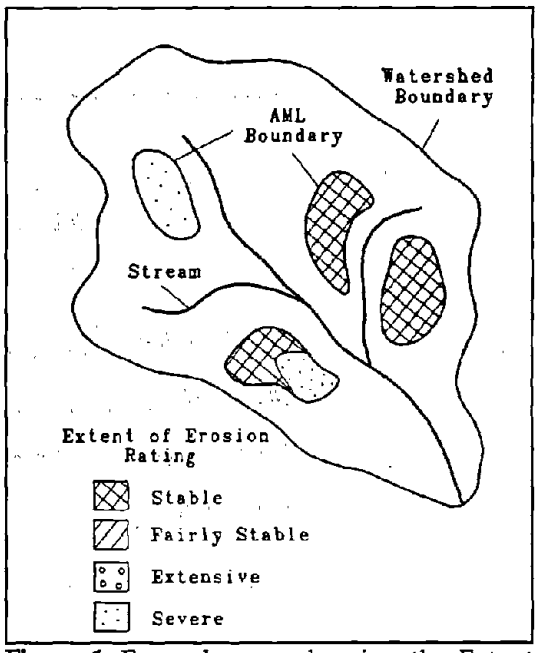


**Figure 1.** Example map showing the Extent of Erosion Evaluation results.

**Application of the Extent of Erosion Evaluation System**

Because of its simplicity and ease of evaluation, the Extent of Erosion Evaluation section of the RSES employs the use of a visual rating system. To ensure a uniform evaluation between sites, an Extent of Erosion Evaluation Form was developed. This form (Figure 2), which is self-explanatory and easy to complete, is completed as each site as it is examined in the field. In addition to completing the form, the location of each site should be noted on the project map, and photographs should be taken for future reference. The number of erosion evaluations required for a project will vary depending on the size of the project area and on the detail required for the study.

The extent of erosion at a project can be evaluated by one of three methods: (1) comparison to a nearby undisturbed site, (2) comparison to a nearby abandoned, unreclaimed site, or (3) evaluation of the information on its own. Where historical information is available, the Extent of Erosion data should also be compared to historical records for pre-mining and/or pre-reclamation conditions.

**Development of the Success of the Vegetative Cover Evaluation System**

The success of the vegetative cover is an important aspect in reclamation because of its impact on other parameters such as soil movement and surface erosion. One method of decreasing erosion is through the interception of raindrops by the vegetative cover. Plants absorb some of the raindrop's kinetic energy, thereby reducing the energy that is available to dislodge soil particles. The degree to which the vegetative cover is effective at reducing erosion is a function of the height and continuity of the plant canopy, the density of the ground cover, and the root density (Morgan 1986). The vegetative cover also dissipates the energy from surface water runoff, thereby decreasing the related erosional force. Since erosion force varies from a cube to a fifth power of velocity, the presence of vegetation is extremely important factor in inhibiting erosion (Morgan 1986).

An increase in the vegetative cover also results in an increases in both the evapotranspiration rate and the infiltration rate (Schwab *et al.* 1981), leading to changes in the water budget. Finally, wildlife diversity and populations respond positively to an increase in available habitat and food supply that is brought on by the establishment of a successful vegetative cover.

**Parameters Used To Measure the Vegetative Cover.** For the development of the RSES, two decisions had to

EXTENT OF EROSION EVALUATION FORM

RSES

INVESTIGATOR_____DATE_____

LOCATION_____

SLOPE LENGTH & DEGREE_____

DESCRIPTION_____

_____

_____

_____

EROSION RATING

| 1 | 2 | 3 | 4 |
|---|---|---|---|
| SEVERE | EXTENSIVE | FAIRLY STABLE | STABLE |

EROSION RATING DEFINITIONS:

SEVERE: More than 70% of the site affected by erosional features (gullies, rills, slumps). Erosional features dominate the landscape. Features do not have vegetation associated with them. In general, they extend the entire slope length. They appear to actively eroding or of recent origin.

EXTENSIVE: More than 30% and less than 70% of the site affected by erosional features. Erosion is an active or recent process. Vegetation is not associated with features.

FAIRLY STABLE: Less than 30% of the site affected by erosional features. Vegetation may be established along rills and gullies. Erosional features do not dominate the landscape--they are not visible from a distance.

STABLE: Erosional features, if present, are vegetated and do not appear active.

**Figure 2.** Extent of Erosion Evaluation Form for the RSES.

be made with respect to measuring vegetation success: the specific parameter to be measured and the method of measurement. The most common parameters measured in other studies include the percentage, density, and frequency of the vegetative cover (Holmberg 1983). The percentage of the vegetative cover refers either to the percentage of ground surface under live aerial parts of plants (the crown cover) or the aerial parts plus the mulch (the basal cover). Density of the vegetative cover refers to the number of plants per unit area. Frequency of the vegetative cover is the number of plots with a specific plant species divided by the total number of plots (Holmberg 1983).

The frequency and density parameters were ruled out for use with the RSES because of the specialization, time and research required to inpliment them. Consequently, the percentage of vegetative cover was chosen as the parameter to measure for the success of the vegetative cover. The Notched Boot Method was selected for determining ground cover percentage because this method does not require any equipment other than paper, a pencil, and footwear to perform and produces reliable results that are easily interpreted.

**Application of the Vegetative Cover Evaluation System**

The directions for the Notched Boot Method as outlined in Oleson (1981) and carried out by Office of Surface Mining (1987) follow:

> The examiner makes a "point" - a white mark or notch - about 1/8 inch wide on the tip of one shoe sole. A course is selected, preferably a straight line, which will cross an average or representative part of the sampling area. A transect usually consists of 100 paces. If there is insufficient space to complete the 100 pace transect in a straight line, it may be folded back and forth across the slope. At each two steps (one pace) a hit is recorded if the shoe sole point touches vegetation and a miss is recorded if bare ground is encountered. The average of more than one transect will increase the accuracy of the results.

For this method, only living and dead plant matter that is in direct contact with the soil is recorded. The hits and misses are then transformed into a percentage of the vegetative cover and the results recorded directly onto a field map along with the location of the transects. Unless there is a large diversity in the apparent vegetative cover, it is suggested that one transect be performed for every 10 acres.

For interpretation, these percentages are then grouped into relevant classification ranges. If all areas in a project contain over 80% vegetative cover, for example, the ranges may be 80-85%, 86-90%, 91-95% and 96-100%. However, in areas where the vegetative cover ranges from 0% to 100%, the classification system must be adjusted to include the total range.

As was done with the Extent of Erosion Evaluation, the classification ranges are then assigned a color code to facilitate interpretation and comparison, and the color for each site is recorded on the project map. Figure 3 is an example map of the results (using symbols instead of colors). Because historical data about past vegetative characteristics are usually very limited, information should be gathered from nearby undisturbed sites and abandoned sites for comparison purposes.

**Success of the Reclamation Success Evaluation System**

To test the applicability of the RSES in the field, it was used to evaluate the reclamation efforts of a watershed in central Pennsylvania. A test case was chosen that contained several unreclaimed abandoned mine sites, active mining sites, and sites that have been reclaimed by BAMR and previous mining companies. The present poor water quality in this area indicated
that AMD is being generated at the mined and/or reclaimed sites.

The RSES proved to be an effective tool for assessing the success of reclamation efforts at the test project. The guidelines that were set forth during the development of the RSES were achieved: (1) The RSES

was easily performed by one person in the field, (2) no heavy or cumbersome equipment was required to perform the evaluation, (3) the evaluation system produced reliable results, and (4) the system did not require an extensive amount of time to perform. The major exception to was the time necessary to locate and analyze the historical water quality data, although locating this data becomes easier with time and experience.

Despite the successes, several minor implementation problems become obvious when the RSES was tested in the field. This led to several modifications of the evaluation system.

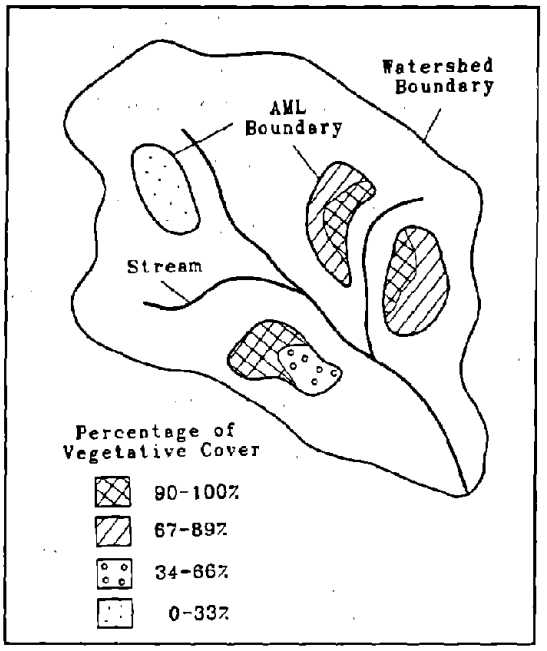


**Figure 3.** Example map showing the Success of the Vegetative Cover Evaluation results.

**Success of the Extent of Erosion Evaluation**

The Extent of Erosion Evaluation was extremely successful at fulfilling the RSES guidelines. The largest weakness is the qualitative, visual nature of the measurement. This could lead to wide variations in the results obtained by different investigators. To help minimize the deficiencies associated with the visual rating system, the description of the ratings on the Extent of Erosion Evaluation Form were made more exact, leaving less room for individual variations in interpretations.

**Success of the Vegetative Cover Evaluation**

The percentage of vegetative cover as the measurement parameter was extremely successful at meeting the RSES guidelines. The Notched Boot Method provided excellent, clear results which are easily interpreted, and it minimized the equipment necessary for the field study. This was important since access to certain areas within the field test site was extremely difficult. The use of color coding provided a system that clearly presents the results for interpretation.

**Success of the Surface Water Quality Evaluation**

The Surface Water Quality Evaluation was the most complex of the three RSES evaluation parameters, involving the collection of historical water quality data, the establishment of sampling locations, the collection of samples, and the comparison of the data. When this evaluation was tested, a major drawback became apparent - the historical data were difficult to find and to interpret. However, despite its difficulties, this step is necessary to effectively evaluate changes in water quality.

Originally, the RSES used a Surface Water Quality Evaluation Sheet, but re-recording data from the laboratory reports was found to be too cumbersome. As an alternative method, the present system was implemented where the sampling point locations are marked on the project map and the water quality data are grouped and presented in tabular form.

## Future Development and Uses of the RSES

The development of the RSES involved analyzing commonly used field practices and adapting the most suitable to the specific requirements of an effective evaluation system. The RSES was designed to be easily performed, yet to be thorough and reliable. These requirements are not easily fulfilled under all potential situations. The RSES, at its present stage of development, does have shortcomings, as any new evaluation

system will.

One shortcoming is the lack of consideration given to seasonal and yearly changes within a study area. To alleviate this problem, if time allows, water samples should be collected over a calendar year at regular time intervals, in order to provide a more distinct profile of the seasonal changes in surface water quality. This detail in the evaluation system was beyond the scope and guidelines of the present version of the RSES.

Similar seasonal variations also exist in the vegetative cover. Numerous events can also affect the results of a field investigation, including drought, a harsh winter, variations in plant species, recent seeding or planting of the area, seasonal variations in plant growth, and insect infestation. For a more detailed study, these events need to be taken into consideration.

The test watershed contained a wide variety of conditions that were useful in evaluating the RSES under varying conditions. However, for further development and testing of the RSES, it should be applied to other AML and reclaimed sites. This will further test its effectiveness and reliability under different conditions.

Several improvements to the system are also possible that would make it more efficient. The Surface Water Quality Evaluation System presently lacks a method of categorizing the overall effect of reclamation on water quality. Further research is needed to determine if such a system can be developed and implemented into the RSES, while still meeting the established guidelines. Another possible refinement of the RSES would be the integration of the agricultural technique of Soil Erosion Mapping into the system. This technique shows the potential for providing an effective method of evaluating erosional feature due to mining activities. Future work should also be carried out to examine any correlations between the Success of the Vegetative Cover and the Extent of Erosion results.

There are several potential applications of the RSES to measure the success of reclamation efforts of AML. Informal field investigations have shown that revegetation and control of erosion on AML do not always result in an improvement in surface water quality. This is especially true in areas where AMD is prevalent. An evaluation system such as the RSES could be used to quantitatively verify this informal theory on a broad scale. The results from these site evaluations could then be compared to the reclamation techniques used on the specific sites. This would allow the identification and expanded use of the specific reclamation techniques that are the most successful at improving surface water quality. Conversely, unsuccessful techniques could be eliminated from reclamation practices.

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# CONTINUOUS LIFT PIERS: DAMAGE REPAIR AND RESPONSE DURING SUBSIDENCE[1]

Eric C. Drumm[2], Richard M. Bennett[2], Guoming Lin[3],
David B. Raaf[4] and Dean Daugherty[5]

**Abstract:** A series of test foundations damaged during a longwall mining operation were left with permanent tilt, curvature, and substantial cracking. Two of the foundations were releveled using continuous lift piers. The continuous lift piers removed the tilt and curvature and significantly reduced the width of the foundation cracks. Following the releveling of the foundations, an adjacent longwall panel was mined, resulting in additional subsidence, although of smaller magnitude. The response of the releveled foundations was monitored and compared with the response of a footing that was not releveled. Although the continuous pier system does not strengthen the structure and distortion was observed during the second event, the system permitted the deformations to be removed in a few hours.

## Introduction

Ground subsidence due to underground mining activities is a significant problem affecting mine operators, the insurance industry, government agencies, and property owners. In the United States, damage to residential structures from underground mining is estimated between $25 and $35 million each year (Gray 1988). With approximately 5.2 million acres of abandoned or inactive coal mines, of which 500,000 acres are in populated urban areas (Dyni and Burnett 1993), damage repair techniques applicable to residential and light commercial structures must be developed. Effective damage remediation methods can only be developed with a good understanding of the mechanisms causing structural distress. Since structures in subsidence prone areas are often subjected to multiple or recurring ground deformation events, the response of repaired structures during subsequent subsidence events must also be evaluated.

Damage to structures from subsidence occurs primarily over inactive or abandoned room-and-pillar mine operations and may occur many years after the coal has been extracted. Since modern longwall techniques result in subsidence that is immediate and can be predicted with some degree of certainty, structural damage can often be reduced or eliminated by premining measures. Although damage due to longwall mining is not as significant an economic issue as damage from abandoned mine operations, the controlled nature of longwall induced subsidence provides a good opportunity to observe the mechanisms governing structural damage.

The response of structures to mining induced subsidence is complex. Structural behavior and damage are dependent upon the relative properties of the foundation and underlying soil, and depend significantly on the magnitude of the ground movements. This coupling of soil and structural response is known as Soil-Structure-Interaction (SSI). Monitoring full-or reduced-scale structures during subsidence has provided useful observations

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regarding damage and SSI effects (Walker 1987, Powell et al. 1988). In a more recent series of field tests (Bennett et al. 1992), some test foundations damaged by subsidence were releveled using continuous lift piers. To evaluate the application of continuous lift piers for subsidence damage remediation, the response during a subsequent subsidence event was observed.

## Test Foundations

Six test foundations were constructed over a longwall panel in West Frankfort, IL (Kane et al. 1990) with the undermining to take place in 1990. The six foundations were built to simulate typical residential or light commercial construction. Constructed of identical size, the six foundations differed in the method of footing construction. Only three of the foundations will be described here: the plain concrete footing, the reinforced concrete footing, and the post-tensioned concrete footing. Details of the other foundations are described by Bennett et al. (1992). A schematic of the test foundations is shown in figure 1, with the residential loading simulated by soil-filled load bins placed on top of the footings. The foundations were built in the anticipated zone of maximum tension, near the edge of the 1990 panel. The location of the foundations relative to the first (1990) and second (1991) longwall panels, is shown in figure 2. The coal seam was about 2.3 m thick, at a depth of about 160 m. A firm layer of shale was located at a depth of approximately 7 m. The longwall panel was 280 m wide and 1950 m long.

## Response to Initial Subsidence Event

The maximum subsidence from the first panel measured at the panel centerline was 1.43 m, while the subsidence in the maximum tension zone near the test foundations ranged from 0.1 m toward the edge of the panel to 0.3 m toward the centerline of the panel. Footing deformation was measured with a precision survey level and tiltmeter. The maximum differential settlement of the plain concrete footing was 0.170 m, which resulted in an average tilt of 0.0139 radians. The reinforced concrete and post-tensioned concrete footings sustained similar deformations. Figure 3 compares the cracks observed in each of the three footings. The plain concrete footing suffered the most damage, cracking in three locations with the maximum crack being 18 mm wide. The post-tensioned footing did not crack and the deformed bar reinforced footing developed small cracks. The block walls

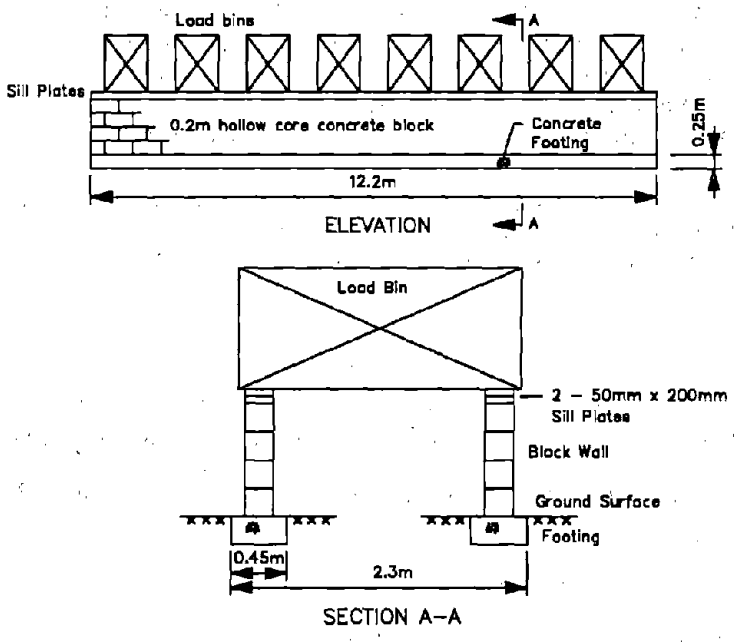


**Figure 1.** Schematic of test foundations and load bins.

cracked at the locations over the footing cracks, and the wall cracks were slightly wider than the footing cracks. The deflection ratio (DR) is defined as the maximum footing deviation from a straight line divided by the foundation length. Values for deflection ratio and average curvature for the test footings and surrounding soil are summarized in table 1.

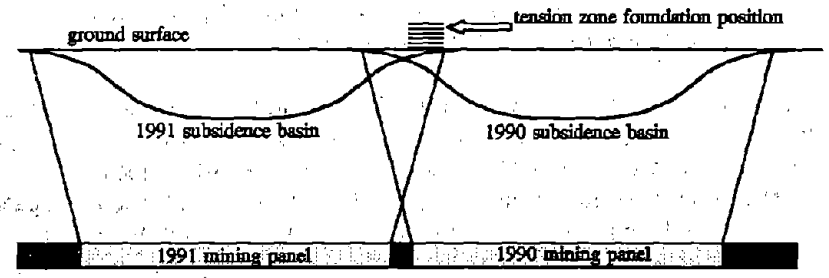


Figure 2. Relative location of tension zone foundations with respect to 1990 and 1991 mining panels.

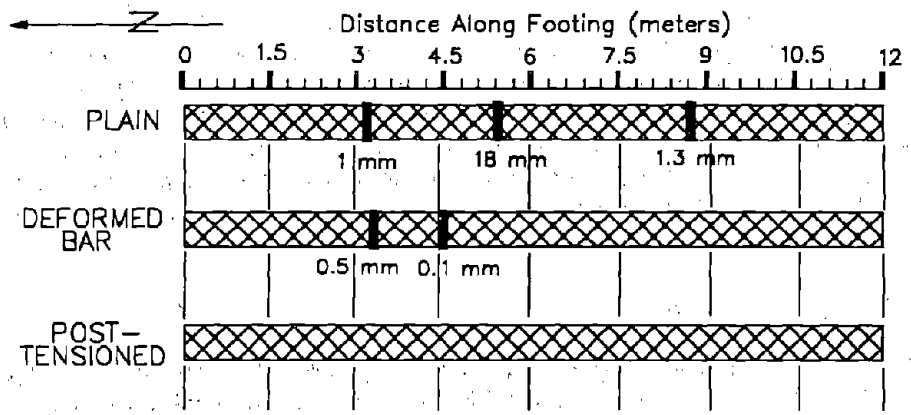


Figure 3. Cracks in foundations due to first subsidence event.

Table 1. Deflection ratio and curvature due to first subsidence event

| Footing | Ground | Plain Concrete | Reinforced Concrete | Post-Tensioned Concrete |
|---|---|---|---|---|
| Deflection Ratio | -0.00386 | -0.00248 | -0.00233 | -0.00212 |
| Curvature (1/m) | -0.00148 | -0.00160 | -0.00154 | -0.00138 |

## Foundation Restoration with Continuous Lift Piers

After subsidence, the plain concrete and reinforced concrete footings were releveled using a system of continuous lift piers (Atlas Systems, Inc. 1990). Once installed, continuous lift piers permit the foundation to be raised or lifted over a wide range, with a high degree of control. The system also permits additional adjustments to the foundation should ground movements occur in the future. The Atlas pier system is a proprietary system (Atlas System 1990), and the installation is described in terms of this method. Other systems with similar capabilities exist, but have not been demonstrated on subsidence-damaged structures.

Releveling the foundation can correct the effects of subsidence such as residual tilt and curvature, which may cause sticking doors and windows, and can return the structure to service. However, the continuous lift pier system does not increase the structural capacity of the foundation and does not stiffen cracked elements. The method should be compatible with structural measures (Marino 1992) to restore the structural capacity. The goal of this investigation was to determine how a subsidence-damaged structure, restored to service with continuous lift piers, would respond to a second subsidence event, and to demonstrate the ability to relevel the foundation after the second event. Foundation restoration with the continuous lift pier system consists of two phases: the installation phase, and the lift phase.

### Continuous Lift Piers - Installation Phase

The footing is excavated at the pier locations to provide access to the sides and bottom. If cracks exist in the footings, steel angles or plates are bolted to the footing to maintain continuity across the cracks during the leveling process. A temporary pier bracket is then bolted to the footing, as shown in figure 4. The cylinder drive assembly is then installed to hydraulically push the pier sections down to a firm bearing strata. If the firm layer is deep or nonexistent, friction must be developed along the mini-piles. To provide adequate reaction for the hydraulic driving forces, supplemental weights can be added to the foundation, or reaction can be obtained from helical outriggers as shown in figure 4. For the test foundations, five continuous lift pier systems were installed on each of the two foundations. The pier sections were pushed to the firm strata at a depth of about 7 m from the original ground surface. Reaction for the pier jacking was provided by supplemental weights added to the top of the foundations.

### Continuous Lift Piers - Lift Phase

After the pier is in place, the supplemental weights, cylinder drive assembly, outriggers, and helixes are removed, and the temporary pier bracket is replaced by the continuous lift bracket assembly (fig. 5). A hydraulic lifting jack is installed at each pier location, and the jacks connected to the hydraulic pump through a manifold. The manifold system permits the simultaneous adjustment of each of the piers to prohibit additional structural distress and ensure that the foundation is releveled.

Using the manifold system, the footings were systematically raised and the foundations leveled. The maximum differential settlement before leveling was 0.24 m measured on March 21, as shown in figure 6. After leveling, June 10, the foundation was essentially level, with the south end slightly higher than the north end. The maximum difference was 0.0524 m on June 10. The footing could have been completely leveled, however, adjustments were made until the sill plate was level within 3 mm. Crack widths in the footings and block walls decreased after leveling. The adjustment heads were left in place to facilitate leveling after the second subsidence event. However, if additional ground movement is not expected, the lifting heads can be removed to restore the original foundation appearance. A significant advantage of the continuous lift pier system is that if additional deformation occurs, the footing can be releveled or adjusted. The installation and releveling were completed in two days. If the helical outrigger system had been used to provide the reaction, the installation period would have been significantly shorter than that required with the supplemental weights.

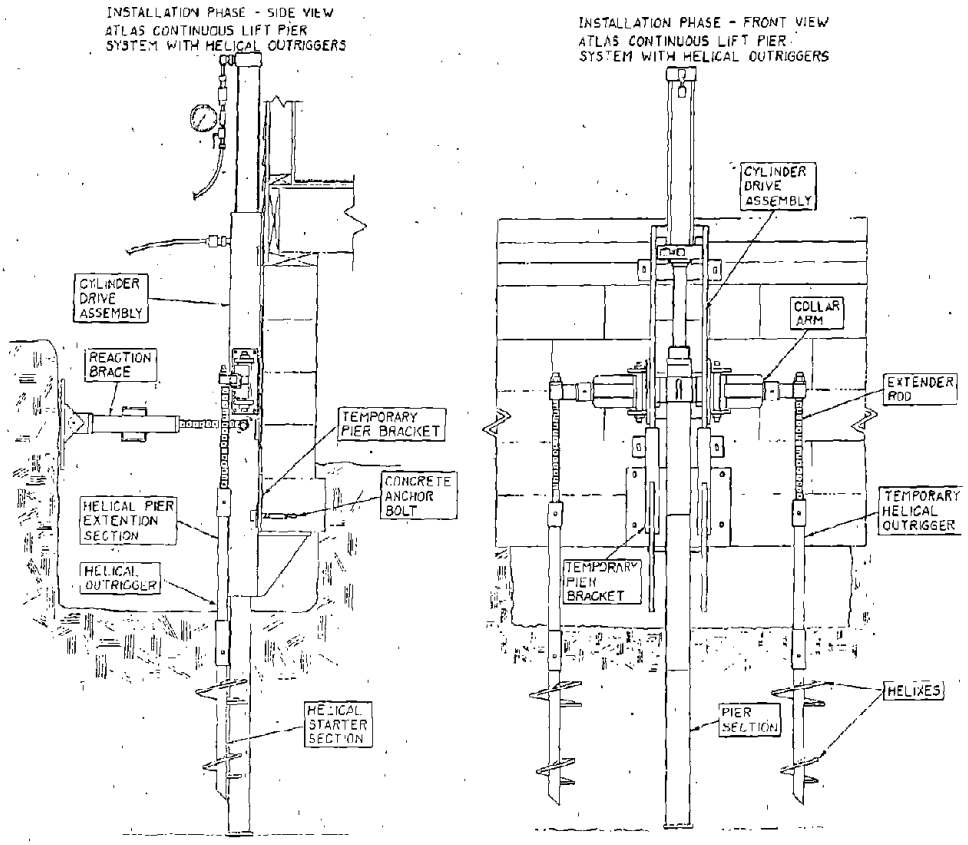


Figure 4. Installation phase-continuous lift pier system with helical outriggers.

## Response of Restored Foundations During the Second Subsidence Event

The foundations were monitored during the second (1991) subsidence event with a precision level and tiltmeter. The deformation of the nearby ground was measured with a level and total station. Subsidence from the first (1990) panel was essentially complete by August 1990, and subsidence from the 1991 panel was not significant until late June 1991. The subsidence of the reinforced concrete footing is shown in figure 6. Maximum subsidence was about 0.26 m, with slightly more settlement at the north end than the south. Unlike the first subsidence event, the second event yielded nearly uniform ground deformations, resulting in minimal curvature in the foundations. During the 1991 subsidence, the foundations were located just outside the zone where maximum tension is expected, as shown in figure 2.

The variation in the deflection ratio over time is shown in figure 7 for each of the three footings. All three footings experienced a significant increase in DR during the second subsidence event, but the reinforced concrete and post-tensioned footing deformed less than the plain concrete footing (table 2). This difference was observed during the first (1990) subsidence event also, except the deflection ratios were much greater, about -0.002. It can be concluded that since the continuous lift piers did not increase the structural rigidity of the footings, they did not improve the resistance to deformation during subsidence. However, since the reinforced footing with continuous lift piers and the post-tensioned footing without continuous lift piers responded in a similar manner, the continuous

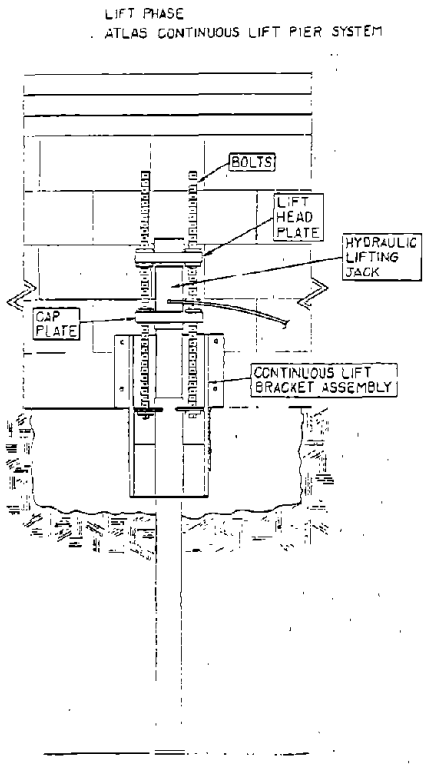


Figure 5. Lift phase-continuous lift piers.

lift piers did not exacerbate the subsidence deformations. Once the continuous piers are in place, the foundation can readily be restored to service.

Tilt of about 0.002 radians was measured at the north end of the reinforced footing, and 0.005 radians at the south end. Except for the plain concrete footing, which had some tiltplates damaged during the pier installation process, similar values were obtained for the other footings. The tilt was calculated with respect to June 10, 1991. The majority of the tilt and deformation occurred between June 19 and July 18. Since the change in tilt was nearly constant, the curvature was assumed to be constant. Curvatures were calculated by obtaining a slope from a linear regression of the tilt values along the footings, and are summarized in table 2. The curvatures experienced by the 1991 event were about one fifth that experienced during the 1990 event, which is consistent with the location of the footings relative to the longwall panel. The footings were constructed within the zone of anticipated maximum tension for the 1990 panel, but were about 50 m away from the zone of maximum tension for the 1991 panel.

According to a damage criterion proposed by Bhattacharya et al. (1984), a brick or masonry structure is liable to suffer functional damage at a deflection ratio of 0.0005 or a curvature of 0.00005 (1/m). The actual relationship between curvature and deflection ratio of a structural element depends on the length of the structural element. For the test footings, it was determined (Lin 1993) that the curvature is about two-thirds of the deflection ratio. The measured deflection ratios and curvatures during the 1990 and 1991 subsidence events also verified the relationship. The deflection ratios caused by the 1991 subsidence were around 0.0003, and no significant additional damage was observed. Therefore, the above proposed criterion in terms of deflection ratio was more appropriate, while the curvature value was overly conservative for the footings.

Table 2. Final curvatures and deflection ratios After second subsidence event.

| Footing | Plain Concrete with Atlas Pier | Reinforced Concrete with Atlas Pier | Post-Tensioned Concrete Not-Repaired |
|---|---|---|---|
| Deflection Ratio | 0.00046 | 0.00029 | 0.00028 |
| Curvature (1/m) | NA | 0.00026 | 0.00019 |

NA = Not available

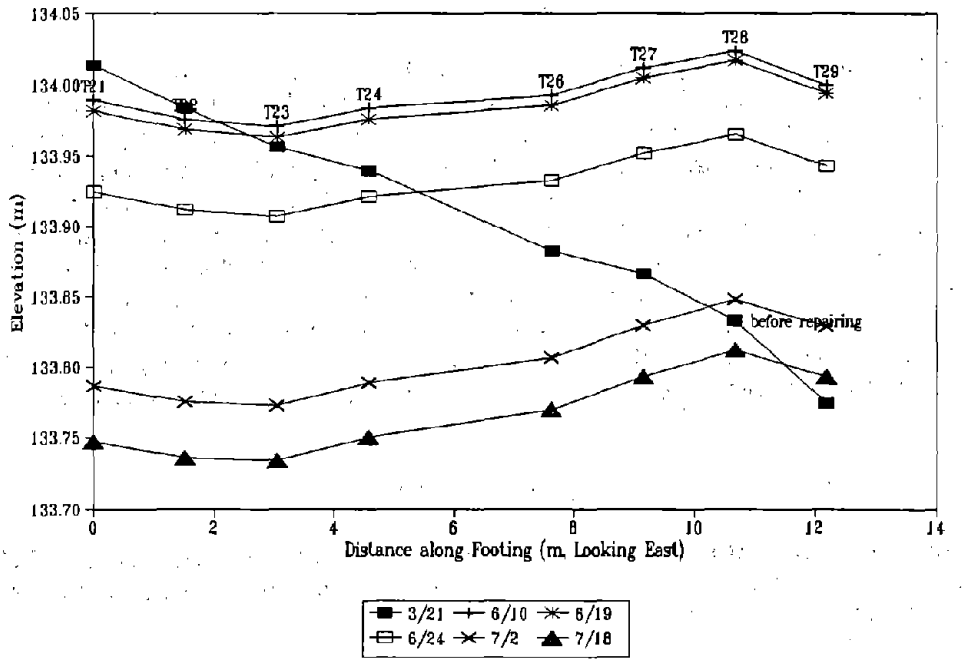


**Figure 6.** Elevation profile of reinforced concrete footing before and after leveling.

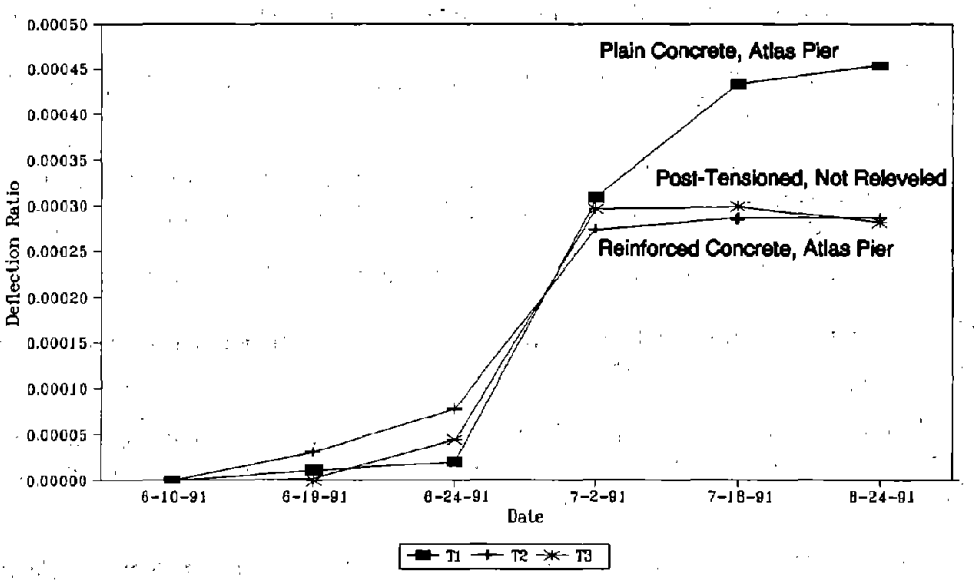


**Figure 7.** Foundation deflection ratio during second subsidence event.

One of the most significant advantages of the Atlas continuous lift pier system is that after installation, the position of the structure can be adjusted repeatedly. The tilt and deformation left in the foundations after the second subsidence event were removed in 1993, restoring the foundation to level. Since the lifting heads were left in place, the releveling process was completed in about two hours, simply by reinstalling the jack and manifold system. Provided adequate travel is provided in the original installation, there is no limit to the number of times the structure can be releveled.

## Conclusions

A series of test foundations monitored during a subsidence event in 1990 were left with permanent tilt, curvature, and substantial cracking. The foundations were releveled in 1991 using Atlas Systems continuous lift piers. The continuous lift piers removed the tilt and curvature and significantly reduced the width of the foundation cracks. In a typical residential structure damaged by subsidence, the restoration of the foundation by the continuous lift piers would have corrected most distorted elements such as sticking doors and windows. Since subsidence often occurs relatively slowly, the continuous lift system could have been used to maintain the structure in a level and undistorted state during subsidence, if desired. The Atlas pier system does not strengthen the structure, and as such is not a structural damage mitigation technique. However, the continuous lift piers were found to be an effective functional damage mitigation technique. The system was not detrimental to the foundation response during subsidence, and permits rapid restoration of structural function after subsidence has occurred. If it had been used in conjunction with repairs to strengthen the foundations, the deformations due to the second event could have been reduced.

## Acknowledgements

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# PIPELINES AND SUBSIDENCE: OVERLOOKED IN SMCRA[1]

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Abstract: In the Federal Surface Mining Control and Reclamation Act of 1977 (SMCRA), provision is made for certain surface improvements to be insulated from the risk of subsidence. These are identified in §1271(e) of the Act, 30 U.S.C. §1201 et seq. The Act identifies areas prohibited to mining, prompted by concern for public safety and welfare. There is no reference to pipelines which transport natural gas, petroleum products and chemicals. For many of these, rupture could be a catastrophe.

The absence of Federal regulation here results in presentation of antagonistic views in State regulation of SMCRA. In the field, State regulators are loath to add to the minimum requirements of Federal SMCRA, frustrating what pipeline operators view as required protection for pipeline safety. This conflict results in costly litigation and uneven administration of safety practices. Interstate pipelines are governed by the Office of Pipeline Safety, U.S. Department of Transportation. The safety regulations of this office would require pipeline operators to confront mine operators in the absence of direction from the Federal SMCRA. It is a costly and ineffective method of safety regulation.

## Historical Perspective

The commercial coal industry is ancient. The Chinese mined coal from surface deposits about 300 A.D. Western coal mining dates its origins in England to the 13th century. Large scale mining in America began around 1900 driven by the engine of industrial development. Deep mining of coal dominated initially; large-scale strip mining gained a foothold in the late 1940's and was challenging deep mining for economic productivity by the early 1970's. In the late 1970's, underground longwall coal mining boosted productivity to rival that of strip mining again.

Drake's well struck oil in 1859 near Titusville, PA. Even though Drake was searching for brine and found oil (an irony), his discovery led to the realization of the potential of petroleum. Natural gas followed close behind. Soon, pipelines began to crisscross the country, carrying oil and gas from the fields to the urban and industrial areas. Conflicts in the use of the land were inevitable.

Strip mining peels the layers of earth away allowing for no other use of the land during active mining. Oil and gas wells and pipelines are either moved or strip mining near them does not occur.

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Deep mining of coal does not present as obvious an example of surface disruption. In years past, subsidence occurred in sporadic isolation, occasionally affecting pipelines. But with the increased frequency of longwall mining and other full-extraction techniques, planned subsidence impacts on pipelines now rival the impact of strip mining.

## Regulatory Control of Subsidence Under SMCRA

In 1977 after decades of abuses, principally by strip mine coal operators, Congress enacted the Surface Mining Control and Reclamation Act (SMCRA) to preempt the regulatory field so poorly maintained by the coal producing States. A wonder of detailed statutory regulation, SMCRA does not address the looming conflict between coal mining and pipeline support as simultaneous land uses.

SMCRA addresses surface impacts of underground mining chiefly in sections 522 and 516; section 522 does not implicate pipelines. Subsidence is one of the surface impacts from underground mining Congress intended to regulate. Concerned about the unpredictable subsidence revealed in the historic record, Congress reported that-

> It is the intent of this section to provide the Secretary with the authority to require the design and conduct of underground mining methods to control subsidence to the extent technologically and economically feasible in order to protect the value and use of surface lands.[4]

SMCRA defines "surface coal mining operations" to include-

> (A) activities conducted on the surface of lands in connection with a surface coal mine or subject to the requirements of section 1266 of this title...and surface impacts incident to an underground coal mine...
>
> (B) the areas upon which such activities occur or where such activities disturb the natural land surface.[5]

That Congress intended to regulate subsidence as a "surface impact incident to underground coal mining" cannot be seriously questioned, notwithstanding the previous administration's one-time contrary view.[6] In House Report No. 218 (95th Cong., 1st sess. (1977) p.125), Congress reported:

> SURFACE IMPACTS OF UNDERGROUND MINES
>
> The environmental problems associated with underground mining for coal which are directly manifested on the land surface are addressed in section 212 and such other sections which may have application. These problems include surface subsidence, surface disposal of mine wastes, disposal of coal processing wastes, sealing of portals, entry ways or other mine openings, and the control of acid and other toxic mine drainage.

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[4] H.R. Rep. No. 218, 95th Cong., 1st sess. (1977) at 125-126. See also, S. Rep. No. 128, 95th Cong., 1st sess. (1977) at 84-85.

[5] 30 U.S.C. §1291(28) (1988).

[6] Solicitor's Opinion, U.S. Department of the Interior, July 10, 1991.

In 1984, the Office of Surface Mining (OSM) denied a permit to an underground mine in the Otter Creek Wilderness Area because "certain [prohibited] surface impacts to the wilderness could not be avoided, namely subsidence and hydrologic effects."[7]

SMCRA addresses the surface effects of underground coal mining mainly in section 516, SMCRA. That section requires underground coal operators to:

> (1) adopt measures consistent with known technology in order to prevent subsidence causing material damage to the extent technologically and economically feasible, maximize mine stability, and maintain the value and reasonably foreseeable use of such surface lands, except in those instances where the mining technology used requires planned subsidence in a predictable and controlled manner: Provided, That nothing in this subsection shall be construed to prohibit the standard method of room and pillar mining...

Section 516(c) authorizes the regulatory agencies to

> [S]uspend underground coal mining under urbanized areas, cities, towns, and communities and adjacent to industrial or commercial buildings, major impoundments, or permanent streams if he finds imminent danger to inhabitants of urbanized areas, cities, towns, and communities.

Section 516(b) permits issued to underground coal operators require that offsite areas be protected from damage and that fire hazards and other conditions that constitute a hazard to the health and safety of the public be eliminated.

The current regulations under SMCRA that address or deal with subsidence and pipelines are found as follows:

| | |
|---|---|
| 30 C.F.R. 778.15 | Right of Entry |
| 30 C.F.R. 783.24(c) | Maps (showing pipelines passing over and through permit area) |
| 30 C.F.R. 784.20 | Subsidence Control Plan |
| 30 C.F.R. 784.23 | Maps (showing "utility corridors") |
| 30 C.F.R. 817.180 | Utility Installation (minimize damage, destruction, or disruption to installations that pass over, under, or through permit area unless approved by owner and agency) |
| 30 C.F.R. 817.121 | Subsidence Control (prevention and repair of damage and suspension of operations causing imminent danger) |
| 30 C.F.R. 843.11 | Cessation Orders (cessation of and correction of any imminent damages caused by condition or practice of coal mining) |

[7] 24 Fed. Reg. 31228 (1984). In accord see, 44 Fed. Reg. 14990 (1979).

| | |
|---|---|
| 30 C.F.R. 817.122 | Notice of Mining (6 months' prior notice to all owners and occupants of surface) |

It is of particular importance to pipeline operators that the subsidence control plan must include detailed information showing the areas of planned subsidence, the measures to prevent, minimize, or repair subsidence damage, monitoring of the subsidence, and the expected effects of any subsidence.

On September 24, 1993, OSM published proposed changes to the subsidence regulations of 30 C.F.R. 784.20 and 30 C.F.R. 817.121 and defined the terms "material damage" and "structure or facilities." The breadth of the definition of structures or facilities arguably includes pipelines. The definition states-

> Structures or facilities means any building, constructed object or improvement whether installed on, above, or below the land surface, including, but not limited to, park facilities, roads, cemeteries, utilities; fences and other enclosures; retaining walls; and septic sewage treatment, irrigation and drainage systems.

When coupled with the material damage standard and the presumption of liability if within the 35° angle of draw, the proposed regulations offer pipelines a positive departure from prior practice. It remains to be seen if the proposed regulations will take effect.

## State Regulations

Upon adoption of SMCRA by Congress in 1977, uniform State regulation of coal mining began. The debates evident in the adoption of SMCRA by Congress were heard again in the State legislatures as each State established mine regulation. In several States in which mining is a prominent activity, the adoption of regulatory controls was accompanied by restrictions on variations. State legislation imposing limitations can be found, for example, in the coal States of Kentucky, Illinois and Indiana.

In Kentucky, the legislation enabling SMCRA regulation is KRS 350.465. A section of that legislation which limits additions to State regulation states,

> The implementation of this section shall contain procedures similar to the Surface Mining Control and Reclamation Act of 1977 (PL 95-87 (sic)) and shall require surface coal mining operation standards no more stringent than provided for in that act.

Similarly, in Illinois, adoption of State controls, identified as Ill.Rev.Stat. ch-96 1/2, ¶ 7901.01 et seq. was accompanied by section 1.02(c) which provides-

> (c) It is also the purpose of this Act to establish requirements that are no more stringent than those required to meet the Federal Surface Mining Control and Reclamation Act of 1977 (PL 95-97)

In Indiana, adoption of State regulation of coal mining was codified as IC 13-4.1. The obstacle to amendment of controls was adopted in 1991 by the Indiana legislature in SEA 46, codified as IC 13-4.1-5:

> Neither the director nor the commission [of the state regulatory

agency] may enforce the following: (1) A rule adopted under this article that is more stringent than corresponding provisions under the Federal Surface Mining Control and Reclamation Act (30 U.S.C. 1201-1328).

That Indiana did not take kindly to the Federal imposition of regulation is evident in the legislative findings in the first passages of IC 13-4.1, in which the reluctance of the legislature is made plain.

> (4) The threat that the federal government will regulate the surface coal mining operations and reclamation procedures, including land use planning and control, if the state of Indiana does not enact the necessary legislation for a state program, is coercive. It makes the overriding consideration whether to prevent further federal encroachment upon and regulation and control of the state, its people and local industry, rather than what is in the best interest of the people of the state of Indiana.

When the State legislation was enacted, the pipeline industry did not show interest. The antisubsidence concerns were mostly to be generated in the future. The principal regulator of interstate pipelines is a Federal agency, the Office of Pipeline Safety, U.S. Department of Transportation. Pipeline operators are attentive to safety regulations of Federal rather than State agencies. The attention of pipeline operators would be drawn to the State SMCRA regulation soon, however, by the onrush of longwall mining, which requires planned subsidence.

The absence of antisubsidence protection for pipelines in SMCRA as adopted by Congress would appear in a court opinion, Shell Pipe Line v. Old Ben Coal Co. This opinion of the U.S. District Court for the Southern District of Illinois was announced in 1988.[8] Old Ben Coal, using longwall mining and the attendant planned subsidence, was mining toward Shell's 40-in petroleum products pipeline. After its requests to Old Ben for payment were denied, Shell spent $750,000 to support the pipeline as the Old Ben mine passed underneath. Shell sued for the preventive expenses to avoid catastrophic damage to the pipeline.

It lost.

The judge in his opinion sympathized with Shell, but citing the Illinois SMCRA regulations, concluded that the preventive costs could not be collected. Shell must first permit damages to occur and then sue Old Ben for the losses. The opinion suggested that the regulation should be changed.

> Perhaps the Illinois legislature will review the current legislation in light of the circumstances of this case. It would be a wise decision, for next time the company or individuals may not act as responsible as Shell and the results could be devastating. (677 F. Supp 572 at 575.)

The SMCRA regulations, adopted by Congress for assumption by each State, do not provide that a pipeline operator above a proposed mine must be given notice that a mine plan is being considered by a State agency. In Indiana, Midwestern Gas Transmission,[9] an operator of a 30-in natural gas pipeline supplying the metropolitan Chicago area, learned that its line lay above a proposed underground mine from a legal notice printed in a local paper. There

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[8] 677 F. Supp 572, (S.D. Ill, 1988).

[9] A member of the Tenneco corporate family.

would be no other notice of the mine to Midwestern Gas. When officials of Midwestern Gas met with representatives of the proposed mine and the controlling Indiana agency, they learned to their dismay that no one in either of the other groups had any idea of the extreme sensitivity of pipelines to subsidence. The coal mine operators were aware of the ruling in the Shell case and, initially, expressed little interest in Midwestern's fears.

Midwestern Gas then began a long struggle to amend the Indiana regulations to provide antisubsidence protection for pipelines. First, by itself, it prepared changes to the Indiana SMCRA and submitted them to the Indiana agency. Although there were several proposals, the added requirement for actual notice[10] was the most significant change.

The second effort by Midwestern Gas to make changes to protect against subsidence was more concentrated. Under the direction of the Tenneco offices, using legal and engineering experts, comprehensive proposals for amendments to the Indiana SMCRA were submitted to the Indiana regulatory agency. At the August 1991 meeting of the governing body of that agency, the proposals were considered. Present and speaking for the proposals were representatives of Tenneco, Amoco Pipeline, Panhandle Eastern, Midwestern Gas, Southern Indiana Gas and Electric, Citizen's Gas of Indianapolis, the Countrymark pipeline, and a public interest lawyer. The theme was public safety and national interest.

Representatives of the coal association were present and spoke against adoption, citing property and other legal rights.[11] The proposals were rejected by the Indiana agency without a dissenting vote. The rationale was Indiana's SEA 46, which would prohibit rules "more stringent" than SMCRA.

The Indiana director of the U.S. Office of Surface Mining then announced that his office would hold a public hearing on the propriety of SEA 46 to determine its compliance with SMCRA. An OSM hearing was held, and again the pipeline, private property owners and coal representatives were heard. The OSM found that SEA 46 was not in compliance with SMCRA,[12] citing the Midwestern Gas comments.

The coal industry association appealed the OSM ruling in the U.S. District Court.[13] Before the matter could be decided, a proposal was submitted to the Indiana legislature to supplant SEA 46 with a measure tailored to the OSM conditions. The measure, Senate Bill 374, passed easily and is Indiana law. The new law contains the "no more stringent" limitation.

SMCRA protects certain improvements, including public roads, occupied dwellings, schools, churches, parks, and cemeteries,[14] from subsidence by prohibiting mining underneath them. The reasons this protection exists may be apparent to the examiner, but the reasons do not include safety or the public interest in the fashion that protection to pipelines would provide.

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[10] 310 Indiana Administrative Code 12-3-106(f)(4) and 12-5-131.1.

[11] The comments were not without acrimony. A coal lawyer contended that the pipeline speakers were deceitful in the request for protection.

[12] 56 Fed. Reg. 64996 (1991).

[13] Indiana Coal Council, Inc. v. Manuel Lujan, Jr., Secretary of the Interior, et al., IP 92-84-C U. S. District Court, Southern District of Indiana, Indianapolis Division.

[14] 30 U.S.C. § 1272 (4),(5).

Presently, the burden is on the pipeline operators, at considerable expense to themselves and risk to the public, to find protection from subsidence. The success of meeting this burden must vary in each State, according to the response of the administrative agencies, courts and legislature of each State. It remains for the Federal Government to provide the uniform standards of antisubsidence protection essential to pipelines.

Regulations enforced by the Office of Pipeline Safety require that pipeline operators pursue the highest standards of safety in the public interest. Experience in Indiana shows that the most concerted efforts of pipeline operators to establish a code of regulated protection speak to an unresponsive audience. The omission of antisubsidence protection for pipelines in SMCRA results in frustration of Federal safety practices. This is a shortcoming that can be remedied only by amendment of SMCRA.

## Deficiencies in Federal and State Programs

SMCRA was intended to set the minimum standards for the coal industry's deep-coal-mining activities. The principal goal of the comprehensive regulatory plan was either to prevent subsidence or to plan subsidence so that the value and reasonably foreseeable uses of the surface were maintained. The failure to establish uniform national standards for the conduct of underground mining where pipelines will be affected has resulted in different standards being applied from State to State. While the proposed subsidence regulations may address some of the concerns, other areas of concern remain unresolved. Examples follow:

1. Notice to pipeline owners of permit application, permit issuance, and commencement of mining near an affected pipeline.

2. Standards for the protection of pipelines.

3. Emergency procedures in case of accident.

4. Hazard zones and prohibited areas.

## Conclusion

SMCRA serves to provide coal mining regulation uniformly throughout the United States. It serves at the same time to frustrate uniform protection from subsidence to pipelines by omission of specific terms. The result is expensive confrontation in the States between competing interests, resulting in uneven compliance with Federal safety standards governing pipelines. The solution is to address the issue through reconsideration of SMCRA.

# OVERVIEW OF MINE SUBSIDENCE INSURANCE PROGRAMS IN THE UNITED STATES[1]

David K. Ingram[2]

Abstract: Because the U.S. insurance industry has determined that mine subsidence is uninsurable, it has decided not to voluntarily offer mine subsidence insurance. The U.S. Bureau of Mines has long been investigating the effects of mine subsidence. These investigations have resulted in Federal regulations and controls of mine subsidence. This Bureau report addresses the development of mine subsidence insurance programs and the eight current mine subsidence insurance programs in the United States. The States that have these subsidence programs include Colorado, Illinois, Indiana, Kentucky, Ohio, Pennsylvania, West Virginia, and Wyoming. The report discusses major aspects of these programs, including history, administrative and operational procedures, insurable structures, recognition of mine subsidence, insurance premiums, and the economic health of each program. Information within this report can be useful for residential and commercial property owners and mine operators. States that are considering starting a mine subsidence insurance program or that have one in place can also use this report as a model for initiating or modifying their programs.

## Introduction

The U.S. Geological Survey has identified 1.7 trillion tons of coal resources in the United States (National Coal Association 1981). As many as 30 States have coal mining operations extracting coal from these reserves (Journal of American Insurance 1985). A 1975 Bureau study (Johnson and Miller 1979) estimated that 7 million acres of land has been undermined in the United States. There still remain about 5.2 million acres of undermined surface area that has not yet subsided. Approximately 500,000 acres is situated under urban, populated areas. The National Research Council Committee on Ground Failure Hazards estimated that from 1925 to 1975 landslides and subsidence caused at least $75 billion in losses. Costs of subsidence damage are three times higher than the combined estimated costs of damage caused by floods, hurricanes, tornadoes, and earthquakes (Insurance Review 1986).

The insurance industry in the United States has investigated the feasibility of offering insurance for damage resulting from subsidence. The Mine Subsidence Task Force of the National Committee on Property Insurance conducted a study in 1986. Their conclusion was "before the insurance industry is called upon to underwrite the devastation that results from mine and other land subsidence, more research is needed in the hazard's prediction, cost, and insurability" (National Underwriter 1986).

The earlier investigations by the Bureau has helped instigate the enactment of a Federal law to protect society and the environment from the adverse effects of coal mine operations (Johnson and Miller 1979). These investigations influenced legislation to create grant monies for States to initiate their own mine subsidence programs.

## Development of State Subsidence Insurance Programs

Until 1977, it was the State's responsibility to regulate and control subsidence-related problems on non-Federal lands. On August 3, 1977, Public Law 95-87, the Surface Mining Control and Reclamation Act

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(SMCRA), was enacted. Provisions within this act help regulate the effects of underground coal mining operations on the surface.

Under SMCRA, coal companies are required to design and implement methods to prevent subsidence or to control and minimize its effects. In addition, underground coal companies are required to pay a 15 cent fee for every ton of coal produced. This fee, along with surface mining operation fees (35 cents per ton), is deposited into the Abandoned Mined Land Reclamation Fund. Money from the Fund helps cover the reclamation costs of underground and surface strip mines abandoned before 1977.

Stabilizing or reclaiming the surface or ground that is subsiding, because of a mine abandoned before 1977, is considered a reclamation cost. Stabilizing or repairing surface structures that are damaged by subsidence is not considered a reclamation cost. Reclamation costs for uncontrolled subsidence caused by mines abandoned after 1977 are the responsibility of the operating coal company. However, liability for costs of subsidence damage to surface structures caused by mining after 1977 depends upon individual State regulations.

In 1984, Congress enacted changes to SMCRA. One of the changes was to authorize grants up to $3 million from the Abandoned Mined Land Reclamation Fund for the establishment of State-operated mine subsidence insurance programs (National Underwriter 1986). To be eligible for the grant, States must contribute to the fund and have SMCRA-approved abandoned mined land reclamation programs (Netzner 1986). States that receive the grant have up to 8 years for their programs to become self-sustaining (Godesky 1990).

There are 21 States eligible for Federal subsidence assistance. Currently, eight States have mine subsidence insurance programs: Colorado, Illinois, Indiana, Kentucky, Ohio, Pennsylvania, West Virginia, and Wyoming. Four of the eight States utilized the Federal grant money to initiate their programs. Three of the States initiated their programs with State appropriations, while one used both State and Federal money. To date, New Mexico is the only other State that is considering organizing a mine subsidence insurance program.

## State Mine Subsidence Programs

### Colorado

Coal mining in Colorado began in the 1860's. The Plan of Operation of the Colorado Mine Subsidence Protection Program reports that there are nearly 50,000 acres of undermined land. This 50,000 acres, in the Front Range corridor, affects more than 7,450 structures and approximately 25,000 people. The Colorado Mine Subsidence Protection Program was established in 1986. Funds to initiate the Trust Fund that finances the program were obtained from the Federal Government.

The Johnson & Higgins insurance brokerage has been designated as the plan administrator of the Mine Subsidence Protection Program. Subsidence insurance policies are only issued to owners of residential structures. The Mine Subsidence Protection Program only insures damage caused by underground coal mine subsidence. Participation in the program is voluntary. The cost to participate in Colorado's program is $160 for the first year and $35 per year thereafter. The first year's fee covers a $125 inspection fee on the insured property. The maximum coverage is $50,000 per occurrence after a $1,000 deductible.

As of late 1992, there were 1,100 policyholders (Gwinn 1992). There have been six subsidence claims filed. Five of the claims have been invalid claims, while the one remaining claim is pending. As of December 1992, the total balance of the program's Trust Fund was approximately $3 million.

## Illinois

Underground mining began in Illinois in 1842. Today, there are about 750,000 acres of surface land undermined in the State (DuMontelle et al. 1981). The Illinois Mine Subsidence Insurance Fund recognizes 34 counties as mine-subsidence-prone areas with approximately 1% or more of the land surface undermined.

In September 1979, Illinois mandated all private insurance companies to offer mine subsidence insurance. The Illinois Mine Subsidence Insurance Fund is the reinsurer for the insurance companies. The Illinois Mine Subsidence Insurance Fund was initiated with a State loan. The Illinois Mine Subsidence Insurance Fund is the only taxpaying, industry-operated subsidence program (Murphy 1992). The fund is guided by a Governing Committee. The Governing Committee is composed of five members from the insurance industry and four members who are not associated with either the insurance industry or the fund.

Mine subsidence insurance coverage can be obtained for residential or commercial structures. The fund recognizes mine subsidence as the collapse of all manmade underground mines. Participation in the program is voluntary. Mine subsidence coverage must be purchased in amounts no less than the amount insured by conventional property insurance, up to a maximum of $350,000 per structure. Premium rates for both residential and commercial insurance range from $8 for $10,000 or less worth of coverage to $53 for $350,000 worth of coverage. The deductible for any paid claim is 2% of the insurance coverage up to a maximum of $500.

As of late 1992, there were over 350,000 structures insured. About 5,000 claims have been filed. Approximately 750 of those claims, or 15%, have been valid subsidence claims (Murphy 1992). The remaining 4,250 claims are either pending or are not subsidence related. As of late 1992, the fund's surplus totaled approximately $1.1 million (Murphy 1992).

## Indiana

Underground coal mining has been occurring in Indiana since 1850. There are approximately 150 square miles, or 100,000 acres, of surface land undermined in the State. The Mine Subsidence Program of Indiana offers subsidence insurance in 26 counties (Indiana Department of Insurance 1988).

Indiana's 1986 Indiana Insurance Code requires insurance companies licensed in Indiana to offer mine subsidence insurance. Indiana's Mine Subsidence Insurance Fund is the reinsurer for the insurance companies. The fund was initiated by a Federal grant (Foy 1992). The Indiana Mine Subsidence Insurance Fund is administered by the Commissioner of Insurance, Indiana's Department of Insurance.

Mine subsidence insurance can be obtained for residential or commercial structures. The fund defines mine subsidence as the collapse of only underground abandoned coal mines. Participation in the subsidence program is voluntary. Subsidence coverage must be purchased in amounts no less than the amount insured by conventional property insurance. The maximum coverage is $75,000 per structure. Annual insurance premiums for residential structures range from $15 for $15,000 or less worth of coverage to $27 for $75,000 worth of coverage. Annual insurance premiums for commercial structures range from $24 for $15,000 or less worth of coverage to $48 for $75,000 worth of coverage. The deductible for a paid subsidence claim is 2% of the insurance coverage; the deductible cannot be less than $250 or more than $500.

Currently, Indiana has about 60,000 policyholders. There have been about 67 claims filed. Thirteen, or 19%, of the claims have been valid, 40 claims have been invalid, and the remaining 14 are pending. The total fund's surplus as of late 1992 was approximately $2 million.

## Kentucky

Coal mining was occurring in Kentucky as early as 1790. Kentucky has two coalfields, one in the eastern part of the State and one in the western part. In western Kentucky, coal reserves underlie about 6,400 square miles in 19 counties. In eastern Kentucky, coal reserves underlie about 11,000 square miles in 33 counties. Past studies have indicated there are approximately 37,000 acres of surface land overlying abandoned coal mines (Rosenbaun 1990).

In 1984, Kentucky established the Kentucky Mine Subsidence Insurance Fund and required insurance companies licensed with the State of Kentucky to offer mine subsidence insurance. The Kentucky Mine Subsidence Insurance Fund (the reinsurer for the insurance companies) became operable November 1, 1986. The Kentucky fund was initiated with a Federal grant.

The Kentucky Mine Subsidence Insurance Fund is administered by the Division of State Risk and Insurance Services of Kentucky's Department of Insurance. Mine subsidence insurance can be obtained for residential or commercial structures. Mine subsidence is recognized by the fund as the collapse of underground coal mines. Mine subsidence insurance can be obtained in 34 counties in Kentucky. Participation in the program is voluntary.

Subsidence coverage must be purchased in amounts no less than the amount insured by conventional property insurance, provided the maximum coverage does not exceed $50,000. Insurance premiums for residential structures range from $15 for $15,000 or less worth of coverage to $22 for $50,000 worth of coverage. Insurance premiums for commercial structures range from $24 for $15,000 or less worth of coverage to $38 for $50,000 worth of coverage. The deductible for a paid claim is 2% of insurance coverage; this amount cannot be less than $250 or more than $500.

Currently, Kentucky has about 37,000 policyholders. There have been approximately 305 subsidence claims filed. Of the 305 claims filed, 29, or 10%, are related to mine subsidence, 260 were invalid, and 16 are still pending. The total fund surplus as of December 1992 was approximately $5.2 million.

## Ohio

Coal mining has been occurring in Ohio since the late 1700's. The U.S. Department of Energy estimated that about 10,000 square miles, or 24%, of the State overlies coal deposits (Energy Information Administration 1989). Currently, there are approximately 3,800 abandoned underground coal and mineral mines (Sommers et al. 1988).

In January 1985, Ohio established the Ohio Mine Subsidence Insurance Underwriting Association, the Mine Subsidence Insurance Governing Board, and the Mine Subsidence Insurance Fund. Insurance coverage became available to the public in 1987. Ohio requires insurance companies licensed in Ohio to offer mine subsidence coverage. The Mine Subsidence Insurance Governing Board administers the Mine Subsidence Insurance Fund. Ohio's subsidence program was initiated with State and Federal appropriations.

Subsidence insurance can be obtained for residential structures only. Mine subsidence is recognized as the collapse of all underground mines. Mine subsidence insurance is mandatory in 27 counties and voluntary in 10 other counties (Leslie 1992). Subsidence coverage must be purchased in amounts no less than the amount insured by the conventional property insurance up to a maximum of $50,000. The annual premium rate for the insurance is $3 for the mandatory insurance and $12 for the voluntary insurance, regardless of the amount of the coverage. For any subsidence claim, there is a 2% deductible, with a minimum of $250 and a maximum of $500.

There were about 13,000 policyholders as of late 1992. There have been 136 claims filed. To date, 16, or 12%, of the claims were valid, 115 were invalid, and the remaining 5 are still pending. Currently, there is no surplus of funds available for Ohio's subsidence program. This is due to a 1992 restructuring of the subsidence program. Projections indicate that the subsidence fund will have a surplus in 1993 and thereafter (Leslie 1992).

**Pennsylvania**

Coal mining in Pennsylvania was first recorded in 1759. Approximately 15,000 square miles, or 33%, of Pennsylvania overlies coal deposits (Energy Information Administration 1989). As of 1985, it was estimated that more than 5 million Pennsylvanians live over abandoned mine sites (Journal of American Insurance 1985).

On August 23, 1961, Pennsylvania established the Coal and Clay Mine Subsidence Insurance Board and the Coal and Clay Mine Subsidence Insurance Fund. The fund for the program was initiated by State appropriations. Subsidence insurance is issued by Pennsylvania's Mine Subsidence Insurance Fund under the Bureau of Mines and Reclamation, Department of Environmental Resources. The Coal and Clay Mine Subsidence Insurance Board implements the rules and regulations of the insurance program.

Subsidence insurance can be obtained in 28 counties throughout the coal or clay mining regions. Insurance is voluntary and can be obtained for both residential and commercial structures. The subsidence insurance covers subsidence due to coal or clay mining operations. The maximum coverage available in Pennsylvania is $100,000. Costs for 1 yr of insurance for residential structures range from $12.50 for $5,000 worth of coverage to $88.50 for the maximum coverage of $100,000. For commercial structures, costs range from $63 for $5,000 worth of coverage to $348 for $100,000 worth of coverage. The maximum coverage will be increased to $150,000 for residential structures and $250,000 for commercial structures pending publication of the regulation. There is a $250 deductible for residential structures and a $500 deductible for commercial structures.

There were approximately 40,000 policyholders as of late 1992. In the bituminous region for the past 10 years, annual claims actually related to subsidence as a percentage of total claims submitted were about 30%. In the anthracite region, this percentage was about 5% (Schnurr 1991, Ruane 1992). These percentages have been highly variable because the number of subsidence events varies from year to year. Currently, the fund has about $11.5 million in surplus.

**West Virginia**

Mining was occurring in West Virginia before it became a State in 1863. West Virginia has the largest coal deposits in the northern Appalachian Coalfield. A total of 40 out of 55 counties in West Virginia have coal reserves that are economically important. In fact, 69% of West Virginia, or 16,800 square miles, overlies coal deposits (Energy Information Administration 1989).

In 1982, West Virginia passed legislation that created the Mine Subsidence Insurance Fund. This legislation also directed insurance companies licensed in West Virginia to offer mine subsidence insurance. The fund was established with State-appropriated money. The Mine Subsidence Insurance Fund, which reimburses the insurance companies, is operated by West Virginia's Board of Risk and Insurance Management.

Mine subsidence insurance can be obtained for residential and commercial structures located in any of 40 counties in West Virginia. Mine subsidence is defined as the collapse of underground coal mines. Mine subsidence insurance is voluntary. The amount of subsidence insurance coverage on a structure must be no

less than the amount of the regular property insurance coverage up to a maximum of $75,000. The cost of subsidence insurance for residential structures ranges from $10 for $10,000 or less worth of coverage to $23 for $75,000 worth of coverage. For commercial structures, the cost ranges from $20 for $10,000 or less worth of coverage to $46 for $75,000 worth of coverage. There is a $250 deductible on all paid subsidence claims.

There were 110,000 policyholders as of December 1992. Since 1983, 1,050 subsidence claims have been filed. Approximately 168, or 16%, of the 1,050 claims have been valid, about 602 have been invalid, and approximately 280 are still pending. Currently, the fund has approximately $10 million in surplus.

## Wyoming

Coal mining in Wyoming started in the 1860's. There are about 15,000 people on approximately 1,000 acres overlying abandoned mines. Wyoming's Abandoned Mined Land Program is working to reduce the potential for subsidence in these towns by backfilling and/or grouting the mine voids.

In 1986, Wyoming created the Wyoming Mine Subsidence Insurance Program. Wyoming's program was started with seed money from the Federal Government (Barnard 1991). The Wyoming Mine Subsidence Insurance Program operates under the Abandoned Mined Land Program of the Land Quality Division, Department of Environmental Quality (Wyoming Mine Subsidence Insurance Program 1987).

Subsidence insurance coverage can be purchased for residential and commercial structures. The program recognizes mine subsidence as collapse of underground coal mines. Wyoming's subsidence program is voluntary. Insurance coverage cannot be purchased for less than 75% of the structure's estimated value nor for more than the structure's estimated value up to a maximum of $150,000. The annual premium rates for structures are $2 per $1,000 for residential coverage and $3 per $1,000 for commercial coverage. Subsidence claims on residential structures are paid out minus 1% of the amount of insurance purchased with a minimum deductible of $250 and a maximum of $500. Deductibles for commercial structures are also 1% of the amount of the insurance purchased with a minimum of $250, but the maximum is $1,000.

There were about 1,000 policyholders as of December 1992. There have been 82 claims filed. About 53 claims, or 65%, have been valid subsidence claims, 24 claims have been invalid, while 5 claims are pending. Currently, the surplus of fund totals about $1.7 million.

## Summary

The objective of this report is to compile and summarize all the mine subsidence insurance programs in the United States. It is important to note that the Bureau's intention is not to evaluate or rank each subsidence program by comparison, but rather to summarize in one report all the programs in existence for a source of reference. Information in this report should be used as a platform for States considering starting a subsidence program or for States that want proven ideas to modify their existing programs. In addition, coal companies should find this information useful in dealings with the local communities. Property owners can use this report as an initial reference to answer questions and/or further pursue insurance coverage. Table 1 summarizes the major aspects of all eight subsidence insurance programs. A list of addresses and telephone numbers of the agencies affiliated with each program can be obtained from the author.

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**Table 1. Summary of mine subsidence insurance program.**

| Item | Colorado | Illinois | Indiana | Kentucky | Ohio | Pennsylvania | West Virginia | Wyoming |
|---|---|---|---|---|---|---|---|---|
| Start of program | 1988 | 1979 | 1986 | 1986 | 1987 | 1961 | 1983 | 1986 |
| Source of funds | Federal | State | Federal | Federal | State, Federal | State | State | Federal |
| Administered by | State insurance company | Insurance association | State insurance companies | State insurance companies | State insurance companies | State | State insurance companies | State insurance companies |
| What is insurable | R | R, C | R, C | R, C | R | R, C | R, C | R, C |
| Type of subsidence | Coal | Coal, clay, limestone, fluorspar, etc. | Coal | Coal | Coal, clay, limestone, salt, etc. | Coal, clay | Coal | Coal |
| Maximum coverage | $50,000 | $350,000 | $75,000 | $50,000 | $50,000 | $100,000 | $75,000 | $150,000 |
| Annual fee: | | | | | | | | |
| Residential | $35[1] | $8-$53 | $15-$27 | $15-$22 | $3-$12 | $12-$88 | $10-$23 | ([2]) |
| Commercial | NAp | $8-$53 | $24-$48 | $24-$38 | NAp | $63-$348 | $20-$46 | ([3]) |
| Deductible: | | | | | | | | |
| Residential | $1,000 | $500 | $250-$500 | $250-$500 | $250-$500 | $250 | $250 | $250-$500 |
| Commercial | NAp | $500 | $250-$500 | $250-$500 | NAp | $500 | $250 | $250-$1,000 |
| Participation | Voluntary | Mandatory[4] | Mandatory[4] | Mandatory[4] | ([5]) | Voluntary | Mandatory[4] | Voluntary |
| Number of policyholders | 1,100 | 350,000 | 60,000 | 37,000 | 13,000 | 40,000 | 110,000 | 1,000 |
| Fund balance | $3 million | $1.1 million | $2 million | $5.2 million | $0[6] | $11.5 million | $10 million | $1.7 million |

C = Commercial structures, NAp = Not applicable, R = Residential structures.
[1]There is a one-time initial $125 inspection fee.
[2]Fee is $2 per $1,000 of coverage.
[3]Fee is $3 per $1,000 of coverage.
[4]Unless waived.
[5]Mandatory in 27 counties, voluntary in 10 others.
[6]Owing to 1992 restructuring of program; a subsequent surplus was projected.

# OVERBURDEN DEFORMATION DUE TO LONGWALL MINING; ITS DEVELOPMENT AND FINAL DISTRIBUTION[1]

David K. Ingram, Michael A. Trevits, and Paul W. Jeran[2]

**Abstract**: The objective of this U.S. Bureau of Mines investigation was to characterize overburden response due to longwall mining. Subsurface strata and surface deformations were monitored during the mining of two adjacent longwall panels in southeastern Ohio. Multipoint borehole extensometers (MPBX) with eight anchors were installed in six boreholes across both panels to measure subsurface displacements. Survey monuments were installed along the center lines and profile lines over both panels to measure surface deformations. During the mining of the first panel, subsurface displacements occurred in a radiating pattern beginning just above the extracted area in the center of the panel and extending upward and outward towards the surface and margins of the panel. Overburden deformation did not project over the gate road entries. During the mining of the second panel, the MPBX's showed almost simultaneous displacement, indicating that the overburden moved as a unit. Again, overburden deformation was confined to the immediate panel area. At both panels, overburden movement began when the longwall face was approximately 25% of the overburden thickness beyond the line of MPBX boreholes.

**Additional Key Words**: subsidence, coal mine subsidence, coal mining, overburden deformation, longwall mining.

## Introduction

Mine operators are mandated by law to predict and reduce the effects of subsidence. Previous subsidence research has mainly focused on surface deformations and correlating mining data with ground movement data. Geologic conditions are usually generalized or overlooked. The goal of this Bureau research effort was to identify and characterize the dynamic response of the overburden rock mass due to longwall mining.

## Site Description

The study area was situated over two adjacent longwall panels in southeastern Ohio. Each panel measured approximately 270.0 m by 2,700.0 m. The panels were separated by a five-entry, four-pillar system (about 105 m wide). The extraction thickness ranged from 1.7 to 1.8 m. Overburden ranged from 64.2 to 84.0 m. The strata have a regional dip of about 1° toward the southeast. There are no major geologic structures, and the topography is mainly of rolling hills with a maximum relief of approximately 26.0 m.

The overburden consists of approximately 30% sandstone, 30% shale, 30% claystone, and 10% coal (fig. 1). For the most part, all of the lithologic units are laterally continuous. Individual units are generally thin, ranging in thickness from less than 1 m to about 6.0 m. Exception to this is a sandstone unit, approximately 15 m thick 33 m above the coalbed.

## Monitoring Program

This project was designed to measure, identify, and characterize overburden strata deformations.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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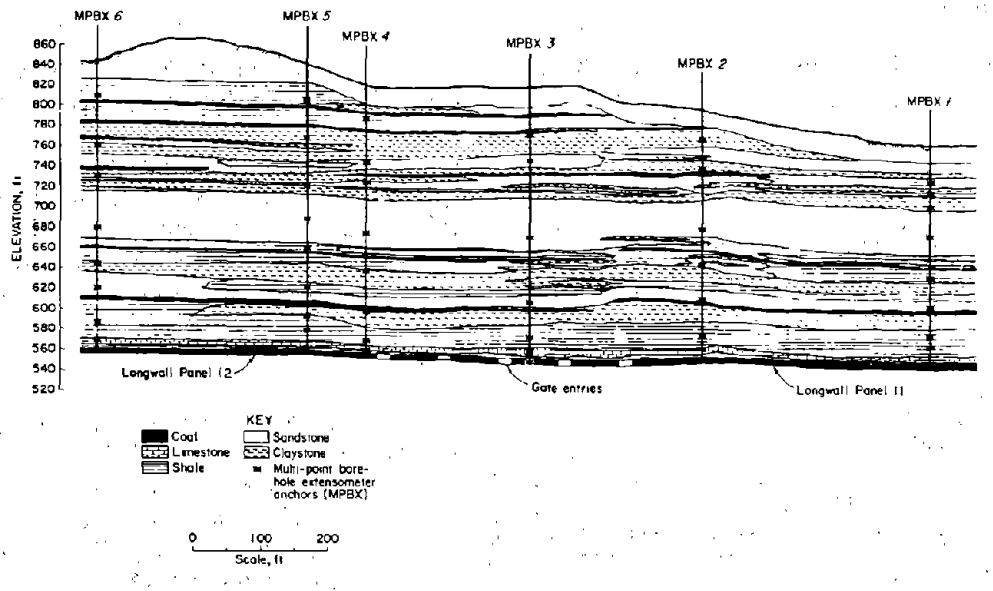


Figure 1. Cross section of the lithology with MPBX borehole and anchor location.

## Overburden Failure

Six 219.1-mm boreholes were drilled along a profile line extending across the two longwall panels. Each borehole was outfitted with an eight-anchor multipoint borehole extensometer (MPBX). During this study, MPBX anchor displacements at each borehole were recorded every hour.

The MPBX anchor locations were selected to verify the theoretical zones of overburden deformation. Two of the eight anchors in borehole 3 were installed inside a coal pillar to monitor any pillar yielding. They are numbered one through eight downward from the surface.

All of the MPBX boreholes were located at strategic locations as dictated by surface subsidence theory (Peng and Chaing 1984). Boreholes 1 and 6 were located in the center of each panel, where the maximum amount of subsidence was expected to occur. Boreholes 2 and 5 were located 30.0 m from the ribline inside each panel at positions where the inflection point was expected to occur. Borehole 4 was situated 3.0 m from the ribline inside panel 12. This location was in the expected zone of maximum horizontal tension. Borehole 3 was located in a pillar in the gate entries between the panels to observe the lateral extent of overburden deformation.

## Surface Subsidence Monitoring

Survey monuments were installed over the study area to monitor the surface deformation during longwall mining. Surface monitoring was necessary to identify the dynamic subsidence and, the final subsidence profile, and to reference the MPBX head assemblies. Survey measurements were scheduled before, during, and, after mining. Frequency of the surveys was increased to twice a week while the study area was being undermined.

## Field Data Results

### MPBX Boreholes

Typically, the first strata to collapse and have the greatest displacement are located immediately above the void created by the extraction. The magnitude of deformation is progressively reduced, moving upward and outward through the overburden from the center of the panel toward the surface. Provided the anchors remain intact during the entire event, the deepest anchor (anchor 8) should displace first and show the largest movement, while the overlying anchors progressively show less and less displacement. Inevitably some of the anchors become loose or fail.

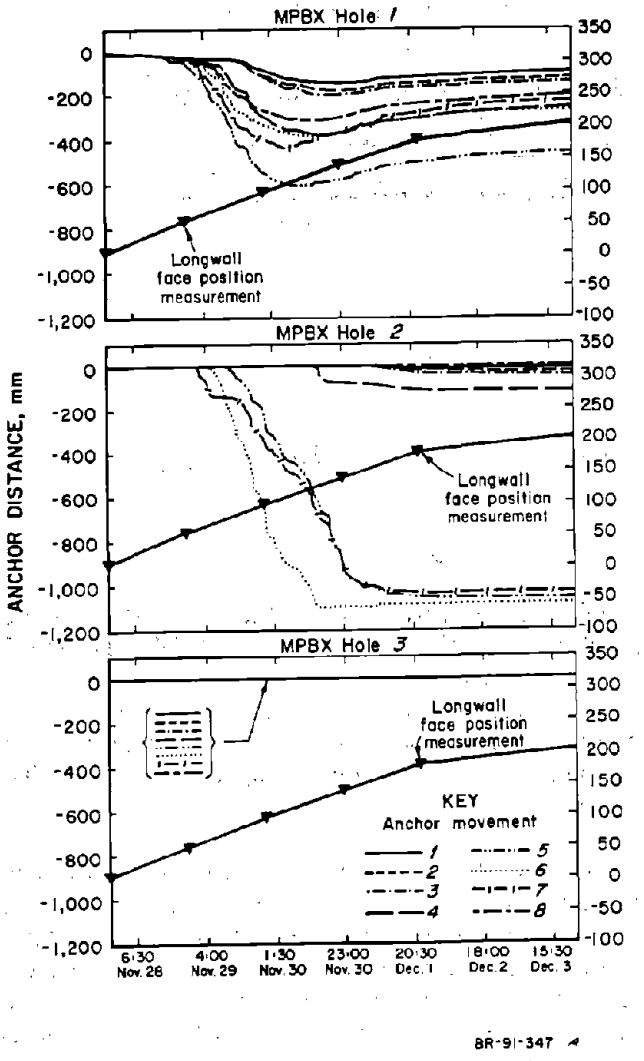


Figure 2. Anchor displacement for MPBX boreholes 1, 2, and 3.

All measurements are relative to differential movement between the MPBX anchors and the surface. No adjustments have been made for surface ground movement except with respect to final displacement. The following observations were made from the MPBX data.

**MPBX Borehole 1.** The total surface subsidence was 1,036 mm. The final order of anchor displacement (in decreasing order of magnitude) was 5 - 6 - 4 - 7 - 8 - 3 - 2 - 1 (fig. 2, borehole 1). Anchor 5 had the largest displacement of 467 mm. Anchors 6, 7, and 8 failed.

**MPBX Borehole 2.** Total surface subsidence at this borehole was 548.6 mm. The final order of anchor displacement (in decreasing order of magnitude) was 6 - 5 - 7 - 4 - 3 - 2 - 1 - 8 (fig. 2, borehole 2). Anchors 6, 7, and 8 failed. Anchor 5 is the deepest reliable anchor with the largest displacement of 1,056 mm.

**MPBX Borehole 3.** The maximum surface subsidence was 27 mm. There was no significant movement of any of the anchors in this borehole (figs. 2 and 3, borehole 3). During the mining of panel 11, the maximum movement of any anchor was less than 2 mm. During the mining of panel 12, the maximum displacement of the anchors was less than 1 mm.

**MPBX Borehole 4.** Total surface subsidence was measured at 152 mm. The final sequence of anchor displacement (in decreasing order of magnitude) was 6 - 4 - 3 - 8 - 2 - 5 - 1 - 7 (fig. 3, borehole 4). Anchor 6 showed the largest displacement of 15 mm. Anchors 8, 7, and 5 failed.

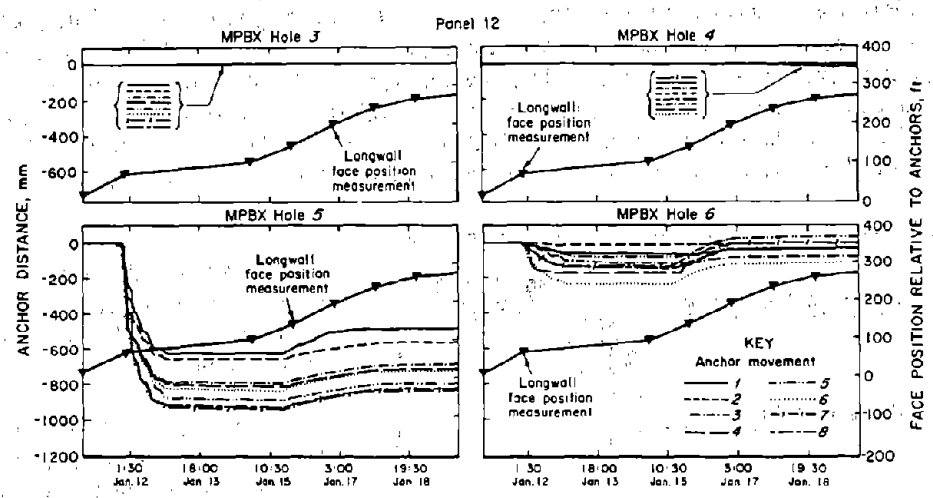


Figure 3. Anchor displacement for MPBX boreholes 3, 4, 5, and 6.

**MPBX Borehole 5.** Final surface subsidence was 701 mm. The final sequence of the anchor displacement (in decreasing order of magnitude) was 7 - 8 - 5 - 6 - 4 - 3 - 2 - 1 (fig. 3, borehole 5). Anchor 7 had the largest displacement of 830 mm. Anchors 8 and 6 failed.

**MPBX Borehole 6.** Final surface subsidence was 1,036 mm. The final sequence of anchor displacement (in decreasing order of magnitude) was 6 - 3 - 1 - 8 - 4 - 2 - 7 - 5, (fig. 3, borehole 6). Anchor 6 showed the greatest amount of displacement of 109 mm. Anchors 8, 7, 5, 4, and 2 failed.

## Surface Subsidence

Thirty five surveys were conducted of the surface monument array during the study. The maximum subsidence for both panels ranged from 0.9 to 1.1 m, or from 55% to 62% of the average extraction thickness. Figure 4 shows the percentage of the final subsidence plotted against the face position in terms of overburden thickness for both baseline surveys and boreholes 1 and 6. Movement of the surface generally began with undermining. Subsidence was over 90% complete when the mining was the thickness of the overburden past any surface point.

Figure 5 displays the final subsidence profile between the two baselines. The profile of the subsidence between the ribline and baseline varies somewhat between the two panels. At panel 11, the inflection point occurs about 33.9 m from the ribline, while at panel 12 it is at 30.0 m. The most pronounced difference between the two panels is the distance between the first point of the maximum subsidence and the ribline. At panel 11, it is approximately 129 m from the ribline, while at panel 12 it occurs at 89.1 m.

## Discussion and Interpretation of Overburden Response Panel 11

When panel 11 was mined, only boreholes 1 and 2 were affected. Deformation was first observed in the strata located immediately above the mined coalbed in the center of the panel. The deformation then radiated outward and upward toward the surface and panel margins. The overburden began to respond when the longwall face was approximately 16.2 m beyond the line of the boreholes. The overburden appeared to reach stability when the longwall face was 54.0 m beyond the line of boreholes.

The lowest three anchors (8, 7, and 6) in boreholes 1 and 2 failed. The deepest reliable anchor in both boreholes was anchor 5, positioned below the 15.0-m-thick sandstone unit. Anchor 4, in both boreholes, was

situated in the lower portion of the same sandstone unit. Anchor 5, positioned 25.5 m above the void area in borehole 1, displaced a total (including surface subsidence) of 1,504 mm, or 84% of the extraction thickness (table 1). For the remaining intact anchors in borehole 1, total anchor displacement uniformly decreased, progressing toward the surface. In borehole 2, anchor 5, located 27.3 m above the mine void area, displaced a total of 1,631 mm, or 92% of the extraction thickness (table 1). Unlike borehole 1, borehole 2 (fig. 2, borehole 2), showed a large difference between the total displacement of anchor 5 and the overlying anchors.

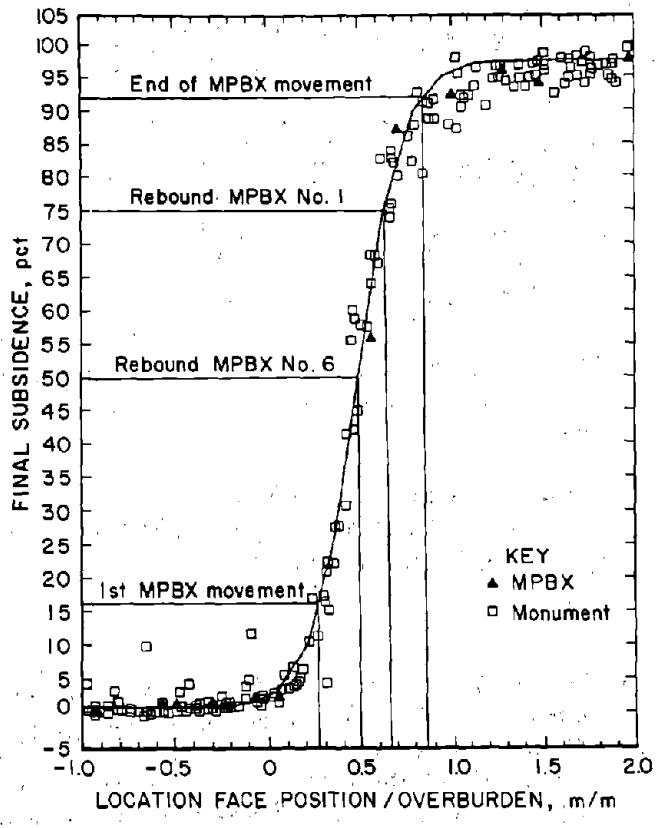


Figure 4. Final surface subsidence baseline data.

This indicates that the sandstone partially supported by the gate road pillars bridged over the collapsed zone at the margin of the mined panel.

Based on all the available data, it is thought that the overburden responded as two zones, a lower caved zone, and an upper zone more representative of a combined fractured and bending zone (fig. 6). It is believed that the lower zone extends from the top of the mined coalbed to the base of the 15.0-m-thick sandstone unit (or a maximum of 18.8 times the extracted coalbed height). The upper zone is interpreted to extend from the bottom of the sandstone unit to the surface.

The lower zone appears to have collapsed into the void area as a unit. The upper zone appears to be influenced by the thick sandstone unit, which is sagging and bending towards the center of the mined area. The lack of movement in borehole 3 indicates that these two zones are confined laterally to the area of the panel.

## Panel 12

When panel 12 was mined, boreholes 4, 5, and 6 were affected. Initial response occurred in the overburden areas along the margin of the panel. This may have occurred because the tail gate side of the longwall face (located close to borehole 5) was approximately 3.0 m ahead of the head gate side. Next, overburden movement progressed rapidly toward the center of the panel. In either case, the overburden appeared to collapse, at least initially, as a unit. Finally, movement was detected along the ribline of the panel (borehole 4). Anchor movement began when the longwall face was 17.1 m beyond the line of boreholes and continued until the face was about 72.0 m beyond the boreholes.

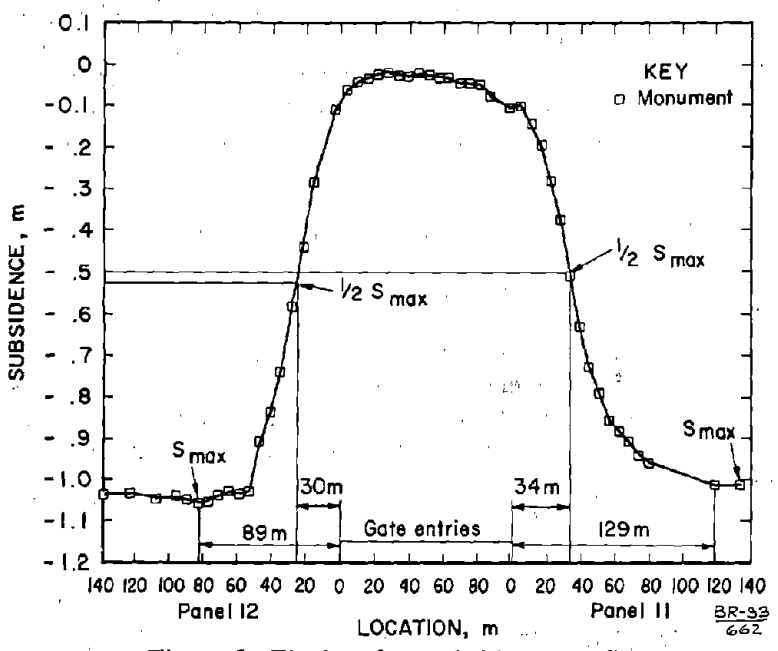


Figure 5. Final surface subsidence profile.

Table 1. MPBX anchor location (height (H)[1] and displacement (D).[2]

| | Borehole No. | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 1 | | 2 | | 3 | | 4 | | 5 | | 6 | |
| | H[1] | D[2] | H | D | H | D | H | D[3] | H | D | H | D |
| Ground surface | 64 | 58 | 74 | 31 | 78 | 0 | 81 | 0 | 82 | 41 | 84 | 58 |
| Anchor 1 | 54 | 64 | 64 | 32 | 69 | 0 | 70 | 0 | 73 | 68 | 74 | 61 |
| Anchor 2 | 50 | 65 | 57 | 32 | 50 | 0 | 58 | 0 | 63 | 72 | 62 | F |
| Anchor 3 | 46 | 66 | 54 | 34 | 36 | 0 | 52 | 0 | 50 | 80 | 50 | 62 |
| Anchor 4 | 38 | 73 | 38 | 39 | 17 | 0 | 36 | 0 | 39 | 81 | 38 | F |
| Anchor 5 | 26 | 84 | 27 | 92 | 7 | 0 | 25 | 0 | 31 | 86 | 27 | F |
| Anchor 6 | 16 | F[4] | 18 | F[5] | 2 | 0 | 13 | 0 | 19 | F | 19 | 64 |
| Anchor 7 | 8 | F | 9 | F | 1 | 0 | 4 | 0 | 10 | 88 | 9 | F |
| Anchor 8 | 6 | F | 5 | F[5] | 0 | 0 | 2 | 0 | 6 | F | 4 | F |

[1]Height expressed in meters extraction area.
[2]Total displacement expressed as a percentage of extracted thickness.
[3]Total displacement percentages less than 1% are expressed as 0.
[4]Anchor failed.
[5]Anchor failure, displaced beyond range of counterweight movement.

Final overburden failure characteristics for panel 12 are not well defined because the only anchors that remained reliable in borehole 6 were Nos. 1, 3, and 6. The displacement data from these anchors show that the strata in the center of the panel moved as expected. Anchors 7 and 8 failed as soon as movement began. Anchor 6 (positioned 19.2 m above the coalbed) had a total displacement of 1,168 mm or 64% of the extraction thickness (table 1). The magnitude of displacement for anchors 3 and 1 uniformly decreased progressing toward the surface.

At borehole 5, anchors 8 and 6 failed. Anchor 7 was the deepest reliable anchor and was positioned 10.2 m above the coalbed. The total displacement was 1,565 mm, or 88% of the extraction thickness (table 1). The displacement of the overlying anchors decreased uniformly towards the surface.

Based on the MPBX data, it is thought that response of the overburden in the center of the panel was different than along the margin of the panel (fig. 6). In the center of the panel, it is believed that the caved zone exists below anchor 6. The strata from anchor 6 and above uniformly responded as a combined fractured and bending zone. Along the margin of the panel, it is believed that the entire column of overburden collapsed as a unit into the void area with no other distinguishable zones of deformation. Finally, based on the lack of movement in boreholes 3 and 4, strata deformation is confined within the panel margins.

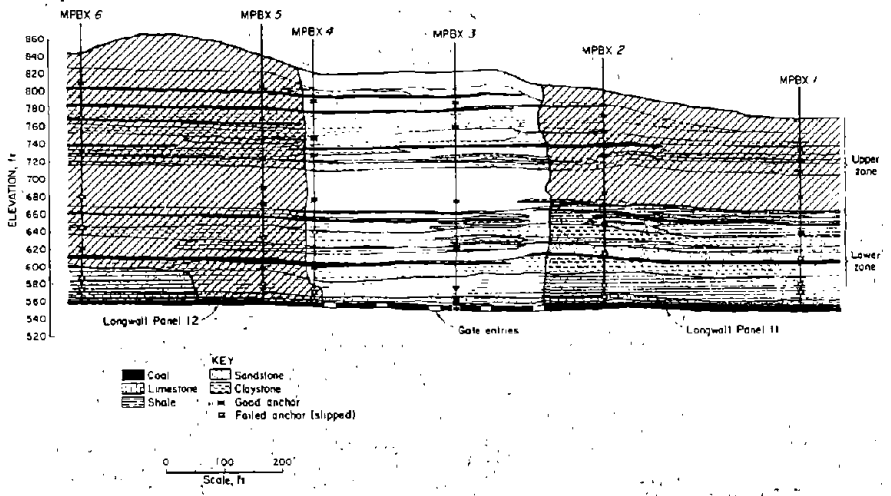


Figure 6. Interpretation of overburden strata collapse bases on MPBX information.

## Surface Subsidence

Several observations can be made when the final surface subsidence from the baseline array is compared with the displacements of the surface assemblages at boreholes 1 and 6. Figure 4 illustrates the combined surface subsidence data from both baseline arrays. By combining the two data sets, a reasonable dynamic subsidence curve is obtained. Plotting surface subsidence from MPBX boreholes 1 and 6 on this curve (indicated by triangles) shows the validity of using the composite curve to describe the dynamic movement for the borehole 1 and 6 surface locations.

When differential movement between the anchors and the surface began in both boreholes (i.e., anchors started to displace), the longwall face was about 25% of the overburden thickness beyond the boreholes. Surface surveys indicate that the surface had already subsided 15% of the total final subsidence at this time. When the surface at borehole 1 began converging on the anchors, the face was approximately 60% of the overburden thickness past the borehole. At that time, the surface had already subsided approximately 75% of the final subsidence. For borehole 6, convergence began when the face was approximately 50% of the overburden thickness beyond the borehole. At that point, the surface at borehole

6 had subsided about 50%. Finally, when the anchors stabilized, or the differential movement between the anchors and the surface ceased in both boreholes, the longwall face was about 85% of the overburden thickness past the boreholes. At this point, less than 10% of subsidence remained to occur.

Additional observations can be made with respect to the final surface profile (fig. 5). The maximum surface subsidence is not at the center of panel 12. This correlates with the interpretation that the strata surrounding MPBX borehole 5 collapsed as a unit rather than as two distinct zones as interpreted in MPBX borehole 6. This is to say that the unit collapse caused greater surface subsidence around borehole 5 by the absence of a caved zone that would have caused more strata bulking and less surface subsidence.

## Summary and Conclusions

Overall, there were significant similarities and differences in the development and final characteristics of the overburden response over two adjacent longwall panels. Both areas had similar mining geometries and geologic characteristics. Other major similarities observed from this investigation include –

1. Initial differential movement between the MPBX's and the surface began when the longwall face advanced 25% of the overburden thickness beyond the MPBX boreholes.

2. Differential movement between the MPBX's and the surface stabilized when the longwall face advanced approximately 85% of the overburden thickness beyond the MPBX boreholes.

3. Approximately 15% of the total subsidence occurred before, and less than 10% of the total subsidence occurred after, differential movement between the MPBX's and the surface.

4. There was no significant movement in the overburden over the gate entries during the mining of either panel.

Major differences that were observed during this investigation include –

1. Initial ground movement occurred in the center of panel 11 and along the margin of panel 12.

2. The development of overburden response to undermining occurred in stages over panel 11 and as a unit over panel 12.

3. Final overburden failure characteristics were different for the two panels. Two zones of deformation occurred uniformly over panel 11. Over panel 12, the failure characteristics in the center of the panel were identified as two zones of deformation and one zone of deformation along the margin of the panel.

While the mining geometries and geology of the two adjacent panels were nearly identical, the response to the subsidence process was not.

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# RESPONSE OF SURFACE SPRINGS TO LONGWALL COAL MINING WASATCH PLATEAU, UTAH[1]

Liane L. M. Kadnuck[2]

**Abstract:** High-extraction longwall coal mining creates zones in the overburden where strata bend, fracture, or cave into the mine void. These physical alterations to the overburden stratigraphy have associated effects on the hydrologic regime. The U.S. Bureau of Mines (USBM) studied impacts to the local hydrologic system caused by longwall mining in the Wasatch Plateau, Utah. Surface springs in the vicinity of two coal mines were evaluated for alterations in flow characteristics as mining progressed. Fourteen springs located above the mines were included in the study. Eight of the springs were located over longwall panels, four were located over barrier pillars and mains, and two were located outside the area disturbed by mining. Flow hydrographs for each spring were compared to climatic data and time of undermining to assess if mining in the vicinity had influenced flow. Heights of fracturing and caving in the overburden resulting from seam extraction were calculated using common subsidence formulas, and used in conjunction with elevations of springs to assess if fracturing influenced the water-bearing zones studied. One spring over a panel exhibited a departure from a normally-shaped hydrograph after being undermined. Springs located over other mine structures, or outside the mine area did not show discernible effects from mining. The limited response of the springs was attributed to site-specific conditions that buffered mining impacts including the elevation of the springs above the mine level, and presence of massive sandstones and swelling clays in the overburden materials.

**Additional Key Words:** underground coal mining, hydrology

## Introduction

Longwall mining involves extracting coal in large blocks called panels using a mechanized shearer. With this method, the mine roof is supported with hydraulic supports that automatically advance as mining progresses. As the supports move, the mine roof is allowed to collapse into the mine void. Strata above the mine level are altered as the mine roof caves behind the shields, creating zones where blocks of rock fill the mine void, or fracture or deform as rock layers warp downward. Recognizing that these alterations in overburden characteristics produce associated changes in the hydrologic system, a study was undertaken to observe and quantify the response of water resources contained in the overburden to mining.

Alterations to the hydrologic system resulting from longwall mining can include (1) increased permeability of the rock mass from fracturing and bed separation, (2) flow loss into mining-induced or dilated fractures or planes of bed separation, and (3) redirection of flow and hydraulic gradients. Studies in the Appalachian and Illinois coal regions indicate that rapid hydrologic response occurs in conjunction with the rapid subsidence induced by longwall mining (Coe and Stowe 1984, Trevits and Matetic 1991). In these studies, dramatic decreases in water levels in wells and flow from springs were observed as the longwall face passed underneath and was followed by subsequent total or partial recovery within months after mining.

Hydrologic response to high-extraction coal mining has become an issue of concern in both Eastern and Western coalfields. Previous studies have examined the effects of longwall mining on aquifers in the Eastern U.S., but few have focused on western U.S. coalfields where the aridity of the region generates concern for the availability of surface and ground water resources. Public sentiment targets coal mining as threat to the available water resources.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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The actual effects and longevity of underground coal mining on water resources in the West are largely unknown at this time. To observe and assess the response of water resources from longwall mining at a western coalfield, 14 springs over 2 mines in the Wasatch Plateau were studied. The springs studied were selected based on proximity to the mines and availability of monitoring records.

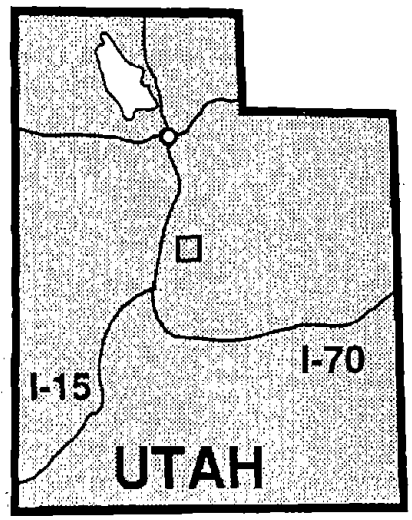

Figure 1. Study site location map.

## Site Description

The study site is located in the Wasatch Plateau coalfield of south-central, Utah (fig. 1).

Two mines, Mines A and B, separated by 26 m, operate at the site. Mine B is located above mine A and partially overlaps it. On figure 2, the outlines of both mines and locations of the springs studied are shown. Mine A is at an elevation of 2,265 m, and Mine B is at an elevation of 2,300 m. Both single- and multiple-seam longwall mining occur within the study area. Overburden thickness averages 520 m, and ranges from 260 m to more than 700 m, dependent on local topography. The topography of the area is typified by rugged, mountainous terrain dissected by stream-incised canyons. Local elevation varies from 2,100 m to over 3,000 m.

Exposed strata in the mine area are of sedimentary origin and range from Cretaceous to Quaternary in age. The stratigraphy consists of freshwater limestones, sandstones, and interbedded sandstones, claystones, mudstones, and shales of fluvial and lacustrine origins. Geotechnical testing of mudstone and claystone cores from the area indicate the presence of hydrophilic or swelling clays in the North Horn and Price River Formations. Economic coal deposits occur in the lower portion of the exposed stratigraphic column. The generalized overburden stratigraphy is shown in figure 3 (Lines 1985).

Recharge to the hydrologic system occurs through percolation of precipitation through matrix flow and into joints, fractures, and faults. Ground water occurs in discontinuous, perched water-bearing zones or in saturated fractures. Springs occur where fractures intercept the surface, or where the perched water-bearing zones intercept canyon walls. The presence of the hydrophilic clays largely retards downward migration of water, and therefore the majority of springs in the area occur in the formations containing the clays.

## Methodology

The springs studied were designated with a letter identifier of A to N. Locations of the springs occurring over the mines were correlated to mine structures such as panels, barrier pillars, or mains as shown in figure 2. Springs A, B, E, F, I, J, L, and M are located over panels, and springs C, G, K, and N are located over pillars or mains. The two remaining springs, D and H, occur in an area that has not been undermined, and were used as controls to reflect precipitation fluctuations.

Spring flows are collected by mine personnel from July through October due to limited accessibility to the springs during winter months. Based on this availability of data, spring flow was used in the form of seasonal recession curves that show flow trends from runoff to baseflow conditions. Spring occurrence based on issuing formation and mechanism (i.e., perched water table or fracture) was determined by the mining company. This information was incorporated into assessing pre- and post-mining flow characteristics for each spring. Spring flow hydrographs were compared to the time of undermining and precipitation to separate climatic effects spring response for the area and to assess if fracture-issued and perched zone-issued springs exhibited similarly shaped recession curves.

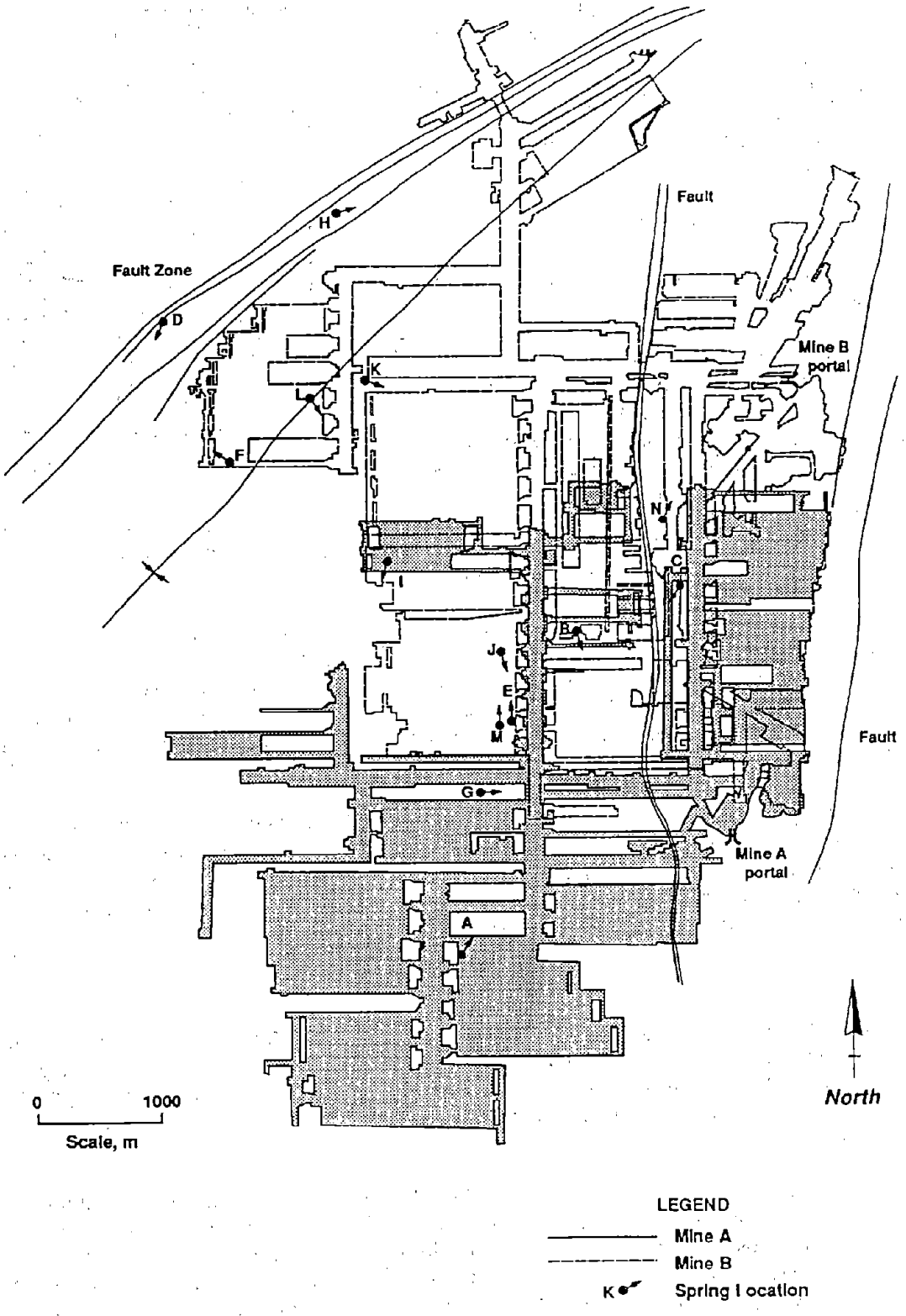


Figure 2. Map showing mine outlines and spring locations.

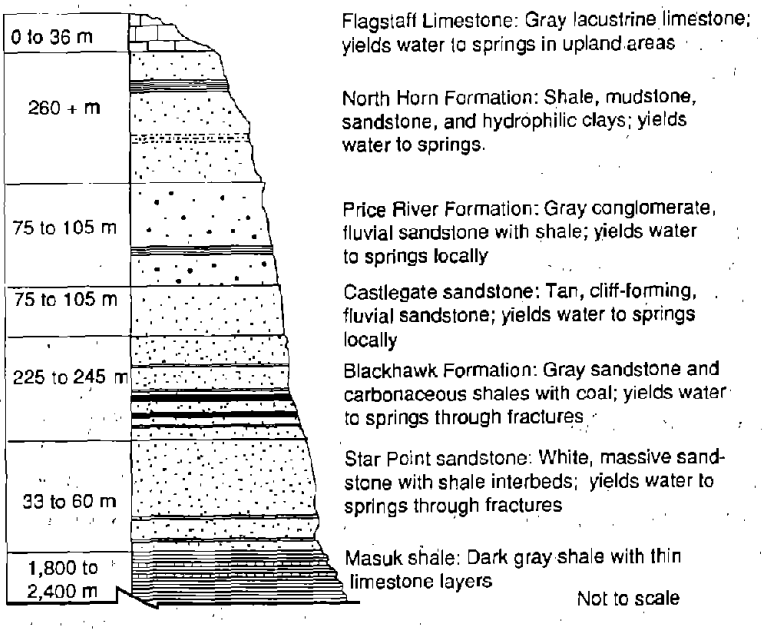


Figure 3. Generalized overburden stratigraphy.

Precipitation trends collected at a weather station at the mine site are illustrated on figure 4. Precipitation and spring flow measurements included data from several years prior to mining and were used to establish a baseline for comparison of spring flow behavior during and after mining. Patterns or trends in the spring flow recession curves were sought to assess if the springs showed similar response to climatic events each year. It is important to note that the region experienced years of decreased precipitation from 1987 to 1992, where precipitation ranged from 70% to 85% of normal.

Spring elevations were compared to anticipated heights of fracturing in the overburden as calculated using widely accepted subsidence engineering formulas (Peng 1992). This comparison allowed for evaluating the overburden fracturing elevation in relation to the elevations where springs were located.

## Discussion

Changes in the recession curves, either increases or decreases in flow that could not be attributed to climatic fluctuations, after mining had occurred would indicate a response to undermining. Based on results from other studies (Coe and Stowe 1989, Matetic and Trevits 1991, and Rauch 1989) dramatic decreases in flows would be expected after undermining. Springs located over panels are expected to show a greater response to undermining than those located over mains or barrier pillars since the majority of subsidence occurs over the longwall panels.

Each spring hydrograph was evaluated for normal response to recharge before

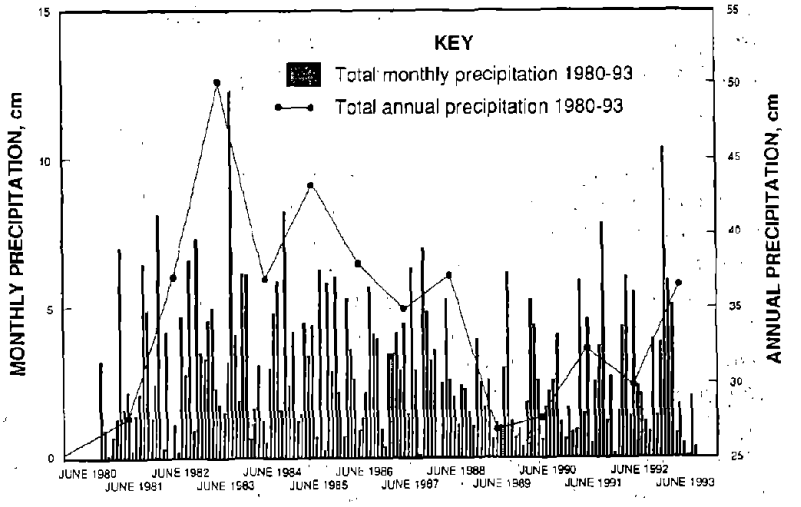


Figure 4. Monthly and annual precipitation, 1980-93.

undermining. The hydrographs of the springs studied all exhibited similar response to precipitation by showing peaks in months of runoff (June or July) grading down to baseflow conditions by September or October. This pattern was repeated each year, but varied dependent upon precipitation. There did not appear to be different climatic response between fracture-issued and perched-zone-issued springs.

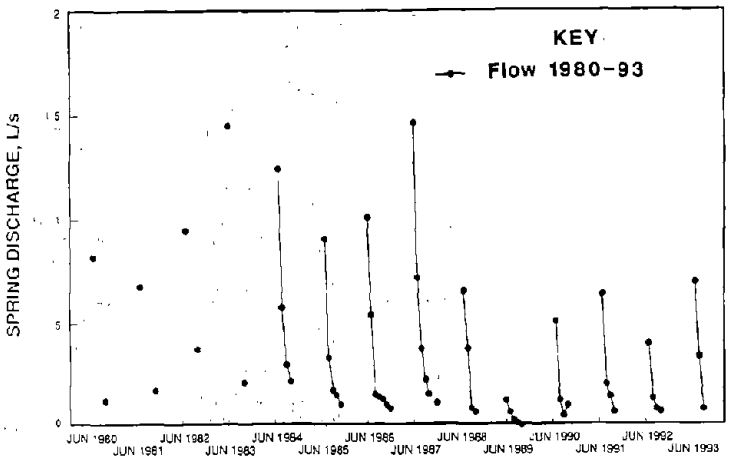

Figure 5. Spring D flow hydrograph.

Springs D and H (which were located outside the area of mining) both exhibited smooth, repeatable recession curves during the course of the study. Since these springs were located in an area that has not been undermined, their responses were used as a baseline for response to precipitation trends in the area. The hydrograph for Spring D is given on figure 5.

Of the springs that were undermined by longwall panels, only spring B showed a departure from the pre-mining hydrograph. Spring B (fig. 6) was undermined by submains in Mine B in August of 1985, and the hydrograph becomes erratic and less repeatable after this time. The spring was then undermined by a panel in Mine A in August 1993. Increased flow was observed after the spring was undermined by the panel, however this was due in part to clearing blockage in a pipe installed at the spring. Spring B occurs at an elevation of 2,822 m, 557 m above Mine A, and 522 m above Mine B. While flow loss was not evident after undermining, the departure of the curves from normal shapes after they had been undermined indicates a response that may be attributable to mining.

Spring elevations were compared to the anticipated elevations of overburden fracturing. To approximate the combined height of fracturing and caving in the overburden, the relationship of fracture height equal to 30 times mining thickness was used (Peng 1992). This relationship was developed for conditions in Eastern U.S. coalfields and may not be wholly applicable to conditions in Western U.S. coalfields. However, no relative value for estimation of caving and fracturing has been developed for western coalfields, thus this estimation was used. The approximated height of fracturing in the overburden correlated to an elevation of 2,600 m for Mine B, and an elevation of 2,500 m for Mine A. To represent both single and multiple-seam mining conditions, it was assumed that fracturing associated with mining the lower seam would be incorporated into the fracturing already present from mining the upper seam (Haycocks 1993). A conceptual subsurface overburden failure schematic is shown on figure 7. The lowest spring studied, Spring E, was at an elevation of 2,737 m, 137 m above the estimated fracturing height.

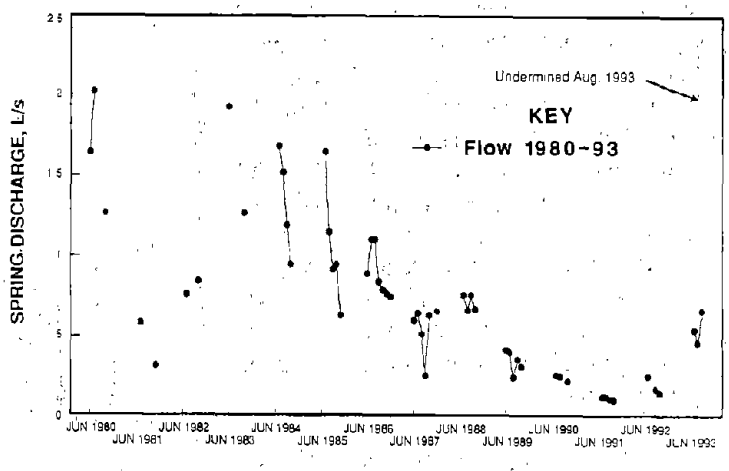

Figure 6. Spring B flow hydrograph.

While Spring B was not the lowest spring in elevation evaluated in the study, it was the only spring to show a response to undermining. The lack of observed response from other springs, especially from those located closer to the mine level, is difficult to ascertain with the data available. The most likely factor limiting response in the other springs is the geologic conditions present, specifically the presence of swelling clays, that buffer the effects of mining on overburden, and hence, hydrologic response. The observed response in Spring B may be

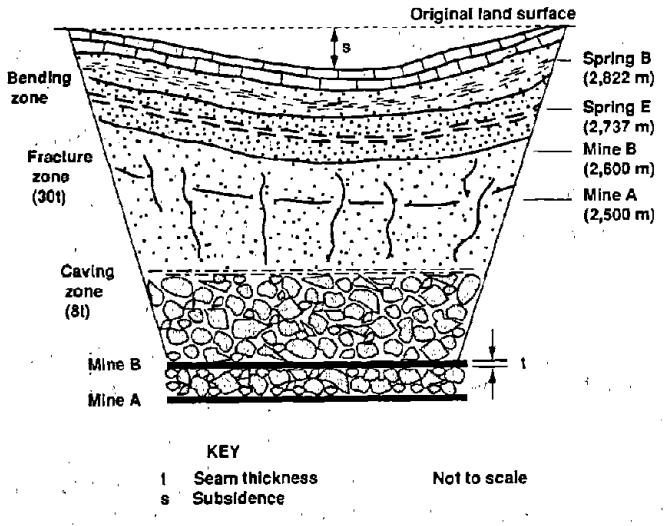


Figure 7. Overburden failure schematic.

attributed to redistribution of stresses in the overburden as subsidence occurred coupled with the fact that surface tension cracks appeared in the vicinity of Spring B.

In contrast to observations made in other studies, the springs studied did not exhibit dramatic effects from undermining. Coe and Stowe (1984) observed rapid and dramatic decreases in spring flow, followed by recovery to pre-mining conditions as the longwall face passed underneath springs 120 m to 145 m above the coal seam. The limited spring response observed at this site may be attributed to several factors; (1) the thickness and composition of overburden present which contained a massive, competent sandstone, which causes bridging in the overburden, limiting subsidence, (2) presence of formations containing swelling clays, and (3) the elevation of the springs above the elevation of estimated heights of fracturing and caving in the overburden.

## Conclusions

The U.S. Bureau of Mines performed a study of longwall mining effects on surface springs above two western coal mines using seasonal spring flow, precipitation, and overburden failure mechanics information. These data sets were used to quantify the response of spring flow to underground mining.

Results indicate that of 14 springs studied, 1 located over a longwall panel showed a departure from a normal recession curve after being undermined. The recession curve for Spring B became erratic after undermining but no flow loss was observed. The spring returned to normal response the following year. The departure from a normally-shaped recession curve may be a response attributable to mining. The other springs located over panels, or other mine structures did not exhibit discernible effects from mining.

Incorporation of commonly used subsidence calculations to estimate overburden failure response showed that heights of caving and fracturing in the overburden would not influence the water-bearing zones studied. However, stresses in the overburden could cause opening and closing of pre-existing fractures or bed separation, creating the temporary alteration of flow curves. The data available do not allow for establishing which mechanism would cause the erratic recession curves.

The observed response of springs to undermining at this study site was different than spring responses observed in studies performed in the Appalachian and Illinois coalfields. At these sites, dramatic decreases in spring flow and water levels in wells were noted. The limited response was attributed to site-specific conditions that buffered the

effects of mining. These conditions included the thickness of overburden between the mine level and the springs, which ranged from 440 m to over 600 m. Additionally, the presence of hydrophilic clays and mudstones in the overburden likely acted to lessen the vertical extent of strata dewatering, as has also been observed in other studies (Rauch 1989). The location of the springs above the computed elevation of combined fracturing and caving heights also contributed to the lack of discernible effects. The results of this study suggest that effects of mining on the springs studied given the site-specific conditions are minor and temporary. Longer term effects due to residual subsidence are not known and will be the target of future studies.

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# THE MINING ENVIRONMENT DATABASE ON ABANDONED MINES, ACID MINE DRAINAGE, AND LAND RECLAMATION [1]

Glen J. Kelly and Ronald J. Slater [2]

**Abstract:** Laurentian University Library has developed an on-line Mining Environment Database. The database provides references and abstracts to journal articles, books and government reports dealing with acid mine drainage, land reclamation, and abandoned mines. The database, created in 1988, now contains over 7,900 citations on reclamation planning, acid mine drainage, sulfide-based tailings, soil and water contamination, mine closure techniques, and other related topics dealing with mining environment studies. Subject coverage is international and focuses on hard rock mining topics. A stand-alone product for IBM-compatible computers is now available. The database is mastered on four high-density diskettes, and special search software is provided to allow full keyword searching of the database citations. Laurentian has acquired copies of all the materials cited in the database. Access to the on-line database is free of charge, with the exception of long-distance costs, and copy, delivery, or Fax charges for requested material. Suggestions for materials not found in the database and donations of pertinent research information from individuals, corporations, institutions, and government departments are welcomed. Private consulting reports (with the appropriate client approval) are especially welcomed, because this category of research literature cannot be purchased or obtained using normal methods, and is often discarded after a mine property is closed. The process of building a comprehensive research database requires a continuing partnership of information specialists and research users to develop a world class research literature database on mining environment and reclamation.

**Additional Key Words:** mining environment database, abandoned mines, acid mine drainage, land reclamation.

## Introduction

Increasingly, as mineral resources either are depleted or are no longer profitable with current conventional mining techniques, governments and the private sectors of both developed and emerging economies will be faced with an environmental problem of major proportions. What should be done with the played-out or abandoned mine site, and how can the land be restored to other, more ecologically friendly, purposes? The current level of funding for research into this problem will increase, as more and more abandoned mine sites will come to represent a significant global environmental, health, and safety hazard for the citizens of many countries. The need for a comprehensive research database on abandoned mines, land reclamation, acid mine drainage, and other mining environment issues has therefore never been greater. In the Province of Ontario, Canada, one such effort has been underway since 1988, in order to assist governments, environmental consulting firms, and mining companies in the planning, implementation, and assessment of effective mining environment policies.

## Background

In 1981, over 200 sq km of the total land surface of Ontario were directly affected by mining operations. These statistics are contained in Ontario's Mines and Minerals: Policy and Legislation, a green paper prepared in 1988 by the Ministry of Northern Development and Mines. The Green Paper outlined a blueprint for policy direction and priorities to be addressed in a revision of the Mining Act. Bill 71, which received Royal Assent in December 1989, significantly altered the Mining Act, in particular Part IX, which focuses on mining operations. All new and

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[1] Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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existing operations are now required to submit a closure plan to the Ontario Ministry of Mines for review and acceptance. Additional provisions related to claim staking, fees, and assessment requirements are contained in regulations accompanying Bill 71. As a result of the legislation, public safety issues associated with mine site hazards such as surface cave-ins, mine tailings sites, and open mine shafts, have created a demand for more information by government, the mining and research communities, and the general public. The regulations include specific requirements for mine managers to follow to stabilize tailings areas and to revegetate and restore mine and mine tailings sites before and after closure (Rev. Statutes of Ontario, 1980).

In December 1988, all levels of government and the mining industry recognized the need to undertake new research initiatives for the management of waste rock and mine tailings sites. The public awareness and concern for the environment, the requirements of new legislation, and the significant increase in research in the areas of abandoned mines, acidic mine drainage, tailings and mine spoils, and land reclamation techniques established a need for a more comprehensive research literature database available to the mining research community. In May 1988, the Ontario Ministry of Northern Development and Mines and the Ministry of the Environment began funding a series of Environmental Youth Corps projects at Laurentian University, mandated to develop a specialized **Mining Environment Database** on abandoned mines and land reclamation. In 1990 and 1991, MEND provided additional database development funding to focus on the global research literature pertaining to acid mine drainage topics. In October, 1993, additional funding was provided by the Ontario Ministry of Northern Development and Mines for data entry and preparation of abstracts during the 36-week period not covered by funding from the Environmental Youth Corps. Salary, equipment rental, supplies, commercial database searches, purchasing, and overhead costs amount to slightly over $225,000 since the project's inception in 1988. Over 6 yr, the database has grown to over 7,900 references and abstracts to journal articles, books, and government reports dealing with abandoned mines, acid mine drainage, and land reclamation. The primary focus of the database is on hard-rock mining topics. Topics include reclamation planning, design, and costs; tailings; heavy metals; disposal of hazardous wastes, including acid mine drainage, sulfide-based tailings, and asbestos particles; leaching; radioactive hazards of uranium tailings; soil and water contamination; soil stabilization; liming; fertilizers; seeding techniques; mine closure techniques; and other related topics. Subject coverage is international in scope.

## Database Access

The Mining Environment Database is accessible either directly, or as a stand-alone product for a PC. Direct access is possible by accessing Laurentian University Library's computer, with a modem and appropriate communications software. Access is also provided on the Internet, either directly at the Telnet address "**laulibr.laurentian.ca**," or via the Liberty gateway from Washington & Lee Law Library, Lexington, VA, "**liberty.uc.wlu.edu**." In September 1993, a stand-alone product for PC and compatible computers was introduced at the Ontario MEND conference in Sudbury, ON.

Database searching is free with the exception of long-distance costs, and charges for photocopy, delivery, or Fax. An IBM-compatible computer, a modem (up to 9,600 baud rate), and appropriate communications software are required. Kermit communications software and a valid user account and password are available free by sending a formatted diskette to Ronald Slater at the address shown below. Requests for materials in the collection should be directed to the Interlibrary Loan section, J. N. Desmarais Library, Sudbury, ON, P3E2C6, (705) 675-1151, ext.3318, or by Fax at (705) 673-6524. Copies of articles, or document delivery by Fax, are available for the cost of reproduction and delivery. Photocopying is limited to Canadian copyright provisions related to fair dealing for the purposes of research, and study and copyright fees may be charged.

**Stand-Alone Version.** A stand-alone product that resides on an IBM-compatible computer is now available. The database is mastered on four diskettes, and special search software is provided to allow full keyword searching of the database citations. The cost is CDN $275 for the search software and the first cumulation, which can be stored on a local hard drive with 20 megabytes of free hard disc space. Yearly updates will be available for CDN $100. An unlimited use site license will be available for each client purchasing a copy of the Mining Environment

Database. The software is available from the CIMMER Office, Laurentian University, Sudbury, ON, P3E2C6, (705) 675-1151 ext. 2280.

## Format, Scope, and Contents of the Database

An on-line catalog format was chosen for the database. A print-based format was considered, but not adopted owing to the cost of publishing the catalog and the delays in obtaining current information between updates. As the database grew, that decision proved to be even more cost effective, in allowing funding to be used for the purchase of more materials, abstracting, and data entry. In future, if there is sufficient demand, the database could be mastered on a CD-ROM disc and sold to clients at a lower cost than printing the catalog.

The Mining Environment Database contains references or citations to materials that have been added by purchase and by donation to Laurentian's collection on abandoned mines, acid mine drainage, and land reclamation. The largest category of material consists of journal articles, followed by papers in conference proceedings, and finally, government publications, reports, and books. The government publications, conference proceedings, and books are integrated into Laurentian's existing circulation and reference collections. The journal articles are housed in the library's periodical collection or, in the case of offprints, in vertical filing cabinets located in the Archives and Rare Books section of the library.

## Methodology Employed in Building the Database

Literature reviews are often time consuming, even when searching commercial on-line databases, because the results normally produce only a list of citations. The researcher must then attempt to obtain the articles, reports, or books from a local library or an unknown source through interlibrary loan. This process can take weeks or months. In contrast, because all citations are in the Library's collections, the Mining Environment Database can save valuable research time in locating and obtaining information.

To improve access to conference proceedings, a decision was made to provide full analytic coverage. This involved providing complete individual citations and subject analysis to all individual papers contained in the proceedings of a conference. The current practice of most libraries is to provide a single citation and general subject analysis for the proceedings of a conference, but not to the individual papers presented.

After consulting with a number of local researchers, it was decided to enter a short four to five-line abstract for each citation in the database, providing the user with a brief glimpse of the contents in each book, article, or report added to the database. Students were chosen to write the abstracts, based on their subject knowledge of the mining environment and their abstracting skills.

## How To Search the On-Line Database

Searching an on-line catalog requires practice, and a useful technique is to search subject terms in the Title Catalog before searching in the Subject Catalog. Entering a subject term in the Title Catalog will often produce a list of citations, and it is then possible to verify the correct subject heading used in the Subject Catalog by checking the assigned subject term used at the bottom of each record. Repeating a search using the correct subject heading will often result in a significant increase in citations, more than using only a Title Catalog search, as the subject term of interest may not have been used in the title of the book, article, or report.

Subject analysis, which has been completed for approximately one-third of the citations, is based primarily on terms found in Library of Congress Subject Headings (LCSH), the subject list used in most academic and research libraries in the United States and Canada. Because LCSH is designed for general research collections, it does not contain many specific terms used in specialized disciplines. Where appropriate, new terms and subdivisions have been created, based on terms provided by researchers and specialists using the database. Database maintenance and

quality control is an ongoing exercise, and the project editors welcome any suggestions from researchers or users for improvements or modifications to subject analysis, the method of presentation, or individual citations.

## Donation of Consulting Reports From Mining Research Community

Private consulting reports often contain essential scientific and factual evidence on particular mine properties and are prepared by individual researchers and consulting firms for individuals, corporations, institutions, and government departments. These reports, with the appropriate client approval, would be especially welcomed for inclusion in Laurentian's Mining Environment Database, because this category of research literature cannot be purchased or obtained using normal methods. Estimates from members of the mining research community indicate that this category of literature is substantial, and the number of such reports could easily be double the literature available through regular commercial sources.

In preliminary research for decommissioned mine sites, environmental consultants and mining companies are looking for answers to the following questions: (1) What environmental impact studies have already been done for the site; (2) what reclamation techniques were suggested for the site, and, if implemented, with what degree of success; and (3) for similar sites in other jurisdictions, what other reclamation strategies were attempted? Too often, during the development of a closure plan, consultants and companies devote considerable time and money to basic research that has already been done, either for the particular site or for another mine facing the identical problem.

At present, even a simple list of the consulting reports on a particular mine property is not available through any public research database. Unfortunately, as mining properties close, many consulting reports are often discarded, in the mistaken belief that no one in the future will ever be interested in reading a report prepared on a mining property that has been closed. The cost and the lack of storage facilities are also given as reasons for disposing of old reports. At a recent conference, one of the principals in a mine that had been closed was asked what he was going to do with the consulting reports that his firm had commissioned for the mine closure. His answer was that he was about to dispose of the material because he no longer had space in his office for the reports, which occupied 6 linear ft, and he had no idea that anyone would be interested in keeping them for future research. When asked if he was concerned about corporate secrets being made available if the reports were in a public research database like ours, he stated that he might be concerned about geophysical exploration studies that were done by the consulting firms, but he certainly was not concerned about releasing the engineering, operational, and land reclamation reports that were prepared either in house or by outside consultants for his firm.

Out-of-print material is another category of the research literature that is difficult to obtain, and is usually acquired through donations from individuals or companies.

## Invitation To Contribute

Building a research literature database and collection is an ongoing process involving concentrated and patient efforts by information specialists and research users. Suggestions for materials not found in the database, and for more appropriate subject terms, are welcome. A number of researchers who have already used the database have provided materials from their private and corporate collections and have offered to assist in providing a more comprehensive list of subject terms that are more meaningful to those working in the field. Suggestions for a more consistent and hierarchical subject catalog and for the addition of the Latin names of organisms for plant and species references will be adopted, with the assistance of specialists who have
volunteered to check and modify the subject heading terms that have already been assigned.

Laurentian encourages suggestions for improving the database, and donations of pertinent research information from individuals, corporations, institutions, and government departments. Income tax receipts may be issued for appropriate donations of materials upon request. If you have suggestions for materials that you feel should be in the database, or donations that you would like to contribute to enrich the collection, please contact the Gifts and

Exchange representative at (705)675-1151, ext 3322; Fax (705)673-6524.

If the materials you wish to donate require special archival conditions, Laurentian has a modern archival facility that has restricted access and special humidity and temperature control. The library can accommodate a donor's request for archival storage of records. For example, a donor may request that a particular document or report can only be released to researchers after a specified number of years. Laurentian can also accommodate requests for release forms signed by researchers, allowing the report to be read but not photocopied or used without the author's, or in the case of private consulting reports, the client's written approval. Laurentian maintains the right to refuse donations of materials that we consider will have little or no benefit to the collection, and that may contain use restrictions or other conditions with which we are unable to comply.

If it is not possible to provide Laurentian with an actual copy of a document, a simple citation listing with information on how a researcher could obtain access to the report would be welcome. As one consultant mentioned, even providing an author, title, date, and mine property list of the private consulting reports would be helpful to future owners of mine properties, consultants, and researchers. Access to a comprehensive and accessible research database on abandoned mines, land reclamation, and acid mine drainage can save governments, consulting firms, and mine companies many millions of dollars in the development of a closure strategy. All three partners must work to ensure that research reports are preserved and shared on a much wider basis than is done currently.

## Conclusion

The database is growing as a direct result of the continued support provided by the Canadian mining industry, federal and provincial government agencies, and mining consulting firms. We are confident that the Mining Environment Database will advertise and promote the efforts of the Canadian mining community in finding solutions to the problems and issues surrounding abandoned mines, acid mine drainage, and land reclamation.

The Mining Environment Database, initially developed for researchers in Canada, can now be made available to other countries having similar research interests, in an easy-to-use, stand-alone version. In addition, the active participation of the international mining community in improving coverage for other countries, where mine closure and mined land rehabilitation is an important environmental issue, is encouraged. The fact that the field of mining environmental studies covers an increasingly wide range of topics, combined with a phenomenal growth rate in the research literature, means that one database cannot hope to offer comprehensive coverage on an international scale. For example, the majority of citations in the Mining Environment Database deal with the environmental aspects of hard rock mining, with a particular focus on Northern Ontario and Canada. While a substantial number of references deal with coal mining, coverage in this area has been done on only a partial basis. Therefore, cooperation is also being sought with other mining research institutions, or libraries with a significant collection in mining environment studies, in the development of an international network of mining environment databases.

Consulting reports, as well as a mechanism for gaining access to and preserving this category of literature for future research, are critical to developing a more comprehensive information network on mining and reclamation. Donating materials that are about to be discarded to Laurentian, or to other local research or public libraries, will assist the library and archival communities in providing the mining community with the materials needed in future research efforts. The process of building a comprehensive database on the mining environment will require a continuing partnership of information specialists and users, in jointly developing a world-class research literature database and collection of materials on abandoned mines, acid mine drainage, land reclamation, and related topics of concern to the mining industry across Canada and around the world.

## ACKNOWLEDGEMENTS

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# HOW TO BUILD A COALITION FOR AMD AMELIORATION IN A WATERSHED. A CASE HISTORY: MILL CREEK OF JEFFERSON AND CLARION COUNTIES, PA[1]

Terry Morrow[1], Peter Dalby[2], Bernie Spozio[3], Gary Swope[4]

Abstract: Since 1990, the Mill Creek Coalition has been actively involved in the recovery of Mill Creek from the effects of AMD. This paper/presentation describes how a coalition of conservation groups, governmental agencies and private interests was formed to coordinate efforts to address the problems of AMD on a 60 square mile watershed in Western PA. Also described is how the Mill Creek Coalition obtained funding and other assistance to build a number of wetland treatment systems that are today improving the water quality of Mill Creek. Characteristics that have made this coalition successful are described and suggestions on how to build a successful coalition are shared.

## Introduction

Clarion and Jefferson Counties are located in an area of western Pennsylvania where surface mining has been occurring for decades. Surface mines have been superimposed on earlier surface mines and drift mines, and regulations and enforcement efforts have changed drastically over the years. Nearly all the smaller watersheds have been adversely affected in this area, with most lacking fish or having only limited diversity and numbers of organisms.

Because of the long history of surface mining, the effects of acid mine drainage (AMD) are of concern to most members of the community, including environmental groups, environmental consulting firms, the biology faculty of the local university (Clarion University of Pennsylvania), Pennsylvania (PA) Game Commission and PA Fish and Boat Commission, U. S. Soil Conservation Service (SCS) and Conservation District personnel and the elected officials in the region.

After some informal contacts with a number of key persons within the above groups, there was ready consensus to address AMD from a watershed approach. Individuals felt that this was feasible because of the development of new AMD treatment technology. The development of this technology, which has low cost and low maintenance requirements combined with long-term effectiveness, made it possible for a group such as the Mill Creek Coalition with its limited resources to attempt an effort of this scale. The specific technology selected by the Coalition involved anoxic limestone drains coupled with successive aerobic and anaerobic wetlands (Hedin 1989, McIntire et al. 1990). Several papers in this proceedings outline the specifics of the Howe Bridge Site treatment system (Hedin and Watzlaf 1994, Kepler and McCleary 1994) constructed on the Mill Creek watershed in the fall of 1991.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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During the time of informal dialogue, one watershed within the area received attention early in the process. This was the Mill Creek Watershed, which drains a portion of both Clarion and Jefferson Counties. There were a number of factors which caused the group to focus on the Mill Creek Watershed :

Mill Creek, with its main stem approximately 15 miles long and its entire watershed encompassing about 60 square miles, had its upper reaches only mildly affected by AMD. Some parts of the main stem and some tributaries had naturally reproducing brook trout populations. In addition, the PA Fish and Boat Commission stocked one section of Mill Creek. The last one-third of the main stem was lifeless owing to the accumulated effects of AMD, primarily from Little Mill Creek, a major tributary. The general consensus was that the watershed had the potential to be improved, and that if nothing was done, Mill Creek would deteriorate further due to a number of untreated discharges which were not on coal company permit areas or that were from mines closed before the current laws were enacted,

Mill Creek had received earlier attention from the U. S. Army Corps of Engineers and the PA Department of Environmental Resources, so the numbers, locations and seriousness of AMD along the watershed were well documented.

The watershed was equally located in both Jefferson and Clarion Counties. This was considered a plus because of the presence and expertise of two Soil Conservation Service (SCS) employees, Conservation Districts, and Resource Conservation and Development (RC&D) organizations (specifically Pennsoil and Headwaters Resource Conservation and Development Councils). Several of the environmental organizations (Trout Unlimited, an Audubon Society chapter and the county affiliates of the PA Federation of Sportsmen) received their membership from the area. Politically, a two-county effort had the potential to be more effective in identifying and acquiring financial support.

The lower half of the main stem of Mill Creek, which has the greatest potential for a recreational cold-water fishery, was located within the public domain as PA Game Commission Gamelands 74, thus ensuring public accessibility.

The watershed would not likely be impacted by future strip-mining activities because much of the area had already been mined and the remaining coal is of low quality. Additionally, the overburden in the area is of such a quality that there would be a high probability of additional AMD problems from any new mining activity. PA DER is currently reluctant to approve mining permits in areas such as this.

Mill Creek is easily within a 5-15 minute drive time for the main population bases of both Clarion and Jefferson Counties, Clarion and Brookville, respectively.

<u>Nature of the Problems in the Mill Creek Watershed</u>

We were able to identify the major AMD seeps on Mill Creek and Little Mill Creek by using Army Corps of Engineers and PA Department of Environmental Resources (PA DER) studies and by actually walking much of the watershed. We identified 18 different discharges. They varied some in chemical characteristics and flows, but they generally had flows of 10-50 gallons per minute and 50-250 parts per million (ppm) of iron, 250-500 ppm acidity, 0-7 ppm of aluminum, 1-15 ppm of manganese and 1000-1600 ppm of sulfates. The discharges are distributed along about ten miles of the watershed. Most of them are discharges from surface mines, but several are associated with abandoned gas wells.

## Mill Creek Coalition

### Organizational Meeting

After the informal contacts in late 1989 and early 1990, a day-long conference was planned for the fall of 1990. The conference was organized by Peter Dalby of Clarion University and the Clarion County affiliate of the Pennsylvania Federation of Sportsmen's Clubs, and Terry Morrow of Clarion University and the Iron Furnace Chapter of Trout Unlimited. The purpose of the conference was to bring groups together for a meeting to discuss the feasibility of improving the water quality of the Mill Creek Watershed, and forming a coalition to pursue these efforts. The following groups and agencies were invited to speak, focusing on their knowledge, expertise and the contributions they might be able to make toward the recovery of Mill Creek: PA DER Bureau of Surface Mining (direct knowledge of past strip-mining activity on the watershed, of existing treatment systems and of potential for future mining activities), PA DER Bureau of Oil and Gas (several gas wells were discharging untreated water into the watershed), PA DER Bureau of Forestry (expertise with Youth Conservation Corps program, a source for potential labor), PA Game Commission, PA Fish and Boat Commission, a local SCS representative, Damariscotta Environmental Consultants (a local environmental consulting firm specializing in AMD treatment), the PA National Guard, and the U. S. Army Reserves (sources of labor and heavy equipment), and the Army Corps of Engineers (authors of one of the studies on Mill Creek and the agency with responsibility for some waterways programs). After reviewing the presentations and a general question and answer period, the consensus was that Mill Creek was a reasonable challenge and that a formal coalition should be formed.

### Formation of the Mill Creek Coalition

The Mill Creek Coalition formed shortly after the conference and includes the following groups and agencies: The Alliance for Wetlands and Wildlife, Damariscotta Environmental Consultants, Clarion County Conservation District, Jefferson County Conservation District, Clarion County Federation of Sportsmen, Jefferson County Federation of Sportsmen, Iron Furnace Chapter of Trout Unlimited, Magic Forest of West-Central Pennsylvania, League of Women Voters of Clarion County, Seneca Rocks Audubon Society and the U. S. Soil Conservation Service. The Coalition eventually developed a constitution and bylaws and became incorporated as a Pennsylvania nonprofit corporation. Because monies could be directed through one of several Coalition member organizations which already had 501(c)(3) status, this tax exempt status was not pursued. Meetings are held monthly and minutes are taken. State agency people are specifically invited to attend particular meetings when their input is important for the business of the Coalition.

### Funding and Support

Funding, generally of relatively small amounts, has come from most of the Coalition members. Some outside organizations in the community have contributed up to several hundred dollars. The Coalition has approached the banking community and solicited funds. Through the Community Reinvestment Act, banks are required to contribute to the local community, and several donations ranging from $100 to $2500 were received.

Grants from Clarion University and the Clarion University Foundation awarded to faculty members involved with the Coalition have provided some support for the hiring of two or three students at any given time. For qualified students, the Pennsylvania Higher Education Assistance Agency (PHEAA) has paid 40 percent of the students' hourly wages. With students working 30 to 40 hours per week in the summer and 10 hours per week during the academic year, this expense is substantial. The students collect water samples and run water chemistry tests monthly, with some

stream and AMD sites being monitored every two weeks, along with the Coalition's treatment systems. Other than the aforementioned duties, the students do some survey work and assist Coalition members with general site preparation before and after heavy construction activities. While the local DER Office of Surface Mining has offered to assist in the sampling (and has done so on occasion), direct control of the students allows the Coalition to monitor with more flexibility and intensity than through DER. Since several Clarion University Biology faculty members are associated with some of the environmental organizations in the Coalition, they supervise the work of the students and the department provides space and equipment for water testing and other uses. Since the inception of field work in 1991, about $15,000 in student wages have been paid.

Larger amounts of money have come from other sources. The Iron Furnace Chapter of Trout Unlimited has donated about $9,000 of chapter funds to the project. Additionally, the chapter has submitted grant proposals to state and national levels of Trout Unlimited and has been awarded about $23,000 for work on Mill Creek. A regional trust approved a grant of $20,000 to support student help and wetland system construction. Four times in the past two years, the local PA DER Office of Surface Mining, along with cooperative coal companies, had fine money totaling about $14,000 given for the purchase of materials through a "reclamation in lieu of civil penalty" program.

Numerous people have given generously of their time, as is typical of many grass roots efforts. Damariscotta Environmental Consultants has donated all its time and some materials to the Coalition. Damariscotta Environmental Consultants' advice on the construction and designs of treatment sites is critical to the success of recovery efforts on Mill Creek. The SCS personnel are important in survey work, soils evaluation, sedimentation and erosion plans and obtaining conservation easements from property owners. The Conservation District offices have handled some of the paperwork and materials purchasing. The PA Game Commission allowed a major AMD discharge to be treated on Gameland 74 and assisted in site preparation.

Of primary importance was an agreement by the PA National Guard during the Coalition's first year to provide nearly all the personnel and heavy equipment necessary to construct one treatment system in 1991. Nearly $25,000 worth of materials were used at this site. The National Guard provided about $50,000 worth of service on the project. At another site in 1992, the National Guard completed two other systems, donating about $10,000 in services. In anticipation of additional, albeit reduced participation by the National Guard in the future, and also perhaps by the Army Reserves, the Coalition has helped arrange the purchase of a tract of land with six AMD sites on it. Since this property is now in the hands of the local RC&D Council, it is open to the public and eligible for work by the Guard or Reserves. Military units are not allowed to work on private property. Both groups receive numerous requests for help, so the Coalition hopes to have them working on Mill Creek occasionally, as time and their other responsibilities allow.

During the past three years, three treatment systems have been constructed on Mill Creek. However, there are another 15 discharges to address. While the Coalition was able to maximize its effectiveness due to the participation of the Guard and others at no or little cost, larger state and federal grants were prepared, thus freeing the Coalition from its dependence on the Guard and others who have given freely of their time. The DER Bureau of Oil and Gas successfully applied for $158,000 under the EPA's 319 program to fund the plugging of at least three flowing abandoned gas wells on the water-shed. This work will occur in 1994. Through the RC&D program and the local SCS, a grant for EPA's 319 program was submitted and approved for $165,000 to address three AMD sites on Little Mill Creek in 1994-95. Also, through the RC&D and the local SCS, a proposal has been submitted to the federal level of SCS to receive funding through its Public Law 566 program, which is designed to treat water quality problems on watersheds the size of Mill Creek and others in the area. The upper level of funding for this program is $5 million per watershed. One very positive feature of this

program is that it allows a group to address an entire watershed, instead of searching for funding to correct one or two discharges at a time. Nearly all of the federal and state programs have a requirement for matching funds or in-kind services. As a result, local financial support will remain a critical component to the Coalition's efforts. Future donated work by the National Guard, Army Reserves and others can be used to satisfy matching requirements of most granting agencies and foundations.

## Accomplishments and Conclusions

In its first two years of existence, the Mill Creek Coalition has been able to accomplish the construction of three major treatment systems on Mill Creek with a significant improvement in the stream's water quality in that area. Biological monitoring over the last two years has shown increases in fish and insect populations. One of these systems, the Howe Bridge Site, is currently removing about 150 pounds of iron and 300 pounds of acidity daily. The other two systems are similar in their performance. These three sites were the major abandoned seeps on Mill Creek. The Coalition is now focusing its attention on Little Mill Creek, Mill Creek's major AMD-affected tributary.

The Mill Creek Coalition has been successful because it has brought a large number of groups and agencies together. The Coalition has been fortunate to have a mix of talented and environmentally-active people and organizations. Ties to entities such as Clarion University, Damariscotta Environmental Consultants, PA DER and the PA National Guard have enabled the Coalition to get several successful projects built to demonstrate the utility of this approach. The initial sites, constructed primarily with grant money and donations, have formed the basis for credibility necessary to successfully compete for larger amounts of money through governmental programs.

Several conservation awards have been given to the Coalition and its member groups in recognition of their accomplishments. These awards have also increased the credibility and visibility of the organization. The news media are constantly informed of major developments through press releases, interviews and field trips. This has kept the local communities informed and interested in the project and its progress.

Other groups in other watersheds can use the Mill Creek Coalition's model to build their own coalition, utilizing the resources in their own area to accomplish similar results. It requires a group of dedicated individuals and a lot of hard work. Based upon our experience, important activities to pursue include the following:

Informally discuss a problem with representatives of other environmental organizations and agencies to determine if there is a serious interest.

Be as specific as possible about identifying the problem, keeping in mind that it should have high visibility, be logically and logistically possible, and benefit the greatest number of people.

Hold a conference (not appropriate for the general public) to bring everyone together to share information and to determine what expertise is available. Attempt to obtain, before the conference ends, a consensus of whether the problem can be realistically addressed.

Identify organizations and representatives of those organizations who are willing to be active members of the coalition and will work well together. Choose interim officers and

hold meetings on a regular basis. For legal purposes and to establish legitimacy, develop a constitution and bylaws and become incorporated by the state.

If not found within the coalition, identify those potential contributors who have the professional expertise to assist with various tasks. Identify sources of heavy equipment and materials which can be donated or acquired "at cost."

Identify and pursue local, as well as state and federal funding sources, and an agency that is willing and capable of handling the administrative paper work and the budgetary matters, including cash flow considerations.

Pursue awards, that if won, call attention to the group efforts and those who are working (paid and unpaid) on the project. Funding sources are also more likely to support projects where groups have established some credibility.

Inform the news media on a regular basis of the group's progress in order to maintain community support and to publicly acknowledge contributors and other key people and agencies.

*Never doubt that a small group of thoughtful, committed citizens can change the world, indeed, it's the only thing that ever has--*

Margaret Mead

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# AN INTRODUCTION TO LAND RECLAMATION PRACTICES IN CHINA[1]

Defu Peng[2] and Boheng Li[2]

**Abstract**: Land reclamation is a brand new field in China. The first part of this paper introduces the concept and the history of land reclamation there. The types of destroyed lands to be reclaimed, the extent of damage, and recent land reclamation practices are also introduced. The second part of this paper concentrates on the administration of reclamation policies and regulations. The third part provides some information about potential land reserves in China, then introduces academic aspects of land reclamation in China.

**Additional key words**: land use policies, reclamation regulations, derelict land

## Introduction

According to the statistical data offered by the United Nations in 1986, the mean of cultivatable and permanent farmland per person in China is about one-third that of the average global value, ranking China 67th in the world. Among the lands to be developed in China, the cultivatable land is only 34 million hectares. Thus, the cultivatable land shortage is very serious in China. Furthermore, China loses a large amount of land every year during its modernization construction. With this development, 2 million hectares of cultivatable land has been lost as a result of mining and industrial activities. During the last several years, the land loss was 20,000 to 30,000 hectares per year. An average land loss of 40,000 hectares in the year 2000 is predicted. The loss caused by mining is important. In some Chinese mining areas, it is reported that 0.4 hectares of cultivatable land will be damaged when 10,000 tons of coal is mined. This situation leads to severe land shortage and environmental pollution. Therefore, mine land reclamation has become an urgent task in China.

## Perspective of Land Reclamation Practice in China

### Concept of Land Reclamation in China

During the 1950's, "land reclamation" was called "restoring farmland" in China, which means "restoring the lands destroyed by mining to farmland." With the development of land reclamation in China, the purpose has become the "reutilization of all kinds of destroyed lands" in order to preserve land resources and improve the ecoenvironment. Thus, the principle of land reclamation in China is that the reutilization of destroyed lands should coincide with local conditions and requirements. Restoring farmland has the highest priority; reclaimed lands can also be used as forest, pasture, fishery, construction, etc.

The reclamation objectives in our country include the following types of destroyed lands:

(1) Abandoned lands destroyed by mining and industrial processes and other human activities;
(2) Abandoned roads, old buildings, and building sites, and the lands used for disposal of garbage and other wastes;
(3) Abandoned pits, pools, low-lying areas in the processes of building drainage systems and other farmlands' capital construction, and the abandoned odd pieces of land;
(4) Abandoned polluted lands;
(5) Lands destroyed due to natural calamity or other reasons.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

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## The Nature of Land Damage

Underground mining can damage land by causing (1) ground fractures and subsidence, (2) changes of hydrogeological conditions, such as the ground water level and waterflow, which may result in the accumulation of water on the land or land drought, and sometimes may produce swamp or salinized soils, or (3) accumulations of mining wastes, which pollutes lands. In China, thousands of mines produce more than 1.2 billion tons of ores per year. Of the mined ores, the coal output is more than 1.0 billion tons. The lands destroyed by mining will soon exceed 30,000 hectares per year.

Underground mining does not directly destroy the land surface. Some production processes such as surface mining, production of bricks, etc., need direct excavation of the lands, which may lead to (1) change of original soil structures, (2) spoil piles, which are subject to runoff and soil erosion, or (3) land pollution due to poisonous elements existing in the spoil piles. China's 100,000 brickyards destroy 70,000-100,000 hectares of lands each year. The total amount of fly ash produced from coal-fired powerplants was 70 million tons until 1990; 230 million tons will be produced in the year 2000. The fly ash does not occupy much land, but can pollute the lands. Finally, it is estimated in the year 2000 that 5,000 hectares of lands will be destroyed during the surface mining of 200 million tons of coal. Lands disturbed by mining and excavation now total 2 million hectares. Including the abandoned lands in the countryside and lands destroyed by mines or factories, the lands to be reclaimed may reach 8 million hectares. Therefore, land reclamation is an urgent task in China and is related to many industries, such as coal mining, oil, industry, urban development, chemical industry, etc. Also contributing to the problem is land damage or degradation caused by irrational production distribution or irrational land use structure and soil erosion due to denudation, inadequate reclamation, or herding. In China, the amount of land damaged by runoff, mud-rock flow, and other natural calamities total about 140,000 hectares per year.

## Land Reclamation Practices in China

Two thousand years ago, there was a large quarry near Shao-hsing city, Chekiang province. The excavated rocks were used for building houses, roads, and bridges. After 400 years of excavation, the quarry produced many sheer precipices and overhanging rocks and deep pools. At the end of the 19th century, Juinyi Tao built a bank for water accumulation and made some sceneries, which became the famous "East Lake" (in Chekiang Province). This is the first example of reclamation practice for tourism in our country. In modern times, land reclamation started in the 1950's. Some mines did some reclamation on their own initiative. For example, the Hengren lead-zinc mine (Liaoning province) reclaimed the abandoned tailings pool in 1957 and Xiaoguan Bauxite constructed 70-80 hectares of lands on mine spoil sites. Bantan Tin Mine used the overburden to fill in the mined area during or as part of the mining process.

The State Council of the People's Republic of China enacted the "Stipulation of Land Reclamation" on November 8, 1988, which formally established a land reclamation system in China. This stipulation has:

(1) Established 12 experimental reclamation sites in 8 provinces, which have different land damage characteristics and use different reclamation methods. This has produced reclamationists with experience in such areas as reclamation organization and management, reclamation funding, reclamation planning, and reclamation techniques.

(2) Spread information on the importance of land reclamation by television, newspapers, and other publicity means. Now land reclamation receives great attention and support from the Chinese people and its government.

(3) Improved reclamation policies and regulations. After the enactment of the state reclamation stipulation, some provinces and departments have made detailed regulations for carrying out the state reclamation stipulation, for example, the "Stipulation of Land Reclamation Planning in Metallurgical Mines," the "Stipulation of Land Reclamation in Gulch-Gold Mines," and "Standards of Land Reclamation."

(4) Established a reclamation management network. In China, land reclamation management works in two ways: (a) state - Province - local - county - country, or (b) state council - industry ministries of state council - mines or factories.

## Policies and Regulations of Land Reclamation in China

The shortage and damage of cultivatable lands have recently received great attention by the Chinese people and government. "Highly valuing and reasonably utilizing every centimeter of land" has been stipulated as one of China's basic national policies. In addition to the "Stipulation of Land Reclamation," some related laws have also been enacted. In 1986, the government of China enacted the "Law of Land Administration of the People's Republic of China," which provides that the land utilized for industrial purposes must be limited and the destroyed land must be reclaimed. In 1989, the "Law of Environmental Protection of the People's Republic of China" was put into effect, which provides protection of land from erosion and damage to protect the Chinese environment. In 1986, "The Law of Mineral Resource Protection" was enacted.

The "Stipulation of Land Reclamation" has 26 articles, the main contents of which state that:

(1) The purpose of land reclamation is for rationally using lands and improving the ecoenvironment.
(2) Land reclamation means the activity of reclaiming the destroyed lands into a reusable state.
(3) The principle of land reclamation is "the destroyed land must be reclaimed by the one who destroyed the land." All the mines (or businesses) or people who destroy lands have the duty of reclaiming the destroyed lands.
(4) The land administration department of local governments manage and control their local reclamation work. The industry ministries of state councils, such as coal, oil, geology, metallurgy, etc., manage and control the reclamation planning and practice in their industries.
(5) Land reclamation planning should coincide with the master land use plan.
(6) The stipulation is overseen by a State Land Administration.

## Land Reserves Suitable for Farming

Taking account of the state's economic development and land use capability, underused land prospects include the following:

(1) The beachy land along the coast of China should be developed. The land near the mouths of the Yangtze, Yellow, and Zhujang Rivers and around embayments such as Bohai Bay total over 16 thousand hectares, have temperate climate, abundant rainfall, and rich soils. Analysis of the unused land along the middle and lower reaches of the Yangtze River indicates that 34% is easily developed, arable lands.
(2) 6.7 million hectares of unused lands adjoining roadsides and around villages and housing areas can become rich farming land if leveled enough.
(3) Derelict land totals 3.4 million hectares and is increasing at a rate of 33.3 thousands hectares each year. Much of this was once cultivatable land. Mining in eastern China has especially led to the loss of a great deal of cultivated land, much of which has subsided as much as 5 m. To further advance Chinese land reclamation, the "Stipulation of Land Reclamation" by the State Council should be enforced.

(4) Over 50% of China's land reserves lie in three remote provinces: Xinjiang Autonomous region, Inner Mongolia Autonomous region, and Yunnan province. This land is relatively sparsely populated but is subject to drought. Potential development requires tapping of its water resources. Analysis indicates that 27.3% of these reserves consist of superior arable land, while the rest is rated moderate or inferior.

## Education and Research on Land Reclamation in China

### Education of Land Reclamation

Some universities such as China University of Mining and Technology (CUMT), Shansi Agricultural University (SAU), and T'ang-shan Institute of Technology have had land reclamation classes and short courses. In addition, some books on land reclamation have been published.

### Land Reclamation Research

Many research institutes and universities such as T'ang-shan Branch of Central Coal Mining Research Institute, CUMT, SAU, Geography Institute of Academia Sinica, and Beijing General Research Institute of Mining & Metallurgy have done research projects on land reclamation. Some of the research results have been used in reclamation practice. The focuses of land reclamation practice are now on mining subsidence, coal waste piles, abandoned mined sites, tailings pool, fly ash, brickyards, abandoned roads, and abandoned village sites. So far, about 2 million hectares of lands has been reclaimed. The reclamation rate is about 6%. The Beijing General Research Institute of Mining & Metallurgy is doing an international project on mine waste reclamation with Australian experts.

### Academic Activities

The China Land Reclamation Society was set up in 1985 as a branch of the China Land Association. It has four technical divisions: surface mining reclamation, underground mining reclamation, policies and regulation of land reclamation, and land reclamation management. This society has an annual meeting to discuss problems and new techniques in land reclamation practice. It produced a book entitled "Selected Papers Related to Land Reclamation, Vol. 1" (the selected papers are from the first through the fourth national reclamation conferences: 1985-1991). The society has more than 300 members and 93 group members. Some members have visited foreign countries, such as the United States, Poland, Germany, and Australia to exchange land reclamation technology.

# DISTURBANCE OF SURFACE STREAM DUE TO LONGWALL MINING[1]

Felicia F. Peng, Zhouming Sun and Syd S. Peng[2]

**Abstract:** Surface streams over longwall panels, like other surface features, may also be affected by underground mining. Disturbed surface stream beds and surface water bodies may cause surface environmental problems and create potential water hazards to the underground mining operation. When a coal seam is extracted under a surface stream, the stream water may form migratory ponds following the surface waves created by the dynamic surface subsidence process. If a thin overburden exists, the water in a migratory pond may fall into the surface dynamic cracks as it travels across the longwall panel. After the surface has reached the final subsidence stage, the migratory ponds will cease moving and stop near the chain pillar area and form a stationary pond. Loss of water may occurs in a stationary pond, because it is located in the high tension zone (or surface crack zone) where its secondary permeability increases. This paper presents a field investigation, including monitoring the stream flow, stream water depth, and surface subsidence. The results discover the phenomenon of pond formation, identify the major factors contributing to stream disturbance and lead to the development of mitigative measures and remediation activities.

**Additional Key Words:** subsidence damage, migratory pond, stationary pond, subsidence mitigation

## Introduction

A recently observed problem over a longwall mining area is the formation of water ponds along a stream valley. The formation of stream ponds may cause problems for both the surface environment and the underground mining operations if one or more of the following conditions exist: (1) disturbance of water supplies for farming, animal, and domestic uses, (2) changes in stream path and natural appearance as well as loss of stream water, and (3) a large body of water has formed on the surface which may eventually enter into the mine workings. The phenomenon of water ponding over the high-extraction mined areas has been described previously (Ackman and Jones 1988; Stump 1992). However, correlation between the magnitude of ground subsidence and the disturbance of surface stream has not been reported.

Field investigation in this project included monitoring stream flow rate, stream water depth, and surface subsidence along a main stream over three successive longwall panels. Considerable data have been obtained as results of the carefully designed monitoring program. The results discover the formation phenomena of stream ponds, identify the major factors contributing to stream disturbance, and lead to the development of mitigative measures and remediation activities.

## Field Investigation

### Site Description

Field investigation for the formation process of a number of stream ponds was conducted along a main stream which ran transversely across five longwall panels. These longwall panels were designated as panel 1 to panel 5 according to the mined sequence, as illustrated in figure 1. The panels were 198 to 229 m (650 to 750 ft) wide and 763 to 854 m (2,500 to 2,800 ft) long. The coal seam was 2 m (6.5 ft) thick and was mined from northwest to southeast at face advance rates ranging from 9 to 20 m (30 to 65 ft) per day. The surface around the study site was

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

[2]Felicia F. Peng, Assistant Professor of Department of Mineral Processing Engineering, Zhouming Sun, Graduate Student and Syd S. Peng, Chairman and C.T. Holland Professor, Department of Mining Engineering, West Virginian University, WV, USA.

slightly hilly with several houses built on it. The overburden depth was 70 to 107 m (230 to 350 ft). There were also several domestic wells and springs. These wells and spring water stored in cisterns were the main drinking water sources for local residents. The stream water and springs also served as the major water sources for some domestic animals and other usage. The main stream bed was about 1.1 m (3.5 ft) high and 0.4 m (1.2 ft) wide, and its water varied from 0.8 to 0.9 m (2.5 to 3.0 ft) deep. It flowed from the tailentry of panel 1 to the headentry of panel 5 at approximately 315° to the mining direction. The first two panels (longwall panel 1 and longwall panel 2) had been mined out before the commencement of this field investigation. Two water ponds were formed along the main stream inside and near the headentry of the two mined-out panels.

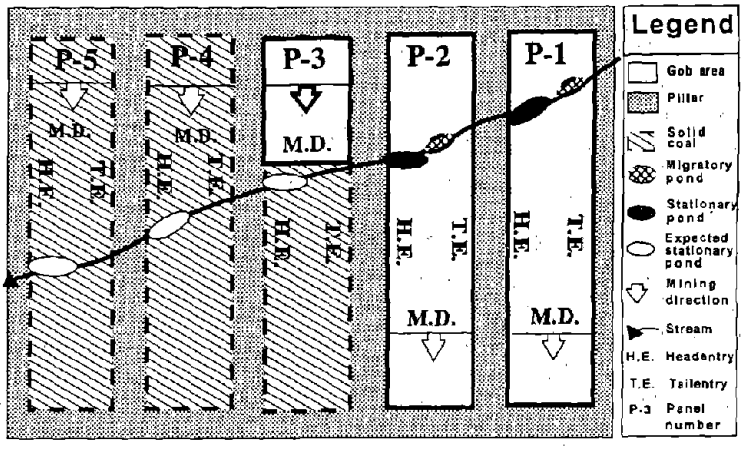


Figure 1. Longwall panel layout and location of stream ponds

To investigate the remaining three panels (panel 3 to 5), three types of monitoring stations, namely, stream flow station, water level station, and subsidence monument, were employed. Two stream flow stations used as reference points were installed in the upstream, $R_{up}$, about 1434 m (4,700 ft) from the headentry and in the downstream, $R_{dw}$, about 305 m (1,000 ft) from the tailentry of panel 5. They were used as the baseline to determine water loss at other monitoring stations.

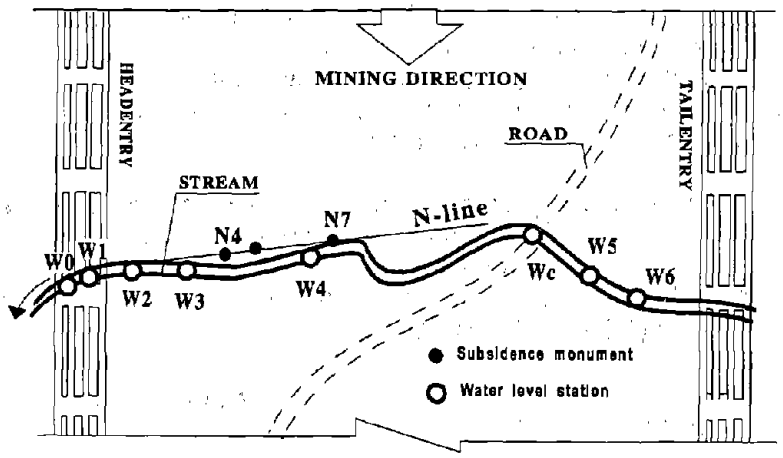


Figure 2. Water-monitoring stations and subsidence line over longwall panel 3

## Installation of Water and Subsidence-Monitoring Stations

The stream flow stations consisted of rectangular and V-notch weirs made of wood and earth dam (Bureau of Reclamation 1984). A subsidence monument was made of wood sticks in size of 2.5x5x30 cm (1x2x12 in). A total station was employed for subsidence survey. Various lengths of the wood sticks attached with rulers were used for the water level stations. The installation of the water and subsidence monitoring on each longwall panel are described as follows: *On longwall panel 3,* twelve stream flow stations along the main stream and seven subsidence monuments along the N-line were installed, respectively, as shown in fig. 2. All subsidence monuments were placed a short distance off the main stream close to the headentry of longwall panel 3 so that they would not be flooded as the surface subsided. *On longwall panel 4,* the layout of monitoring stations over the panel is shown in fig. 3. It was a long dry period during the earier stage of the measurement period. A low flow rate at the upstream, $R_{up}$ was recorded. The stream water in the previously

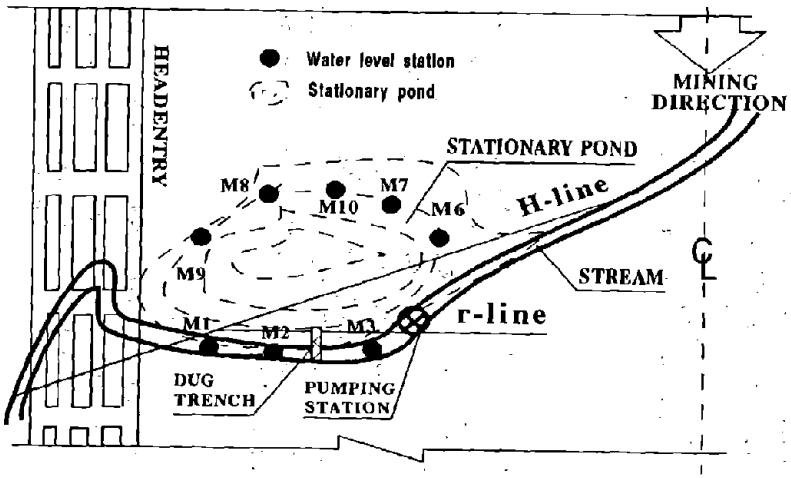


Figure 3. Water-monitoring stations, subsidence lines, and stationary pond over longwall panel 4

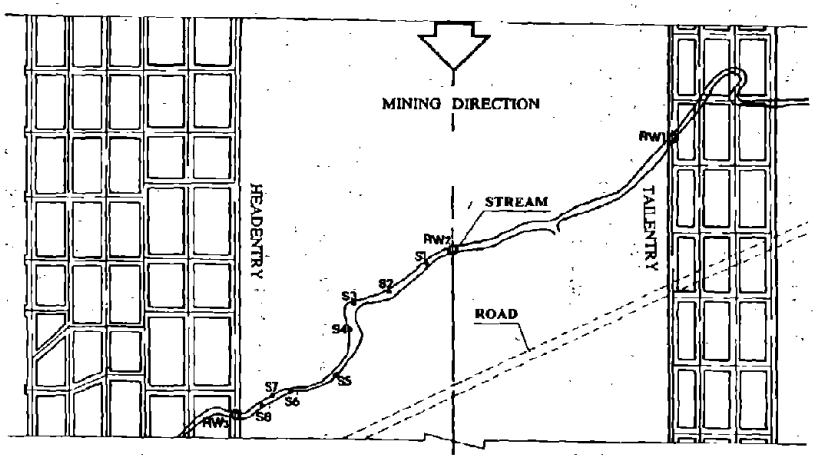


Figure 4. Water-monitoring stations along S-line over longwall panel 5

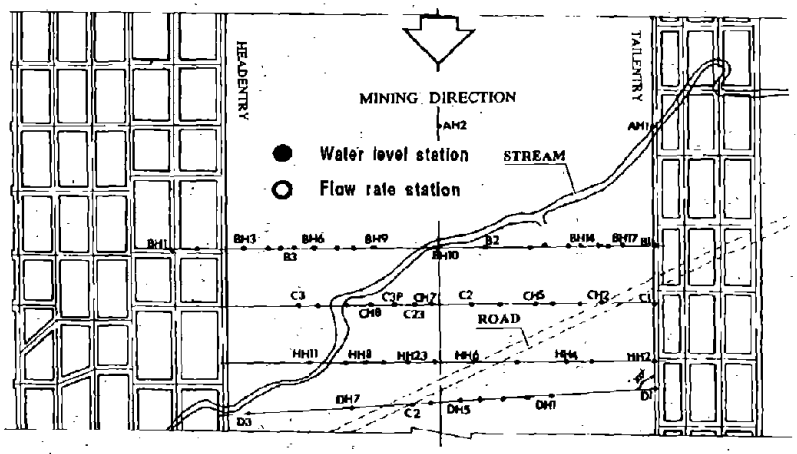


Figure 5. Subsidence monuments over longwall panel 5

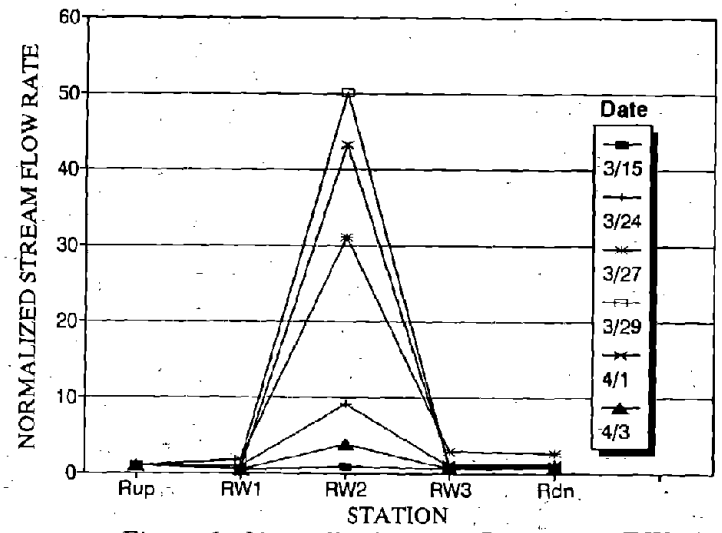


Figure 6. Normalized stream flow rate at RW1, RW2 and RW3 over longwall panel 5

mined-out longwall panel 3 was also dry out due to the lost of the water to the surface cracks. In later stage of the measurement period, the accumulated stream water near the headentry of longwall panel 4 was observed after the crack closures in longwall panel 3. As a mitigative measure, the accumulated stream water was pumped over to the adjacent longwall panel 5 using the sump, pump and a 10 cm (4 in) ID vinyl hose. The monitoring was focused on the flow rate of the springs and the reference stations, the changes of water levels, and surface subsidence on the r-line along the main stream. *On longwall panel 5*, the stream flow stations, stream water level stations, and subsidence monuments were systematically installed over the longwall panel 5, which covered a rectangular area 229 m (750 ft) long by 122 m (400 ft) wide, as illustrated in figures 4 and 5. Periodical surveys of stream ponds and subsidence were performed over the area.

## Results and Discussion

### Normalized Stream Flow Rate

The measured stream flow rates over the three panels have considerable fluctuation from day to day and from station to station (Sun 1993). The unstable stream flow rates may be caused by the following factors: movement of stream bed due to surface subsidence, change in precipitation, varying sources of stream water, feeding of small tributaries into the main stream, and errors in measurements of the head of stream water. Among these factors, surface subsidence plays a major role in the fluctuation of the stream flow rate.

The normalized stream flow rate (*NSFR*) is employed to represent the relationship between any station of interest $i$ and a reference station $r$ in any given day $j$. *NSFR* is defined as $Q_{i,j} = q_{i,j}/q_{r,j}$, where $Q_{i,j}$ is the normalized stream flow rate at station $i$ on day $j$; $q_{i,j}$ is the measured stream flow rate at station $i$ on day $j$, and $q_{r,j}$ is the measured stream flow rate at reference station $r$ on day $j$. Before the longwall face approached the stream, the initial flow rates at every station were measured. If *NSFR* at any given station $i$ is greater than the initial *NSFR*, the stream water is considered to have water accumulated around that station. On the other hand, the stream water is considered to have water lost if the *NSFR* is less than the initial one. The *NSFR* of longwall panel 5 is shown in figure 6. The results indicate that the *NSFR* at stream flow station RW2 was greater than that at other stations. It increased first and then gradually decreased. This phenomenon was contributed by the movement of the body of water along the main stream as a result of the moving surface subsidence waves. When surface subsidence was over, the *NSFR* for RW2 was still higher than its initial measurement. This is because RW2 was located at the panel center, where the maximum subsidence occurred.

### Migratory and Stationary Ponds

The migratory (or transient) and stationary (or relatively stable) stream ponds were observed over all five longwall panels during the longwall mining. The development of these ponds can be described by the variations of the water depth measured in the vicinity of the stream, as shown in figures 7 to 8 for panel 4 and 5, respectively. The data show that the water depth increased first, and then gradually decreased until it reached a relatively stable level. This process results in the formation of the migratory and stationary ponds. The migratory pond moves from the tailentry (upstream) until it reaches the stable stage to form a stationary pond close to the headentry (downstream). The water depth of a stationary pond may change, but the geometrical center of the pond is relatively constant. For example, the formation of the stream ponds (i.e. pond 1 to pond 5) in panel 5 is schematically shown in figure 9. Pond 5 is a stationary pond settled by the headentry of panel 5.

### Impact of Surface Subsidence on Stream Ponds

The disturbed stream flow rate and water depth over longwall panels are quite different from a pure channel flow. A significant movement of the stream bed caused by surface subsidence leads to a pronounced change in stream flow rate and water depth. Surface subsidence was surveyed for all three panels as shown in figures 10 to 12. Surface subsidence propagates like a wave and causes the formation of the stream ponds. Correlation of the

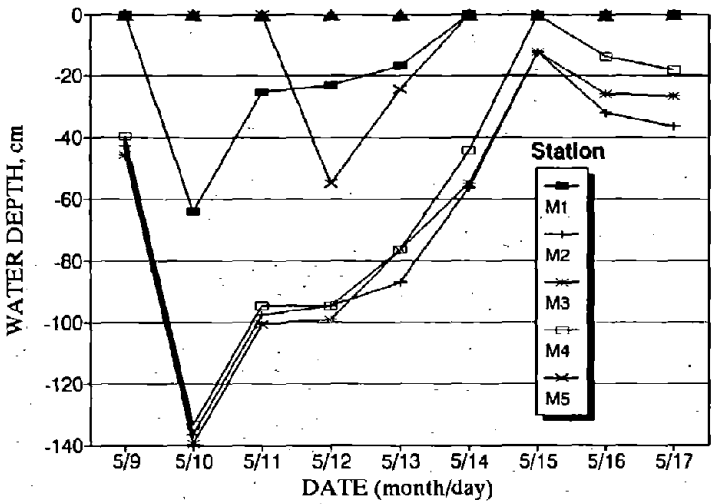


Figure 7. Water depth as function of time along M-line over longwall panel 4

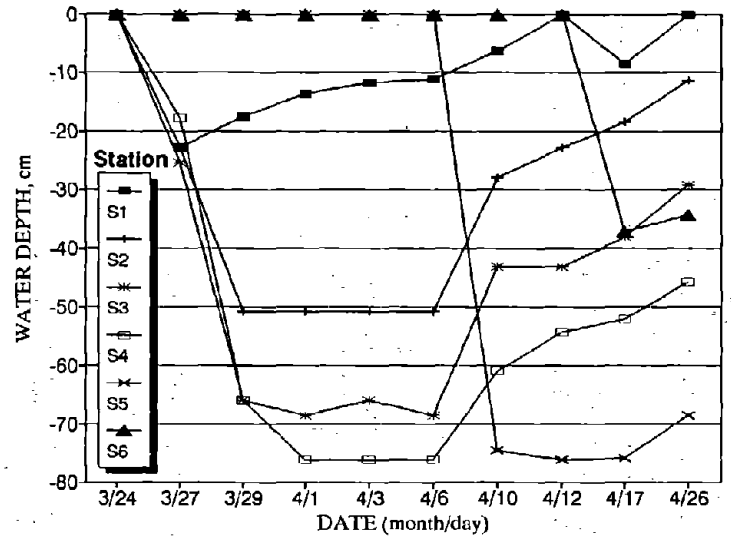


Figure 8. Water depth as function of time along S-line over longwall panel 5

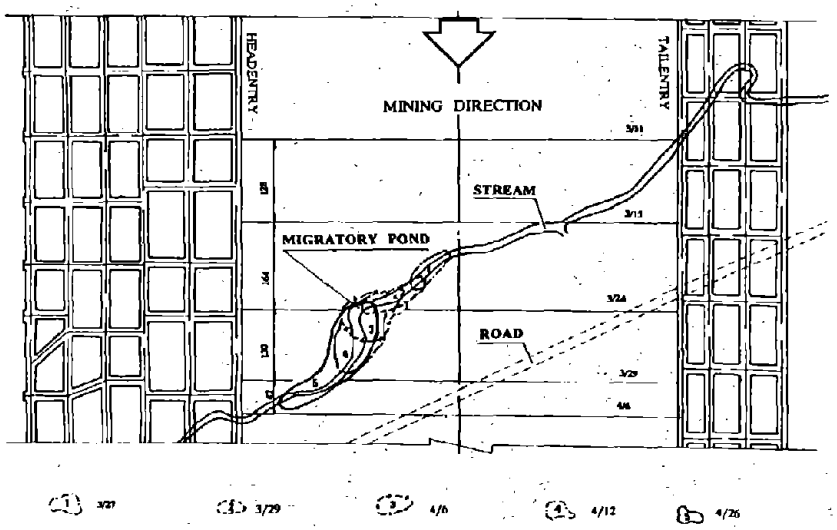


Figure 9. Formation process of migratory ponds over longwall panel 5

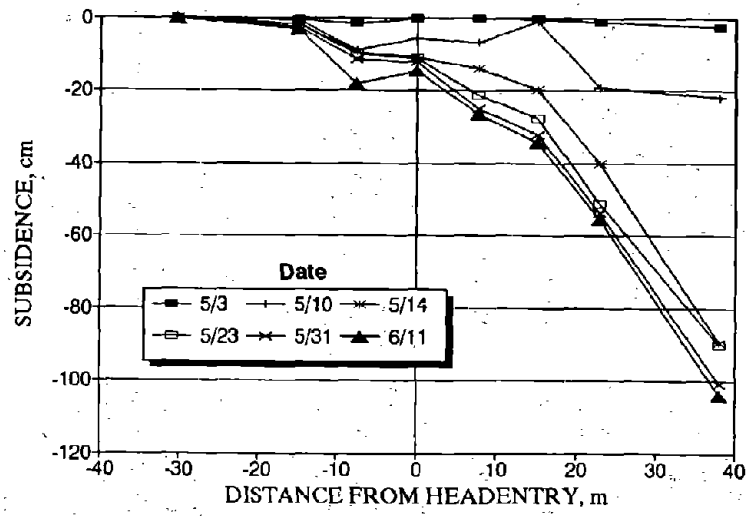


Figure 10. Subsidence profile along N-line over longwall panel 3

formation process of a stream pond at W4 with the surface subsidence at monument N7 over longwall panel 3 shows that the formation of the migratory pond corresponds to the stage 1 of the dynamic subsidence process (Peng 1993) as illustrated in figure 13. At the end of stage 1, surface subsidence at N7 had reached about 85% of its maximum amount, and the migratory pond approached a stable stage. Both the pond and subsidence were stabilized during stage 2 (85 to 90% of maximum subsidence). Surface subsidence increased slightly during stage 3 (more than 90% of maximum subsidence), but the water depth of the stationary pond kept almost the same as the previous level, or decreased slightly at the end of stage 3.

## Effect of Angle of Stream Flow

The orientation of a flowing stream over a longwall panel can be classified into four types of patterns based on the angle of stream flow. The angle of stream flow is defined as the angle between the advancing direction of the longwall face and the flowing direction of the surface stream (measured clockwise) as shown in figure 12. From these four types of stream flow patterns (table 1), the formation of the migratory and stationary ponds for a given longwall panel can be predicted.

Accordingly, the defined angle of stream flow takes into account the surface stream flow conditions, and surface subsidence as well as the mining direction. It is a method to determine the severity of stream ponding under various mining conditions. The development of the angle of stream flow and its application have been described in detail elsewhere (Sun 1993).

Table 1 Relationship between angle of stream flow and stream ponds

| Stream flow pattern | Angle of stream flow $\varphi$(°) | Migratory pond | Stationary pond |
|---|---|---|---|
| Flow 1 | $0 <= \varphi < 90$ | Yes | Yes |
| Flow 2 | $90 <= \varphi <= 180$ | No | Yes |
| Flow 3 | $180 <= \varphi <= 270$ | No | Yes |
| Flow 4 | $270 < \varphi <= 360$ | Yes | Yes |

For example, if the angle of stream flow belongs to the flow 1 (or flow 4) pattern and the adjacent new panel is on the south (or north) side of the panel being mined, the migratory pond will occur and travel along the stream. The stationary pond will then be formed by the headentry of the panel being mined. The migratory pond will disappear if the angle of a stream flow is in flow 2 or flow 3 pattern. In flow 2 pattern, the stationary pond by the headentry has more influence on mining the adjacent new panel than the flow 3 pattern, if the adjacent new panel is on the south side. The optimum angle of stream flow is 90° or 270°, provided the adjacent new panel is located in the opposite direction to the stream flow. In this configuration, the migratory pond will not occur and the stationary pond will settled by the tailentry instead of the headentry.

Based on the field data, a subsidence-controlled computer model has been developed for predicting the formation process of the stream ponds (Sun 1993). The trend surface analysis technique and Knothe's theory (Knothe 1957) are employed to determine the static stream water elevation. By using this model, the location, size and volume of the stream ponds can be predicted. Furthermore, the potential water problems caused by stream ponding can be assessed. The results may be used to (1) serve as a guide for designing the longwall panel layout to minimize the influence of stream ponding, including prevention of water loss, and (2) develop mitigative measures for restoration of the stream and land to meet required environmental standards.

## Conclusions

From the field investigation and data analysis, the following conclusions can be reached:

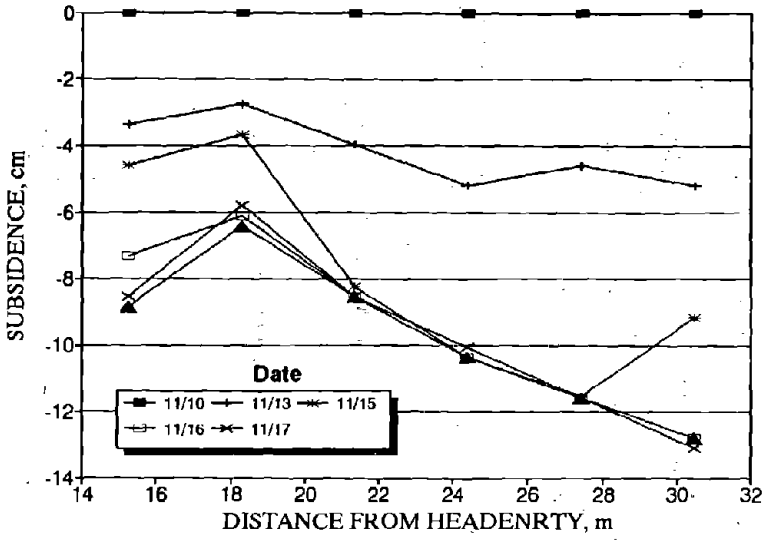


Figure 11. Subsidence profile along r-line over longwall panel 4

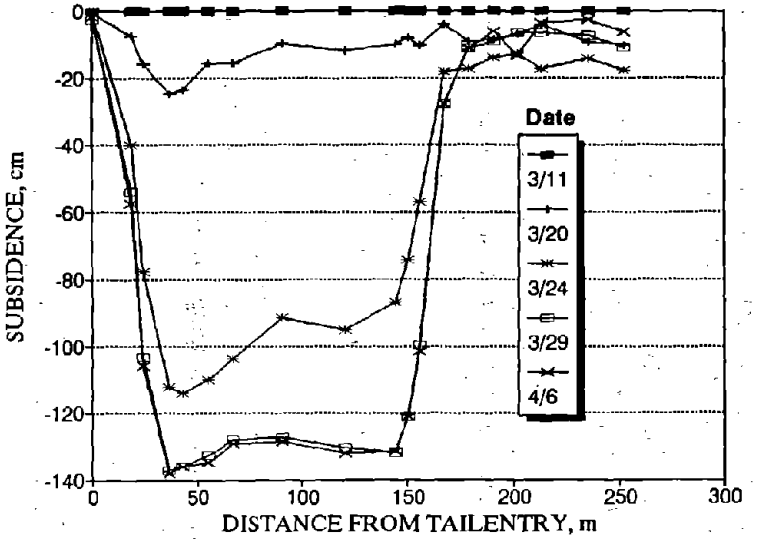


Figure 12. Subsidence profile along BH-line over longwall panel 5

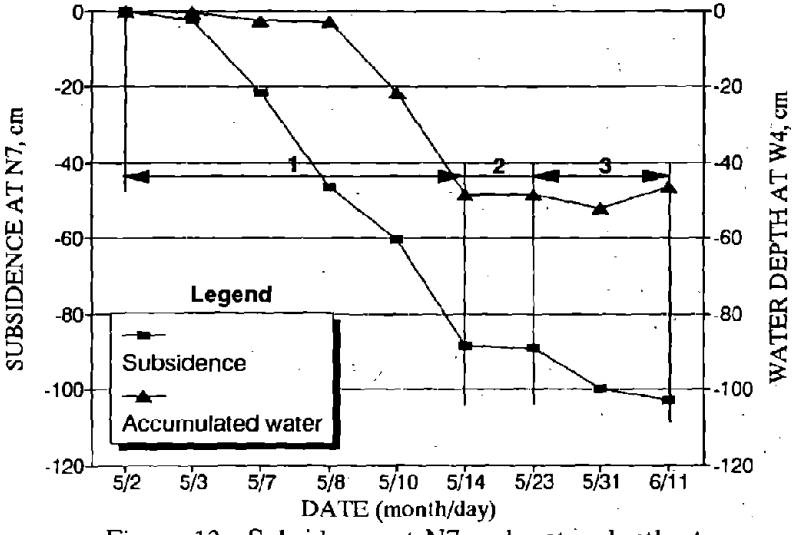


Figure 13. Subsidence at N7 and water depth at water station W4 over longwall panel 3

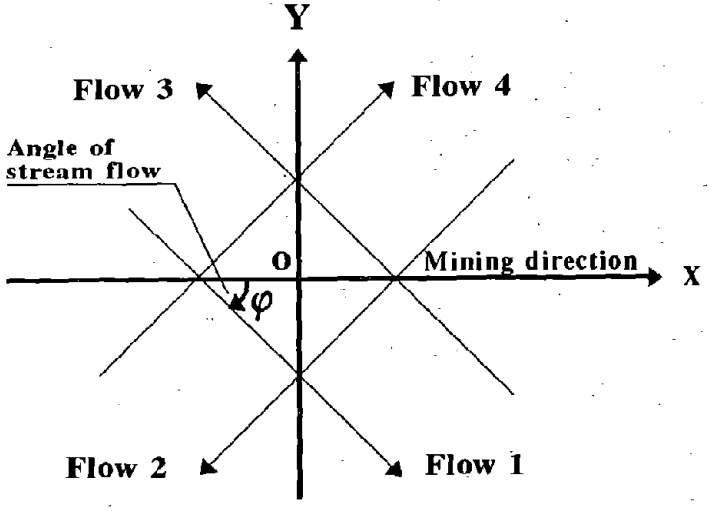


Figure 14. Angle of stream flow and flow patterns

• The presence of stream ponds may create water problems for surface environment and underground mining operation, especially when a large stationary pond is formed in the tension zone by the headentry.

• The formation process of the stream ponds may be affected by many factors, including precipitation, soil properties, vegetation, variation of water source upstream, etc. However, the topographic change resulting from surface subsidence plays a major role in governing the distribution of the stream ponds.

• The migratory pond is created by a dynamic surface subsidence process. If the dynamic strain exceeds the limiting value, water loss to dynamic cracks may occur in the migratory pond when it travels across the longwall panel.

• A stationary pond is formed as the migratory pond reaches a stable stage, when the local subsidence reaches about 80% to 90% of the final subsidence. The location and size of the stationary pond depend mainly on the topography and magnitude of the final subsidence basin.

• The defined angle of stream flow can be used to guide the longwall panel design to minimize water problems caused by a disturbed surface stream. The optimum angle of stream flow is 90° or 270°, based on the configuration of the adjacent new longwall panel. The mitigative measures-pumping the water, digging trenches, sealing the bottom of the stream bed, etc.-can also be designed based on the study.

## Acknowledgements

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# ELECTROMAGNETIC AND GRAVIMETRIC MAPPING OF A PARTLY COLLAPSED UNDERGROUND LIMESTONE MINE IN CLARK COUNTY, IN[1]

R. M. René, C. H. Ault, and S. S. Frushour[2]

**Abstract:** The Falls City underground limestone mine in Clark County, IN, was abandoned in 1903 ± 3 yr. The mine is now flooded and partly collapsed. A map of the mine was needed because undermined areas would be unsafe for buildings. We used this mine as a test site to evaluate gravimetric surveying to map other underground mines in the region, some of which cannot be entered. We first mapped the Falls City Mine using a commercially available "line finder," normally used to locate underground utilities such as pipes and telephone lines. Part of our crew carried the continuously operating, 82.5-kHz vertical-dipole transmitter through the underground mine in a boat. Aboveground we followed the boat using a receiver which generated an audio pulsing tone with higher frequency indicating greater magnetic field intensity. Gravity was measured at 4- or 8-m intervals in 10 profiles (total length = 2.5 km). In four of these profiles, inflection points of 300-$\mu$gal Bouguer gravity anomalies located edges of the uncollapsed portions of the mine. Gravity profiles around the perimeter of the collapsed mine were used to search for any tunnels inaccessible from the surface. None were detected. The gravity method should be practical in mapping similar underground mines in the region.

## Introduction

The underground Falls City Mine in the southeast corner of Clark Military Grant 89, Clark County, IN, (fig. 1), was abandoned in 1903 ± 3 yr. (Ault 1992, Siebenthal 1901). The mine exploited the Silver Creek Member of the North Vernon Limestone (Devonian) for production of a natural cement. The mine is now flooded and partly collapsed. A map of the mine was not available. The uncollapsed part of the mine is accessible by boat through entrances from a lake which occupies an adjacent quarry. We mapped a small part of the mine with a compass and a measuring tape stretched between a boat and a raft. Holding walls to stretch the tape was difficult and might have been dangerous. We therefore mapped the perimeter of the mine using an underground oscillating vertical magnetic dipole transmitter and an aboveground receiver. We used a commercially available 82.5-kHz "line finder," normally used to locate underground utilities such as pipes and telephone lines. Previous workers (e.g., Bell and Bell 1987, Charlton 1966) have used this method to locate points in limestone caves using noncommercially manufactured equipment with frequencies of 2 to 38.4 kHz.

Siebenthal (1901) recorded 11 shallow underground limestone mines in Clark County. Most of these mines are inaccessible from the surface, and the uncertainty of their extent may impede development of lands in their vicinity. We conducted a gravity survey of the Falls City Mine to test the application of gravimetric mapping to other similar mines and to detect any inaccessible galleries that might emerge from the collapsed areas of the Falls City Mine. Only the Falls City Mine is known to have partly collapsed (Ault 1992).

## General Description of the Mine

Patton and Dawson (1955), Shaver (1974), and Droste and Shaver (1975) described the stratigraphy of the region. In a quarry wall at the Falls City Mine, Siebenthal (1901) described 1.5 m of soil overlying 3 m of the New Albany Shale (Devonian-Mississippian), 2 m of the Beechwood Member of the North Vernon Limestone (Devonian);

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-29, 1994.

[2]Ray René, geophysicist, Curtis Ault, geologist, and Sam Frushour, laboratory supervisor, Indiana Geological Survey, Indiana University, Bloomington, IN.

and 4.7 m of the "cement rock" (the Silver Creek Member of the North Vernon Limestone). Beneath these strata lie the Speed Member of the North Vernon Limestone, the Jeffersonville Limestone (Devonian), and older Paleozoic strata. Most or all of the thickness of the Silver Creek was mined. Our measurements of mine heights from 3.7 to 4.5 m indicate that much of the flooded mine floor is covered with debris. The roof of the underground mine is the base of the Beechwood Member. Because the strata dip slightly southwestward, the roof of the mine is 3.7 m above water level near the northeastern end of the underground mine and 0.8 m above water near its southwestern edge. Galleries in the Fall City Mine were as much as 300 m long and 12 to 15 m wide. Roughly 75% to 80% of the limestone was extracted from the mined interval.

In most of the mine few joints were apparent in the roof, thus accounting for the strength of the mine, which was not shored by timbers. Near entrances from the quarry, however, a few masonry posts provide additional support. In one area a 2- by 4-m block of the Beechwood had collapsed into the mine, and the New Albany Shale was exposed in the roof. Near the northeast end of collapse area *D* (fig. 1) a portion of the roof broke away along joints. Many of the pillars have a concave surface with the overhanging part of the pillar having partly fallen away from the mine roof along the bedding plane between the Silver Creek and Beechwood Members. The collapsed areas are wooded. A 1940 aerial photograph showed that most of area *A* (fig. 1) had collapsed, but that areas *B*, *C*, and *F* were still being farmed without evidence of collapse. Collapse in area *A* is essentially complete; however, there is one sinkhole at the eastern edge of the area that is separated from the collapsed galleries by a short section of one gallery that remains uncollapsed. A 1960 aerial photograph showed that areas *B* and *C* had collapsed, perhaps shortly before that time. Of 11 entrances to the mine from the southeast wall of the quarry, 4 have collapsed (area *D*). Additionally, there are now two entrances from collapsed area *C* and several entrances that resulted from the collapse of area *B*. Areas *D* and *E* are in wooded areas, and it is difficult to date the collapse. Area *F* apparently collapsed after 1960. A small underground mine (area *E*) along the west wall of the quarry is only partly collapsed.

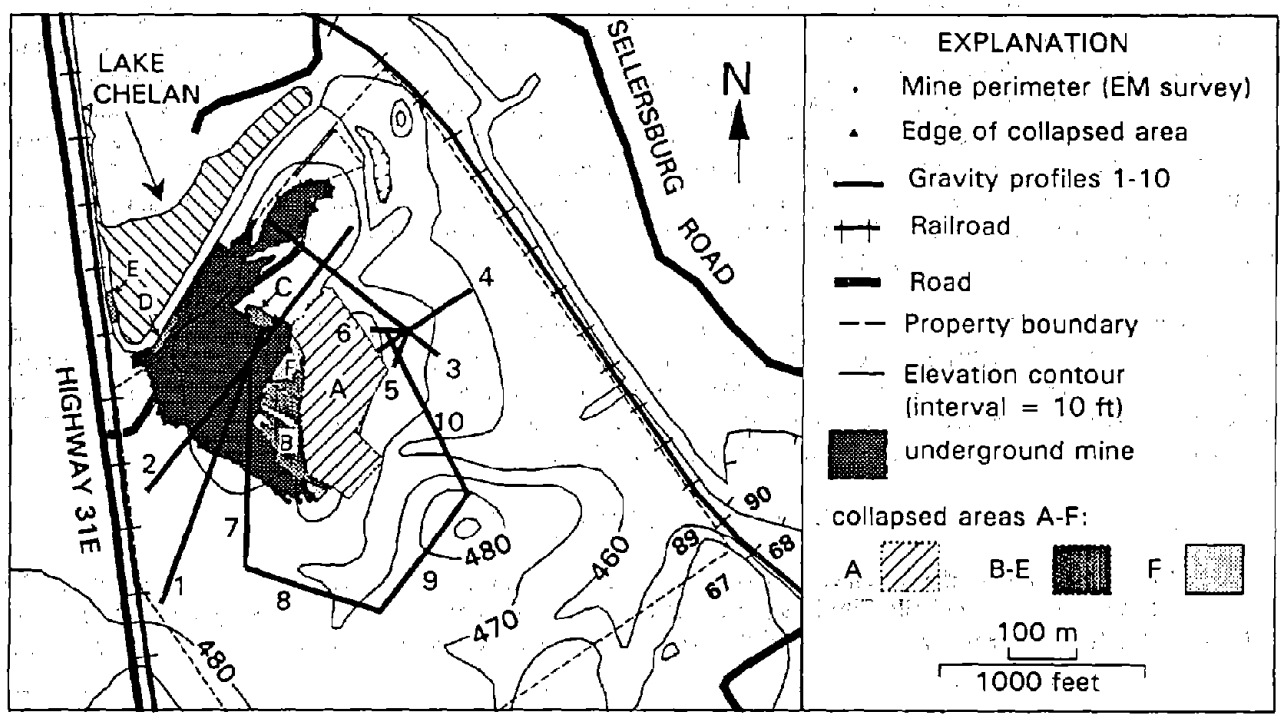


Figure 1. Map of the Falls City Mine showing the uncollapsed portion of the mine as mapped with an underground electromagnetic transmitter and aboveground receiver, collapsed areas, and gravity profiles 1-10. The indicated property boundary includes the southeast and northeast boundaries of Clark Military Grant 89. Collapsed areas were mapped with the aid of aerial photographs.

## Basic Principles of Electromagnetism

In a vacuum the quasi-static (low-frequency) magnetic field of a sinusoidally oscillating vertical magnetic dipole (horizontal current-carrying loop of small dimensions) is a dipolar field with zero azimuthal component. The radial and tangential components are

$$H_r = I_o A \cos(\Theta) \sin(2\pi ft)/(2\pi R^3) \quad (1)$$

and

$$H_\theta = I_o A \sin(\Theta) \sin(2\pi ft)/(4\pi R^3), \quad (2)$$

respectively (Wait 1982), where $I = I_o \sin(2\pi ft)$ is the current in amperes, f is the frequency in hertz, t is the time in seconds, A is the loop area in meters squared, R is the distance from the dipole in meters, and H is the magnetic field intensity in amperes per meter squared. The quasi-static approximation applies for distance much less than the wavelength of the electromagnetic radiation ($R << \lambda$), where wavelength $\lambda = c/f$ and c is the velocity of light. Directly above the dipole, the magnetic field is vertical and of greater intensity than elsewhere on a horizontal plane at a given height above the dipole.

In the presence of a conductor, the primary magnetic field (equations 1-2) induces secondary electric currents and a secondary magnetic field. If electrical properties are cylindrically symmetric about the vertical dipole source, including the case of a horizontally layered earth, there will be no azimuthal component of the secondary magnetic field, and the total (primary plus secondary) field directly above the source will still be vertical, although it may be attenuated. For a plane electromagnetic wave transmitted through a conductor, secondary currents cause the total magnetic field to be attenuated by a factor $e^{-x/d}$, where x is the distance transmitted, d is the skin depth,

$$d = (\pi \mu \sigma f)^{-1/2}, \quad (3)$$

$\sigma$ is the conductivity in siemens per meter squared, and $\mu$ is the magnetic permeability in henries per meter. In most rocks $\mu$ is approximately equal to the permeability of free space $\mu_o$ ($=4\pi\ 10^{-7}$ H/m). To aid detection or lessen distortion of the total magnetic field, a low-frequency source is used to ensure that the source-receiver distance is less than several skin depths in the intervening media.

## Field Procedure of Electromagnetic Mapping

We mapped the Falls City Mine using a "line finder." The transmitter unit includes a 0.15- by 1.07-m coil, which we oriented horizontally in the mine. The 82.5-kHz signal was modulated at 382 Hz and pulsed at 4.8 kHz. The continuously operating transmitter was centered abeam in a 3.7-m boat in which we traced the mine's perimeter by following the left wall of the mine so that no galleries at the periphery of the uncollapsed portion of the mine would be missed. The receiver unit included an audio speaker, a sensitivity control knob, and an attached wand containing a small sensing coil. The frequency of an audio pulsing tone increased with increasing magnetic field intensity at the sensing coil. The receiver was carried aboveground by the operator who followed the transmitter as it moved through the mine below. The receiver could detect the underground signal from a horizontal distance of about 10 m. The movement of the in-mine transmitter was halted at prearranged time intervals of 5 or 10 min. The location directly above the transmitter where the magnetic field was vertical, the pole position, was then determined by orienting the axis of the receiver coil horizontally so that a null response would be obtained.

Excluding stations at the edge of collapsed areas *B* and *C*, 55 mine-perimeter stations were located and marked with paint on the walls of the mine at an average interval of about 20 m along the mine's perimeter. We then used a level-transit to locate the aboveground mine-perimeter stations relative to property lines and features that were identifiable in aerial photographs. After a preliminary map of the mine was plotted, we reentered the mine to sketch details of the walls and to measure elevations of the roof and floor relative to the water surface. A video tape recording was made to document the condition of the mine's interior.

## Results of Electromagnetic Mapping

We first successfully tested the applicability of the line finder to underground mapping in a shallow limestone cave (Buckner Cave) in Monroe County, IN. Although previous applications of electromagnetic mapping in caves have used much lower frequencies, use of a 82.5-kHz signal in the Falls City Mine was successful because the mine is very shallow. Electric logs of the Indiana Geological Survey drill hole 290 in Clark County, about 20 km north of the Falls City Mine, indicate that the maximum conductivity of the basal Blocher Member of the New Albany Shale is 0.05 S/m. The corresponding skin depth at a frequency of 82.5 kHz is 7.8 m. The Beechwood limestone is about 10 times less conductive than the overlying shale, and the soils at the surface are probably less or only slightly more conductive than the shale. The attenuation of the electromagnetic field by strata above the mine roof is, therefore, tolerable.

Because a suitable nonelectrically conducting boat was not available, we carried the transmitter through the mine in an aluminum boat. We placed the transmitter near the center of the boat so that an axis of symmetry for secondary currents in the boat would be vertical through the approximate center of the transmitter loop. We tested the method near an entrance to the mine from collapse area *C* (fig. 1). There we placed the in-mine transmitter near the center of the boat and then located the pole positions on the surface above the mine relative to a point directly above the transmitter. By placing the transmitter near the center of the boat with reasonable care, whether fore-and-aft or abeam, the pole position at the ground surface was displaced by less than 1 m. When the transmitter was placed near a corner of the boat, the pole position was displaced by 5 m.

Figure 1 shows the mine perimeter locations on a map generated by digitization of selected features of a 1964 7.5-min quadrangle topographic map and a 1960 aerial photograph. The mine encroaches within about 10 m of a house and a stone barn, but no existing buildings are undermined.

## Principles of Gravity Surveying

A gravimeter measures spatial changes in the gravitational field due to the Newtonian attraction of the Earth's mass and the centrifugal acceleration associated with the Earth's rotation. So that gravity anomalies associated with local density contrasts can be identified and interpreted, drift, latitude, free-air, Bouguer, and terrain corrections are successively applied. Drift corrections, which remove effects of instrument drift and tidal effects of the Moon and Sun, are generally estimated by frequent repeated occupations of a base station. The drift is assumed to vary linearly with time over short intervals of time. The latitude correction, which removes effects of the Earth's equatorial bulge and centrifugal acceleration, is $C\,\chi\,\sin(2\Phi)$, where $\Phi$ is the latitude of a reference station, $\chi$ is the distance north of that station, and $C = 0.8121\ \mu\text{gal/m}$ (1 gal = 10 mm/s$^2$). The free-air correction, which removes effects of variation in distance from the center of the Earth, is $k\,h$, where h is the elevation above a datum and $k = 308.5\ \mu\text{gal/m}$. In local surveys a datum at, or just below, the lowest point in the survey is commonly used. The result of applying this correction is called the "free-air" gravity.

The Bouguer correction, which removes the gravitational effect of an infinite horizontal slab lying between the gravimeter and the datum, is $-2\pi G\rho h$, where $\rho$ is the Bouguer density, or assumed density of this layer, and G (= 66.7 pN m$^2$/kg$^2$) is the gravitational constant. The result of applying this correction is the "Bouguer" gravity. The terrain correction, which removes the effect of deviation of the terrain from a horizontal surface through a given station, is $T\,\rho$, where T depends on differences in elevation between the gravimeter location and nearby points on the Earth's surface. Often this correction is negligibly small, or varies gradually enough within a survey area that it is omitted. A regional gravity field (due to density contrasts extending beyond the bounds of the survey area or of larger lateral dimensions than are of interest in a given survey) may be estimated by mathematical methods (e.g., low-order polynomial fitting) or by visual inspection of the gravity field. The result of subtracting an estimate of the regional field from Bouguer gravity is called the residual Bouguer gravity.

Except on a regional scale, free-air gravity, $g_F$, will generally be correlated with topography owing to the gravitational effect of the near-surface rocks or soil. This correlation can sometimes be used to estimate the Bouguer

density. For a homogeneous near-surface stratum and no anomalous gravitational fields,

$$g_F/(2\pi G) = \rho\,[h\text{-}T/(2\pi G)] + K_F, \qquad (4)$$

where $K_F$ is a constant. In reality, lateral variations of density above and below the datum are present. If, however, these density variations are not correlated with the topography, then $K_F$ becomes a random variable. In Parasnis' (1962) method, equation 4 is the basis of determining the Bouguer density as the slope of a straight line fit to points plotting $g_F/(2\pi G)$ versus $h\text{-}T/(2\pi G)$.

## Field Procedure in Gravity Surveying

Ten gravity profiles at 4- or 8-m station intervals were located with a level-transit. Although the electromagnetic survey was conducted in the fall of 1992 and the gravity survey was conducted in the spring of 1993, most of the survey stakes and flags were preserved through the winter. Thus, the two surveys could be accurately tied, and profiles 1, 2, and 7 were placed directly through stakes at mine-perimeter stations. Gravity was measured at station locations using a temperature-regulated astatized gravimeter in which a micrometer screw adjusts tension on a "zero-length" spring to balance the gravitational force on a weighted horizontal beam. We determined the null position of the beam using a galvanometer driven by an electronic capacitance beam position indicator. One dial mark interval on the micrometer screw corresponded to a variation of gravity by about 10 μgal. Gravimeter readings were obtained by estimating the nearest tenth of a dial mark.

The gravimeter was leveled on a concave leveling disc, or baseplate, set firmly against the ground by pressing its three 55-mm legs into the soil. One baseplate was left at the base station at the intersection of gravity profiles 1, 2, and 7. An identical baseplate was used at other gravity stations. The base station was reoccupied at about 1-h intervals. In part of profile 9 inconsistent data were recorded owing in part to gusty winds and loose soil on a hillside, which made firm planting of the baseplate difficult. We did not reoccupy these stations since our survey was limited by the need to clear the ground before spring cultivation of the field.

## Results of Gravity Surveying

Figures 2-7 show the residual Bouguer gravity and elevation above datum in profiles 1-10 (fig. 1).

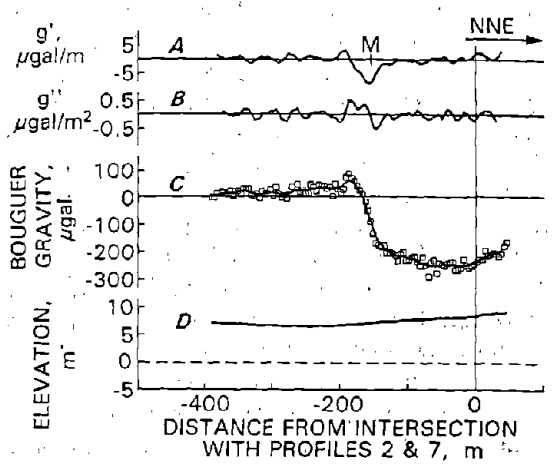


Figure 2. Profile 1: *A*, smoothed first horizontal difference, g′; *B*, smoothed second horizontal difference, g″; *C*, residual Bouguer gravity, g, and smoothed gravity (solid curve); *D*, elevation above datum. The minimum of g′ occurs above the known mine edge (M) located by electromagnetic mapping.

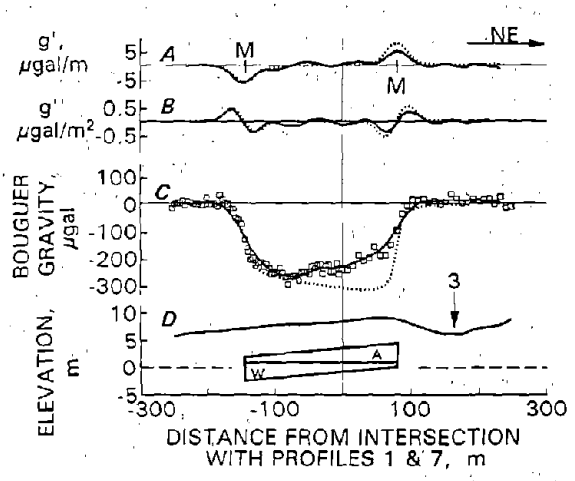


Figure 3. Profile 2 [measured (solid) and modeled (dotted) profiles]: *A*, smoothed first horizontal difference, g′; *B*, smoothed second horizontal difference, g″; *C*, residual Bouguer gravity, g, (open squares) and smoothed gravity (curves); *D*, elevation above datum and model of the mine (A = air-filled, W = water-filled; with density contrasts of 2.12 and 1.32 mt/m³, respectively). The extrema of g′ occur above the known mine edges (M). An arrow locates the intersection with profile 3.

Latitude corrections reduced the data to the latitude of the base station, and free-air and Bouguer corrections used a datum 8.33 m below the base station and elevations from a level-transit survey of profiles 1-10. Terrain corrections were not applied. The Bouguer density of 2.05 mt/m$^3$ was obtained by Parasnis' method using linear regression to obtain the slope of a straight line fit to points in a plot of normalized residual free-air gravity ($g_p/2\pi G$) versus elevation above datum, h (fig. 8). Only off-mine stations in profiles 1-10 that were more than 20 m from collapsed areas and more than 50 m, along the direction of profile, from the uncollapsed mine perimeter were used. A regional field was removed from the free-air gravity before application of Parasnis' method to obtain a more reliable density estimate.

The residual free-air and Bouguer gravity fields (figs. 2-8) were obtained by subtraction of a regional field estimated by examination of Bouguer gravity at least 50 m distant from the mine in profiles 1-3, 7, and 10. The estimated regional gravity is constant in an east-west direction and increases southward at the rate of 0.4 μgal/m. Smoothed residual Bouguer gravity profiles (figs. 2-7) were derived by application of a triangular-weighted nine-point smoothing filter (1,2,3,4,5,4,3,2,1) /25, with shorter filters used at the ends of profiles. In profiles 1-10, the root-mean-square difference between smoothed and unsmoothed residual Bouguer gravity is 14 μgal, excluding end effects of the smoothing filter. This difference results partly from measurement errors and "geologic noise" due to shallow subsurface density changes not related to the Falls City Mine.

Gravity profiles 1-3 and 7 (figs. 2-4 and 6) cross the edge of the uncollapsed portion of the mine as determined by our electromagnetic mapping with an underground transmitter. In these profiles a negative Bouguer anomaly of about 300 μgal is observed. Figures 3-6 also show the smoothed first and second horizontal differences, which are approximations of first and second horizontal derivatives [$g'$ ($=\delta g/\delta x$) and $g''$ ($=\delta^2 g/\delta x^2$)] of the residual Bouguer gravity. The smoothed first horizontal difference was obtained by application of a seven-point filter comprising the (-1,1)/Δx differencing filter convolved with the averaging filters (1,1,1)/3 and (1,1,1,1)/4, where Δx is the station spacing of 4 or 8 m. The smoothed second difference was obtained by application of a nine-point filter comprising the (-1,2,-1) /$\Delta x^2$ differencing filter convolved with (1,1)/2, (1,1,1) /3, and (1,1,1,1)/4 averaging filters (Blackman 1965).

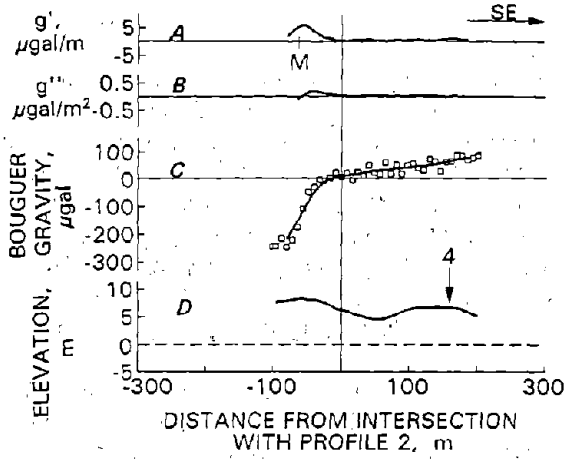


Figure 4. Profile 3: *A*, smoothed first horizontal difference, $g'$; *B*, smoothed second horizontal difference, $g''$; *C*, residual Bouguer gravity (open squares) and smoothed gravity (solid curve); *D*, elevation above datum. The maximum of $g'$ occurs above the known mine edge (M). An arrow locates the intersection with profile 4.

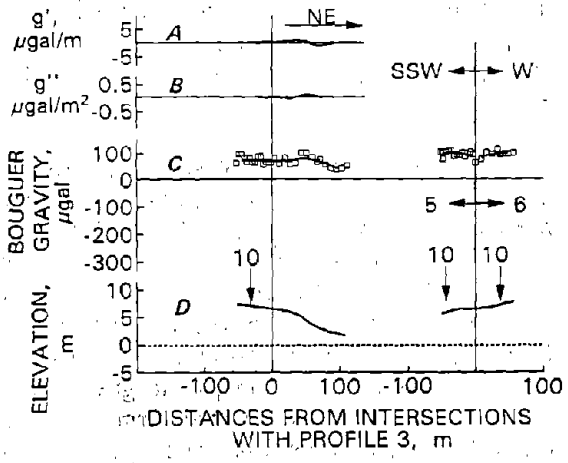


Figure 5. Profiles 4, 5, and 6: *A*, smoothed first horizontal difference, $g'$; *B*, smoothed second horizontal difference, $g''$; *C*, residual Bouguer gravity (open squares) and smoothed gravity (solid curve); *D*, elevation above datum. Arrows locate intersections with profile 10.

For a two-dimensional gravity field, the second horizontal derivative is proportional to the second vertical derivative ($\delta^2g/\delta z^2$) since the gravity field satisfies Laplace's equation

$$\delta^2g/\delta x^2+\delta^2g/\delta y^2 = -\delta^2g/\delta z^2, \qquad (5)$$

where y is the horizontal coordinate measured perpendicular to the profile. The points of inflection of the gravity anomalies, as defined by extrema of the first-difference curves and zero crossings of the second-difference curves, are approximately coincident with the edges of the mine (figs. 2-4 and 6). This result is consistent with the fact that the zero crossings of the second vertical derivative, and therefore any second horizontal derivative of gravity resulting from a homogenous (constant-density), two-dimensional rectangular prism with vertical sides and a width greater than the depth to its center, occur very near to points directly above the edge of the prism (Elkins 1951). Figure 3 shows a comparison of measured and modeled profiles of residual Bouguer gravity, g, g′ (=$\delta g/\delta x$), and g″ (=$\delta^2g/\delta x^2$), and the corresponding density contrast model of the mine. In drill hole 290 a density log indicates that the average density of the basal Blocher Member of the New Albany Shale is 2.42 mt/m$^3$. The underlying North Vernon Limestone has densities of 2.62 to 2.79 mt/m$^3$ with the most dense strata occurring in the upper 2 m of the formation. The density model (fig. 3*D*) assumes a mine height of 4.3 m and a density contrast between the Silver Creek ($\rho$ = 2.65 mt/m$^3$) and a mined area comprising 80% air or water and 20% unmined rock. The relative amounts of air and water in the model correspond to measurements made in the mine near profile 2. Although the beds dip slightly, density contrasts along bedding were used. The gravitational effect of the structure related to variable thickness of shale and density contrasts above and below the mine level was not considered since this effect may be partly removed by subtraction of the regional field. The modeled gravity was computed on the sloping ground surface (fig. 3*D*). The residual Bouguer gravity in line 2 is less than the modeled gravity near the northeast edge of the mine, where the mine is poorly approximated as a two-dimensional body and where debris has washed into the mine from the adjacent collapsed areas. Profile 2 crosses above a large debris cone which entered the mine through one of the two entrances from collapse area *C*.

Negative gravity anomalies of about 50 and 70

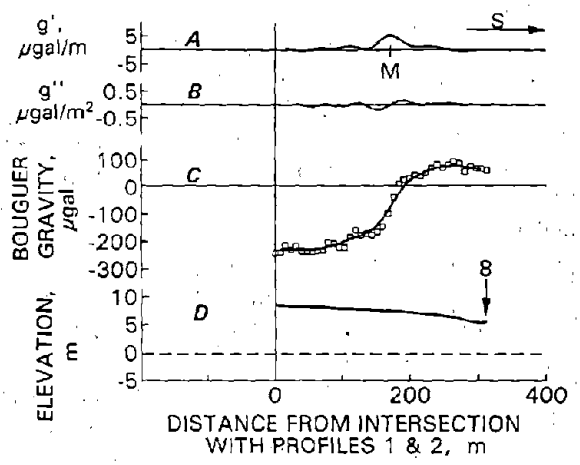


Figure 6. Profile 7: *A*, smoothed first horizontal difference, g′; *B*, smoothed second horizontal difference, g″; *C*, residual Bouguer gravity (open squares) and smoothed gravity (solid curve); *D*, elevation above datum. The maximum of g′ occurs above the known mine edge (M). Profile 8 starts at the southern end of profile 7.

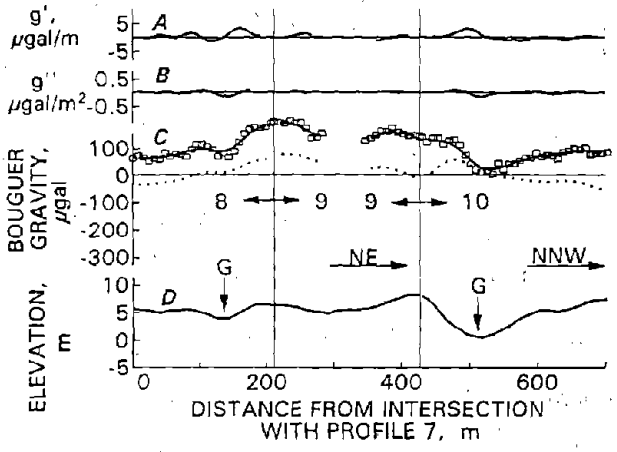


Figure 7. Profiles 8, 9, and 10: *A*, smoothed first horizontal difference, g′; *B*, smoothed second horizontal difference, g″; *C*, residual Bouguer gravity, g, (open squares) and smoothed gravity using Bouguer densities of 2.05 (solid curve) and 2.50 (dotted curve) mt/m$^3$, respectively; *D*, elevation above datum. Relative minima of g (using the lesser Bouguer density) occur in the gully (G) traversed by profiles 8 and 10.

$\mu$gal occur in profiles 8 and 10 (fig. 7*C*), respectively. These anomalies are coincident with a gully south of collapsed area *A* (fig. 1). Use of a Bouguer density of 2.5 mt/m$^3$, instead of 2.05 mt/m$^3$, effectively removes these small anomalies (fig. 7*C*). It is probable that this gully has eroded some of the shale with a density of about 2.42 mt/m$^3$ and perhaps even some of the underlying limestone. In comparison, a 4.3-m high tunnel at a depth of 8 m would create a 60-$\mu$gal anomaly if it were 4 m wide and air-filled ($\Delta\rho$ = 2.65 mt/m$^3$) or 6.5 m wide and water-filled ($\Delta\rho$ = 1.65 mt/m$^3$). The negative gravity anomalies in profiles 8 and 10 are, however, relatively wide and positively correlated with the terrain and are, therefore, not interpreted to result from tunnels associated with the Falls City Mine. On the hillside beneath profile 10 to the south of this gully, a small positive anomaly may indicate relatively thin soil on the hillside.

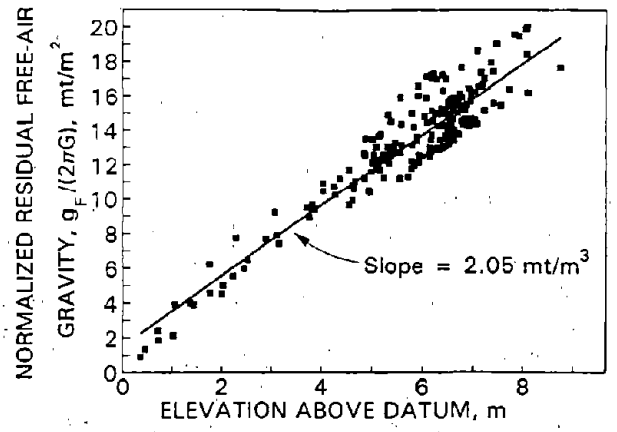


Figure 8. Normalized residual free-air gravity, $g_F/2\pi G$, in profiles 1-10 versus elevation above datum. The plotted straight line, obtained by linear regression, has a slope of 2.05 mt/m$^3$. The data are from all off-mine stations more than 20 m from collapsed areas and more than 50 m, along the direction of profile, from the uncollapsed mine perimeter.

## Conclusions

The method of electromagnetic mapping a very shallow limestone mine with an underground dipole transmitter and aboveground receiver is practical and efficient. Gravimetric mapping of the Falls City Mine successfully located the edges of the mine at inflection points of 300-$\mu$gal Bouguer gravity anomalies. No inaccessible galleries leading from the collapse area of the mine were detected. Gravimetric mapping of similar underground limestone mines in southeastern Indiana should be practical.

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# THE POTENTIAL AND PROSPECTS FOR ALLEVIATING LONGWALL-MINING CONCERNS THROUGH THE USE OF BACKSTOWING[1]

Michael J. Sanzotti, and Susan B. Bealko, and Christopher J. Bise[2]

**Abstract:** This study examined the potential for controlling longwall-induced subsidence by pneumatically backstowing the void created by mining with waste rock, and having the process scheduled as a unit operation of the mining activity. After a review of background literature, interviews with representatives of various mining companies, public-interest groups, and State and Federal regulatory agencies, a case-study evaluation of pneumatic stowing in a southwestern Pennsylvania mine site was conducted. The significant result of this investigation indicates that current stowing technology cannot keep pace with the production potential of U. S. longwall systems. At the case-study mine site, concomitant stowing would reduce subsidence by 50% of that incurred during full longwall caving, but the longwall coal production rate would decrease from 4,000 st per shift in the non-stowing case to 1,500 st in the stowing case. A preliminary cost analysis reveals that pneumatic stowing would add $6 to $10 to the operating cost of producing a ton of coal at the case-study site.

**Additional Key Words:** longwall mining, subsidence, backstowing, overburden, pneumatic stowing, caving.

## Introduction

In recent years, the increasing importance of longwall mining to U.S. underground coal production has been due to its more favorable levels of production, productivity, and safety when compared to room-and-pillar mining. In 1975, for instance, longwall mining accounted for 3.1% of the total underground coal production (Barczak 1992). Since that time, a dramatic increase in longwall production is reflected in the fact that longwall mining currently accounts for 37% of the total U. S. underground output (Merritt 1992).

While the future for longwall mining appears promising, one issue that continues to limit its acceptance is surface subsidence. The underground mining of coal creates a void which, under certain conditions, may result in the collapse and settling of overlying strata into the void. The effects on the overlying strata can propagate up to the surface and cause surface subsidence which, in turn, affects the natural and cultural environments; the resulting severity is primarily dependent upon the magnitude of subsidence. However, the extent to which these effects are experienced by humans is a function of the mine's location relative to various land uses, such as housing or agriculture.

It should be obvious that a mining engineer can exercise control only over the siting and dimensions of the void. With this in mind, the National Coal Board (NCB) conducted extensive empirical studies in England over 25 yrs ago to determine the relationship between the various parameters. The NCB found, among other things, that a relationship existed between the ratio of the void width to the depth of cover and the ratio of the maximum amount of subsidence observed to the seam thickness and that subsidence damage is negligible where the width-to-depth ratio is below 0.25 and can be significant when it exceeds 0.40. If the NCB findings can be extrapolated to U.S. conditions, which is often done, then it is obvious that longwall mining, as practiced under depths of cover less than 1,000 ft with face lengths currently averaging 714 ft (Merritt 1992), is conducted under conditions that will result in surface subsidence.

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One alternative in underground coal mining is to leave coal underground to protect surface structures. This practice is reasonable for room-and-pillar mining, but not for the full-caving longwall method. This leaves backstowing, the act of replacing the mined material with some type of fill material so that the newly created void is packed, as a method to restrict the movement of strata within the mine workings. Backstowing has been used successfully, both here and abroad, to limit subsidence damage in isolated cases. Debate as to its potential for limiting deep-coal-mining-induced damage has generated both information and misinformation. Recent legislation (Pennsylvania House Bills 628, 632, and 2514), testimony (the Pennsylvania Coal Association's testimony regarding H.B. 2514 --- "The Deep Mine Mediation Bill" --- October 9, 1990), and positions taken by groups such as the National Wildlife Federation and the Environmental Policy Institute further underscore the need for clear understanding of the issues and the development of accurate information.

Where stowing is applied successfully in other countries, generally three requirements have been met: (1) It is technically and economically the most desirable alternative for mining the resource, (2) the stowing activities are subsidized by the respective governments, and (3) an abundance of otherwise non-usable material is available for stowing.

This study examined the potential for controlling longwall-induced subsidence using backstowing as a unit operation of the mining activity, and is a distinct departure from past studies of after-the-fact stowing under structures to be protected, such as schools and churches. The U. S. Bureau of Mines has, for example, conducted several studies with regard to abandoned anthracite mines in Pennsylvania and in bituminous mines in the Illinois basin (Whaite and Allen 1975; Wade and Conroy 1977). The problem currently faced in Pennsylvania regarding longwall mining is how to limit subsidence while the panel is actively being mined, not after. Thus, this study of the potential for concomitant backstowing during mining focused particular attention on (1) ease of incorporation, (2) potential effectiveness, and (3) anticipated costs.

## Case-Study Site Selection and Technical Analysis

To evaluate the technical and economic feasibility of backstowing in southwestern Pennsylvania longwall mines, a stowing system was incorporated into a case-study mine. The mine site selected should be representative of an active longwall mine in southwestern Pennsylvania, and backstowing material should be readily available in the immediate area.

### Site Description

The 16,700 acre site is located in eastern Washington County and is characterized by flat-lying to gently dipping sedimentary strata. Geological formations within the area that overlie the Pittsburgh seam at the selected site, from youngest to oldest, are stream alluvium (sand, gravel), the Carmichaels terrace deposits, and the Greene, Washington, Waynesburg, Uniontown and the Pittsburgh Formations.

The Pittsburgh Formation consists of alternating limestone and shale, Sewickley coal, carbonaceous shale, Pittsburgh sandstone, and the Pittsburgh coal, which averages 63 in at the minesite. The strata immediately above the Pittsburgh coal consist of interbedded shale and thin coal, while the strata immediately underlying the coal varies between calcareous shale, limestone, shale, sandy shale, and clay. Mine rock, which includes immediately overlying and underlying strata, may result from materials removed from the mine because of roof falls, floor heaving, excavations for mine water sumps, and other underground operations.

The depth of cover ranges from 80 to 500 ft, with 450 ft being the average depth over the more recently extracted longwall panels. The mining methods practiced in the mine are continuous mining and longwall mining. Continuous miners develop the main and panel entries and may be used for room-and-pillar mining with pillar

removal. Three entries are driven for longwall gateroad development, and the width and length of the longwall panels are 600 ft and 6,000 ft, respectively. Production from the longwall is in the range of 4,000 st per shift.

Two refuse disposal areas are at the site: Disposal Area A, which has been in existence for approximately 25 yrs, is completely filled, and Disposal area B, which is currently active, occupies 241 acres. Coarse coal refuse from the preparation plant is loaded into mine refuse rail cars and transported approximately 3 miles overland to the area. There, the refuse is discharged into a hopper, and is deposited within the disposal area in a layer thick enough to support the weight of the equipment hauling and spreading the refuse. The targeted refuse discharge rate from the preparation plant is 4,000 st per day, and the maximum vertical thickness of the refuse pile is 115 ft. To demonstrate the characteristics of the refuse, an effluent water sample was taken from the sedimentation pond that receives runoff water from the refuse area. The samples comply with the limits specified by the Pennsylvania DER as being safe for discharge into the surrounding water system, ensuring that the coal refuse is suitable for reintroduction underground.

At the mine site, subsidence is anticipated over room-and-pillar and longwall mining, but can be planned and controlled with regard to the timing and areal extent. Through the use of field surveys, the subsidence factor observed varied from 0.4 to 0.6. Anticipated effects on the surface may include minor rolls in surface topography and surface tension cracks.

Past experience has revealed that the surface over a longwall panel subsides in a trough-like, gentle depression. It has also been noticed that the major settlement occurs during the first five weeks of mining. Minor structural damages have been observed over the site's longwall workings. The surface over the mine is primarily used for farming and pasture land.

## Subsidence Estimation

The subsidence prediction model chosen was first evaluated under conditions without backstowing, and then compared to subsidence profiles with the backstowing operation in place. Strains due to differential settling of the surface were predicted, and the extent to which surface structure damage can be reduced using stowing as a unit operation was estimated.

The method of subsidence prediction used in the study is applied by many mining companies operating in the case study area. Recent work (Hartman 1992) indicates that the hyperbolic tangent function is applicable in the bituminous coal fields of southwestern Pennsylvania:

$$S(x) = (0.5)\,((S_{max})\,(1 - \tanh cx/b) \qquad (1)$$

Subsidence surveys over a previously mined panel were obtained from the case-study area, permitting a comparison between anticipated subsidence using the model and actual conditions. The values used in the calculation were $S_{max}$ = 3.45 feet, c = 2.0 ( a subsidence constant), x varied between -250 ft to 325 ft, and b = 150 ft.

While estimating the effect of backstowing on surface subsidence, the indicator of the percentage loss in seam thickness after the roof has settled is known as the stowing factor or convergence. For pneumatically stowed faces, a convergence of 50% is common (Munjeri 1987; Reinshagen 1986; Voss 1983). Thus, the prediction model can then be developed for backstowed panels using a seam height of 50% of the original extraction height. A method detailed in the NCB's Subsidence Engineer's Handbook can be used to obtain a new value of $S_{max}$. It was estimated that a subsidence factor of 0.46 would be experienced at the case-study mine and, using this information, a new maximum subsidence was determined to be 1.73 ft. Then, using this $S_{max}$ in equation 1, a new profile was constructed. Values used for the calculations were similar to the previous calculation, except $S_{max}$ = 1.73 feet.

Accordingly, the maximum subsidence can be reduced by 50% using full-face pneumatic stowing. This alone is not enough information to determine the effectiveness of stowing to alleviate subsidence damage since damage to structures from subsidence is the result of differential horizontal movement which causes a strain (extension, +E;

compression, -E) in the ground surface. Extension is characterized by the opening of joints in brickwork and compression is characterized by the squeezing of openings, such as doors and windows, and the horizontal movement of brickwork (Munjeri 1987).

To analyze the strain experienced as a result of subsidence in the case-study mine, a method of predicting strains was used as outlined in the Subsidence Engineer's Handbook. The predicted maximum strains can then be classified with regard to the extent of damage to be experienced using tables developed and presented in the handbook. Figure 1 presents the complete strain and subsidence profiles, while Table 1 presents the associated damage classification.

## Stowing System Layout

The +5 in material will be fed into a hammermill crusher using a front-end loader at the coarse coal refuse site. To achieve the required capacity, two 350-st/h crusher units will be needed to ensure the overall stowing-system efficiency. After the material is crushed to -3 in, it is transported, by means of a belt conveyor, to a 32,000-cu-ft storage silo which will ensure a steady feed of material to the shaft gravity pipe and will also reduce the amount of moisture in the material. A screw conveyor at the silo will allow the material to be fed to the shaft opening at the desired rate.

The infrastructure of a pneumatically stowed mine provides the stowing operation with material and other supplies. The layout incorporated the existing design aspects, and the specific configuration complies with guidelines suggested by Voss (1983). Material from the surface silo will be sent into the mine through a 20-in-diam shaft pipe having a spiral guide to reduce particle breakage on impact at the bottom. Recent designs have used cast steel in the construction of the spirals and are interlocked in such a way that smooth conveying of material down the pipe is realized. Advantages of gravity shaft pipes are high-capacity throughput, small space requirement in the shaft cross-section, and ease of maintenance. A throughput of 1,000 st/h of stowing material can be realized using the various pipe designs.

At the base of the shaft, material will be loaded onto a main belt conveyor for transport to the working section. This belt will be 36 in wide, have a maximum capacity of 700 st/h, and will be driven by a 400-Hp motor (Bise 1986). For the case-study, a length of 10,000 ft was used to simulate the access distances between multiple longwall faces. The main belt will discharge into an underground storage bunker, and the material will then be

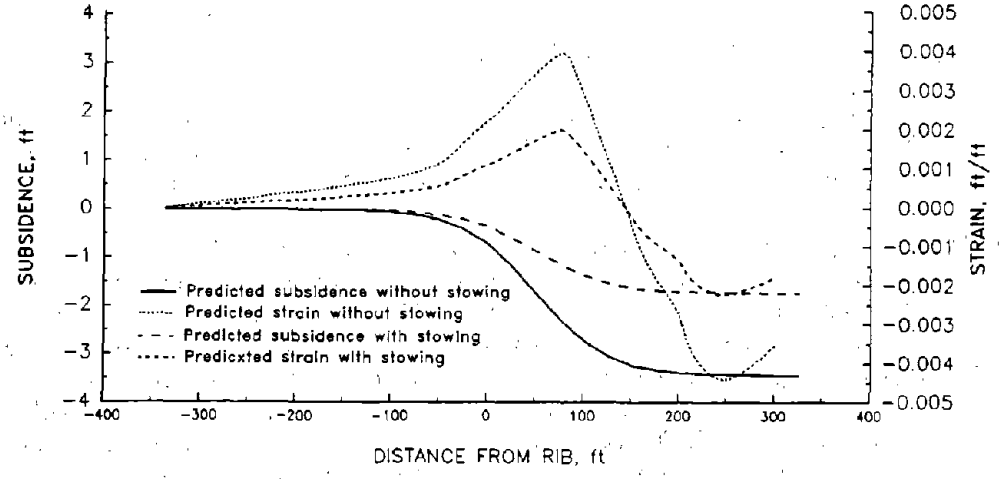


Figure 1. Graph showing predicted subsidence profile and predicted strain profile without pneumatic stowing compared with predicted subsidence profile and predicted strain profile with pneumatic stowing

Table 1. Results from strain calculations and the associated damage classification for a 100-ft-long structure located over the panel being mined.

| Subsidence control | Maximum extension (+E) | Maximum compression (-E) | Damage classification (from NCB) |
|---|---|---|---|
| With pneumatic stowing (full-face). | +0.00195 ft/ft | -0.00221 ft/ft | Class I to II: Hair cracks in plaster, no visible cracks on the exterior, to doors and windows sticking slightly, Repairs to decoration probably necessary. |
| Without pneumatic stowing (caving). | +0.00390 ft/ft | -0.00441 ft/ft | Class IV. service pipes disrupted and door frames distorted, floors sloping noticeably. |

supplied to the stowing machine by a second conveyor similar to the longwall stage loader and conveyor unit utilized in the coal haulage operation.

Once the stowing material is delivered to the section, it is ejected using a stowing machine with a rated output capacity of 9,800 cu ft/h. The stowing machine discharges material to a 9-in-diam roadway line consisting of lengths of rigid pipe and a hydraulic telescoping section, all lined with cast basalt. From the roadway line, the material flows through a $90^{o}$ bend into the face stowing line which is suspended from rear cantilevers on the longwall shield supports. Each 15-ft pipe section is coupled with specialized couplings and a discharge unit is incorporated (every 25 to 30 ft). There are basically two different discharging methods used in pneumatic stowing, the lateral discharge and the mechanized front-end discharge. In this study, the mechanized front-end discharge is used because it is a newer development and is reported to obtain a better and more unrestricted placement of stowing material in the void. As the cut advances, the face pipe is displaced toward the gob by hydraulic rams and is left suspended from the supports in such a way that it does not interfere with the next stowing section.

As previously mentioned, the average production from the existing caving longwall face is approximately 4,000 st per shift. With concomitant backstowing, the ability to pack the newly created void will become the factor affecting coal production since, within a given time period, the amount of coal mined must be balanced with the ability to fill the void created by mining that coal. For example, if 6 h of production time is available per shift and the stowing system is designed using a stowing machine that has a discharge rate of 9,800 cu ft/h, the amount of void that can be created and fully stowed per shift is approximately 37,500 cu ft. At current longwall mining rates and optimistic backstowing performance, the production from the longwall face after implementing backstowing would drop to about 1,500 st/shift or 4,500 st per day. Although a stowing machine exists which will stow material at a rate of 12,300 cu ft/h, the lower capacity machines are more common, primarily due to compressed-air requirements (Rauer and Voss 1983; Reinshagen 1986; Voss 1987).

## Case-Study Economic Analysis

The economic analysis section consists of a capital cost estimation, an operating cost estimation, additional labor considerations for the stowing operation, and a cash flow analysis to incorporate all of these factors, including the production loss, into the economic impact of implementing such a system in an existing longwall mine.

### Capital Cost Estimation

The capital costs needed for the stowing installation total $12,546,624, which consist of surface equipment ($1,302,480), infrastructure equipment ($5,430,500), and stowing equipment ($5,813,644). It was assumed that the mine already owns a front-end loader to do the job of loading the refuse into the crusher. As indicated earlier, the

materials-handling system used for this case-study was dictated by the inherent features of the chosen site. Also, in an effort to use the most economical methods, waste drop pipes in the shaft and belt haulage in the mains were chosen. Obviously, other methods could be used and should be evaluated according to established site-by-site criteria. The equipment listed represents the minimum requirements for stowing one longwall face.

The total capital cost is based on the most recent manufacturer's equipment prices and the most recent available literature on issues involving the stowing and mining industries. A $1 million contingency has been added to accommodate any minor construction needed to install the stowing system and any further roof supports required, bringing the total capital required to $13,546,624.

## Operating Cost Estimation

The USBM Cost Estimation System Handbook (1987) was used to estimate the operating cost of surface and infrastructure operations. The stowing segment of the operating cost was calculated using a method outlined by Singh and Courtney (1975), due to a lack of published literature and limited information from overseas mine operators and manufacturers. In this method, the operating cost represents the cost of replacement or repair divided by the recommended life of the equipment in tons of stowed material. Results from the calculations are presented in Table 2.

## Additional Labor Requirements for Pneumatic Stowing

Sources of labor information in stowing systems are mostly European and data have been modified for U.S. conditions. Surface personnel needed include a front-end loader operator and an operator for the crushing operation. This crusher operator will be able to monitor the material moving to the storage silo and will also be able to detect problems associated with the movement of material to the shaft waste pipe. Maintenance and emergency situations will be somewhat dependent on the existing personnel of the mine; therefore, the estimate is fairly conservative. For the underground segment of the operation, two additional miners will be needed to maintain the waste belt line. In the face area, the longwall crew will be responsible for running the stowing operations.

In all, an additional eight workers will be needed to sustain the stowing operation. This number will be incorporated into the economic analysis to show the impact on the entire operation. Labor cost per day is estimated to be $219 and is based on an average hourly wage for a typical underground crew.

## Cost Analysis

In the cost analysis, information for the case-study mine was obtained to simulate the production costs for a longwall mine in southwestern Pennsylvania. The analysis was conducted for a 20-yr mine life incorporating one longwall unit and three continuous miner sections. The economic analysis used a labor schedule, production schedule, operating-cost detail, capital-expenditure detail, and a depreciation schedule. Once the base-case analysis was created, information obtained from the previous capital-cost, operating-cost and labor-requirement sections could be used to analyze the effect of adding a pneumatic stowing operation to the mine. Table 3 summarizes the information obtained in the analysis.

Three cases were studied in the cost analysis. In the Base Case, the production cost of a mine incorporating one full-caving longwall section was analyzed. In Case 1, the various cost items and production impacts for pneumatic stowing were added to the Base Case. In Case 2, an additional longwall section and an additional continuous-miner section were added to the mine to attempt to return to the production levels in the Base Case; further, it was assumed that continuous-miner production could increase to 450 st per miner shift when the machines are less constrained by longwall development.

Table 2. - Summary of operating costs for the pneumatic stowing operation.

| Segment of operation | Cost per ton of coal mined |
|---|---|
| Surface facilities: | |
| Loading | $0.3641 |
| Crushing (Power and materials handling included) | 0.1019 |
| Infrastructure: | |
| Shaft pipe | 0.0825 |
| Main waste conveyor | 0.1760 |
| Loading conveyor | 0.1163 |
| Stowing operation: | |
| Stowing machine | 0.0531 |
| Air compressor | 0.2500 |
| Road pipe | 0.0460 |
| 90 degree bend | 0.0457 |
| Face pipe | 0.1728 |
| Telescoping pipe | 0.2897 |
| Discharge units | 0.5000 |
| **Total Operating Cost** | 2.1981 |

Table 3. Summary of economic analysis for a 20-yr-life longwall operation with and without the incorporation of pneumatic stowing.

| Production: | Base case | Case 1 | Case 2 |
|---|---|---|---|
| Annual....st | 3,243,000 | 1,518,000 | 3,0011,500 |
| Daily....st | 14,100 | 6,600 | 13,050 |
| Days/year | 230 | 230 | 230 |
| Tons/shift - CM....st | 350 | 350 | 450 |
| Tons/shift - LW....st | 4000 | 1500 | 1500 |
| Plant recovery....% | 80 | 80 | 80 |
| Labor: | | | |
| Hourly workers/day(avg) | 236 | 244 | 296 |
| Absenteeism....% | 12 | 12 | 12 |
| Overtime....% | 15 | 15 | 15 |
| Salaried workers/day | 66 | 66 | 66 |
| Tons per miner/day | 46.7 | 21.3 | 36.0 |
| Economics: | | | |
| Labor cost (hourly + salary) | $4.69 | $10.28 | $6.08 |
| Operating cost | 13.90 | 16.10 | 16.10 |
| Total cash cost | 18.59 | 26.38 | 22.18 |
| Total operating cost (includes depreciation) | 20.13 | 30.64 | 25.39 |

## Summary and Conclusions

This study has addressed the use of concomitant pneumatic stowing to alleviate the concerns of longwall mining. The use of pneumatic stowing as a subsidence-control measure must rely greatly on the technical and economic feasibility of incorporating the stowing system as an additional unit operation of the existing system. The study included a hypothetical case-study evaluation of pneumatic stowing at a southwestern Pennsylvania mine site to determine its feasibility.

The use of pneumatic stowing in the southwestern Pennsylvania mining district will not completely eliminate surface subsidence caused by longwall mining. For the case-study mine, the maximum subsidence can be reduced to 1.73 ft from 3.45 ft, a 50% reduction. The predicted subsidence profiles are shown in graphical form. To determine the extent to which subsidence is causing damage, the strains associated with the horizontal and vertical displacement were calculated using a method from the NCB. With these strains, a classification of the damage caused can be chosen. Table 1 shows that with stowing, the subsidence damage is reduced from Class IV to between Class I and II.

In the cost analysis of the case-study mine, a Base Case without stowing was compared to the case with stowing (Case 1) and another (Case 2) where production losses due to stowing were accommodated with additional production units. In none of the cases were financing costs, profit margins, taxes on profit, royalties, and cost depletion considered. In addition, stowing material costs and costs to transport stowing materials to the mine site were not considered for Cases 1 and 2.

The significant result of this study indicates that current stowing technology cannot keep pace with the production potential of U. S. longwall systems. The case study indicated that (1) stowing would reduce subsidence by

50% of that incurred during full longwall caving, (2) longwall coal production would decrease, and (3) stowing would add at least $6 to $10 to the operating cost of producing a ton of coal.

## Acknowledgments

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# EFFECTS OF LONGWALL MINING SUBSIDENCE ON GROUND WATER LEVELS WITHIN A WATERSHED HYDRAULICALLY ISOLATED FROM MINE DRAINAGE[1]

Bogdan Staszewski[2]

**Abstract:** Surface and ground water resources are effectively preserved from depletion by underground mine drainage if impervious deposits of sufficient thickness and extent that underlie the aquifer avoid fracturing and undergo only plastic deformation resulted from the strata flexure. This does not mean, however, that these waters are not subjected to the effects of mining disturbance. Differential vertical settlement of the mine overburden and the ground surface can significantly affect flow pattern and water retention within a watershed area. This is evident, for example, in the areas of multiseam coal mining where the longwall method is used. The effects of this type of mining on water level were tested in a selected watershed where shallow water bearing deposits were entirely isolated from underground mine pumpage. Results of more than 7 years of field investigations were compared with data collected from other coal mining areas of various hydrogeological conditions. The study revealed a variety of water table responses on the postmining subsidence. Basically, changes of water table height in a given site above the point located at the top of aquifer base depend mainly on alteration of that point position against the local drainage base in the hierarchic structure of flow system. The relationship between the magnitude of ground subsidence and water level decline varies within the area of the subsidence trough, within the watershed, and among various watersheds of different hydrogeologic conditions. Except for the situation of hydrostatic flow conditions, lowering of water table elevation in response to the settlement of aquifer base was observed.

**Additional Key Words:** monitoring of watershed, induced ground water fluctuations, mining damages, inundations.

## Introduction

Underground mining accompanied by deformations and fracturing of the overburden can cause partial or complete dewatering of the overlain water-bearing strata. The vertical range of the drainage influence can be restricted by a layer of waterproof sediments if it occurs above the zone of caving and does not undergo fracturing. Then all the overlying water-bearing layers will remain hydraulically isolated from mine drainage. This does not mean, however, that shallow ground water horizons are free from mining disturbance. Local inundations of land surface are the most characteristic effects of postmining subsidence in such a case. These effects become a major environmental problem in some coal mining areas where the longwall system with caving is carried out. Using this system for extracting the multiseam deposits results in lowering of the aquifer base by several or even tens of meters.

It is often claimed that subsidence of the aquifer base does not involve change of the absolute (sae level) elevations of ground water level, which means that depth to ground water decreases of the value equal to the magnitude of subsidence. In that case, the aquifer is considered to be an underground reservoir and the assumption of hydrostatic law is valid. The question still exists as to the impact of subsidence on water level in dynamic flow conditions. It was tested in a selected watershed. Results of field investigations were compared with data collected from other coal mining areas of various hydrogeological conditions.

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24÷29, 1994.

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## Investigations in the Selected Watershed

### Objective and Scope

The objective of the investigation was to record the behavior of ground water table as a response to subsidence resulting from longwall mining within dynamic water flow system free from the influence of mine drainage. It was aimed at better understanding the processes leading to the development of land inundation due to uneven subsidence in watershed scale.

The field investigations comprised baseline data collection, setting up an observation network for monitoring subsidence and water level fluctuations, observations of changes in flow pattern in the course of ground movement and after its stabilisation. Some results of these investigations within one of the selected watersheds in the Upper Silesian Coal Basin are presented below.

### Description of the Project Site

The watershed, with an area of 3.5 sq km, is covered with Quaternary sediments composed of silts, sandy clays and sands. Their total thickness lies between 10 and 30 meters. The water bearing horizon is associated with the occurence of 1-2 layers of sands. The upper layer of continuos extent is covered on prevailing part of the watershed by semipermeable silts of 1-3 m in thickness and it forms an unconfined or semiconfined, leaky aquifer. The second layer occur locally at the bottom part of Quaternary deposits and is of confined flow conditions.

The Quaternary deposits are underlain by Tertiary formations of 140-300 m in thickness and build up with nonpermeable Miocene clays and claystones that separate shallow aquifers from the impact of coal mine drainage. Coal seams are mined under the watershed with the longwall system at depths 350-750 m, and about 80% of the watershed area is at present subjected to postmining subsidence.

### Results of Monitoring Ground Water Level Fluctuations and Mining Subsidence

The effects of mining subsidence on the change of ground water level is superimposed on the response due to natural factors. To identify the induced modification to the elevation of water level, adequate long periods of observation are necessary and sometimes the reference to water level fluctuations in other points situated outside the range of mining influence is required.

The low ground water level stage recorded in October 1982 was chosen as the reference stage on the beginning of observation. Next, which proved to be the best comparable natural water level stage during the investigation period, was the stage of November 1986. Under the comparison of both stages, the change in ground water elevation within basinwide scale was approximately defined (fig.1). Over almost the entire drainage basin, decline of ground water elevation was observed. It ranges between 0.5 m to over 2 m. Relative to the corresponding ground surface subsidence that occurred during this period, changes of depth to ground water were found (fig.2).

The rise of water table relative to ground surface occurred in the central part of the subsidence trough and in areas where subsidence tilt is counter to the direction of ground water flow. Depth to ground water increased in the upper wing of the subsidence trough and in areas where subsidence tilt is the same way as the flow. Areas of positive and negative changes of water level with regard to ground surface are separated by zones where the magnitude of water level drop and subsidence are approximately equal to each other. It is illustrated by profiles of ground surface subsidence and water level fluctuations in well s. 19a (fig.3).

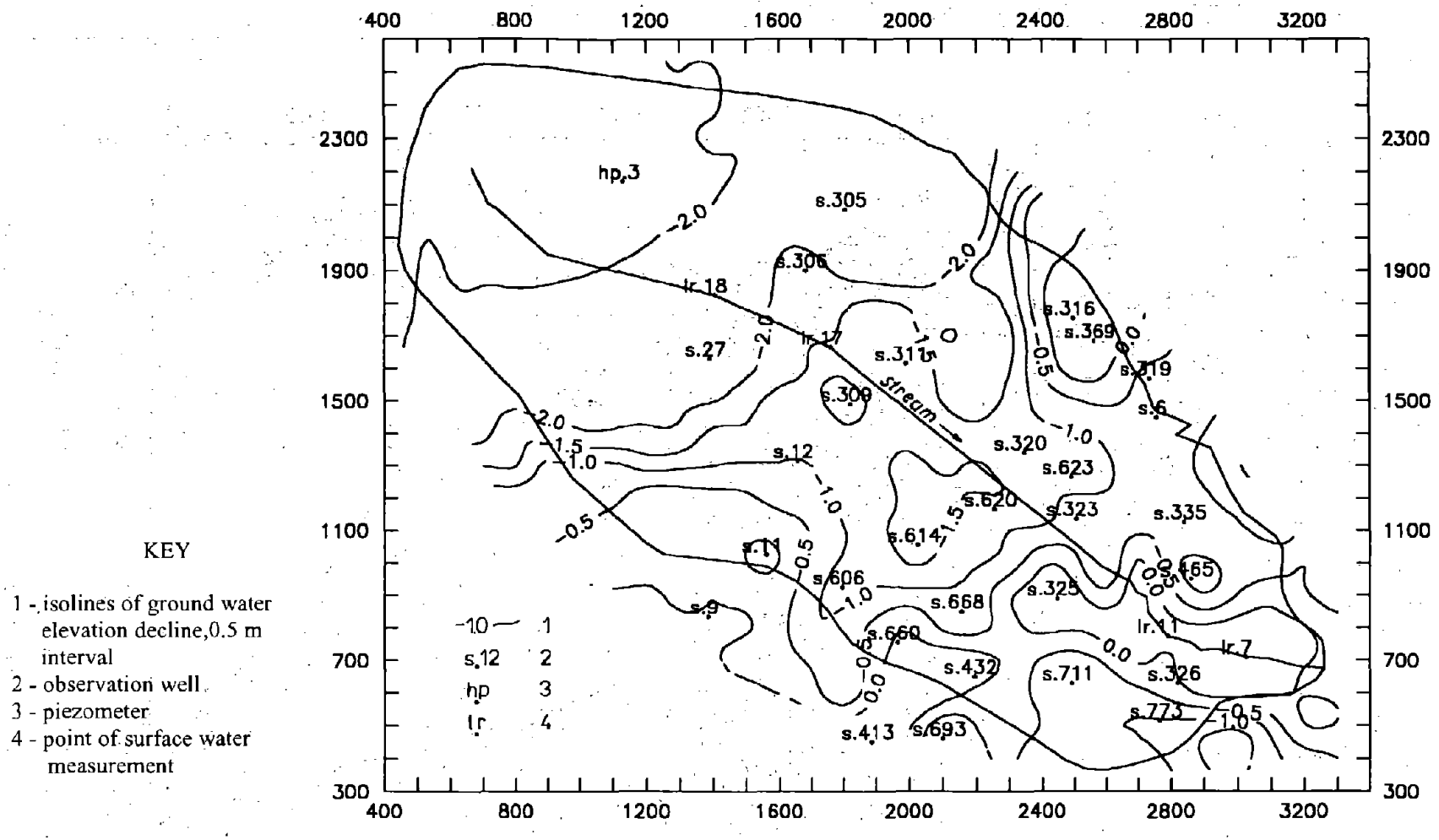


Figure 1. Changes in elevation of ground water level, October 1982 - November 1986 within the research watershed

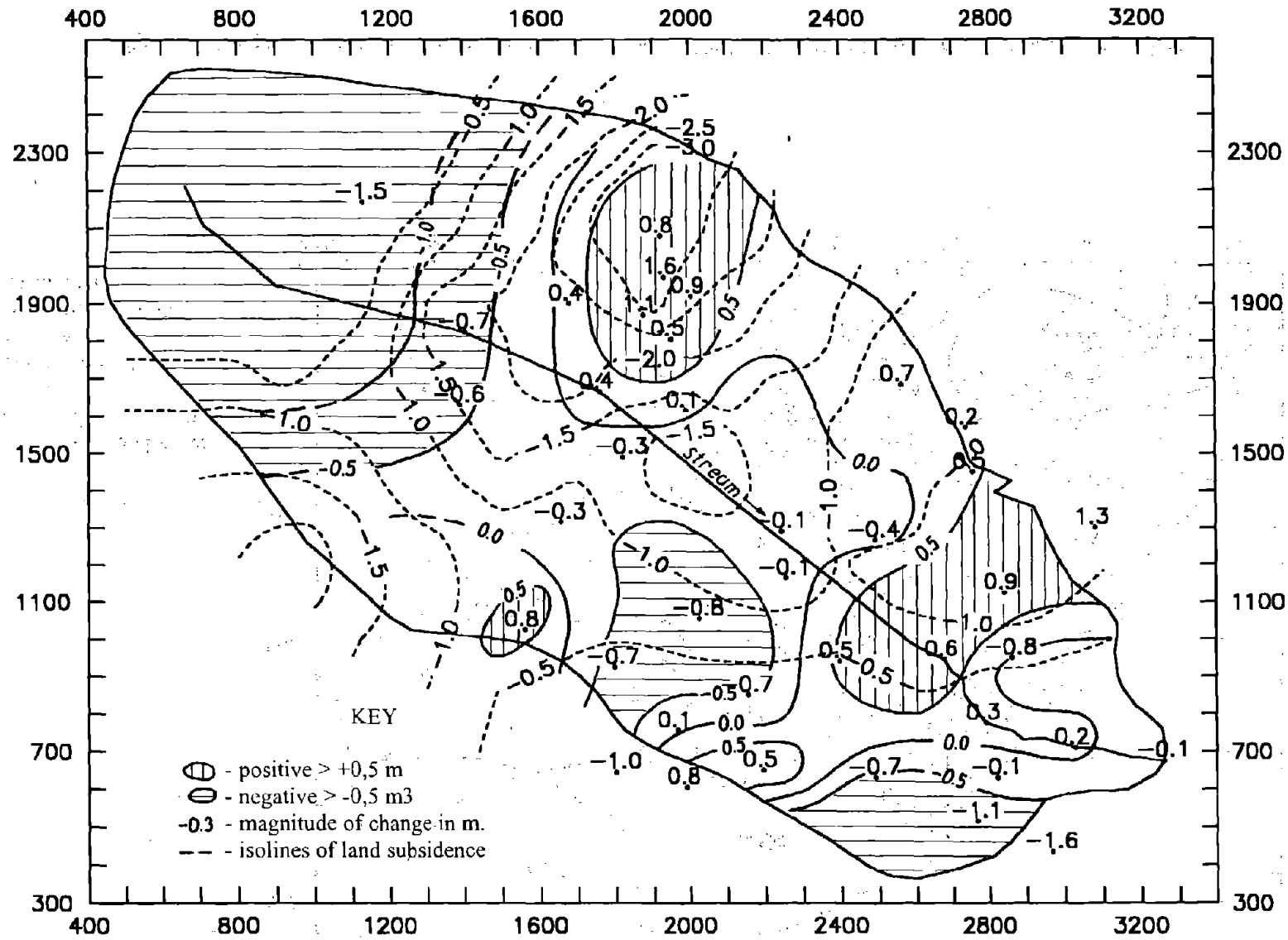


Figure 2. Changes in depth to ground water level, October 1982 - November 1986 within the research watershed.

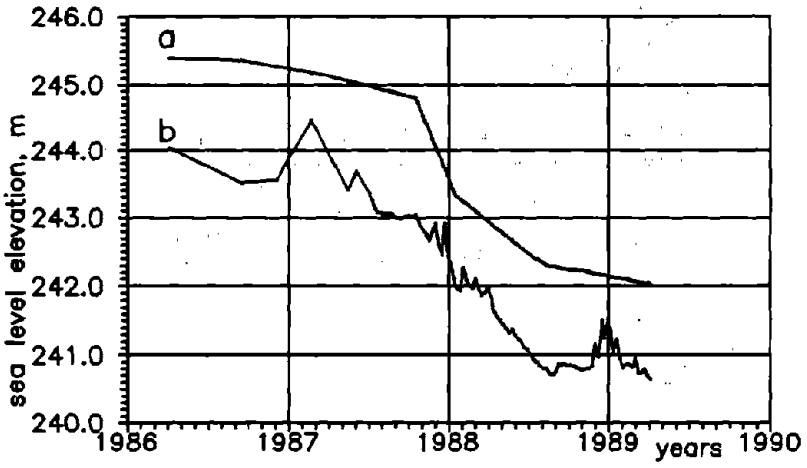


Figure 3. Profiles of surface subsidence (a) and fluctuation of water elevation (b) in well s. 19a.

## Relationship between the magnitude of mining subsidence and decline of ground water level

Results from other test watersheds and other drainage basins of different hydrogeological conditions have revealed a variety of water level responses to mining subsidence.

The magnitude and character of these changes depend on the depth, range, and geometry of a subsidence trough, as well as on natural conditions of a drainage basin in which this trough is being developed. The differentiation of individual features of a waterflow system causes the same subsidence in various drainage basins to create various effects. For example, the impact of mine subsidence on ground water level over the collieries in Da Tun Coal Mining Area, Jiangsu Province, China, was analysed. Subsidence troughs that have developed there in a vast drainage basin of infinite water resources within the alluvial plain of the Yellow River have not caused a change in ground water elevation. Depth to ground water has decreased by a value equal to subsidence (Staszewski, 1988).

Another example comes from small watershed bounded by the outcrops of water-bearing layer and entirely covered by the subsidence trough over supercritical extraction panels. Here, the ground water elevation has decreased by a value approximately equal to the magnitude of subsidence (Staszewski, 1992).

Essential to the problem discussed here is temporary position of a given point located at the top of aquifer base against the local drainage base. The local drainage base is understood to be the elevation of an aquifer base or streambed at the outlet of basin, subbasin, or catchment of the just developed subsidence trough, while the elevation of water table above the drainage base is called outflow level.

It is obvious that the relationship between subsidence and drop of water elevation at a given point is changing over the course of subsidence development. This is illustrated in figure 4, where curve H denotes a decline in ground water elevation and curve z- subsidence of the base of a flow path over time. Figure 4a presents the situation of a stable drainage base. In this case, after the base of flow path is lowered down to the level of the drainage base, water table reaches its lowest position (outflow level) and further settlement of the base of flow path has no impact on the water level elevation. The effect is an increase in saturated thickness of the aquifer h, and the flow path is transformed into a stagnant water body. Within small watersheds, drainage bases also undergo subsidence, as does the outflow level (fig. 4b). Even then, lowering of water elevation is limited by the elevation of a drainage base of the higher-order drainage basin (fig. 4c).

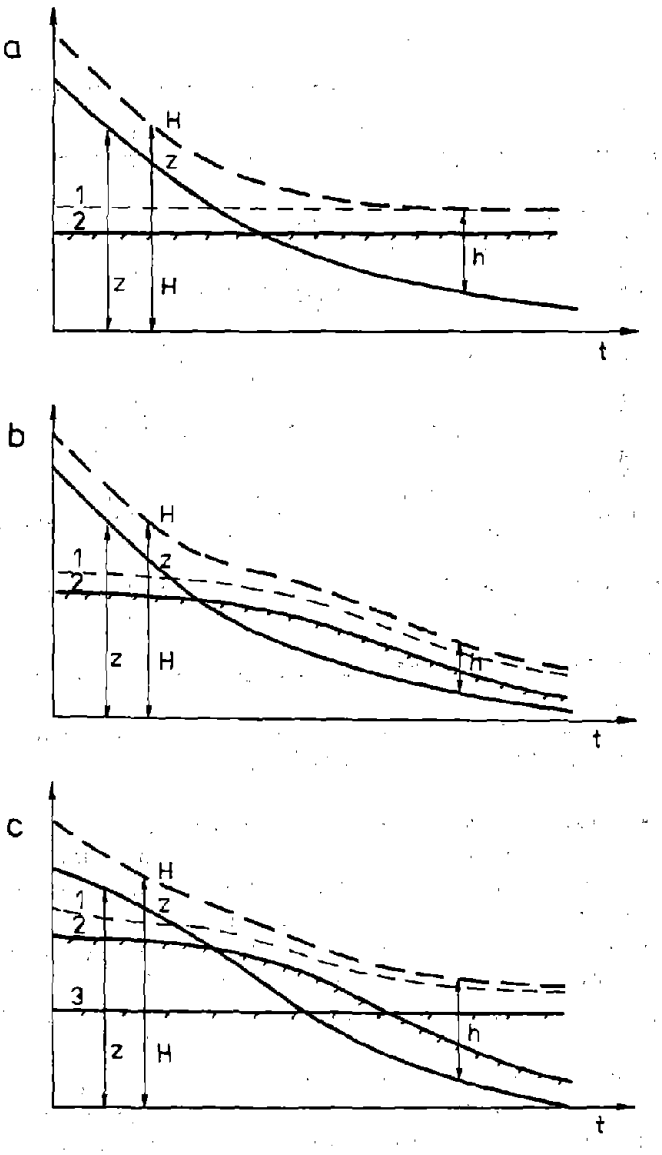


Figure 4. Diagrammatic representation of decline of ground water elevation (H) in course of subsidence of the aquifer-base (z) in a given point.

a - under stable drainage base, b - under lowering of the local drainage base, c - under lowering of the local drainage base and stable drainage base of the higher order flow system.
1 - outlow level, 2 - local drainage base, 3 - drainage base of the higher order flow system.

## Conclusions

As a result of underground mining, settlement in the base of flow path causes, with few exceptions a drop in water level elevation. The magnitude of this drop in relation to subsidence varies within the subsidence trough, within the drainage basin, and among different drainage basins. It also varies with time depending on to what degree the deformations of the base surface of an aquifer modify the initial hydraulic gradient. The magnitude of the drop is controlled by the reduction of elevation head over the local drainage base. Lowering of the base of the flow path down to or below elevation of the drainage base leads to a situation in which hydraulic gradient reaches its critical value. Further settlement of the base has no impact on position of water level, and the flow path transforms into a stagnant water body.

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# CONSOLIDATION BEHAVIOR OF OIL SAND FINE TAILINGS [1]

Nagula N. Suthaker and J. Don Scott[2]

**Abstract:** In northern Alberta, oil sand deposits are mined and processed to recover heavy oil. Oil sand plants produce about 650,000 mt/d of tailings. Approximately one-half of the fines in the tailings streams flows into tailings ponds to form fine tails deposits which presently total 325 million $m^3$.

A slurry consolidometer has been developed to allow for large deformations during consolidation and to allow permeability tests to be performed on the fine tails. Development of a top cap clamping system has permitted permeability testing to be conducted on the high-void-ratio slurries without having seepage-induced consolidation. The equipment, the fine tails material, and the testing methods are described.

The permeability test results revealed a time-dependent flow with a constant hydraulic gradient. The flow velocity drops up to two orders of magnitude before reaching a steady-state value. Unlike normal soils, it was found that the amount of compressibility of the fine tails is influenced by the initial void ratio of the sample, which suggests that aging changes the microstructure of the fine tails. This suggests that there is a time-dependent parameter involved in the compressibility of fine tails. Therefore, a single void ratio-effective stress relationship is not sufficient to describe the consolidation behavior of fine tails. The permeability-void ratio relationship, however, is not influenced by the initial void ratio. These experimental results suggest that permeability and compressibility are controlled by different elements of pore structure.

**Additional Key Words:** tailings, compressibility, hydraulic conductivity, mine waste.

## Introduction

In mining operations, the waste material from a mineral processing plant is generally in the form of a slurry. The slurry is deposited hydraulically and contained in some type of pond-dyke arrangement. Design problems associated with these ponds include storage capacity, embankment stability, seepage, and land reclamation. The physical properties of the waste material that affect the design of the waste storage facility are compressibility, permeability, and shear strength. Many waste mineral slurries contain significant amounts of silt and clay fraction. The presence of clay can result in a very high initial void ratio and subsequently very large volume changes during consolidation.

In northern Alberta, oil sand deposits are mined and processed to recover heavy oil from two oil sand plants, Syncrude Canada Ltd. and Suncor Inc. The tailings streams are composed of

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[1]Paper presented at the International Land Reclamation and Mine Drainage Conference and the Third International Conference on the Abatement of Acidic Drainage, Pittsburgh, PA, April 24-20, 1994.

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about 85% sand and 15% fines by weight plus a small amount of heavy oil that bypasses the extraction plant. Syncrude and Suncor produce, respectively, about 480,000 mt/d and 170,000 mt/d of tailings, which range in solids content from 40% to 60%. The tailings sand forms the pond dikes and beaches and accumulates at a rate of 225,000 $m^3$/d. Approximately one-half of the fines in the tailings streams flow into the pond to form a fine tails deposit. After sedimentation and approximately 2 yr of consolidation, the fine tails form a deposit at about 30% solids, which accumulates at a rate of 45,000 $m^3$/d. Approximately 325 million $m^3$ of this fine tails are presently held in the tailings ponds. The material, by economic necessity, must rely on self-weight consolidation for its densification. Self-weight consolidation is very slow in the tailings ponds. The result is a necessity to continually enlarge the containment ponds and dikes to hold the rapidly accumulating large volumes of material. Therefore, it is necessary to understand the consolidation rates and behavior of the fine tails before an assessment of the effectiveness and feasibility of long range disposal plans can be made.

The consolidation behavior of the fine tails is controlled by the compressibility, permeability, thixotropic strength, and creep properties of the material. Laboratory tests have been performed to measure these properties and their variations with porosity and time. The fine tails material is unique because of its properties and the vast volume that must be disposed in an acceptable environmental and economical manner. The consolidation and permeability equipment, the fine tails material, and the testing methods are described in the following sections.

## Consolidation Testing

The standard oedometer test is developed for small-strain consolidation problems and generally adopts small-strain theory. In the oedometer test, constant loads are applied to the sample, and the settlement with time is observed for each load. This allows a relationship between stress and void ratio, coefficient of compressibility, and, by using theory, a coefficient of consolidation and then the hydraulic conductivity to be obtained.

There are two significant experimental difficulties in using the standard oedometer test for the fine tails. The main problem is the large strain that takes place. The other problem is the determination of hydraulic conductivity. Permeability testing may cause seepage-induced consolidation. There are several methods of determining the coefficient of consolidation and calculating the hydraulic conductivity from the laboratory data. All incorporate the assumption that the compressibility and the coefficient of consolidation are constant during each load increment. This assumption presents the major restriction to the applicability of the analytical procedures. The accuracy of the determination of the coefficient of consolidation is inversely proportional to the size of the load increment (Znidarcic et al. 1984).

## Modified Consolidation Testing

Advanced testing techniques have been developed to overcome the problem of the test duration of large-strain oedometer tests. Such tests include the constant rate of deformation (CRD) test (Lee 1981), the constant hydraulic gradient (CHG) test (Lowe et al. 1969), and the constant rate of loading (CRL) test, all of which are faster than the step-loading oedometer test. These test procedures employ an inversion of the Terzaghi infinitesimal consolidation theory to yield usable results. Therefore, their applicability is restricted by this theory's assumptions (Znidarcic et al. 1984).

The CRD test has to be done at the field deformation rates since the test results are dependent on the rate of deformation. The assumptions required for the reduction of data include

infinitesimal strain, constant coefficient of consolidation, and the assumptions regarding the relationships between either the void ratio and time or the void ratio distribution in the sample. Neither assumption can be validated (Znidarcic et al. 1984).

In the CHG test, the loading rate is continually adjusted through a feedback mechanism, such that the pore pressure at the undrained boundary remains constant; hence the hydraulic gradient is constant within the sample throughout the test. The assumptions include infinitesimal strains, constant permeability, a linear compressibility, and a constant void ratio throughout the sample (Znidarcic et al. 1984).

The CRL tests (Aboshi et al. 1979) and constant loading tests (Janbu et al. 1981) have similar assumptions and restrictions (Znidarcic et al. 1984). The seepage test (Imai 1979) does not rely on Terzaghi's theory for consolidation parameters. However, sample rebound at the completion of this test may lead to erroneous results.

The above-mentioned tests, except for the seepage test, with their analyses have been set up for soils that obey the assumptions involved in Terzaghi's infinitesimal consolidation theory. These tests are not applicable to highly compressible soils such as dredged materials and mine waste slurries, which undergo large deformations during consolidation. To overcome such setbacks, slurry consolidation tests have been developed for these very soft materials.

## Slurry Consolidation Testing

Slurry consolidometers have been developed to allow large deformations during consolidation and to allow permeability tests to be performed on the samples. The test is conducted similar to the oedometer test in which a step-loading procedure is applied, allowing for the direct determination of the effective stress-void ratio relationship. The permeability-void ratio relationship can be determined by conducting permeability tests on the sample after each load increment. Seepage-induced consolidation can be prevented using a load-fixing device. Therefore the major difficulty remaining with the procedure is the test duration.

Though several procedures have been developed to decrease the time involved in consolidation testing of clayey soils, they involve inverting the complex finite-strain consolidation theory, and therefore their usefulness depends on the assumptions used in the inversion process. Therefore in this research, a long test duration was accepted and a slurry consolidometer with step loading procedure was used.

Imai (1981) noted that for highly active soil slurries that undergo hindered settling during the sedimentation phase, the initial void ratio at the start of consolidation is strongly dependent on the initial void ratio of the slurry. In Imai's study, several compression curves were found in the low-pressure range (less than 0.1 kPa). Above this effective stress value, there was a unique compression curve. To investigate the effect of the initial void ratio on the consolidation of oil sand fine tails, three different initial solids contents were employed in this research.

## Permeability Testing

To understand the consolidation properties of a soil, laboratory testing is necessary to determine its hydraulic conductivity. For soils undergoing large changes of void ratio, it is necessary to determine the hydraulic conductivity as a function of the void ratio.

## Indirect Methods of Determining Hydraulic Conductivity

Indirect methods of determining the hydraulic conductivity are carried out by inverting a consolidation theory and applying it to the rate of compressibility data obtained from a consolidation test. The most common method of indirectly determining the hydraulic conductivity in the standard oedometer test is either from the logarithmic time plot or square root time plot of the consolidation of a soil under a constant stress. These methods of determining hydraulic conductivity have been found to be unacceptable for slurry consolidation because of experimental (e.g., high strain rates) and analytical reasons (Terzaghi's assumptions).

Tavenas et al. (1983) indicated that the back-calculated values of permeability underestimated the measured values by up to six times for soft clays. The reason for the difference is the assumptions in Terzaghi's consolidation theory. The assumptions that are mostly violated are constant permeability, constant compressibility, and hence a constant coefficient of consolidation. Tavenas et al. concluded that indirect methods are unacceptable in determining the permeability characteristics of soft clays.

## Direct Methods of Measuring Permeability

The direct method involves forcing a fluid through the soil and monitoring either the rate of flow through the soil or the hydraulic head changes induced by it. The two most widely used methods of directly determining the hydraulic conductivity of a soil are the constant-head and the falling-head permeability tests.

The advantages of these two tests are their simplicity in testing procedure, apparatus, and evaluation of the test data. The disadvantage of the conventional methods is the length of time required when performing low-gradient permeability tests. When direct methods are used for highly compressible slurries, low gradients must be used since large gradients induce consolidation due to seepage. The falling-head test is unfeasible with low gradients.

A constant-head technique was chosen for this research because it allows for extremely small head drops during the test and because of its ability to monitor time effects on flow rate. A variable-head test is not feasible for both of these reasons.

## Experimental Program

To determine the compressibility and permeability data for the oil sand fine tails, a large-strain slurry consolidometer was used (fig. 1). This apparatus allows a void ratio-effective stress relationship and a permeability-void ratio relationship to be obtained.

A step-loading procedure, similar to a standard oedometer test, was employed owing to its simplicity and the ability to perform permeability testing after the completion of consolidation under each load increment. A constant-head technique was employed for the permeability portion of the test. Since a change in pore fluid chemistry may affect the permeability, tailings pond water was used as permeant to achieve consistent results with the field conditions.

Common to all slurry consolidation test methods is the danger of consolidating the slurry during the permeability test. Consolidation occurs when the stresses introduced by the seepage force of the permeant are greater than the stress under which the sample was previously consolidated. To overcome this problem, a clamping system was used to fix the loading ram, which sits on the soil, to prevent its movement. This system then allows an upward hydraulic

gradient to be applied to the sample, up to a gradient where the induced seepage stress is equivalent to the previously applied consolidation pressure.

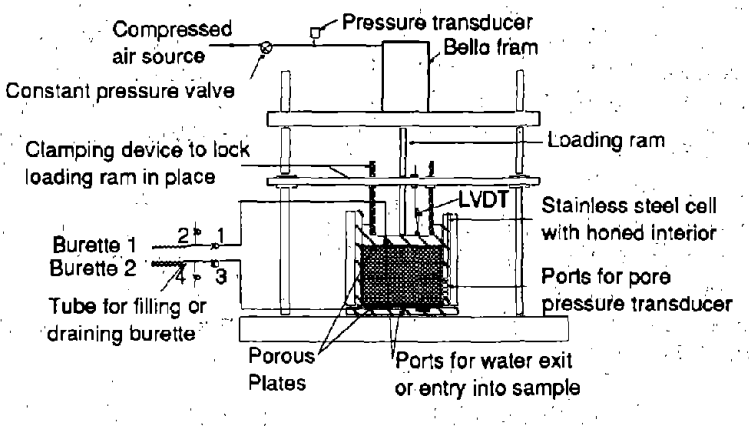


Figure 1. Slurry consolidometer.

Oil sand fine tails with three different initial solids contents (20%, 25% and 30%) were tested in the slurry consolidometers. Liquid and plastic limits of fine tails are approximately 59% and 23%, respectively. Bitumen content by mass of the fine tails is 3% of the mineral solids. The applied stress increments were equal to the previously applied stress. The consolidation tests and permeability tests lasted for up to 2 yr. This elapsed time is from the start of the self-weight consolidation stage to the end of the last permeability test of the final load increment.

## Results and Discussion

Figure 2 shows a typical change of void ratio with elapsed time. For some of the smaller applied increments, it was necessary to stop the consolidation before the displacement had leveled out on a logarithm time plot. Although on the log time plot the material appears to be rapidly consolidating, the material has entered the $10^4$-min log cycle, and any additional change in void ratio would be very small.

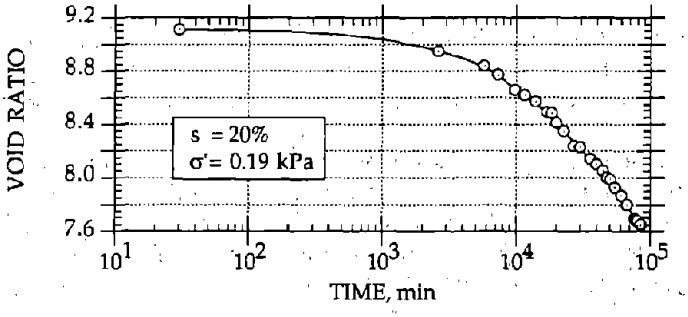


Figure 2. Void ratio variation with time.

Excess pore pressures were measured at five ports using a pressure transducer. The ports are numbered from the top (fig. 1). Figure 3 shows a typical excess pore pressure variation with time.

Initially, the consolidation stress is the midheight self-weight in the sample. Subsequently, the stress is the midheight self-weight plus the stress applied by the loading ram.

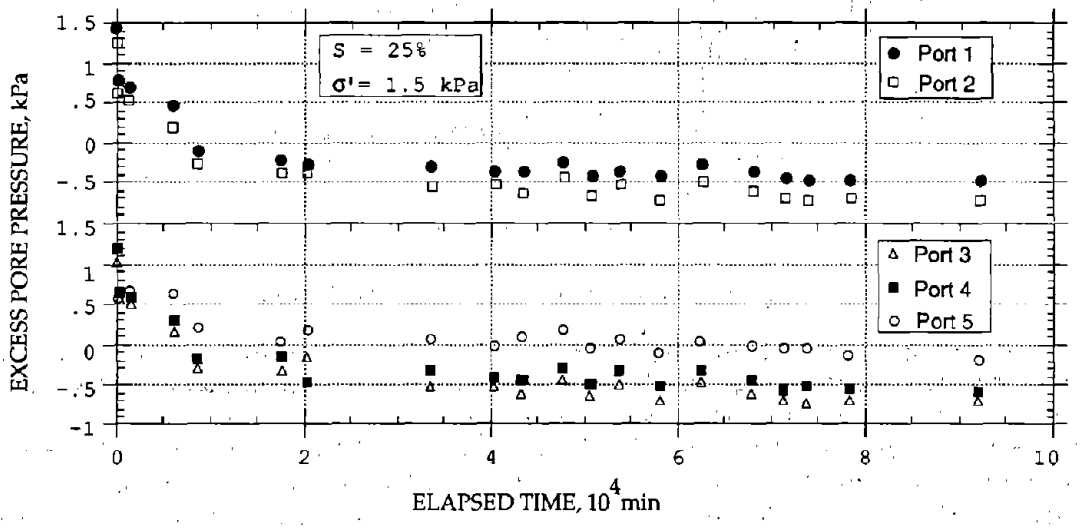


Figure 3. Excess pore pressure variation with time.

Figure 4 shows a typical relationship of flow velocity with elapsed time during a permeability test. The flow velocity decreased until it approached a steady-state value. The flow velocity at this steady state was used to determine the hydraulic conductivity. Several permeability tests were run at different gradients, and this phenomenon was found to occur repeatedly. This finding suggests that the decrease in flow velocity may have been triggered by seepage forces and is reversible. The bitumen in the fine tails might account for this phenomenon. Although considered as a solid in calculations, the bitumen is deformable under stress and not totally rigid. This deformable quality could allow bitumen to move to block pore throats while being subject to a seepage stress. Once the seepage has stopped, the bitumen would revert to its initial position or shape. A similar drop in flow velocity was also experienced at lower void ratios; however, the drop in flow velocity from initial to steady state became less as the void ratio decreased. It would be expected that the flow velocity drop would be less, because at very low void ratios very little movement of the bitumen can take place.

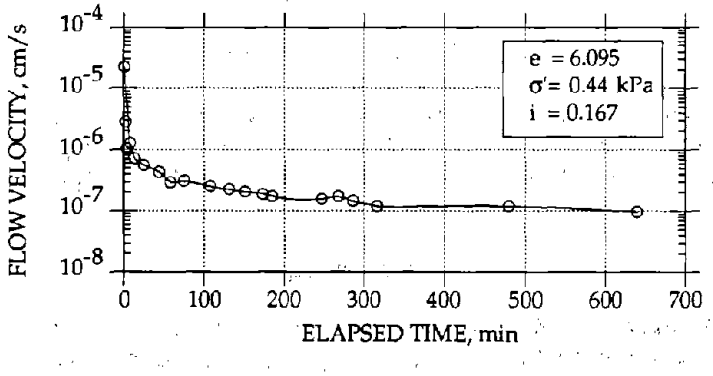


Figure 4. Variation of flow velocity with time.

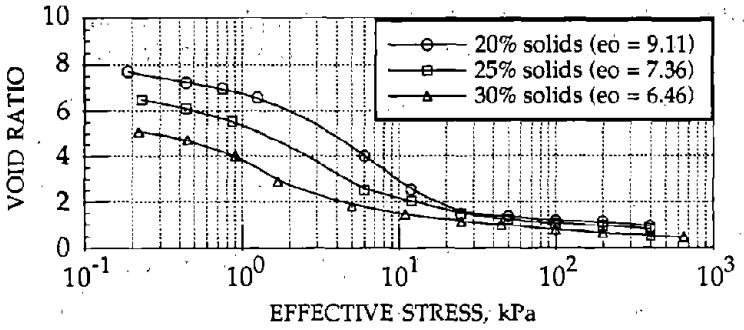


Figure 5. Compressibility of oilsand fine tails.

Figure 5 depicts the compressibility of the fine tails. The results for the 30% initial solids content test were obtained from Pollock (1988). It is clear from figure 5 that the effect of initial void ratio is substantial in the low effective stress range that exists in the tailings pond. When the effective stress reaches about 100 kPa, the effect of initial void ratio becomes small. The conclusion is that the compressibility of the fine tails is dependent on the initial void ratio of the sample. Usually, in soil mechanics, the recompression of overconsolidated soils follows a different void ratio-effective stress relationship until the virgin compression state is reached. However, fine tails are not overconsolidated soils; they are underconsolidated and therefore should follow the virgin compression line irrespective of the initial void ratio. The experimental results suggest that the process of aging develops different fine tails microstructures. Therefore a time-dependent factor has to be taken into account which connects the individual void ratio-effective stress relationships.

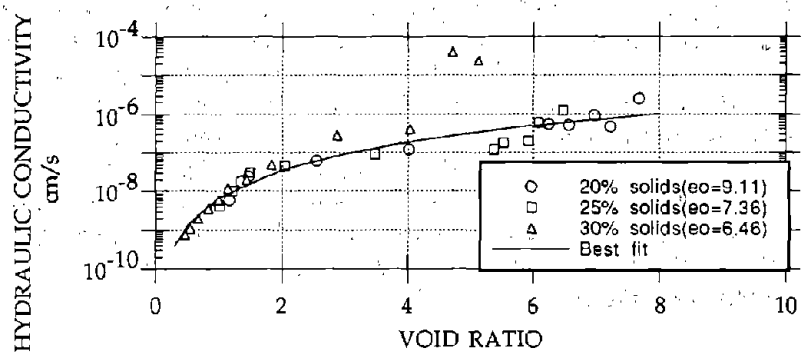


Figure 6. Permeability of fine tails.

Figure 6 shows all the hydraulic conductivity values for different void ratios. Unlike the void ratio-effective stress behavior, void ratio-permeability behavior does not appear to be influenced by the initial void ratio. It suggests that different micropore structures control the permeability and the compressibility.

## Conclusions

A slurry consolidometer has been developed to allow large deformations during consolidation and to allow permeability tests to be performed on the fine tails. Development of a top cap clamping system has permitted permeability testing to be conducted on the high-void-ratio slurries without introducing seepage-induced consolidation. The permeability test results revealed a time-dependent flow discharge at a constant gradient. The flow velocity would drop up

to two orders of magnitude before reaching a steady-state value. The steady-state values were used to calculate the hydraulic conductivity.

Unlike normal soils, the compressibility of the fine tails is controlled by the initial void ratio of the sample, which suggests that aging changes the microstructure of the fine tails and hence the compressibility. The above analysis suggests that there is a time-dependent parameter involved in the compressibility of fine tails. The older the fine tails, that is, the longer they have consolidated in the tailings pond, the smaller the void ratio they will reach under an applied effective stress. Therefore a single void ratio-effective stress relationship is not sufficient to describe the consolidation behavior. On the other hand, the permeability is not influenced by the initial void ratio, which suggests that permeability and compressibility are controlled by different elements of pore structure.

The above properties have been measured to allow the development of an analytical model for the long-term consolidation behavior of fine tails. The design of the eventual disposal and reclamation procedures and facilities for the oil sand fine tails requires such an analytical model.

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# COMPUTER AIDS LAND RECLAMATION PLAN IN MINE AREAS[1]

Bian Zhengfu[2], Wang Junsheng[3], Wang Tongji[4]

**Abstract.** Land reclamation plan is a basic and complex work for land reclamation engineering. This paper introduces the technique of a computer-aided subsided land reclamation plan (**CASLRP**) and focuses on the suitability evaluation of subsided land. Through suitability evaluation, how much land is suitable for development can be determined and the border of different land utilization and the program chart of land use can be drawn by computer. A practical example is given in which the **CASLRP** technique is applied to a mined area of Jiangsu Province.

**Additional Key words:** reclamation plan, suitability evaluation, subsidence.

## Introduction

Land reclamation planning is basic and complex work for mine land reclamation engineering (Bian Zhengfu and Guo Dazhi 1990). The Stipulations of Land Reclamation (State Council of the People's Republic of China, Novemeber 1988) clearly stipulates that a land reclamation plan should determine the direction of land use according to economical principle, natural conditions, and the state of land destruction; the plan should be in tune with the overall plan of land use, and integrated with the mine plan. Therefore, appropriate evaluation of the features of land destruction is necessary before the land reclamation plan is made, and suitable measures to harness subsided lands must be taken into account. Thus, a land reclamation plan must be made based on a full investigation, precise data, and adequate evaluation. The investigation should include the features of climate, the state of land destruction, the distribution of water bodies and other resources, the state of soil erosion, the amount and types of plants, etc.

The basic data for a land reclamation plan includes not only social, economic, environmental, geological, and mining aspects, but also some basic charts such as a physical chart with the natural features of subsided land, statistical curves with information about such things as the output of a mine, reclamation conditions, a surface-underground contrast plan with the terrain and coal face on it, a present map of land utilization, and a distribution map of water bodies. To reasonabaly evaluate the suitability of land, it is necessary for land undermined with potential subsidence to predict its subsidence and a map of subsidence prediction results must be drawn. Therefore, a great deal of work processing data and drawing charts is necessary. It is inconceivable that this work be completed by hand.

Over the past years, we devoted ourselves to the development of a Computer Aided Subsided Land Reclamation Planning (**CASLRP**). This technique provides a highly efficient and reliable method of generating a reclamation plan for subsided land. The **CASLRP** technique comprises of five parts (Bian Zhengfu et al. 1993): 1)creating the basic charts; 2)processing the data; 3)evaluating the suitability of the site; 4)optimizing the reclamation plan and 5)outputing the final plan. In the past, except for the part of evaluating the suitability of the site, the other parts have been researched in some detail (Lin Jiacong and Bian Zhengfu, 1989). This paper will concentrate on

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the suitability evaluation of subsided lands.

## Model for Evaluating the Suitability of Subsided Land

Suppose there are m factors that affect a certain land use. For example, they are $X_1$, $X_2$, $X_3$, ..., Xm. Their state vector is $\mathbf{C}_i$,
where:

$$\mathbf{C}_i = (c_{i1}, c_{i2}, \ldots, c_{ij}, \ldots, c_{in}) \quad (1)$$
$$i = 1, 2, \ldots, m;$$
$$j = 1, 2, \ldots, n.$$

i represents a factor, j represents a suitability grade of a certain land use. For each land use every state vector is a totally ordered set ranging from an optimal grade to a bad grade. That is:

$$c_{i1} > c_{i2} > \ldots > c_{ij} > \ldots > c_{in} \quad (2)$$

Suppose $a_i$ means the weight of influence factor i on a certain land use ($0<a_i<1$), $p_{ji}$ in hundred-mark system represents the influence of a different factor on the grade j of the land use object. If a grided data structure is used, every grid block has a corresponding index vector $\mathbf{UN}(O_k)$.

$$\mathbf{UN}(O_K) = \sum_{i=1}^{m} a_i p_{ji} = (un(O_k))_j \quad (3)$$

Suppose $\mathbf{U}(O_k) = 1 - |\mathbf{UN}(O_k) - p_j/100| = (u(O_k))_j$
Where, $O_k$ represents the land use target, k = 1~t. Then, j making the value of $u(O_k)$ maximum is the grade of the grid block.

Suppose $\mathbf{C}_j(O_k)$ represents the state vector of target $O_k$ to be evaluated which can be used for scaling the target $O_k$, $\mathbf{U} = \{O_t, t \in \mathbf{T}\}$, $\mathbf{T}$ is target set, then the step to determine which grid block is suitable for the target and which suitability grade it belongs to is as follows:

(1) Choose targets. Choose t targets $O_1$, $O_2$, ..., $O_t$ from $\mathbf{U}$ as following rules, that is, $c_i(O_1) > c_i(O_2) > \ldots > c_i(O_t)$ is always right under the condition of $1<i<m$.

(2) Evaluate the grade of suitability. The suitability grade can be regarded as a fuzzy subset within its universe of discourse $\mathbf{U}$, written as $\mathbf{N}$. The membership function of $\mathbf{N}$ must be determined before the grade of grid block is determined by its membership grade of $\mathbf{UN}(O_k)$. The mark of the reference target can be determined by Delphi Method (Huang Xingyuan 1989, Huang Xingyuan and Ni Shaoxiang 1993).

(3) Construct the matrix $\mathbf{A}$.

$$\mathbf{A} = \begin{bmatrix} a_1p_{11} & a_2p_{12} & \cdots & a_mp_{1m} \\ a_1p_{21} & a_2p_{22} & \cdots & a_mp_{2m} \\ \cdots & \cdots & \cdots & \cdots \\ a_1p_{n1} & a_2p_{n2} & \cdots & a_mp_{nm} \end{bmatrix} \quad (4)$$

The row vector of matrix **A** is a quantitative form of n reference vectors in every class.

(4) Compute the membership grade of grid block. The method is as formula(3). $\mathbf{UN}(O_k)$ is the mark of target $O_k$ also.

(5) Calculate and compare $u(O_k)$.

$$\mathbf{U}(O_k) = 1 - | \mathbf{UN}(O_k) - p_{ji}/100 | = (u(O_k))_j \qquad (5)$$

Grid block, which makes the value of $u(O_k)$ maximum, can be classified as grade j.

## Case Study Example of CASLRP Applied to Mine Land in Jiangsu Province

### General Situation of Reclamation Area

**Geographic Location.** The reclaimed area is 30 km northeast of Xuzhou City shown as Figure 1. The Tuntou River, which is the main river in this area, is located south of the reclaimed area, and the Jiawang district of Xuzhou City is at the north. The subsided land resulted mainly from mining of the Hanqiao and Xiaqiao coal mines, Which are state-run. In this area, many lands tend not to subside, and only few areas will continue to subside. Parts of subsided lands have been reclaimed by local residents without planning.

**Social and Environmental Problems.** The population is about 110,000 in this area, and with increasing coal production, cultivated land is decreasing which results in increased unemployment. The ammount of waterflooded land is vaste because of subsidence. The percentage of afforestation is lower than that of the adjacent area.

According to practical conditions and the demands of the local residents, the prescribed future planned land uses in this area are only for agriculture and fishery.

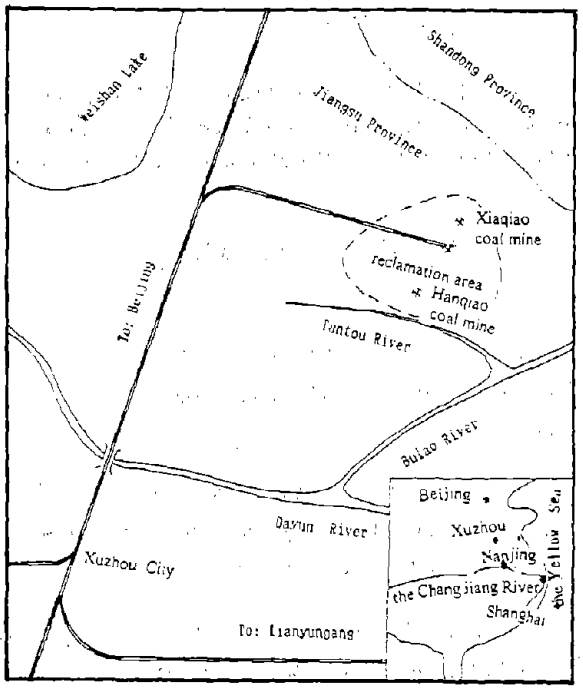


Figure 1. Map of geographic location of the subsided land.

### Evaluation Factors and Standard for Grading

Before evaluation, climate condition, soil texture and soil structure do not vary much, these factors are not chosen for evaluation; in contrast, present state of land use (F1), flooded nature (F2), vertical drop in elevation due to subsidence (F3), condition of irrigation and drainage (F4), and location of the unit to be evaluated (F5) are important, therefore were selected as part of the analyses.

The standards used for grading are listed in Tables 1 and 2.

Table 1. Standard for grading agriculture land

| Class | Evaluation factor | F1 | F2 | F3 | F4 | F5 |
|---|---|---|---|---|---|---|
| | Weighting | 0.2 | 0.25 | 0.20 | 0.25 | 0.10 |
| 1 | Mark | 20 | 25 | 20 | 25 | 10 |
| | Condition | rice-field, vegetabe plot , irrigated land | No | <0.5 | Good | Good |
| 2 | Mark | 10 | 16 | 13 | 20 | 6 |
| | Condition | nonirrigated land | T | <1.5 | Or | Or |
| 3 | Mark | 5 | 8 | 6 | 10 | 3 |
| | Condition | wasteland, beach | S | <3.0 | Poor | Poor |
| 4 | Mark | 0 | 0 | 0 | 0 | 0 |
| | Condition | wastehill, site of fly ash kiln | P | >3.0 | W | W |

T=temporary, Or=ordinary, S=seasonal, P=perennial, W=worst

## Flowchart of the Program

As was described in the evalua-tion model, a program was developed to determine the total suitability grade for each grid block in the study area. The flow chart for the program is shown in Figure 2.

**Reading of Data**. The data can be obtained by means of a grided data structure for the map of the present land utilization. Every grid is not only the point of data-collected, but also the point of data-processed.

**Coding of Data.** Data of every grid block is stored in a three-dimensional array named F(I,J,K), I and J represent the abscissa and ordinate from the corner west-south of the grid. K represents the order of the data for the grid.

## Results

**Grading Map of Land Use and Evalua-tion Results.** Figure 3 shows the grading map of subsided lands after evaluated. Table 3 shows the results of the evaluation.

**Advantages and Disadvantages.** The main advantage of the method presented above is that it is easy to change the factors and to adjust their weights. Secondly, by means of the technique of **CASLRP**, the

Table 2. Standard for grading fishery

| Class | State of flooding | Ground subsidence | source of water |
|---|---|---|---|
| 1 | Perennial | >3.0 m | Good |
| 2 | Seasonal | >1.5 m | Ordina |
| 3 | Temporary | >0.5 m | BE |
| 4 | No | <0.5 m | Poor |

BE=barely enough

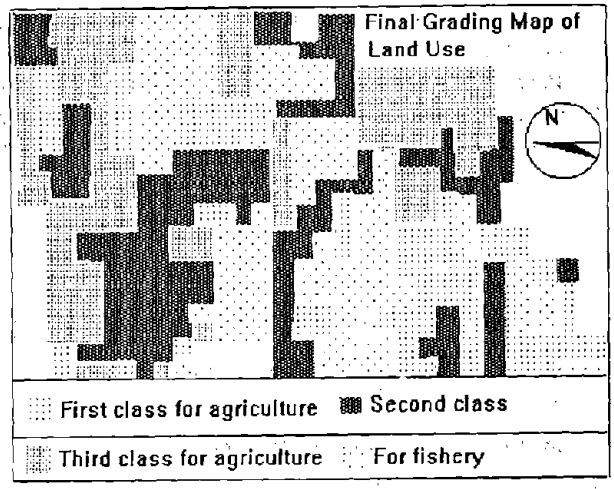


Figure 3. Final grading map of land use

results of evaluation can be output directly to figures and tables.

The main disadvantage is that the weight of every factor is subjective. This can be overcome by a combination of experts' grading and theoretical deduction based on statistical data.

Table 3. Area of land use in every class (unit: ha.)

| Item | Area |
|---|---|
| Total area | 1575 |
| First-class for agriculture | 292.6 |
| Second-class for agriculture | 449.7 |
| Third-class for agriculture | 335.1 |
| For fishery | 260.8 |
| Others | 236.8 |

## Conclusions

The model of subsided land suitability evaluation for reclamation as a part of the **CASLRP** technique was discussed, and a practical example in Xuzhou coal mine area was presented. It was demonstrated that the **CASLRP** technique is an effective tool for mine land reclamation, and that the results of evaluation are reliable and practical.

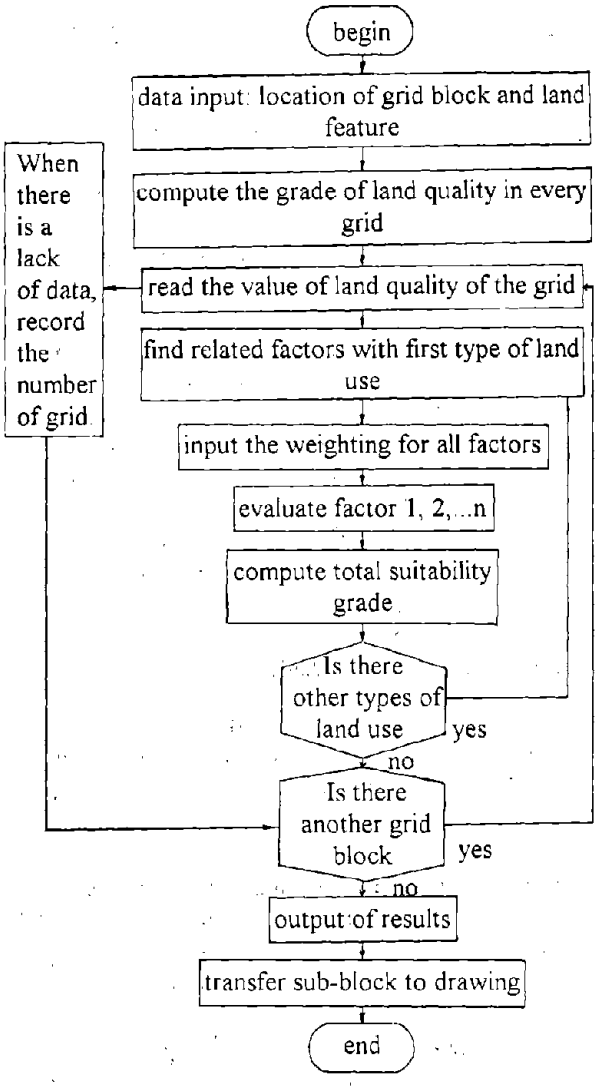


Figure 2. Flow chart to determine the total suitability grade

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## A SOLUTION FOR PERMANENT CLOSURE OF UNDERGROUND MINES LOCATED IN URBAN AREAS

Uroš Bajželj and Jakob Likar, (Department of Mining Engineering, University of Ljubljana, Slovenia).

More complex geological and geotechnical conditions with long period of mining activities required a special solution for closure of mines in urban areas. A typical example is the Idrija Mercury Mine (Slovenia), which in 1990 celebrated its 500th anniversary of underground operations. The low mercury content of the known mercury ore reserves within the Idrija ore deposit does not allow profitable exploitation. The stoppage of mercury production and the closure of the mine in this connection requrie the execution of very exacting operations for the technical conditions, improvement, and consolidation. The further subsiding of the roof layer overlaying the worked-out ore deposit areas is to be prevented. Consolidation of the old mine workings will have a great influence on the existence of the town of Idrija, which developed above the mine during the period of 500 years of mining. On the basis of detailed observations of movements, deformations, and the analysis of its results, it was ascertained that roof layers were subsiding into the mined-out ore deposit areas. The complex geology and unfavorable surface morphology in the region of the ore deposit or urban area caused the creeping of rock masses , which are endangering the town of Idrija. Analyzing the stress-strain relations, measures were examined closely, taking into account the rock mass stabilization above the Idrija ore deposit or town. The following measures are to be taken into account: consolidation of old fillings and particular rock pillars of poor strength between the fillings, moving of certain ore bodies using the sublevel caving method with cemented backfill, dewatering of the upper part of roof layers of the Idrija ore deposit.

## USING GIS TO EVALUATE MINE SUBSIDENCE POTENTIAL

Richard Balogh/William Morrison (OSMRE, Pittsburgh, PA)
Arthur Hoffmann/Mitchell Weber (Gannett Fleming, Inc. Pittsburgh, PA)
Barry Evans (Geo Decisions, Inc. State College, PA)

At the request of the U.S. Department of the Interior, Office of Surface Mining Reclamation and Enforcement (OSMRE), Gannett Fleming, Inc. has performed a study to identify areas of the Borough of Munhall, Pennsylvania which have the potential to be affected by deep mine subsidence. Records of all available mine subsidence reports, mine subsidence insurance claims and associated investigations, available mine mapping, published geologic information, and subsurface data including 123 exploratory boring logs were reviewed and evaluated. Base maps were developed using Geographic Information System (GIS) capabilities and ARC/INFO GIS software. Aerial photographs obtained in February 1993 were digitized and used to develop "photograph like" base mapping referred to as digital orthophoto base mapping. Features such as coal crop lines, mine boundaries, coal structure contours, streets, political boundaries, significant structures, ground contours and subsidence events were shown on the base mapping. Associated thematic maps were developed which included isopleths of coal, overburden and roof rock thickness as well as sandstone to shale overburden ratio. The GIS mapping made it possible to spatially relate a wide variety of different types and presentations of data. Three parameters were found to be most closely related to mine subsidence potential. For areas of the Borough underlain by the previously mined Pittsburgh coal seam, the thickness of overburden, roof rock type and mining method were found to be the most significant factors. Mining method was by far the most significant. It was found that areas of the mine that were totally extracted showed very low susceptibility to subsidence. Most of the documented cases of mine subsidence in the past 20 years were found to have occurred in areas of partial extraction. Interestingly, the few subsidence events that did occur in totally extracted areas occurred near main headings where chain pillars were typically left in place. The conclusion of the study is that where coal was totally extracted, subsidence has probably already occurred. In partially extracted areas, a potential for mine subsidence exists today.

## THE AUSTRALIAN CENTRE FOR MINESITE REHABILITATION RESEARCH

L.C. Bell. (Director - ACMRR).

The Australian Centre for Minesite Rehabilitation Research (ACMRR) is a joint venture between the Australian mining industry through the Australian Mineral Industries Research Association Ltd (AMIRA) and three of the organisations working most actively in this area in Australia: **CSIRO** Minesite Rehabilitation Research Program; **University of Queensland** Centre for Mined Land Rehabilitation; and **Curtin University** Mulga Research Centre.

The ACMRR was established in July 1993 to provide a national framework to conduct **Strategic Research** into minesite rehabilitation. It is an industry led and funded initiative.

The Goals of the Centre include: to conduct strategic research into minesite rehabilitation to provide sustainable environmental solutions which are acceptable to industry, government and the community; to be recognised as a centre of excellence undertaking commissioned research on minesite rehabilitation in an independent and thorough manner; to provide scientific and technological foundations to facilitate industry and government in setting acceptable standards; to act as a networking and communications focus; and to enhance education and training in minesite rehabilitation.

Strategic Research Programs in:

**Water Systems** - downstream surface and groundwater quality;

**Land** - the long-term behaviour and stability of constructed landforms;

**Ecosystems** - the long-term sustainability of constructed landforms;

**Waste** - the long-term treatment and disposal of waste products;

will allow the ACMRR to achieve these goals through specific research projects in these areas, developed with industry sponsors. This paper will discuss our progress to date, research projects underway, and plans for the future.

## TEN-PERCENT SET-ASIDE ACID MINE DRAINAGE ABATEMENT PROGRAM

J. Michael Bielo. (Bureau of Abandoned Mine Reclamation, Harrisburg, PA)

The Ten-Percent Set-Aside AMD Program was authorized as part of a recent amendment to the federal Surface Mining Control and Reclamation Act of 1977. The amendment allows the Department to spend up to ten percent of the abandoned mine lands annual allocation to address environmental degradation caused by acid mine drainage. There are some restrictions in the use of the funds. For example, currently accepted or proven technologies in AMD abatement are to be used, and a significant improvement to the affected hydrologic unit should result from the reclamation work. On the other hand, the amendment does not provide for performing research or granting money to other parties for their use. The limited funds available under the Ten-Percent Set-Aside Program for AMD abatement make it imperative that the funds be used judiciously to make the most positive impact possible. Consequently, the site evaluation committee will be looking for project sites where the proposed AMD abatement will restore the greatest length of polluted stream for the least amount of money. They are not considering sites where large-scale active treatment and associated long-term maintenance and operations costs to the Commonwealth will be required. To insure a thorough evaluation of potential reclamation sites using the most current information available, a one-page AMD Project Form has been developed to solicit pertinent information from interested parties. Much of the material and information previously provided by various parties is general in nature with relatively old technical support data. Thus, we are asking each petitioner to reconsider the site or watershed they have suggested, and to complete a form or forms for those areas, being as specific and detailed as possible.

## BEHAVIORAL PROPERTIES OF GRANULAR HALITE BACKFILL

Brett C. Davidson and Maurice B. Dusseault. (University of Waterloo, Waterloo, ON, Canada)

Granular halite is presently being used as a backfill material in potash mines in Canada to optimize extraction ratios, reduce environmental concerns associated with wastes, enhance mine safety and reduce surface subsidence. Laboratory investigations on granular halite have shown that compaction processes are primarily dependent on grain size, porosity, moisture content, stress and temperature. Compaction creep rates ($\varepsilon$) at stresses $\leq\approx$4 MPa is proportional to $1/d^3$ and at stresses $\approx$8 MPa this relationship becomes $1/d^2$, where **d** is the grain size. Distribution of the various grain sizes in a granular material can affect a specimens initial porosity. Specimens having a wide range of grain sizes will tend to have lower initial porosities, however, the total amount of volumetric strain which can be achieved will be less. The time dependent porosity reduction, or compaction creep of granular halite is controlled by several mechanisms which dominate creep rates in intact salt specimens. This is based on the premise that backfill is comprised of many small pieces of intact salt and depending on the conditions of placement, it is subjected to the same creep processes. The two mechanisms which dominate compaction creep in a mine situation ($\sigma$ <2 MPa, T <100°C) are pressure solution and grain boundary sliding. Pressure solution occurs when dissolved ions diffuse through a liquid film from areas of high stresses on grain contacts to areas of low stresses in pore spaces. For pressure solution to occur there must be the presence of moisture (brine) in the backfill to provide a liquid film layer. Grain boundary sliding is a process which allows the individual grains of halite to slide past one another into the pore spaces of the backfill. This process usually coincides with pressure solution and accommodates for changes in grain shape.

## SOLUTION CAVERN DISPOSAL OF SOLVAY PROCESS WASTES

Brett C. Davidson and Maurice B. Dusseault. (University of Waterloo, Waterloo, ON, Canada)

The Solvay Process generates $Na_2CO_3$ and $CaCl_2$ from saturated NaCl brine and finely ground limestone ($CaCO_3$); the sodium carbonate comprises a major constituent of glass and other products. At the General Chemical Canada Ltd. operation in Amherstburg, Ontario, the NaCl is obtained by solution mining from the Salina Formation at a depth of 450-500 m, and the calcium carbonate is obtained from shallow limestone quarries. The waste from the process is a non-toxic fine-grained (silt-sized) mineral waste in a saturated brine. Since the establishment of the operation 85 years ago, mineral wastes have been permanently stored in surface tailings ponds, expanded as required.

Continued development of waste impoundments as the primary and permanent waste management system is becoming proportionately more costly because of higher land costs, stricter environmental controls, transportation distance, and possible decommissioning and rehabilitation costs. Old waste pits cannot be rehabilitated in a conventional manner because the shear strength of the wastes remain low, saline liquids have to be isolated from the groundwater resources, and in dry weather wastes can even become an airborne, wind-blown local nuisance.

A more economical solution is to place the wastes into the solution caverns in the form of a thick slurry. Over time, the caverns close and the wastes become permanently entombed in the salt stratum, isolated from the hydrosphere for geological time periods. No surface ponds are created, and old ponds can be emptied as required. No chance of surface water contamination exists, and additional advantages are associated with the filling of the cavern with a solid material that will reduce the chances of cavern collapse into superior strata. The geomechanical information required for design include the creep behaviour of the intact salt strata, the porosity, permeability, and compressibility of the Solvay wastes.

## SCIENTIFIC ASSESSMENT OF THE EFFECTS OF MINING ON AQUATIC ECOSYSTEMS IN CANADA: A REGULATORY REVIEW

Sheila F. Forsyth and William Blakeman. (Environment Canada, Ottawa, Ontario, Canada).

Metal mine effluents are regulated federally in Canada by the *Metal Mining Liquid Effluent Regulations* (MMLER, 1977) under the *Fisheries Act*. The federal government and other stakeholders, including representatives from the mining industry and environmental groups, agreed that the effectiveness of the MMLER to protect aquatic ecosystems should be examined prior to substantive revisions. A multistakeholder process, the Assessment of the Aquatic Effects of Mining in Canada (AQUAMIN), to do a science-based assessment of the documented biological, physical and chemical effects of metal mines on aquatic ecosystems in Canada, has been initiated by Environment Canada. The scope of mines includes operating, decommissioned and abandoned base metal, precious metal and uranium mines. Sources of information for AQUAMIN include studies conducted at specific mine sites, scientific literature and information from other sectors (e.g., coal, industrial minerals) that provides insight into metal mining effects. Effects data will be correlated with regulatory compliance records, the technology used and the monitoring procedures at the site for case studies. This approach differs from some traditional water protection activities which use findings from laboratory or controlled field experiments to develop regulations to protect natural systems. The AQUAMIN report will provide direction and information on: ways to amend the MMLER should any amendments be required, the design of an environmental effects monitoring program for metal mines in Canada and the identification of information gaps.

## ABANDONED MINED LAND INVENTORY US BUREAU OF LAND MANAGEMENT

Jennifer Fox (1), Teresa E. McParland (2), and Robert H. Robinson (3). (Bureau of Land Management (1) Washington DC, USA, (2) Salt Lake City, UT, USA, (3) Denver, CO, USA).

For resource management purposes, the US Bureau of Land Management (BLM) requires a national inventory of abandoned mines on public lands that are within the stewardship of the agency. The concerns involve environmental damage, physical hazards, liability torts, land use planning, and future remediation. In 1992, BLM commissioned a task force to develop an inventory methodology for abandoned mined lands (AML). The charge to the task force is to develop a methodology that uses existing BLM resources, has consistent nationwide procedures, and minimizes duplication of effort. While details of the methodology continue to evolve, the main features include 1) collection of existing data, 2) pilot inventory projects, 3) development of a BLM data base, 4) coordination with the BLM programs for hazardous materials and stormwater, 5) BLM state office implementation of AML inventories, and finally, 6) collation of the state inventories into the national BLM inventory. BLM expects to complete data collection and pilot studies in 1994. Each BLM state office will prepare implementation plans in 1995 and start field work the same year. The target for completing the national inventory is 1997.

## EXPLORATION OF THE DISCIPLINE SYSTEM OF LAND RECLAMATION SCIENCE

Zhenqi Hu (China University of Mining & Technology, Xuzhou, Jiangsu, P.R. China)

Land reclamation is generally taken as an engineering. This paper studies the land reclamation science from the viewpoint of the discipline development first. The authors think that land reclamation is an applied discipline with multidisciplinary nature, which is in close relationship with Geology, Ecology, Agronomy, Land Science, Environmental Science and other disciplines. It is the science dealing with the formation mechanism and characteristics of destroyed lands due to human disturbance, as well as the policy, theory, method and technique taking place in processes of reclaiming the destroyed lands to a desire land-use target. The discipline has its specific research objective ---- the destroyed lands due to human activities, and its kernel issues are reutilization of the destroyed lands and rehabilitation of the destroyed ecosystem. The main study tasks of this discipline are: (1) the law and characteristics of the destroyed lands, (2) the techniques of reclaiming the destroyed lands and rehabilitating the destroyed ecosystem, (3) land reclamation management (which includes the reclamation policy, law and regulation and other management issues). Being an applied discipline, land reclamation science consists of fundamental theory, methodology and applied technology. The applied technology of reclamation includes the techniques of evaluation and prediction of destroyed lands, the planning techniques of land reclamation, the techniques of civil engineering for the land reclamation, the reclamation techniques of biological engineering, administration techniques of land reclamation, etc. In short, this paper presents a complete discipline system of land reclamation science.

## MINE CLOSURE IN THE PROVINCE OF ONTARIO - CANADA

J.A. (Sandy) McIntosh. (Ministry of Northern Development and Mines, Sudbury, Ontario, Canada)

A display featuring the activities of the Mine Site Reclamation Section of the Province of Ontario will be presented. Included in the display will be information on the mine closure aspects of the revised (June, 1991) Mining Act together with highlights of the abandoned mines program in Ontario.

## THE RULES OF WORK AND DAMAGE PREVENTION IN LINEAR STRUCTURES ON MINE SUBSIDENCE AREAS

Rudolf Mokrosz (1), Szczepan Wyra (2). ((1) District Design Office, Zabrze, (2) Silesian Technical University, Gliwice, Poland).

The paper deals with the issues due to influence of mining induced ground deformations on the technical infrastructure, such as, waterlines and gaslines, sewage collectors, and hot water pipelines. The interaction between the above mentioned structures and the deforming surface in all stages of deformation is considered, but restricted to sag type of subsidence. The results were derived, using the analytical model based on the Budryk and Knothe theory of continuous surface deformation. They are presented as rules and recommendations for designers and operating maintenance crews. This paper covers application of preventive means in newly designed structures for their safe usage, as well as an outline of methods for mitigation of mining induced loading on existing structures. The review of the current state of knowledge in the subject enables the determination of the problems that require solutions and indicates recommendations for future research.

## LAND-SAVING PROBLEMS IN COAL STRIP MINING OF KUZBASS STRATA

Alexander G. Netsvetaev (1) and Vadim V. Mikhalchenko (2). ((1) JV Carbo-KH, Kemerovo, Russia, (2) Technical University of Kuzbass, Kemerovo, Russia)

The Kuznetsk coal basis contains approximately 63 billion metric tons of coal reserves. Open-cast mining will be the main mining method used for its further development. However, a deterent to this type of mining is the high population density in this area. To solve the problem, it is suggested to work steeply-inclined seams locating the pit of the first stage where the overburden from other sections is placed later on. The effect of land saving is much greater if the pit of the first stage is built using the technology of stage-deepening of mine operations. In this case, the mine operation front is located across the strike of the deposit and moves along the strike with new horizons being cut in certain time and space intervals. As a result, the pit bottom of the first stage is formed at an angle providing the possibility of placing the overburden rock in the goaf starting from the first years of the deposit exploitation. Upon achieving the designed pit depth, the deposit is exploited with constant parameters of the working zone and stable transportation distance. The investigations established that the use of stage-deepening technology makes it possible to cut the devastated area by 2.5 - 3.5 times, to reduce transportation distance by 3.5 - 4.5 times which results in increasing economical efficiency of the technology and, as a result, increases the limit overburden coefficient by 1.5 - 2 times.

## HIGH VOLUME – HIGH VALUE USAGE OF DRY GAS DESULFURIZATION (FGD) BY-PRODUCTS IN UNDERGROUND MINES

Thomas L. Robl (1) and Mary B. Ashbaugh (2). ((1) University of Kentucky, Center for Applied Energy Research, Lexington, KY, USA, (2) U. S. Department of Energy, Morgantown Energy Technology Center, Morgantown, WV, USA).

Highwall mining is a promising development in the mining industry that extends the recovery of coal behind existing and future highwalls. This recent technological breakthrough has not only broadened the recovery of coal resources but also has opened the door for investigating the feasibility to control acid mine drainage and mine subsidence by filling abandoned underground mine voids with alkaline, advanced coal combustion by-products such as fluidized bed combustion material. Highly successful systems of highwall mining consist of a continuous miner connected to a series of long belt conveyer cars that is remotely controlled and camera operated. The FGD by-products can be hydraulically or pneumatically placed in the highwall mines. Their cementitious characteristics exhibit relatively high compressive strengths which enable the material to be used as a roof support structure when placed into the highwall mine tunnels, freeing up the existing coal for recovery that was previously left for support. As a result of advanced dry flue gas desulfurization technologies to reduce pollutant emissions more ash by-products will be produced. These by-products can be utilized in underground mines. This project investigates the use of FGD material for subsidence control and acid mine drainage as a promising technology that is facilitated by the highwall mining technique. Major project participants include the University of Kentucky Center for Applied Research and Addington Resources. The Morgantown Technology Center is sponsoring the project.

## SYSTEM APPROACHES TO EFFECTIVE MINE WASTE HANDLING BASED ON SLURRY PUMPING

Anders Sellgren and Åsa Sundqvist. (Div. of Water Res. Eng., Luleå Univ. of Technology, Luleå, Sweden)

Slurry transport is a feasible and effective method for handling and disposal of mine waste. Waste rock particles with sizes of 100-200 mm can be transported cost-effectively with slurry pumps. It has been found that the investment needed for pumping was less than half of that of a conveyor belt system in an example of transportation of 2.4 Mtonnes of waste rock per year over a horizontal distance of 3.5 km. About 1000 tonnes per hour of a mixture of a waste rock and a fine-grained tailings product can be pumped horizontally 1 km at an approximative total power consumption of 300 kW in a pipe with a diameter of 0.3 m. In an integrated system where waste rock i transported together with wet-handled fine-grained tailings, the extra cost of transporting waste rock will only be a fraction of the ordinary wet handling cost of tailings. Furthermore, the handling of waste rock and fine-grained tailings is limited to one disposal area. Slurry pumping also possesses a great potential at smaller old mine waste deposits, where a mobile slurry pumping "module" can be used together with a flexible hose arrangement. Underground disposal may be an interesting environmental solution in certain areas. In the neighbourhood of abandoned mines or of on-going mining activities there is often underground space available. However, hydrological and geochemical factors need to be evaluated locally to ensure an environmentally feasible storage.

## ANALYSIS OF KINEMATICS AND LOADS FOR BRIDGE STRUCTURES IN MINING AREAS

Jerzy Śliwka. (Department of Bridge Construction Silesian Technical University, Gliwice, Poland).

Ground deformations in the vicinity of bridge structures are becoming more and more extensive due to mining systems favoring the robbing of thick seam workings without either pneumatic or hydraulic filling. Bridge structures built in the past are not adapted to ground deformations. Also, methods eliminating unfavorable incidents in construction processes were not known. The paper presents problems concerning the behavior of medium span bridges in mining areas. The problems will be discussed on the example of a road bridge running over a railway line. The bridge in question has been subjected to mining effects since 1981. Since that time, there have been three attempts to adapt it to accommodate mining effects. The first attempt consisted of changing the bearing system and formation of expansion joints. The second attempt consisted of formation of a new expansion joint on one of the piers. The third attempt consisted of shifting the bridge spans in such a way as to ensure the formation of expansion joints that could accommodate the forecasted effects of mining. During the observation of viaduct performance, certain phenomena were perceived that were to be explained through appropriately planned research work on site. The research had to be canceled because of financial reasons (i.e., no sponsors).

## SUBSIDENCE AND RECLAMATION IN COAL MINE AREAS OF CHINA

Hong Yu, Fang Guo. (Department of Geology, Xiangtan Mining College, Xiang tan, Hunan, PRC).

Coal mining has caused widespread ground surface subsidence and deterioration of soil structure in China. Approximately 8,500,000 hectares of land have been affected by subsidence. Two major causes of subsidence are the mining technique used and the pumping of ground water to facilitate coal extraction. About 60% of all coal mines in China have been included in a reclamation study. This study includes filling underground mines with gangue, coal ash, and garbage and use of subsided areas as fish ponds and wastewater treatment ponds.

# ECOENVIROMENT CHARACTERISTICS OF SUBSIDENCE LAKES IN COAL MINE AREAS AND WAYS TO HARNESS THEM

Bian Zhengfu, Zhang Guoliang, and Lin Jiacong. (China University of Mining and Technology, Xuzhou, the People's Republic of China)

With the increase in coal output, the environment in coal mine areas, especially land destruction, has become increasingly worse. A main form of land destruction is subsidence lakes, which have disreputed people's lives and coal production and which change the ecological system of mine areas into water-land type or aquatic system. Two conditions forming subsidence lakes are surface subsidence and high phreatic level. The ratio of waterlogging in China's eastern coal mine areas is higher than that of other areas because of high phreatic level, and eastern mine areas are agriculture-based at the same time. It is a fact that the structure and function of an ecological system after mining is different from that before mining when we harness subsided land and subsidence lakes. It is not a good way to resume the original structure and function of the ecological system. Therefore, before harnessing subsidence lakes, it is necessary that their ecoenvironment characteristics be investigated thoroughly, then new structure and function of ecology affected by coal mining and ways to harness subsidence lakes are posed. These ways include: subsidence lakes used as fly ash, coal waste and other wastes deposit, land reclamation by raising the shallow land with digging the deep, recreating new aquatic ecosystem, etc.

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