

Bulk infilling of underground mines

The backfilling of underground mines is a well known, if not commonly practised, method for disposing of mine waste while providing structural support. In recent years, increasing environmental pressure against surface methods of waste disposal and concern about subsidence have resulted in an increase in the popularity of infilling of mine adits. For example, backfilling of abandoned mines in the U.K. is seen as a means of reducing the possibilities of subsidence, thereby opening up 'under-mined' areas for potential development.

This apparently ideal solution for the disposal of mine wastes is now being questioned in the

U.S. on environmental grounds. The concern is that the wastes might be geochemically unstable and lead to contamination of the soil and groundwater. This is particularly the case when the infill being used has a high metal content.

Two articles on this topic follow. The first article discusses research that was carried out by the U.S. Bureau of Mines in response to the Underground Injection Control Regulatory Programme. This regulatory programme, which encompasses the bulk infilling of underground mines, was set up by the Environmental Protection Agency under the auspices of the Safe Drinking Water Act (1974).

The second article looks at the use of colliery spoil as a bulk material for infilling abandoned limestone mines in the U.K. There are already tight regulatory controls on these activities and proposed legislation could impose further restrictions. At present the bulk infilling of U.K. mines is covered by the Mines and Quarries Act, the Water Resources Act and the Environmental Protection Act. Although a proposed European Union Council Directive on waste landfill is not designed to encompass the bulk infilling of mines, it could do so by default. This may well result in the imposition of more stringent controls on minefill operations.

Mine waste sandfill

The Safe Drinking Water Act was enacted in 1974 by the U.S. Congress to develop regulations to protect the nation's underground sources of drinking water. This led to the Underground Injection Control (UIC) regulatory programme which was promulgated in 1981 by the U.S. Environmental Protection Agency (EPA).

Within the UIC, a Class V well includes "sand backfill and other backfill wells used to inject a mixture of water and sand, mill tailings or other solids into mined out portions of subsurface mines whether what is injected is a radioactive waste or not". A well is broadly defined as either a dug hole or a bored, drilled, or driven shaft whose depth is greater than its largest surface dimension.

Because placement of mine waste backfill in underground openings is considered to be underground injection under UIC regulations, the impact on mine backfilling practices could be extensive. Currently, over 20 metal mines in the U.S. use backfill materials ranging from quarried rock to cemented mill tailings. The major purpose of the backfill is to support the ground during mining; however, backfilling is increasingly considered as a means to minimise surface disposal of mine waste.

Water affected by backfilling during active mining is usually pumped to the surface where its disposal is regulated as surface discharge. Backfilled areas may become flooded after a mine is closed, possibly resulting in contamination of ground water. Therefore, in contrast to regulations addressing surface disposal, regulations concerning the contamination of ground water after mine closure will be a primary focus under the UIC programme.

Returning mine waste to the original excavation is an apparently attractive option because the original minerals are stable in that environment. However, chemical transformations that occur during the milling process, as well as after placement underground, may produce less stable minerals. Also, the hydraulic conductivity of uncemented sandfill will probably be several orders of magnitude greater than the surrounding rock because of the mechanical destruction of the original rock during milling. Uncemented sandfill with higher hydraulic conductivity than the surrounding rock may provide preferential pathways for groundwater flow, which may enhance metal leaching.

The mechanisms controlling dissolution of minerals in sandfill and the mobility of heavy metals change when sandfill becomes flooded after mine closure. The nature of these changes must be considered in order to estimate the

impacts of mine waste backfill on groundwater. The U.S. Bureau of Mines (USBM) has carried out research aimed at studying the possible impacts of mine waste sandfill on the quality of groundwater in the rock surrounding a stope.

METHODOLOGY

This research is part of a larger effort in which the impacts of mine waste backfill on water quality are being investigated at four underground sulphide mines. Backfill materials used in the four mines included cemented total gradation tailings; uncemented, classified sand; and uncemented waste.

Water samples were gathered monthly from both above and directly below a sandfilled stope in an underground mine in northern Idaho. Cation and anion concentrations in samples were compared to determine if differences could be detected and attributed to the backfill material. Mineral assays of sandfill from the test stope, published mineralogic descriptions, and information from mine records were used to determine the minerals that might have been sources of the major ions in solution. Samples of the sandfill were also leached with acid to determine the degree to which metals were available for dissolution.

Two geochemical computer models, WATEQ4F and BALANCE, were used to investigate the geochemical reactions involving the major ions. WATEQ4F is a thermodynamically based model that computes element speciation and mineral saturation in water samples. Saturation indices provide a basis for evaluating the potential for dissolution and precipitation reactions between minerals in backfill and infiltrating water. Knowledge of reaction kinetics and mineral stability ranges are necessary for interpreting the validity of the thermodynamic approach used by WATEQ4F.

BALANCE is used in conjunction with WATEQ4F to define and quantify the mass

Stabilising and backfilling undermined areas releases land for development and reduces surface waste disposal

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transfer between infiltrating water and backfill materials. The primary use of WATEQ4F and BALANCE is to investigate whether dissolution of carbonate minerals is buffering acid produced from sulphide oxidation. Again, kinetic controls on mineral dissolution are not addressed in WATEQ4F or BALANCE.

Laboratory research on mechanisms controlling mineral dissolution and investigations of the subaqueous disposal of tailings provide a basis on which to predict the long-term hydrochemical impacts of sandfill after mine flooding. Important factors to be considered include the mechanisms that control mineral solubilities in sandfill, the effect of oxidation of sandfill prior to flooding, and the potential for migration of heavy metals from flooded sandfill.

The test site, located in the Coeur d'Alene Mining District of northern Idaho, is a moderately deep underground lead-zinc mine. The geologic formations encountered in the mine consist of Precambrian metasediments, including quartzites, argillites, siltites, and minor amounts of carbonates. Galena and sphalerite are the principal ore minerals, while quartz, siderite and pyrite are the principal gangue minerals. Quartz, sericite or illite, and iron-rich carbonates are the principal minerals found in the wall rock. Ore was mined by conventional cut-and-fill methods, and backfill consisted of tailings classified using cyclones.

The unfractured rock surrounding the mine has very low hydraulic conductivity; consequently, water infiltrates the bedrock along the fractures and faults and enters the mine where these are intersected by backfilled or open mine workings. The hydraulic conductivity of the fractured bedrock was estimated at 1×10^{-7} m/s. For comparison, the hydraulic conductivity of the very fine grained sandfill used in the mine was calculated from its grain-size distribution to be 1×10^{-5} m/s using the Hazen equation.

Acid mine drainage, containing high concentrations of dissolved solids, is produced within parts of the mine through oxidation of pyrite and subsequent dissolution of other minerals. Table 1 shows concentrations of dissolved constituents detected in water as it enters the mine and in water from an acid-producing area of the mine. Water from the acid-producing areas is characterised by high concentrations of SO_4^{2-} , Fe, Ca, Mg, Mn, Al and heavy metals, as well as by low pH. The primary controls on acid production are the mineralogy of the rock and the availability of O_2 and water. Water quality within the mine is highly variable because acid production from pyrite oxidation is significant only in isolated areas.

The stope selected for this study is the 17-level stope on the Barr vein, which is also known as the Barr stope. The stope consists of three intervals separated by horizontal pillars that were mined between 1973 and 1981, overlain by other sandfilled stopes. Rocks in the vicinity consist of sericitic or vitreous quartzites interbedded with infrequent argillite layers. Approximately 21,700 t of ore were extracted from a discontinuous, fault-controlled, sulphide orebody. Average ore grade was 5.4% Pb, 77.5 g/t Ag and 0.7% Zn. The Barr stope is not located in the most serious acid-producing part of the mine. One monitoring site (B1) was

Table 1.
Concentrations of dissolved elements in test mine (mg/l).

Element	Recharge	Acid mine drainage
Al	<0.01	412.00
Ca	9.70	574.00
Cl	0.08	0.28
Fe	<0.01	17,900.00
Mg	0.43	2,230.00
Mn	<0.01	3,030.00
Na	0.24	0.01
S	0.25	28,300.00
Si	3.59	63.30
Zn	0.02	10,000.00
HCO ₃	0.53	0.01
pH	7.40	2.20
EC (S)	0.07	55.20

EC: Electrical conductance

located near the top of the Barr stope and two others (B2 and B3) near the bottom. The upper and lower levels are approximately 425 m and 490 m, respectively, below the main haulage level in an inactive portion of the mine.

Samples of backfill were collected from the stope and from the underground secondary cyclone discharge circuit at the mine. All samples were sent to the USBM's Reno Research Center for chemical assay and mineral identification by X-ray diffraction. The chemical assay method used was lithium borate fusion followed by acid digestion using hydrochloric acid and inductively coupled plasma analysis. A maximum leachability assay of samples of each backfill material was also conducted, using a mixture of HCl (22 cm³), HNO₃ (4 cm³), and H₂O (20 cm³) at the USBM's Spokane Research Center, to determine the maximum leachable quantities of metals. The maximum leachability assay was not intended to replicate natural conditions, but to show the presence of metals that could be released under highly acidic conditions.

Water samples were collected both before and after the water came into contact with the sandfill. Sampling was initiated in November 1990 and continued on a monthly basis through May 1991. The sampling programme was constrained by the difficulty of access to the backfilled stopes, by the presence of water seeps, and by the cost of obtaining and analysing additional samples.

ANALYSIS

Chemical assays show that approximately one-third of both the backfill from the Barr stope (old sandfill) and the backfill from the secondary cyclone discharge circuit (new sandfill) consists of silicon. Significant quantities of Al, Ca, Mg, K, Mn, and Fe, as well as measurable quantities of As, Ba, Na, Cd, Co, Cr, Cu, Ni, Pb, and Zn were also detected in samples from both backfills. Analysis by X-ray diffraction indicated both sandfill samples consist of quartz with traces of muscovite, pyrite and siderite.

The most notable differences between assays of the old and new sandfill samples were higher concentrations of As, Cu, and Zn, and lower

concentrations of Mn in the new sandfill. These differences can be explained either by differences in metallurgical methods, differences in ore mineralogy, or by leaching of As, Cu and Zn from the old sandfill. Calcium, magnesium, and sulphur were the most prominent major ions leached from the backfill samples using a mixture of HCl and HNO₃ (table 2). Significant quantities of Pb, Zn, Mn, Fe and As and lesser quantities of Cd, Co, Cr, Ni and Al were also leached from the backfill samples.

The most notable differences between leachates from the new and old sandfill samples were higher concentrations of As, S and Zn, and lower concentrations of Fe and Mn leached from the new sandfill. The higher SO_4^{2-} concentrations in leachate from the new sandfill may be the result of previous removal of SO_4^{2-} from the old sandfill as a result of sulphide oxidation and subsequent leaching of SO_4^{2-} . The higher Fe and Mn concentrations in leachate from the old sandfill may have been derived from the more-soluble, oxidised secondary minerals in the old sandfill. Arsenic and zinc may have already been leached from the old sandfill, explaining why concentrations of these elements were lower. However, because concentrations of other metals, including Cd, Co, Cr and Cu, were not lower, or were only slightly lower, in the old sandfill, it is possible that differences in ore grade or metallurgical methods caused the differences in metal concentrations leached from the old sandfill.

WATER QUALITY

Concentrations of dissolved constituents detected in water from the sampling sites were significantly lower than concentrations detected in the worst acid-producing portions of the mine (table 1). However, during one or more visits, concentrations of many constituents in water at the sampling sites exceeded concentra-

Table 2.
Maximum leachability of backfill materials (mg/l).

Element	Old sandfill	New sandfill
Ag	0.02	0.03
Al	1.97	1.87
As	5.95	10.45
B	0.84	0.51
Ba	0.03	0.02
Ca	35.40	38.80
Cd	0.21	0.34
Co	0.13	0.17
Cr	0.40	0.25
Cu	0.56	0.69
Fe	311.50	186.25
K	0.90	0.75
Mg	12.37	6.45
Mn	29.45	9.52
Na	0.12	0.41
Ni	0.16	0.21
P	0.49	0.39
Pb	21.55	27.79
S	37.65	85.75
Si	2.22	1.68
Zn	14.91	35.45

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tions in the mine recharge water, as well as maximum concentration levels (MCL) allowed under the Safe Drinking Water Act. Of these constituents, only SO_4^{2-} consistently exceeded the MCL (table 3).

Electrical conductance was higher at sites B2 and B3 than at B1. Increased concentrations of Ca, Mg, SO_4^{2-} and HCO_3^- accounted for most of the increase. No relevant changes in pH or concentrations of heavy metals were detected. The only meaningful differences between water quality at sites B2 and B3 were lower levels of HCO_3^- and Mn and higher levels of Fe at B3.

Concentrations of major ions (Ca, Mg, HCO_3^- and SO_4^{2-}) in water samples from all collection sites showed the effects of oxidation of sulphides. The increases in Ca and Mg, in conjunction with near-neutral pH values, suggests acid production is being buffered by the dissolution of carbonates. Concentrations of heavy metals were low, possibly as a result of buffering. Slow dissolution reaction kinetics of primary minerals containing heavy metals, combined with a high rate of pore water flushing, could also have limited metal concentrations in discharge water.

The statistical significance of the differences between water quality at the upper and lower level sampling sites cannot be determined because of the small number of samples collected. However, water sampled on the upper level probably reflects the poorest quality recharge to the Barr stope, poorer than recharge from the surrounding bedrock, because this water comes from overlying sand-filled stopes. Also, water quality on the lower level reflects a more advanced degree of sulphide oxidation than does water at the upper level. This suggests that the quality of water at B2 and B3 is poorer than that from all individual sources of recharge to the stope.

Saturation indices calculated using WATEQ4F provide a basis for identifying the dissolution and precipitation reactions that may control the concentrations of elements in water samples (table 4). A saturation index greater than 0 indicates that a mineral will tend to precipitate, while a saturation index less than 0 indicates that a mineral will tend to dissolve. As suggested previously, the water at all sampling sites appears to be in equilibrium with carbonates.

A list of minerals considered to be sources and/or sinks of the major ions detected in water samples from B1 and B3 were entered into BALANCE. For input to BALANCE, a saturation index between -1 and +1 is assumed to equal 0, indicating equilibrium conditions. BALANCE was used here to test whether a proposed set of dissolution and precipitation reactions could explain the water quality changes.

With the exceptions of dolomite and gypsum, minerals whose saturation index is 0 ± 1 were considered to be both potential sources and potential sinks. Dolomite was considered only as a potential source because it does not precipitate readily under the conditions existing at the mine site. There is no gypsum in the mineralogy; therefore gypsum was considered only as a potential sink. Illite occurs in significant quantities in the wall rock, but was not considered a potential source because its saturation index is much greater than 0.

Table 3.
Concentrations of dissolved elements above (B1) and below (B2, B3) the Barr stope (mg/l).

Element	Dec. 90		March 91		May 91		MCL
	B1	B2	B1	B3	B1	B3	
Al	0.60	0.67	0.61	0.55	0.27	0.30	NAP
As	0.30	0.20	<0.10	<0.10	NA	NA	0.05 ¹
Ba	0.02	0.02	0.02	0.02	0.01	0.03	1.00 ¹
Ca	185.10	391.80	151.90	251.30	154.40	321.60	NAP
Cu	0.04	0.03	0.06	0.06	<0.01	0.03	1.00 ²
Fe	0.06	0.01	0.66	0.03	0.49	1.23	0.30 ²
K	3.40	11.00	4.20	4.20	4.00	5.50	NAP
Mg	43.70	111.30	46.60	79.60	41.00	82.40	NAP
Mn	0.83	2.36	0.89	0.04	0.73	0.10	0.05 ²
Na	1.12	1.85	0.22	0.86	0.89	1.57	NAP
Ni	<0.01	0.02	<0.01	<0.01	<0.01	<0.01	NAP
P	<0.05	<0.05	<0.05	0.16	0.23	<0.05	NAP
Pb	0.16	0.10	0.21	<0.05	<0.05	0.12	0.05 ¹
Si	9.42	8.59	7.28	7.01	7.68	8.14	NAP
Zn	0.06	0.35	0.09	0.13	0.03	0.12	5.00 ²
HCO_3^-	NA	NA	250.00	305.00	264.00	305.00	NAP
SO_4	373.20	1,140.60	360.90	797.10	340.20	863.70	250.00 ²
pH	7.79	7.36	6.89	7.09	7.34	7.32	NAP
Eh	244.00	303.00	218.00	212.00	118.00	164.00	NAP
EC	0.97	1.76	0.93	1.55	0.94	1.48	NAP

¹ Primary standards.

² Secondary standards.

MCL: Maximum concentration levels.

NA: Not analysed.

NAP: Not applicable.

EC: Electrical conductance.

Eh: Redox potential.

Two generalised models explaining the changes in observed water quality, with the constraints described above, were obtained using BALANCE. The dominant reactions of the first model included oxidation of pyrite, accompanied by dissolution of calcite and dolomite, precipitation of either ferrihydrite or siderite, and release of carbon dioxide gas. Dissolution of minor amounts of albite and K-feldspar, accompanied by precipitation of minor amounts of amorphous silica and montmorillonite, were also included in the first series of reactions. The second model substituted dissolution of ankerite

for calcite, while the other reactions remained the same. Inspection of an Eh-pH diagram (Fig 1) and the presence of deposits of yellow boy (ferrihydrite, goethite or jarosite) indicated that ferrihydrite was precipitating instead of siderite. On the basis of known mineralogy, the second of the generalised models best represents existing conditions.

Geochemical models indicate that dissolution of carbonate minerals can explain the observed buffering of acid produced during pyrite oxidation. Concentrations of metals in water discharging from the Barr stope may be limited by

Table 4.
Saturation indices for Barr stope water samples.

Mineral	Site	Date			Average
		12/90	03/91	05/91	
Albite	Upper	-1.4	-2.4	-2.1	-1.8
	Lower	-1.3	-2.0	-1.8	-1.7
Calcite	Upper	0.8	-0.2	0.3	0.4
	Lower	0.6	0.2	0.5	0.6
Ca-montmorillonite	Upper	5.6	6.9	5.3	5.9
	Lower	6.4	6.3	5.5	6.0
Dolomite	Upper	0.7	-1.1	-0.2	0.0
	Lower	0.5	-0.3	0.3	0.3
Ferrihydrite	Upper	1.7	1.1	0.6	1.1
	Lower	0.9	0.1	1.6	0.6
Gypsum	Upper	-0.8	-0.9	0.9	-0.8
	Lower	-0.3	-0.5	-0.4	-0.4
Siderite	Upper	-2.4	-0.8	-0.4	-0.9
	Lower	-3.3	-2.0	-0.1	-1.5
Silica gel	Upper	-0.4	-0.5	-0.5	-0.4
	Lower	-0.5	-0.6	-0.5	-0.5

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this buffering action. WATEQ4F and BALANCE are thermodynamically based computer models that do not account for reaction kinetics.

DISCUSSION

The most notable differences between assays of the 10-year-old and recently deposited sandfill samples were higher As, Cu and Zn, and lower Mn in the new sandfill. These differences can be explained either by differences in ore grade or by the leaching of As, Cu and Zn from the 10-year-old sandfill. The most notable differences observed from maximum leachability tests of the new and old sandfill samples were higher As, total S and Zn, as well as lower Fe and Mn in the new sandfill. The higher S concentrations in the leachate from the new sandfill may have resulted from previous leaching of SO_4^{2-} from the old sandfill as a result of sulphide oxidation. The higher Fe and Mn concentrations in leachate from the old sandfill were probably derived from more soluble, oxidised secondary minerals in the old sandfill. Arsenic and zinc may have been leached previously from the old sandfill; however, differences in ore grade is a more probable explanation.

Concentrations of major ions (Ca, Mg, HCO_3^- , SO_4^{2-}) in water samples from all sampling sites show the effects of sulphide oxidation. The increases in SO_4^{2-} , Ca and Mg, in conjunction with near-neutral pH values (table 3), suggest that acid production may be buffered by dissolution of carbonates.

The dominant reactions determined using the computer models WATEQ4F and BALANCE included oxidation of pyrite, accompanied by dissolution of ankerite and dolomite, precipitation of ferrihydrite, and release of CO_2 . Thus, dissolution of carbonate minerals can explain the observed buffering of acid produced during pyrite oxidation. The concentrations of metals in water discharging from the Barr stope may be limited by this buffering. Alternatively, slow dissolution reaction kinetics of primary minerals containing metals may limit concentrations.

The average concentrations of dissolved constituents detected in water from above and below the Barr stope were much lower than the

concentrations detected in the worst acid-producing areas of the mine. During one or more sample visits, concentrations of many constituents in water from the three sampling sites exceeded the concentrations in mine recharge water as well as the MCL set forth in the Safe Drinking Water Act. However, only SO_4^{2-} consistently exceeded these. Also, water from the lower level had higher concentrations of many of

they would have been leached along with Fe and Mn during the acid leachability test.

The degree to which the sandfilled stopes constitute preferential groundwater flow pathways depends on their degree of interconnection and their hydraulic conductivity relative to the surrounding bedrock (the natural aquifer). In the case of the Barr stope, the hydraulic conductivity of the sandfill is expected to be approximately two orders of magnitude greater than that of the bedrock.

In addition, the Barr stope is connected to an extensive system of sandfilled stopes. Groundwater flow will be channelled preferentially through the sandfilled stopes when the groundwater flow system reaches a new equilibrium. Nonetheless, flushing of the sandfill by groundwater will be much less after the stope is flooded because the mine will no longer act as a drain.

The overall impact of this specific sandfilled stope is expected to be small because of the combination of decreased sulphide oxidation, the apparent lack of metals contained in secondary minerals in the sandfill, and decreased metals transport resulting from lower groundwater flow rates.

The Spokane Research Center is continuing its research into the impacts of both cemented and

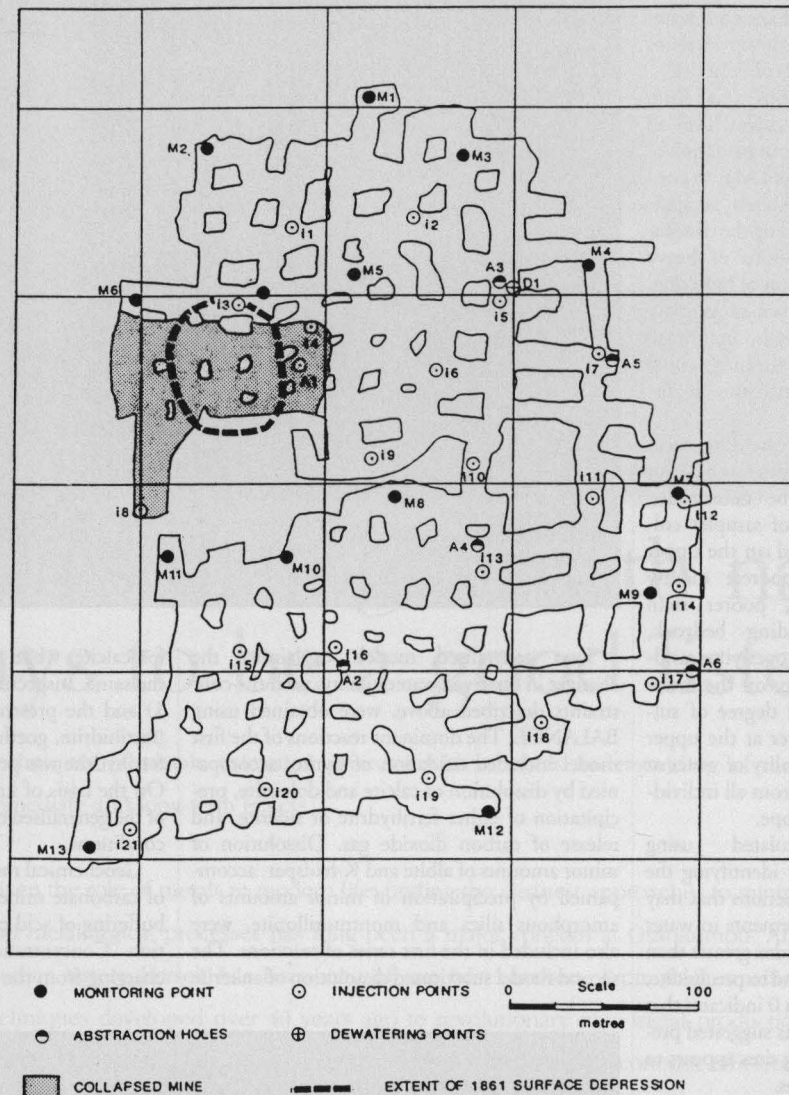


Figure 1. Plan of Littleton Street limestone mine showing location of infill holes.

the products of sulphide oxidation than did water from the upper level.

Extrapolating the results of this study to the future when the sandfilled stope is flooded requires an understanding of changes in the controlling geochemical reactions and the rate of groundwater flow through the sandfill. Once the sandfill is flooded, the rate of oxidation of sulphide minerals will be much lower, resulting in less acid being produced and therefore less associated mineral dissolution. Metals contained in secondary minerals in oxidised backfill may be released after the backfill is submerged. However, if significant amounts of metals were contained in secondary minerals in the sandfill,

waste backfill on the quality of groundwater at two mines in the U.S. Monitoring is being planned at a third field site, an abandoned mine containing sandfilled stopes that have largely flooded; in this mine, water samples will be collected from monitor wells installed from the sandfill surface. Coincident with field sampling, laboratory permeability and leaching experiments are being conducted in order to study the leaching characteristics of different backfill materials.

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Colliery spoil rock paste

As the demand for developable land becomes more intense, and the desire to conserve our agricultural and 'greenfield' land grows, methods of reclaiming land that has fallen into dereliction due to its past use are becoming increasingly important and technically more innovative.

The U.K.'s West Midlands is an area long affected by the risk of collapse due to underlying limestone mines. The mines range from being just a few metres below the surface to up to 280 m in depth with the original mine cavity ranging from 2 m to 20 m in height. Collapse of these mines has given rise to localised voids at the surface or crownholes, or to a more widespread general subsidence similar to that experienced from coal mining. This has caused concern for public safety and for the integrity of the overlying infrastructure. In addition, the land overlying the mine workings often cannot be redeveloped and therefore becomes derelict.

In 1982, as a result of the widespread nature of these problems, the Department of the Environment instigated a programme of investigation and subsequent remedial works to ensure public safety and bring the land back into beneficial use. One solution is to bulk infill the mines to prevent further collapse.

As the West Midlands is also affected by the problem of surface dereliction caused by colliery spoil heaps from coal mining, a means of using this waste material to infill the limestone mines was developed. The aim was to achieve the dual benefit of reclaiming dereliction at both the site of the limestone mine and at the colliery spoil site.

Infilling of the Littleton Street Mine was the first contract and included an initial trial phase to ensure that the desired material properties could be produced. Ove Arup & Partners was involved in the development and implementation of the programme, with Fairclough Civil Engineering Ltd as the contractor.

DESIGN PHILOSOPHY

The principle requirement of colliery spoil rock paste was that it should be cheap to produce, would flow for long distances, and have sufficient strength to support blocks of rock spalling from the mine roof thereby promoting 'choking' of any collapse chimney before a crownhole or local surface subsidence could occur.

The design strength of rock paste is typically about 12 kPa to 20 kPa and is varied to suit each application. This is achieved by the addition of a small percentage of pulverised fuel ash (a waste product of coal-fired power stations), lime and water to the colliery spoil. The contractor was required to achieve mix proportions to within $\pm 5\%$ of specifications and extensive monitoring was undertaken using continuous computerised displays of data. Tests to determine the shear strength of the rock paste were carried out several times per shift.

Initial pumping trials carried out by the Building Research Establishment indicated that

rock paste was capable of flowing for long distances over shallow gradients. This gave the advantage that relatively few surface injection holes had to be constructed as compared to other methods of bulk infilling; for example using grouting or hydraulic flushing.

The development of rock paste as a controlled low strength bulk infilling material involved a series of laboratory tests followed by full scale trial infilling of dry mine workings using 29,000 m³ of rock paste. The ideal colliery spoil for the production of rock paste was found to be a well graded, friable material with moisture content just above the plastic limit and a clay percentage of between 10% and 30%. A wetter material or one with a higher clay content is likely to be 'sticky' and difficult to handle and mix. The principle design criteria are that the paste should be pumpable at strengths of between 2 kPa and 4 kPa, flow for distances in excess of 100 m, have minimal consolidation and achieve a long term strength of between 12 kPa to 20 kPa one year after placement.

LITTLETON STREET MINE

The Littleton Street limestone mine is situated in the West Midlands of England and located to the north of Walsall. Workings ceased near the end of the nineteenth century. The room and pillar workings are located at depths of between 35 m to 60 m below ground level

and occupy a plan area of approximately 8 ha (see figure 1). Mine cavity heights vary between 4 m and 8 m and the void space totals about 550,000 m³. The area that could potentially be affected by surface subsidence was calculated to be about 18 ha.

A previous desk study and specifically developed site investigation techniques facilitated comprehensive mapping of the mine, with pillar positions and mine boundaries located with a high degree of certainty. The approximate position of the shafts used to extract the limestone was also known as was the location of a back-filled crownhole which had occurred as a result of local collapse within the mine.

Colliery spoil, obtained from three sources, had been tested prior to letting the infilling contract to establish the general suitability of the material. The source sites at Little Wyrley, Grove and Baddesley collieries were located between 10 km and 40 km from the infilling site, and provided 377,000 t, 247,000 t and 165,000 t of spoil respectively.

All sites had previously been used for agricultural purposes but because of the stockpiling of colliery spoil had fallen into dereliction. Transport of the spoil to the infilling site was by road and was subject to controls designed to ensure minimum disturbance to the environs of the infilling site, Walsall town centre and residential areas.

The contractor was required to provide colliery spoil material of maximum 50 mm size and falling within a stipulated grading envelope having not less than 10% clay (see figure 2). Burnt

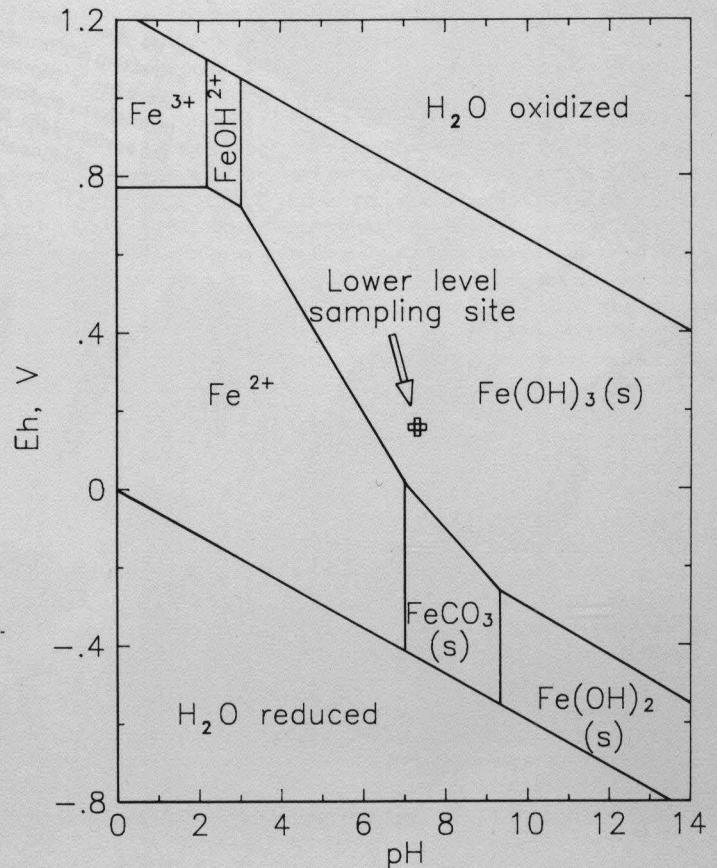


Figure 1. Eh-pH diagram for ionic species and hydroxides of Fe in water from site B3 on 17 level. Ferrihydrite [Fe(OH)₃], siderite [Fe(CO₃)], and Fe(OH)₂ are solid phases.

material was excluded and limitations on combustibility, sulphate, and chloride content were also stipulated. The additives used in the production of rock paste consisted of about 4% pulverised fuel ash, 1.5% to 3% lime and water.

The plant requirements for the production and placing of colliery spoil rock paste are such that a conventional civil engineering plant can be used. The materials receiving and handling facilities at the infilling site comprised a weigh-bridge, data recording station, storage for the colliery spoil, pulverised fuel ash and lime and a wheelwash facility. Two spoil transport conveyors were installed, one to feed the control weigh hopper with spoil from the delivery or storage areas, the other to move the spoil at controlled feed rates from the hopper to the elevated mixer platform.

Mixing was carried out via a twin shaft, continuous paddle mixer fed with bulk colliery spoil. The ash, lime and water were combined in a single fine slurry prior to introduction into the mixer. The resulting paste was discharged to an elevated holding tank from whence it was drawn off and fed by a single manifold to two electrically powered, hydraulic, double acting displacement pumps. Each pump directed paste into a pipeline which could be connected in isolation, or in tandem, to the network that delivered paste to the injection points.

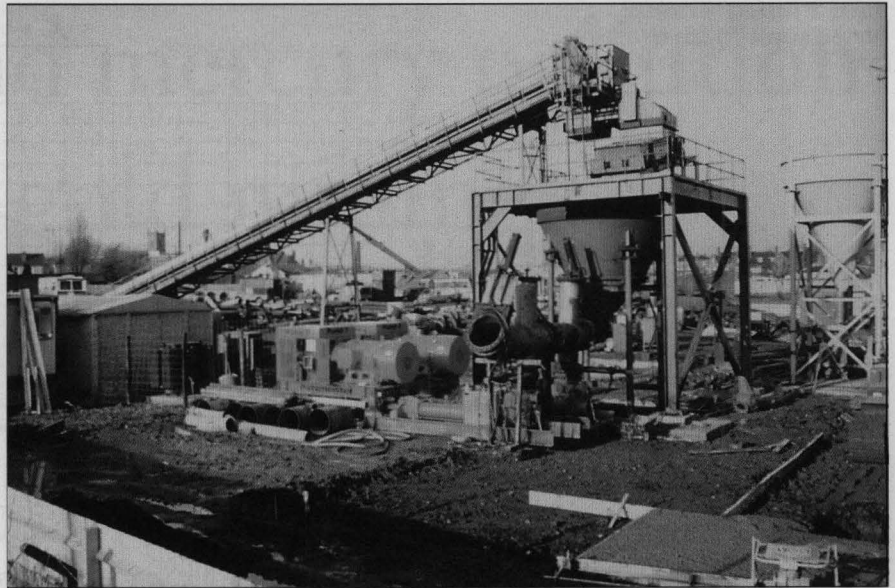
Electric power was controlled by switchgear and the supply was from an on-site sub-station connected to the local electricity supply. Independent compressed air for operating the valves on the materials feed systems and paste pipelines was supplied by on-site compressors. Control of all the equipment was centralised, with a single computer controlling the running of the plant.

All the works had to be carried out remotely from the surface as the mine was inaccessible. As this was the first flooded mine to be infilled using rock paste, a trial production of the material was specified to confirm that a suitable mix could be manufactured with the required strength gain characteristics.

Prior to trial paste production, the contractor had commissioned the infilling plant and carried out calibration of all material feed elements. A period of 28 weeks was allowed for site establishment, drilling of boreholes and trial paste production. The trial specification required that a number of design mixes should be pumped through a 200 m length of piping and discharged into a surface heap under controlled conditions.

The geometry of the heap was used to determine the strength and flow characteristics of the paste. In addition, a number of tests examined pore pressure, settlement and consolidation factors. Five mixes were produced in sufficient quantities to undertake the above sequence of operations and paste from each batch was put into twin containers for longer term monitoring.

Once operations commenced the specified 50 mm maximum size was not enforced, with 65 mm to 75 mm screens being used to minimise the loss of coarse suitable material. Generally, only about 3% of the material was found to be oversize. Unsuitable material such as burnt shale material, and that with a high clay content was



excavated and placed in discard heaps or bypassed when it was believed that such material extended to the base of the heap.

General view of infilling plant at Littleton Street site.

RESTORATION OF SOURCE SITES

Towards the end of winning operations at the Little Wyrley and Grove source sites, earth works were carried out to achieve a final landscaped form, using the previously stored overburden and unsuitable material. The Baddesley source site was still part of a working colliery and was therefore not restored.

The 2.6 ha occupied by the Little Wyrley Colliery spoil tip was described prior to spoil winning operations as "large and prominent" within a landscape of well managed farmland and woods. Thus, in terms of the landscape, a reduction in height of the tip was considered to be beneficial to provide a gentler, more integrated landform with more scope for conservation of the peripheral areas and establishing vegetation. The land has since been restored to agricultural use.

The 4.2 ha occupied by the Grove Colliery spoil tip was described prior to winning operations as a "predominantly harsh and alien feature" within a traditional English landscape, although part of the tip supported well established tree growth and was described as providing a more positive attribute. The removal of the tip provided the basis for development of the site for leisure activities by others whilst retaining the woodland features. One development proposal was for the creation of a country park at the site containing canal-based activities, tourist camping, riding and informal countryside recreation.

MINE INFILLING

As the mine was flooded, all infilling operations had to be carried out remotely from the surface. In order to facilitate this, twenty two 450 mm diameter injection boreholes were constructed into which a 300 mm retrievable injection downpipe could be installed (see figure 1).

Five water abstraction holes of 200 mm diameter were drilled and lined with permanent plastic casing to mine roof level. Pumps were installed to maintain a relatively constant mine

water level as rock paste was introduced, thus ensuring that there was no flooding of basement areas or significant changes in the groundwater regime.

Ten permanently lined boreholes, from previous site investigations, were used for monitoring purposes. An additional five observation holes were also provided during the infilling contract.

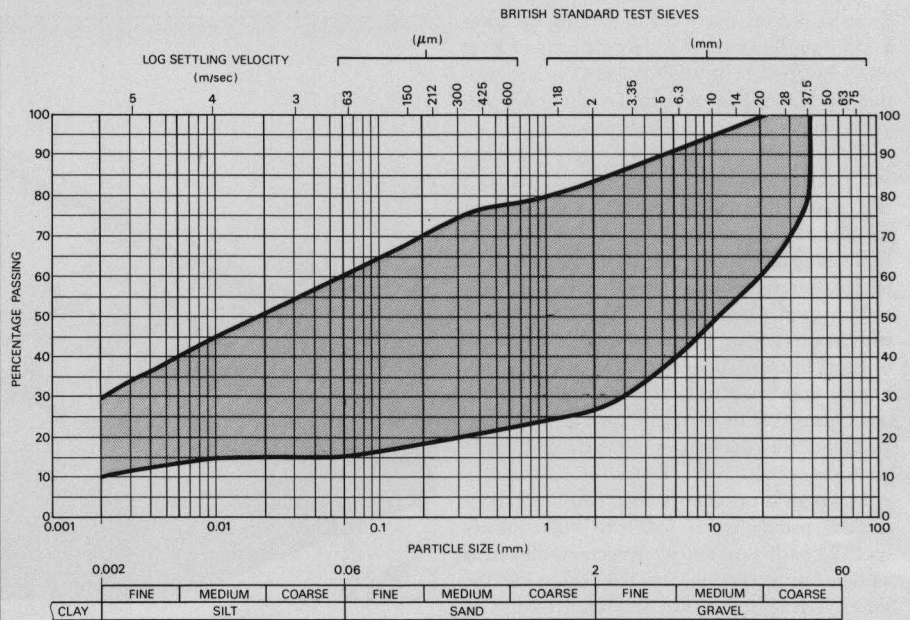
Injection of rock paste was carried out from one injection point at a time with infilling generally progressing from the deepest to the shallowest part of the mine. At each point an injection tube was placed to 1m above the mine floor. The injection tube was raised periodically as infilling progressed. Although 20 potential injection points were constructed, only 14 were required to fill the mine workings. During the last few weeks of infilling a second injection of rock paste was attempted at most injection holes to prove that the mine was filled. Paste was pumped by a small mobile pump located adjacent to each hole. At most locations only enough paste to fill the downpipe was placed with pressures of 1,500 kN/m² to 2,000 kN/m² being held to prove that the mine in the area of the injection hole was filled.

It was necessary to carry out additional grouting at only one location to complete infill operations. In all, a total of 790,000 t of paste was injected in 76 weeks (just over 10,000 t/week average) to fill a mine void of some 570,000 m³.

Prior to commencing infilling it had been possible to survey the mine geometry using ultrasonic surveying techniques. The basic method of monitoring the progress of infilling was to record the depth to the mine floor or rock paste surface at any monitoring or other borehole adjacent to the injection point in use. During the early stages, closed circuit television was used and the transverse movement of the paste could clearly be seen, the movement being pulse-like in response to the piston stroke of the pumps.

After infilling, the long term strength of the paste in the mine was measured using static cone

Figure 2. Acceptable range of particle size distribution for colliery spoil.



penetrometer (piezo cone) via the observation holes which had been cleared of paste to mine roof level. A continuous measurement of pore pressure, sleeve friction and cone resistance was thus recorded.

BENEFITS OF USING COLLIERY SPOIL ROCK PASTE

In terms of mine infilling the flow characteristics of rock paste allow treatment to be carried out from relatively few injection points thus reducing disruption of the surface and saving drilling costs. At the time the work was carried out, colliery spoil was a readily available and cheap source of low strength bulk infill material. The re-use of a waste material has the obvious advantage of not depleting stocks of non-renewable resources. The process created no additional wastes and the introduction of the materials into the mine cavities did not result in pollution of the ground or groundwater.

The dual benefit of allowing simultaneous reclamation of both the surface source site and underground mine workings is perhaps the most obvious advantage of using colliery spoil rock

paste. Such benefits could hold anywhere in the world where the pressure for development land is high and such dual dereliction exists. This form of reclamation also takes the pressure off greenfield site development by allowing economic development of derelict land. As a result of the work described, approximately 18 ha of derelict urban land has been released for redevelopment. In addition about 7 ha of rural derelict

land has been reclaimed and is available for beneficial use.

By S.T. Jarvis and P.A. Braithwaite

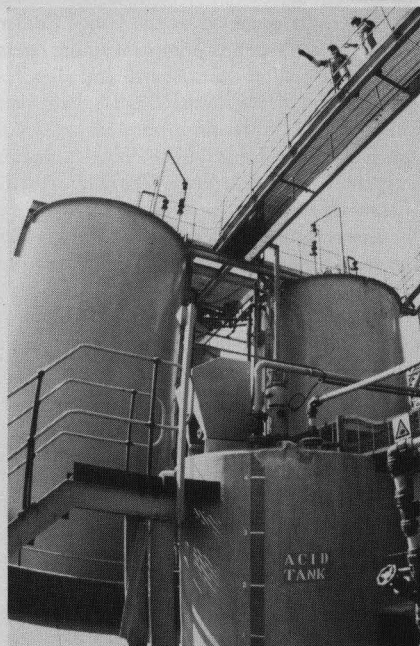
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December 1994