THE DIFFERENCES IN UNDERGROUND MINES DEWATERING WITH THE APPLICATION OF CAVING OR BACKFILLING MINING METHODS

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ABSTRACT

Zambia Consolidated Copper Mines, Ltd. (ZCCM) is planning a substantial increase in ore production in several of their underground mines on the Zambian Copperbelt over the next 10 years. The future production strategy is based on development of productive and economic mining methods through the application of mechanization and backfilling. Mechanization is designed to provide the production capability and the backfilling is designed to reduce water flow into the mines.

A similar trend can be seen in world-wide changes in mining methods from open stoping and sub-level caving to cut-and-fill stoping. Backfill is being employed worldwide, including in Australia, Canada, Sweden, Latin America, Zambia, and the U.S.A. Plans for backfill mining methods are underway for future operations in Chile, Canada, Zambia, and Mexico. The principal reasons for these changes in mining methods is twofold:

- Increased ore recovery, and
- Decreased environmental impact.

The main difference in the environmental impacts between mining with sub-level caving or open stoping and mining with backfilling methods is the reduction in subsidence or the potential for subsidence. Backfilling reduces ground movements in the rock overlying and adjacent to mine openings as well as subsidence at the surface. Reduced ground movement decreases the number and size of fracture-controlled hydraulic flow paths into a mine and, thereby, the impact of mining on surface and ground water resources. This paper deals with: 1) The impacts caused by open stoping and sub-level caving in comparison to backfilling methods; 2) The approximate impact of backfill on dewatering strategies, and; 3) The environmental benefits of backfill mining. The differences in mine drainage strategies are supported by case histories from various mines.

INTRODUCTION

Most of the ore recovery in underground mines on the Zambian Copper Belt has been mined with the application of the open stoping and sub level open stoping mining methods. Similar mining methods, allowing caving of the hangingwall, have been used world wide. Caving mining methods cause substantial, and often uncontrollable subsidence effects. Subsidence caused by open stoping impacts, to a variable degree, the strata between the mined zone and surface.

Application of any kind of backfill reduces the distance above the stope which will eventually collapse in response to mine extraction. The reduction of subsidence limits the impacts on land surface, including impacts on surface and ground water resources. The type of backfill and the method of its placement greatly influence the magnitude of subsidence reduction.

Mining methods using backfill were introduced to the mining industry mainly to increase the ore recovery and to improve the grade factor by decreasing the dilution. However, many additional beneficial mining and environmental factors accompany the introduction of backfilling: reduced ground movement, prevention of ground failures, reduced volumes of waste rock hoisting, improved in-mine ventilation and refrigeration, reduced fire hazard, reduced ground water inflow, reduced impact on surface water, reduced damage to surface structures, and better land use of undermined areas. The only drawback of the backfilling is the substantial increase in the cost of mining.

CAVING MINING METHODS

There are two potential surface subsidence effects of underground mining, chimney collapse and trough subsidence. Chimney collapse involves the collapse of the immediately overlying rock into the mined opening whereas trough subsidence involves the downward deflection of the overlying and adjacent rock toward the mined opening. Chimney subsidence is potentially the most hazardous to surface structures. Collapse chimneys develop when the roof rock progressively collapses into the mine opening. If the opening is sufficiently high or sufficient rock is extracted by caving mining, there is a risk that chimney collapse will breach the ground surface. The height that a chimney will develop for a given height of extraction depends on the swell of the roof rock when it collapses. As indicated on Table 1, the typical percent volumetric free, or unrestrained, swell of hard rocks considerably exceeds that for soft rocks. Piggott and Eynon (1977, p 764) presented a conservative geometric method of predicting the maximum height of a chimney collapse dependent on the percent swell of the rock and the height of rock extracted. An adaptation of their method is presented as Figure 1. Rectangular chimney collapse progressively develops in the rock over a longwall coal panel as the face advances. In this case both plan dimensions are large with respect to bed thickness, joint spacing and depth. Wedge type chimney collapse develops over a single underground opening where one plan dimension is small, such as over a drift in a hard rock mine or a single room in a room and pillar coal mining panel. Conical collapse, which can penetrate through the greatest thickness of overlying rock, develops over a stope of roughly equal plan dimensions in hard rock block or sub-level caving and over room and breakthrough intersections in coal mines.

The conical collapse height prediction of Piggott and Eynon appears to be limited in rock to approximately ten times the mining height. This predictive method assumes free, unrestricted, swell of the collapsing rock. The method ignores the consolidation of the collapsed rubble. Collapse chimneys are capable of penetrating through great thicknesses of overlying rock. This was demonstrated by the initial block cave breach of the surface at the Henderson Mine, which occurred after a rock column equal to 485 feet (148 m) had been extracted upward into the 3800 feet (1160 m) of overlying granite gneiss. The percent of free swell could not have been more than approximately 44 percent in this worst-case of conical collapse. It is probable that the pressure of the overlying collapsed rubble prevented the granite gneiss from expanding at a more realistic 60 percent free swell for granite.

The deformation of the rock above and adjacent to a mine opening and the immediately overlying collapse chimney, referred to as trough subsidence, extends the mining induced subsidence effects laterally. Unlike chimney subsidence, trough subsidence effects extend outward from over the center of mine extraction. The limit of trough subsidence effects at the ground surface defines an angle from the limit of mining called the angle of draw. Abel and Lee (1980) related the angle of draw to rock type for coal measure rocks, with the greatest angle of draw associated with the weaker argillaceous rocks. The variation in measured angles of draw for the same rock type indicates that other factors must impact the angle of draw and the magnitude of associated subsidence effects. Thomas (1971) presented subsidence measurements that demonstrated the outward extent of trough subsidence at the San Manuel Mine. All rock between the trough subsidence limit and the collapse chimney is distorted by the locally imposed deformation.

A collapse chimney represents a potential path for surface and ground water to enter a mine. The drainage area exposed within a collapse chimney considerably exceeds the drainage area of the physically mined openings in both block caving and sub-level caving. In addition, the trough type deflection subsidence above and adjacent to a collapse chimney must increase the permeability of the fractures along which the movements take place in the affected ground. All mining induced ground movements should be expected to increase water entering a mine. Backfill has the effect of decreasing the magnitude of subsidence in direct proportion to the effectiveness of the backfill in reducing the effective mining height.

Most of the caving mining methods assume substantial collapsing of the undermined rock masses. The propagation of subsidence and fracturing of rock above the cave mined zone usually reaches the ground surface. Uncontrolled (chimney) subsidence can cause severe damage to surface structures (buildings, bridges, and roads). Land use in the chimney collapse subsidence areas is limited for many years. Various degrees of fracturing, within the through subsidence, above and adjacent to the mined stopes usually interconnects the hangingwall water bearing strata and causes increased water inflow into mines. Surface water bodies (rivers, lakes, and ponds) can contribute to mine inflow via subsidence fractured rocks. Substantial costs were incurred at many mining projects (including the Konkola Mine) for stream relocation, impermeabilization, and increased pumping of water from the mines.

In the underground mines at the Zambian Copperbelt caving mining methods required an extensive mine drainage of the footwall and hangingwall aquifers within an angle of caving (cave line) ranging from 60 to 75 degrees, and large areas had to be dewatered with the increasing depth of mining. Such dewatering requires extensive drainage drive mining, drilling, and pumping. Extensive mine dewatering develops large zones of influence within which severe impacts on surface and ground water resources can occur.

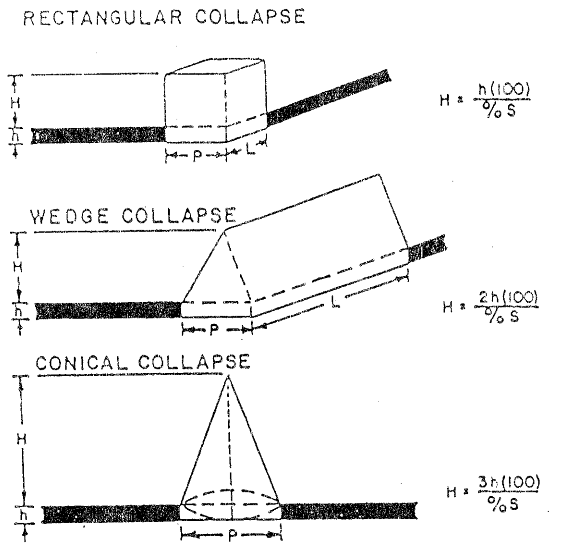
TABLE 1

BANK DENSITY, SWELL FACTOR AND PERCENT OF FREE SWELL FOR SELECTED ROCKS AND SOILS

ROCK OR SOIL	BANK DENSITY	SWELL FACTOR	FREE SWELL
Basalt	185 PCF	0.67	50%
Bauxite	119 PCF	0.75	33%
Caliche	141 PCF	0.55	82%
Carnotite, primary uranium ore	137 PCF	0.74	35%
Clay, natural	126 PCF	0.82	22%
Coal, anthracite bituminous	100 PCF 80 PCF	0.74 0.74	35% 35%
Concrete	120-155 PCF	0.72	40%
Conglomerate	153 PCF	0.72 - 0.63	40 - 60%
Copper ore, altered siliceous	141 PCF	0.74	35%
Earth, wet loam	126 PCF 96 PCF	0.79 0.81	27% 23%
Granite, quartzite, fresh typical values	163 - 170 PCF	0.67 - 0.56 0.61	50 - 80% 64%
Gravel, pit run	135 PCF	0.89	12%
Gypsum	198 PCF	0.57	75%
Iron ore, high grade hematite limonite magnetite taconite	174 - 237 PCF 174 - 237 PCF 204 PCF 192 - 226 PCF	0.58 - 0.55 0.58 0.55 - 0.58 0.58	72 - 82% 72% 72 - 82% 72%
Limestone typical values	155 - 163 PCF	0.57 - 0.60 0.59	67 - 75% 69%
Marble, metamorphic	170 PCF	0.57 - 0.60	67 - 75%
Montmorillonite, chlorite, kaolinillite, smektite	141 PCF	0.77	30%
Pyrite	189 PCF	0.85	18%
Sand, dry damp wet	100 PCF 120 PCF 130 PCF	0.89 0.89 0.89	12% 12% 12%
Sandstone	153 - 157 PCF	0.60	67%
Shale, mudstone	104 PCF	0.75	33%
Siltstone, hard soft	153 - 157 PCF 126 PCF	0.57 - 0.60 0.82	67 - 75% 22%
Slate	170 - 180 PCF	0.77	30%
Traprock, basic, igneous	163 PCF	0.67	50%

NOTES: Free swell (%) = change in volume broken as a percent of original bank volume
Swell factor = broken density/bank density

Adapted from: Caterpillar, Inc., 1987, Caterpillar performance handbook, p 740 and Euclid Road Machinery Co., 1953, Estimating production and costs.

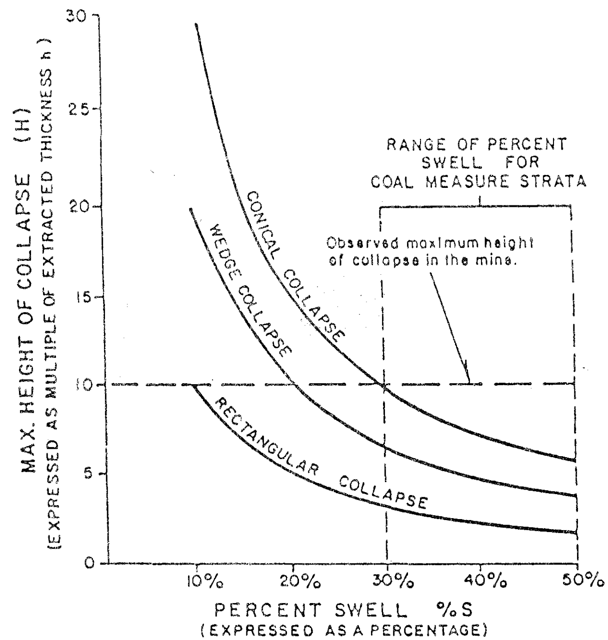


$\%S$ = PERCENT SWELL = $\frac{Vc - Vo}{Vo}(100)$

Vo = ORIGINAL VOLUME OF UNBROKEN STRATA

Vc = VOLUME OF COLLAPSED ROOF BEDS

Diagram showing notation for calculating maximum height of collapse (H) in relation to geometry of collapse.
(modified from Piggott & Eynon, 1977)



Graph showing variation in maximum height of collapse for different modes of failure and bulking factors.
(modified from Piggott & Eynon, 1977)

BACKFILLING MINING METHODS

Backfill was initially developed to permit more economic mining of relatively weak to incompetent vein-type ore deposits. Backfill has permitted mining of weak orebodies under incompetent hanging wall rock at considerable depth, such as the Magma Mine at a depth of nearly 4,800 feet (1460 m) near Superior, Arizona. Delayed backfilling has been applied to stabilize large open stopes in competent ground. Delayed backfilling with cemented rockfill has permitted mining of adjacent 100 m high, 50 m wide by up to 75 m long blocks of ore at depths to 1,000 m at Mt. Isa in Queensland, Australia.

The method was adapted by coal mining to permit high-extraction mining beneath built-up areas on the ground surface by reducing subsidence effects. In the U.S., backfilling of abandoned mine workings through boreholes from the surface has been more recently applied as part of the Abandoned Mined Lands (AML) program of the Office of Surface Mining (OSM). One of the requirements of the Surface Mining Control and Reclamation Act (SMCRA) for operating a coal mine in the US is the filing of a Subsidence Control Plan with OSM. This plan must predict the worst-case subsidence effects on surface improvements and surface and ground water resources. Backfilling can significantly reduce subsidence effects, both at the ground surface and on overlying aquifers. Backfilling can reduce the inflow of water into a mine by increasing the flow resistance to water entering the mine.

Backfilling has been applied to halt a progressively advancing squeeze at the White Pine Mine in Michigan and to increase ore recovery and reduce the potential for flooding at El Mochito Mine in Honduras. Details on these applications are provided in the case histories at the end of this report.

Subsidence and ground movements induced by mining can be significantly reduced by backfilling. The magnitude of subsidence and ground movement reduction achievable with backfill is dependent on the completeness of the backfill, the backfilling material, the competence of the backfill and the method of fill placement. It is nearly impossible to achieve 100 percent backfilling of a mine opening. Competence in this context refers to the strength and stiffness of the fill. The competence of backfill can be enhanced by the use of autogenous cementing minerals and(or) Portland cement. The method of fill placement can range from manual packing, to hydraulic, to pneumatic. Hydraulic fills are usually the most effective in reducing mining induced ground movements.

Fill Characteristics

Mill tailings are the most common backfill material and the most common method of placement is by slurrying mill tailings, i.e. hydraulic backfill. In 1971 the USBM listed 40 U.S. mines, primarily base metal, utilizing hydraulic backfill. Bulkheads are required to initially contain and drain excess water from hydraulic backfills. In the case of mill tailings backfill, drainage is essential to eliminate the potential for subsequent liquefaction. The authors have observed tailings runs from stopes filled with unclassified tails at the Cole Mine at Bisbee, Arizona and the Ontario Mine at Park City, Utah. At the Cole Mine unclassified tails were released as tailings runs more than a decade after placement, when lagging in underlying timber chutes rotted out. At the Ontario Mine unclassified tailings backfill was actually forced upward to an overlying level when a combined pillar and hanging wall failure occurred. Hydraulic backfill is usually classified to remove the extremely fine fraction. The fine fraction of the mill tailings, typically less than approximately 400 mesh (0.037 mm), is removed in order to permit drainage and, thereby, prevent later liquefaction under stress changes or vibration. The process of liquefaction was described in the Aberfan Tribunal Report, (British Government, 1967, p 115), as follows: "This can occur in a heap of loose sand or in an uncompacted tip of mine rubbish. If the lower part of the tip contains water, filling the spaces between the particles, and if a sudden load or shock is applied (such as the slipping of the upper part) the water then supports the particles and the whole saturated body behaves as if it were a liquid." The magnitude of the "sudden load" necessary to liquefy saturated unclassified tails can be as little as one of the authors walking over the center of a stope, at the Ontario Mine.

The critical requirement to prevent liquefaction is the reduction of the water content of the backfill to below saturation. Unclassified tails tend to drain extremely slowly because of barriers to water migration developed by segregated fine silt and clay particles present in most mill tailings, typically greater than 30 percent minus 400 mesh (0.037 mm) particle size. Generally, hydraulic backfill is classified to remove the fine fraction and, thereby, to facilitate drainage and prevent later liquefaction. At the North Broken Hill Mine in Australia, the classified uncemented mill tailings backfill is sufficiently drained through decant raises that LHDs can operate on the top of the fill four hours after completing a fill lift.

Portland cement can be added to reduce the potential for or even prevent subsequent liquefaction. If the fine fraction is not removed, there will be limited void space present for the nominal less than 325 mesh (0.044 mm) Portland cement added to solidify, stiffen and strengthen the backfill. At the Magma Mine (Murray, 1973) the tailings were deslimed at roughly 400 mesh (0.037 mm) and cement added at an approximate 12 to 1 by weight sand-cement ratio for the lower 3 feet (0.9 m) and 20 to 1 for the upper 9 feet (2.7 m). The 28-day mean compressive strength of the fill was statistically 28 psi (0.19 MPa), regardless of the cement content. At El Mochito Mine in Honduras an extremely weak, but effective backfill was produced by classified minus 1/4-inch (6.4 mm) and plus 150 mesh (0.0504 mm) crushed limestone backfill with a 40 to 1 by weight sand/cement ratio. El Mochito Mine intermittently dumps coarse development rock into the stopes during stope filling. The coarse rock backfill may explain the ability of the weak backfill to withstand blasting vibrations from mining the adjacent ore and for exposed fill walls 120 feet (36 m) high and up to 125 feet (38 m) long to stand unsupported.

The increase in the seven-day compressive strength achieved by varying the sand-cement ratios of eight classified tails in use as hydraulic backfill was reported by Corson (1970). He reported the range of seven-day mean compressive strengths for 20 to 1 sand-cement ratios ranged from 5.8 psi (40 KPa) to 99.3 psi (403 KPa). It is interesting to note that the lowest strength was for the coarsest of the eight classified tails and that had the lowest proportion of minus 400 mesh (0.037 mm) particles. The low strength tails was otherwise not significantly different from the other seven stronger classified backfill tails. The seven-day strength of the strongest of these eight cemented hydraulic backfills had the highest proportion of minus 400 mesh (0.037 mm) particles and was the most poorly graded. There are, obviously, a number of factors involved in the strength of cemented hydraulic backfill, in addition to the cement content. Prediction of the strength of a cemented hydraulic backfill can be considerably in error. The compressive strength of a cemented backfill can only be determined by testing.

Coarse rock fill has a long record of successful application. Coarse rock fill is utilized as delayed stope fill at El Cubo Mine near Guanajuato, Mexico. At El Cubo the rock fill is development muck trammed to the top of the stopes and dumped. Coarse rock fill and limited unclassified mill tailings are dumped at the top of one end of stopes at the Erzberg Mine in Austria as broken ore is removed from the stope floor the other end. The backfilling program at the White Pine Mine involved initial placement of coarse waste rock in 12-foot (3.7 m) high by 28-foot (8.5 m) wide rooms and crosscuts. The LHDs used were only able to place the coarse rock to within 1 to 3 feet (0.3 to 0.9 m) of the flat-roofed openings. Nearly complete fill was accomplished by halting the coarse waste filling operation every 50 feet (15 m) to pneumatically blow relatively fine backfill over the just placed coarse backfill. This costly backfilling operation was driven by the need to protect a major section of the mine from an advancing squeeze. A borehole was drilled 2,200 feet (670 m) from the surface to introduce the pneumatic fill near the filling area. The backfill pipes had to be repeatedly assembled and broken down.

Paste backfill is the most recent innovation in backfill technology. Brackebush and Edwards (1993) define paste backfill as, "a high density mixture of water and fine solid particles with a relatively low water content (10-25%)". The low water content prevents segregation, or settling, when stationary. The required consistency of the paste is slump in the concrete cone test of less than 12 inches (305 mm). Paste mixtures are non-Newtonian fluids, i.e. they have a significant yield stress but have a relatively constant viscosity with respect to flow rate. Paste backfill has been dropped in pipes as much as 5100 feet (1.5 km) vertically at the Lucky Friday Mine in Idaho and pumped as much as 7545 feet (2.3 km) horizontally using an intermediate pumping station at the Bad Grund Mine in Germany. At the Greens Creek Mine in Alaska, filtered paste is batched with cement and hauled underground, just like concrete. At Greens Creek the paste is mechanically placed. Patchet (1978) presented the results of a prototype paste backfill at the Western Holdings No. 1 Shaft in the Orange Free State, S. Africa. The unclassified tailings were dewatered by a centrifuge near the stope and then mixed with cement in a wet-mix shotcrete machine and pumped to the stope.

In practice paste mixtures range from 73 percent to 90 percent solids by weight. Fine particles are essential to the development of a paste. In most cases, pastes must contain at least 15 percent by weight of particles less than 20 microns in diameter. Therefore, unclassified mine tailings are frequently usable for making a paste, provided the slimes are not lost during dewatering. Conventional gravity thickeners are the first step in producing a paste. The normal 30-35 percent water content of the underflow from a thickener must be further reduced to produce a paste. Field applications employ either filtration or mixing with dry alluvial material to reduce the water content. Concrete mixers are frequently used to thoroughly mix the paste components. Cement can be added to the mix, for strength and positive prevention of subsequent liquefaction, either in the mixer or injected into the paste pipeline a short distance before placement in a stope. Compressive strengths of 100-300 psi (0.7-2.1 MPa) have been achieved with 3-5 percent Portland cement by weight. Precise control is necessary to produce and maintain the consistency of a paste.

Paste backfill has several advantages. Pastes cannot by liquefied without the addition of water. Pastes utilize the fine fraction of mill tailings. High compressive strengths can be achieved with less cement because segregation does not occur within a paste. There is little water to decant from a filled stope. The biggest factor limiting the wider utilization of paste backfill has been the necessity for precise control of the water content.

Ground Movement Effects

Backfill reduces the magnitude of ground movements that can result in the rock mass overlying or adjacent to a mine opening. The space occupied by backfill is not fully available for ground movements. Table 2 indicates the range of consolidation reported with different backfill materials and adjusted to an approximate loading stress change from 0 to 2000 (14 MPa) when confined in an undrained laboratory consolidometer. The range of percent volumetric consolidation is partly due to the types of material involved and partly to the initial moisture content. Coarse, low- density, low-strength, brittle and uniformly graded materials in the dry state underwent the greatest volumetric consolidation.

TABLE 2

CONSOLIDATION OF MINE BACKFILL MATERIALS

Consolidation Pressure (psi)	Material	Range of Volumetric Consolidation(%)	References
2070	Sand	2 - 30%	(1) (2) (3) (4)
2070	Sand & Gravel	3 - 5.6%	(1) (3)
2070	Sand, Ash & Refuse	8.6 - 35.2%	(1) (3)
2070	Ash	15 - 52%	(1) (3) (4)
2070	Washery Reject	35 - 38.5%	(2) (3)
2070	Crushed Stone	22.8 - 42%	(2) (3)
2070	Sand & Washery Reject	6.4 - 23.2%	(2) (3)
2070	Tailings	21 - 24%	(4)

Notes: 1) Maximum range from saturated to dry and from poorly to uniformly graded.
2) Initial densities not considered. 2070 psi = 14.3 MPa.

References for Data: (1) Dierks, 1933; (2) Whetton & Sinha, 1950; (3) Singh & Gupta, 1968; (4) Mickle & Hartman, 1961.

Field measurements indicate a somewhat different picture. Corson and Wayment (1967) reported a maximum 22 inches (0.56 m) of closure across an approximately 12-foot (3.7 m) wide hydraulically sand- filled stope at a depth of approximately 6600 feet (2000 m) below the surface in the Star Mine in Idaho. This stope is only a section of a very long stope on the strike of the vein. The wall rocks are considered incompetent because of hydrothermal alteration. The stope walls were initially supported by 12-inch (0.3 m) diameter round timbers. The uncemented backfill was classified between 40 mesh (0.416 mm) and 0.003 mm. The measured closure took place over 36 months and closure was continuing when last measured. The closure rate had significantly decreased over the convergence measurement period. The closure resulted in over 540 psi (3.7 MPa) of stress in the fill when last measured. The measured fill stress was increasing at a decreasing rate at the time of the last measurement. The major portion of the horizontal load originally carried by the excavated ore was apparently being carried by the crown and sill pillars above and below the 200-foot (61 m) high stope. The measured compaction of the backfill was 15.3 percent over 36 months. Zahary (1960) reported deformations along one wall of a 60-foot (18 m) wide, 70-foot (21 m) long, original minimum span, and 250-foot (76 m) high rock-filled vertical stope at a depth of 1050 feet (305 m) in the Geco Mine near Elliot Lake, Canada when 50 feet (15 m) of the 100-foot (30 m) width of the adjacent 350-foot (107 m) high pillar was blasted. The wall rocks are variably competent gneisses and shists. Blasting was performed in two stages, 300 days apart. The maximum overall wall deformation was 1.8 inches (46 mm), indicating fill compression of only 0.5 percent, assuming both stope walls moved inward the same amount. The rate of stope wall deformation decreased to a steady state within 50 days after each blast, but was continuing at a steady rate 680 days after the initial blast. The loose rock fill could not have been subjected to much consolidation pressure by the stope wall deformation following the blasting the stope to a length of 120 feet (36 m), new minimum span. The major portion of the horizontal load originally carried by the rock excavated was apparently carried by adjacent pillars and the backfill was not carrying much load. In both of these cases the critical minimum width of stoping which would transfer the full ground stresses to the backfill had not been reached.

Longwall coal mining provides some insight into the backfill consolidation that takes place when the full ground stresses are imposed on the backfill. The surface subsidence measured over large areas of coal extraction indicate the effectiveness of backfill in limiting ground movements. Table 3 presents the maximum surface subsidence measured over such wide, super-critical, areas of extraction. Most of the cases in Table 3 were probably not cemented. It would appear that hydraulic backfill is capable of reducing subsidence to less than one-third of what would develop when the roof is permitted to cave, in the absence of backfill. Piggott and Eynon (1977) provided a mathematical method of calculating the height of caving above a longwall mined coal seam, based on the free swell of the immediate roof rocks. The roof collapse following the advance of longwall face support provides a loose rock backfill. The consolidation of this collapsed roof-rock backfill under the overburden loading never reaches the density of the original in place roof rock. This explains why measured surface subsidence never equals the seam height mined, as shown under the "None" column in Table 3.

A major limitation on the potential effectiveness of backfill in limiting ground movement is the fact that the backfill is normally placed with a "loose" or "honeycombed" structure (McNay & Corson, 1975, p16). Loosely-placed low-density backfill is subject to considerable consolidation before it develops significant load carry capacity. Table 2 indicated the consolidation potential of various backfill materials under approximately 2000 psi (14 MPa) of fully confined compressive stress. Hydraulic backfill appears to have the potential to reduce maximum surface subsidence over longwall panels to approximately 29 percent of that which would develop over longwall panels without backfill. Pneumatic backfill appears to have the potential to reduce maximum surface subsidence over longwall panels to approximately 57 percent of that which would develop over longwall panels without backfill. Comparatively, hydraulic backfill appears to reduce subsidence reaching the surface to 52 percent of the subsidence that would reach the surface using pneumatic backfill. This compares to Orchard's (1961, p 261) statement concerning the surface subsidence reduction from hydraulic backfill, as follows: "It is estimated that the proportional subsidence for a full critical area would have been about 23 per cent of the seam thickness extracted, an efficiency almost twice that of pneumatic stowing." The probable reason for the differences reported between the efficiency of hydraulic and pneumatic backfill is the difference in packing achieved. Pneumatic backfill is apparently more "honeycombed", i.e. more loosely packed, and, therefore, more compressible.

TABLE 3

MAXIMUM MEASURED SURFACE SUBSIDENCE AS A PERCENT OF COAL SEAM THICKNESS EXTRACTED AND BACKFILL METHOD

Hydraulic	Pneumatic	Manual Packing	None	References
0.3 - 8%	--	25%	30 - 70%	(1)
2%	--	--	--	(2)
8 - 30%	30 - 70%	30 - 70%	60 - 95%	(3)
15 - 20%	--	--	70 - 90%	(4)
--	50%	--	90%	(5)
10 - 18%	22%	36 - 50%	63 - 72%	(6) (11)
22 - 45%	--	--	72 - 88%	(7)
10 - 25%	40 - 50%	45 - 76%	76 - 85%	(8) (9)
10 - 30%	25 - 60%	25 - 62%	80 - 95%	(10)
25 - 35% (*)	45 - 55%	65 - 75%	85 - 99%	(12)
10 - 15% (**)	- -	- -	- -	(12)

Notes: Assumed super-critical extraction area, i.e. minimum width of extraction exceeds 1.4 times the depth.
(*) Sand and tailings
(**) Sand and tailings - cemented (up to 3% of cement).

References for data: (1) Knox, 1913; (2) Rellensman & Wagner, 1957; (3) Brauner, 1973; (4) Wardell, 1969; (5) NCB, 1975; (6) Salamon, 1964, Polish experience; (7) Salamon, 1964, Hungarian experience; (8) Salamon, 1964, German experience; (9) Singh & Gupta, 1968, German experience; (10) Singh & Gupta, 1968, worldwide experience; (11) Orchard, 1964, Polish experience; (12) Neset, 1984, Czech experience.

Ground Movement Reduction Potential of Backfill

Backfill of any type is capable of reducing the ground movements that would otherwise result from mining. Every underground opening is in the process of closing and if the opening is filled with another material the magnitude of the closure will be reduced. How much the closure of the opening will be reduced is subject to conjecture, but tests can be performed to estimate the magnitude of the reduction. Hydraulic backfill with classified tailings would appear to have the potential of reducing ground movement associated with closure to somewhere between one-forth and one-third of the volume occupied by the hydraulic tailings backfill. Pneumatic backfill appears to have the potential of reducing ground movements associated with closure to somewhere between one-half and two-thirds of the volume occupied by the pneumatic backfill.

The laboratory measured consolidation of coal mine backfill materials have been reported by various researchers. The reported method of testing was the soil testing procedure referred to as a "consolidated undrained test". The range of volumetric consolidation percentages for the same apparent material probably represents, in part, varying moisture contents, from dry to saturated, as well as different testing laboratories. Some very general inferences can be drawn from these results. More poorly graded materials exhibit more uniform consolidation. More uniform materials undergo more consolidation. It would appear that the presence of a wide range of particle sizes provides fine particles to fill the interstices between the larger particles.

The behaviors of the rock masses above the mined and backfilled stopes is of crucial importance for the assessment of the necessity to dewater the hangingwall aquifers. In the mining industry, computer models are widely used to simulate the performance of rock masses in an active mining environment.

The Rock Mechanics ground of ZCCM used the Universal Distinct Element Code (UDEC) model to simulate the response of the hangingwall rock masses to mining at the Zero Fold in the Nkana Mine with and without backfill (Spivey, 1991). A typical section of the Zero Fold area at mine section 2000 North of the Central Shaft was used for the modelling. Results of the modelling indicated the great difference between mining with and without backfill on the behaviors of the hangingwall strata. The contour plots indicate closure/displacement decreasing towards the mining face and the magnitude of surface subsidence decreasing to the southwest. During modelling of the Zero Fold mining, both with and without backfill, the history of vertical displacement subsidence was monitored at 10 locations. The monitoring points were located immediately beneath the Far Water Dolomite, some 200 meters above the orebody, 20 meters thick. Results of modelling indicated that the reduction in subsidence at the Far Water Dolomite with the use of backfill would reach 90.5 percent. Such reduction of subsidence would assure the integrity of the aquiclude strata between the orebody and the Far Water Aquifer.

Impacts of Backfill on Ground Water Inflow

Ground water inflow into mines with good quality backfill is generally greatly reduced. To our knowledge, there are no metal mines using exclusively mining methods with backfill, with a significant ground water inflow, and/or with any major problems related to ground water inflow. Several mines with ground water inflow are in conditions where the water bearing strata are adjacent or near the orebody (for example: Neves Corvo Mine, Portugal; Murray and Real, 1990), or where major inflow occurred during mine development, prior to ore recovery (Hilton Mine, Australia; Mutton and Whincup, 1988), or where, prior to backfill application, mining methods which permit caving of hangingwall were used (El Mochito, Honduras; Halley, 1981). Our extensive research of mining case histories (Appendix) indicates that the experience with any adverse conditions associated with cut-and-fill mining and ground water is limited.

Ground water inflows are generally less for backfill mining methods than for methods which permit caving of the hangingwall, because the backfill material fills the mined out void. The backfill reduces the distance above the stope which will eventually open up in response to mine extraction. Therefore, the zone of increased fracture permeability will be considerable reduced. If one assumes that the ground water entering a mine is dependent upon the surface area of the volume of disturbed mining ground, the use of backfill should reduce ground water inflows in direct proportion to the compressibility of the backfill in relation to the mine extracted volume. Ground water inflows should be reduced by 90% using 10% compressible uncemented backfill. Cemented backfills have the potential of reducing ground water inflows to what would normally flow into the same surface area of open tunnels. This simplified calculation assumes that there are no aquifers to be tapped in the hangingwall. If there are such aquifers in the hangingwall, the distance that ground water would be drawn toward the stopes will be proportionately reduced by the elimination of hangingwall collapse, and by the reduction of the hangingwall deflection. Simply filling stopes with any solid material reduces the potential hangingwall movement that could occur in case of open stopes.

CHANGES OF THE DEWATERING CONCEPT IN BACKFILLED MINES

The discussion presented in the previous sections indicated that with the introduction of backfill mining method the requirements for mine dewatering are reduced. With a good quality backfill, which would reduce the subsidence by 80 to 90% in comparison with caving mining methods, the mine dewatering can be substantially reduced. Typically, with backfill mining methods it is necessary to dewater nearby footwall and hangingwall aquifers which are adjacent to the ore zone. However, hangingwall aquifers approximately 40 meters or more above the mined ore would not probably drain into the mine (Straskraba, 1991). The need for drainage is highly site specific and the presence of water-bearing faults and anticlines has to be considered. Hangingwall aquifers with high hydrostatic pressure may need limited drainage to reduce the pressure. Caution should be exercised in mines where caving methods of mining were initially applied. The location of caved areas, development of subsidence, and fracturing should be considered prior to substantially reducing drainage efforts in areas of backfill application.

The reduction of dewatering effort in mines with extensive drainage of hangingwall aquifers high above the mined horizon, as for example in the ZCCM mines (Konkola, Mufulira, and Nkana), could bring significant cost reduction. Any decrease of drainage drift mining, drilling of deep drainage boreholes, and reduced pumping due to changed dewatering strategy, can help offset the increased cost of the backfilling operation. However, any substantial change in dewatering strategy should be based on geotechnical and hydrogeologic studies, and supported by rock mechanics and hydrogeological monitoring.

Surprisingly, there is limited amount of technical literature dealing with the impacts of backfilling on mine water inflow. More research is definitely needed in this field.

ENVIRONMENTAL IMPACTS OF CAVING AND BACKFILLING MINING METHODS

Caving Mining Methods

Hard rock mining with the use of caving has been substantially limited in Canada and the USA during recent years. The main reason of this restriction is not only the application of more productive mining methods but also the implementation of more stringent environmental laws. The exception is longwall mining for coal, where the total percentage of coal mined by longwall mining methods is still increasing. However, this is due to the controlled and predictable subsidence during longwall mining and increased knowledge of the potential environmental impacts of this mining method.

Caving mining methods with the development of fracturing between the mined horizon and surface cause several environmental impacts. Following are the most typical cases:

- Open cracks and fractures on the ground surface;
- Limited land use in areas of severe subsidence;
- Potential impacts on surface water bodies;
- Potential drainage of shallow aquifers, impacts on vegetation and wildlife;
- Impacts on water quality due to interconnection of aquifers;
- Interconnection of several aquifers;
- Increased mine inflow, larger zone of influence;
- Impacts on water wells in large areas; and,
- Impacts on mine safety due to the increased water inflow.

Backfilling Mining Methods

Experiences with backfilling mining methods indicate that the reduction of subsidence and consequent limited fracturing of the hangingwall strata, reduces the potential for surface and ground water inflow into the underground mine. Most of the mines using backfill are relatively dry and none of the mines known to the authors experienced excessive mine inflow. Benefits of backfilling for the reduction of environmental impacts of mining are numerous. Following are some of the advantages of backfill mining:

- Limited fracturing above the mined strata;
- Limited land surface subsidence;
- Low potential for impacts on surface water resources;
- Limited impact on hangingwall aquifers
- Reduced mine water inflow and reduced impacts on ground water resources;
- Improved mine ventilation and refrigeration; and,
- Limited potential for mine fires.

CONCLUSIONS

The introduction of mining methods with backfilling significantly improved the ore recovery and grade control in most of the hard rock mines where it has been utilized. Backfilling mining methods have many other beneficial factors in addition to ore recovery and grade. Improved safety, decreased potential of fire hazards, improved ventilation and refrigeration, and decreased dewatering costs for mining, drilling and pumping were reported at many mining operations. However, a substantial reduction of the environmental impacts of mining with the introduction of backfill is seldom considered as a benefit and cost factor in feasibility studies. Feasibility studies for the introduction of backfilling mining methods should include the factors of the reduced impacts on land surface and surface and ground water resources.

CASE HISTORIES

1) White Pine Mine, in the upper peninsula of Michigan, USA, Between January 9 - 13, 1988, a 2.1 acre (8400 m^2) area roof fall of immediate roof Brown Massive shale occurred in a storage area at what proved to be the maximum 2100 foot (640 m) depth end of the initial major pillar and roof collapse. On January 14, 1988, 12:25 P.M., 171 acres (690000 m^2) of initial pillar and roof collapse occurred at depths ranging from 2100 feet (640 m) to 1985 feet (605 m). The resulting "Big Boom" was felt throughout the mine, the plant site and the town site. Two air doors were blown out and an air blast with a cloud of dust swept through the Southwest Shaft and Area V shops area. Surface subsidence monuments measured the following morning showed 0.18 feet (5.5 cm) of maximum vertical subsidence over the previous three days versus 0.52 feet (16 cm) the previous 12 years. The cave area continued to progressively expand, ultimately covering an area of 384 acres on June 30, 1988, at depths ranging from 2100 feet (640 m) to 1750 feet (530 m). Maximum surface subsidence increased to a total of 1.08 feet (33 cm) by August 31, 1988. The initial failure was centered on 31.4 acres (127000 m^2) of room and pillar mining, to heights from 22 feet (6.7 m) to as much as 28 feet (8.5 m). The planned mining height was from 12 feet (3.7 m) to 18 ft (5.5 m). Ninety-eight percent of this area was mined in 1980-1982 with the remaining 2% mined in 1986. Aerial extraction in this area was approximately 68%.

The cave progressed rapidly toward the No. 67 Belt and despite standing 1200 cedar stulls, wrapping 9 pillars with wire rope and installing twenty-four 25 ton jacks the belt line for two-thirds of the mine production was lost February 4, 1988 at 5:30 A.M. The cave was also progressing toward the critical Southwest Shaft and Area V Shops. Convergence monitoring ahead of the advancing cave provided ample warning of impending local collapse. No one was injured during the cave. Cave progression toward the Southwest Shaft and Area V Shops was halted by August 30, 1988. Four rows of rooms and crosscuts on the west and south sides of the cave were backfilled during July and August 1988. Coarse aggregate was placed to within 3 ft (0.9 m) of the roof with LHDs and pneumatic backfill blown over the top of the coarse fill to completely fill the openings. Convergence rates measured in the Area V Shops slowly decreased to instrument sensitivity by August 30, 1988.

No significant increase in water entering the mine occurred as the result of the cave. This collapse was simply the only unplanned and the most serious of six major area caves of ground. The other five were planned as part of the pillar retreat mining at shallower depths. The mine had 27 smaller planned caves of ground during retreat mining. Hydraulic backfill had been successfully utilized to permit mining near important mine workings at 17 locations in the mine. Coarse waste rock and pneumatic backfill were used for the backfill protective barrier for the Area V Shops and the Southwest Shaft in order to speed the backfilling operation by eliminating bulkhead construction and because of limited availability of unfrozen fine backfill.

2) El Soldado Mine, about 84 miles (135 km) northwest of Santiago, Chile. The two largest open stopes collapsed to surface in 1992, the California Stope and the Santa Clara/Chuquichico Stope. There was sufficient warning that no injuries resulted from either collapse. The maximum height of the California Stope at the time of the collapse was 1070 feet (325 m) , the maximum width 400 feet (120 m) and the maximum length 980 feet (300 m). At the time of its subsequent collapse the Santa Clara/Chuquichico Stope maximum height was 890 feet (272 m) , maximum width 430 feet (130 m) and maximum length 1310 feet (400 m). In both cases the maximum crown pillar thicknesses were roughly 260 feet (80 m). Additional ore remained adjacent to the post-collapse rubble-filled California Stope. Cut-and-fill mining was evaluated as a method of recovering this ore adjacent to the extremely coarse rubble. Backfill is also being considered as a means of more safely extracting the Valdivia Sur orebody, whose maximum planned ultimate dimensions are 660 feet (200 m) height, 400 feet (120 m) width and 670 feet (205 m) length. There are over 30 additional smaller open stopes scattered around and between the two largest stopes that are candidates for backfilling. Mine drainage has not been a problem because of the haulage and drainage level approximately 410 feet (125 m) below the lowest mining level.

3) El Monte Mine, about 15 miles (21 km) north of Zimapan, Mexico. The open stope, which is the mine, collapsed to the surface in April 1990 after two months of minor falls of ground. The maximum dimensions of the open stope at the time of the collapse were 750 feet (229 m) in height, 165 feet (50 m) in width and 490 feet (150 m) in length. The overall height of the potential orebody is approximately 1090 feet (333 m). The orebody rakes at approximately 70 degrees. The oxidized crown pillar was approximately 165 feet (50 m) thick before collapse and had successfully spanned the open stope for 20 years before the collapse. Mining the approximately 340 feet (104 m) of ore underlying the collapsed rubble obviously presented a problem. Cut-and-fill mining was considered, but has tentatively been rejected due to the amount of ore that would be lost and the development that had already been completed within the lower part of the orebody. The collapse was not accompanied by any increase in mine drainage.

4) Rocanville Mine, 130 miles (210 km) east of Regina, Saskatchewan, Canada. The soluble Prairie Evaporite Formation, Esterhazy Member potash ore lies approximately 80 feet (24 m) below the 2nd Red Bed marlstone and dolomite and 105 feet (32 m) below the Dawson Bay limestone and dolomite aquifer at a depth of approximately 3150 feet (960 m). Hydraulic pressure in the Dawson Bay aquifer is 1250 psi (8.6 MPa). Panel extraction, at the typical 8-ft (2.4 m) mining height is normally 35 percent, with a maximum 40.5 percent. The low extraction has resulted in haulage distances of up to 8 miles (13 km). In 1984 PCS Mining Ltd. completed a study of the potential for backfilling as a possible means of increasing extraction while maintaining the same or less subsidence induced strain in the Dawson Bay aquifer. In addition, the halite backfill will significantly reduce surface disposal of rock salt waste. The conclusion of this extensive and expensive study was that up to 840-ft (256 m) wide backfilled longwall panels could be mined with equal safety to the then current panels of three 65-ft (20 m) wide rooms separated by two 25-ft (7.6 m) wide yield pillars with 125-ft (38 m) wide panel barrier pillars.

The backfilled-longwall mining study was completed on October 3, 1984. On November 18, 1984 an inrush of brine started in a an 8-ft high (2.4 m) by 28-ft wide (8.5 m) development drift where it intersected a natural collapse structure. The 181 million Imperial gallon (823000 m^3) total inrush eventually reached a maximum flow rate of 5000 Igpm (22.7 m^3/min) on December 5, 1984 before being contained by an 87-foot (26.5 m) long bulkhead. The 1984 inrush put a halt to planning for backfilled longwall mining at the Rocanville Mine.

5) El Mochito Mine, 50 km southwest of San Pedro Sula, Honduras. The fault-controlled San Juan orebody has been developed over a height of 1125 ft (343 m) at the top of the 1400-ft (425 m) thick lower limestone member of the Atima Formation. It is immediately overlain by the approximately 350-ft (105 m) thick middle shale member, followed by the 1700-ft (520 m) thick upper limestone member. The middle shale functions as a barrier to water present in the upper limestone. The orebody is generally elliptical in shape, with horizontal dimensions ranging from 280 to 450 ft (85 to 140 m) E-W and 100 to 250 ft (30 to 75 m) N-S, between the 1600 and 2100 Levels.

The upper 625 ft (190 m) of the San Juan orebody was erratically mined with open stopes which were later incompletely filled with uncemented coarse development rock. It was obvious that the middle shale was breaking up and moving downward following the coarse rock fill. The potential exists for rupture of the middle shale and an inrush of water into the mine from the overlying upper limestone. Open stoping with delayed cemented fill is being used for mining the lower 500 ft (150 m) of the orebody.

The systematic open stopes are N-S, normal to the long axis of the orebody. Stope lengths are up to 120 ft (36 m) long, 40-ft (12 m) wide and 12-ft high (36 m). Thicker parts of the orebody are mined using two independent stopes, end to end. The backfill is surface quarried, crushed and cycloned limestone slurried, hydraulically transported and introduced at the top of a completed stope. Cement is added to the fine, +150 mesh - 1/4 in, fill in the proportions of 30 to 1. Mine development rock is dumped by LHDs into the top of the stope under fill. The weak fill wall that is exposed on at least one stope wall of all but the initial stope on a level has stood reasonably well, despite heavy blasting.

Considerable effort was expended in designing the 1600 Level crown pillar to support the uncemented coarse rock fill resting upon it. Cemented fill is planned for use for future mining of the crown pillar and upper ore remnants. Considerable ore was bypassed during initial open stope and rock fill mining of the upper 625 ft (190 m) of the San Juan orebody.

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