Design of backfill as support in Polish Coal Mines

by J. Palarski*

Introduction

Tailings backfill was introduced into Polish coal mines in 1893. Over the past hundred years, considerable work has been done on the assessment of optimum mixtures, the development of flexible delivery system, and the adjustment of fill technology to a wide variety of mining and geological conditions. Sand or industrial waste material for the backfilling of underground workings is used as a support mechanism to control ground deformations and surface subsidence.

Underground Mining Methods

The mining methods commonly used in Polish coal mines are illustrated in Figure 1. The long-wall method is employed in most coal seams, where coal is extracted from advancing or retreating faces, the faces usually being more than 150 m wide and 500 m long. The shape and advance of the winning face are defined in Figure 2. The fills used in coal-mining operations have evolved from loosely dumped rock (pillars constructed of rock and roll fill) and hydraulically placed sand fills, through pneumatic and stowing, to today's hydraulically transported, densified paste fills containing fly ash.

Nowadays, fly ash with a binder is injected, through a hole into cavity zones of the roof and small voids, to support the roof, to create an artificial hangingwall, to eliminate the movement of the rockmass, or to eliminate the fire hazard in abandoned mine workings.

Backfill Materials

The backfills currently used in Polish coal mines are produced from four raw materials:

- sand
- crushed and milled development waste rock
- processing-plant tailings (flotation waste and slime)
- power-station ash, fly ash, and slag.

Different combinations of these materials are also used. In 1992, about 86% of the backfill placed underground in longwalls was sand with crushed waste rock, 5% was crushed waste, 8% was a mixture of ash, fly ash, slag, and tailings or crushed waste rock, and 1% was drainage-free hydraulic fill.

Various combinations of materials are characterized to their particle-size distribution and optimum pipe-transport and placement properties.

The required properties of a backfill material depend on the geological and mining conditions. It is fundamentally important that cognizance is taken of the following in situ conditions:

- the confined compression and stress-strain behaviour, and closure in the stopes
- the percolation ratio
- the spreading of a backfill mixture and filling voids and the fill stability along the longwall
- the fill pressure on a backfill fence.

Stress-strain Behaviour and Compressive Strength of Backfill

The amount of strata deformation is reduced in proportion to the amount of backfill material added to the voids. The packing density and fill ratio of the material can vary significantly for any given particle-size distribution, and depend on the amount of water in the backfill mixture, the drainage technique, and the manner of placement.

It is generally agreed that the main factors determining the deformation of the overburden strata in longwall mining with backfill are as follows:

- the stress acting on the overburden strata, resulting from the redistribution of geostatic stresses during the mining of coal
- the seam height
- the compressibility of the backfill and the fill ratio
- the layout of the longwalls.
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Several different mixtures were tested to show the influence of backfill composition on its compressibility. From Figure 3 it is apparent that backfill materials containing a binder behave differently from the stress-strain behaviour of cohesionless backfill.

The maximum compressive strength of backfill containing binders depends on:
- the type of material, i.e. its particle-size distribution
- the type of content of the binder
- the water-to-solid ratio.

The test results showed that a mixture with low porosity, a minimum water content, and adequate active binder produces stronger backfill for a given amount of binder (Figures 4–6). Mixes stabilized with cement displayed higher early strength than those stabilized with materials containing CaO. However, the 5 to 6 day strength values were higher for the mixes containing binders other than cement.

Based on the results summarized in Figure 7, the highest strength for the backfill was obtained when alpha semi-hydrate (CaSO₄·½H₂O) was used as a binder. Mixes with 50% alpha semi-hydrate and 50% crushed waste rock or 50% fly ash achieved compressive strengths of 8 MPa and 16 MPa respectively in one hour.

Rapid setting of the backfill eliminates the need for drainage by locking water within the backfill mass. An addition of fly ash to the alpha semi-hydrate improved the setting time (hardening time), as shown in Figure 8.
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Figure 3—Stress-strain curves for various backfills

Figure 4—28-day strengths of a backfill containing: 5 per cent fly ash, 95 per cent tailings, cement, and water

Figure 5—Strength of a cemented rock-waste fill versus the fines content

Figure 6—Compressive strength of stabilized backfill

Figure 7—Compressive strength of stabilized backfill

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At the beginning, the fill is usually in a saturated state. During the fill placement water drains continuously through the fill and the highly permeable fences (barricades). Water is removed from the placed fill by two mechanisms. After the fill materials have settled and consolidated soon after their placement, excess water accumulates on the fill surface. The ‘drainage windows’, (timber raises), allow the surface water to drain away rapidly. It is known that drainage reduces the pore-water pressures within the settled fill. The distribution of pore-water pressure is a function of the fill composition, fill capacity, geometry of the paddock, and the location of drainage gates or ‘drainage windows’.

The tests showed that the pore-water near the high permeable fence were low. The maximum pore-water pressures were recorded at the centre of the backfill area, about 25 to 40 m behind the fill paddock (Figure 9). Further away, the pore-water pressure dropped sharply and became negative.

Backfill in a coal mine frequently consists of crushed or milled waste and sand, i.e. cohesion-less material. The shear strength of a granular material is determined directly by the pore-water pressure according to the effective stress law. Therefore, in fill design and mine planning, it is important to ensure that significant pore-water cannot develop in a backfill area. High pore-water pressure leads to a complete loss of shear resistance and to subsequent liquefaction of a fill material.

The permeability of a fill determines the drainage condition. Investigations indicate that successful fill materials have a permeability coefficient in the range (2 to 10) x 10⁻⁵ m/s.

Drainage of the Fill in Mine Stipes

Hydraulic fill is transported to longwalls through pipelines in the form of a slurry typically containing 50 per cent water (by volume). It is not desirable for the water introduced with the slurry to be retained within the fill void. Small particles and binders added to increase the strength of the fill are leached away as the water percolates downwards. Variations in material properties, such as porosity and hydraulic conductivity, may significantly effect the drainage process.

In longwall mining, backfill is placed in narrow, extensive paddocks built of timber and geotextile, where high stope compression and, consequently, high stresses in the fill air generated. The backfill paddocks are built and filled behind mechanized support as the face advances every 1.8 to 3.6 m. The longwall moves up dip. It is known that, as the porosity of a backfill under pressure is reduced, the fill materials display stiffer response.

Figure 8—Setting time as a function of water-to-solids ratio

Figure 9—Distribution of pore water pressure in filled void
Longwall: width 180 m, dip 18°, height 2.8 m
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The investigated fill materials were produced from fly ash, slag, and processing waste, or a mixture of these components. Hydraulic placement of these materials results in a loose fill structure with a porosity of these materials results in a loose fill structure with a porosity of 62 per cent. Figure 10 shows the percolation rates for different backfill materials.

1. Flotation tailings 0-3mm 75%, slag 25%
2. Sand 100%
3. Slag 100%
4. Flotation tailings 1-3mm 100%
5. Flotation tailings 3-6mm 100%
6. Flotation tailings 3-6mm 50%, fly ash 50%
7. Flotation tailings 0-6mm 33%, slag 33%, fly ash 33%

Figure 10—Percolation rates for various backfill materials

Underground, a backfill of low water content does not display any apparent cohesion, which is necessary to allow free-standing vertical walls of fill. The slurry is discharged into the paddock every 8 to 12 m to fill up the void homogeneously completely.

Various processes occur with hydraulic fill, and variations in the backfill material can lead to a non-homogeneous fill. During the placement of crushed waste rock mixed with fine materials, segregation occurs, with the coarser particles settling close to the discharge point. Further heterogeneity of the fill arises from different local settling rates and from the differences in the impact and compaction of coarse and fine particles (Figure 11).

Fill Pressure

The drainage gates are fenced from the fill area through side barricades or packs, which allow the fill to drain and are designed to give resistance to the backfill pressure. The packs are built to isolate the gates from the fill area, where dangerous materials are disposed. In that case, water is prevented from draining out of the fill area. The packs are designed to give sufficient resistance to the roof and to the fill pressure.

The total pressure after drainage in the hydraulic backfill acting normal to the fence is a sum of the silo effect and the convergence components. Pressure loads that were measured at the barricades during the placement of fill are reported in Figure 12.

The measurements show that, for a small height of fill, the pressure increases rapidly (hydrostatic head) when the face is within 25 to 40 m away (at a seam dip 18 degrees), and reaches its peak values from 0.75 \( \rho_g \) to \( \rho_o \) \( \rho_g \) \( \rho_o \) \( \rho_o \) (\( \rho_o \) = density of the mixture). Further, the pressure decrease with the placement of fill. After reaching a minimum at a fill distance of about 50 to 60 m behind the face, the pressure rises slowly again. This increase of pressure at the barricade is caused by the weight of the overlying strata, which is incompletely transferred to the backfill in the fenced void. However, the location of the peak pressure varies from area to area owing to differences in backfill compositions and strata characteristics.

Distribution of Vertical Stresses and Closure around a Longwall

Backfill was first used to control surface subsidence. The modern structural function of backfill is to facilitate full mining of coal without loss of control of the rockmass.
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As local support, backfill can prevent spatially progressive disintegration of the fractured rockmass, reduce rockburst damage, and provide better convergence control. If backfill is properly placed and confined, it can act as a global support element in the mine structure.

Figure 13 shows the distribution of vertical stresses and closure around longwall coal faces with caving and with backfill. In an infilled longwall, the vertical stress rapidly increases with the distance into the yield zone in the unmined coal, and reaches its peak 1 to 5 m ahead of the face. The peak pressure ranges from 3 to 6 times the overburden stress. In filled stopes, the stress reaches its peak 8 to 20 m ahead of the face. The peak stress so produced is two to four times the overburden stress, and depends on the stiffness of the backfill. In both cases, with increasing distance into the unmined coal, the vertical stress decreases towards the overburden stress.

In the mined-out area in unfilled and filled stopes, the vertical stress increases with distance from the face and rib side. The pressure increases more slowly in an unfilled area than in a filled area.

The closure rates of the immediate roof strata in a filled longwall are significantly lower than the closure of the main roof in an unfilled longwall.

Fill Technology

Preparation of the Fill

Various types of fill and placement methods are used in Polish coal mines. Details of the fill types, the preparation, and the required properties are given in the literature\(^1,2\) and only a brief description of the fill technology is given here.

A backfill mixture that contains sand or crushed waste is prepared within a simple plant, where the material is washed out of a tank by water jets or is fed mechanically onto screens to which water is added. The mixture thus obtained flows through screens to a hopper, where water is supplied in the quantity needed to give the required concentration.

A multi-component slurry containing binders is prepared in mixers to which all the components are supplied in the required quantities. From the mixer, the slurry is fed into a hopper and then flows by gravity through pipelines to the mine workings.
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Gravitational Transportation of Backfill Slurry

In coal mines, pipelines connect the backfill plant on the surface to the underground workings. These pipelines form a network to the shafts, which are equipped with at least two pipelines, one of them being a reserve. Horizontal pipelines can branch but, during backfilling, the stream must not divide into two or more parts. In workings such as a longwall, the pipeline branches out every 8 to 16 m, and the backfill mixture is directed through these branches to the void that is to be filled (Figure 14).

The flow of slurry through a gravitational pipeline can be described by the modified Bernoulli equation on the assumption that the concentration of slurry does not change along and across the pipeline.

\[ p + h \rho_m g + \frac{v^2}{2} \rho_m + \Delta p = \text{const}, \]  \[ \text{[1]} \]

where:
- \( p \) = pressure
- \( h \) = depth
- \( \rho_m \) = density of mixture
- \( v \) = velocity of flow
- \( \Delta p \) = elementary resistance of flow
- \( l \) = length
- \( g \) = gravity

When the formula for the conservation of energy, equation [1], is applied to the input and output of a pipeline, the equation for the flow of backfill slurry changes to

\[ H \rho_m g = \sum_{i=1}^{n} \Delta p_i l_i. \]  \[ \text{[2]} \]

It was established by the derivation of this formula that the difference in atmospheric pressures between the input and output section of the \( (P_i, P_o) \) and the value of the pressure \( \frac{v^2}{2} \rho_m \) is extremely small. Based on the same principle for an input at a certain point in the pipeline \( P(P_i, h_p) \), the equation for pressure can be presented as

\[ H \rho_m g = P_i + h_p \rho_m g + \frac{v_p^2}{2} \rho_m + \sum_{i=1}^{k} \Delta p_i l_i. \]  \[ \text{[3]} \]

where:
- \( H \) = total depth of the pipeline
- \( h_p \) = depth of point \( P \) in the pipeline
- \( l_p \) = length of pipeline from an input to point \( P \)
- \( v_p \) = velocity of low in point \( P \)
- \( n \) = total number of parts into which the pipeline is divided
- \( k \) = number of parts of pipeline from an input to point \( P \)
- \( P_i \) = pressure at point \( P \).

To solve these equations, the equation describing the elementary resistance of flow in a pipeline, \( \Delta p \), must be known. In the gravitational transportation of backfill material, the flow is turbulent. The models for particular mixtures, which were established on the basis of laboratory measurements at industrial plants, are given in equation [4] to [6], the parameters being defined after equation [6].

(a) For a slurry containing and and up to 50 per cent crushed waste rock with a maximum particle size of 50 mm.

\[ \Delta p = \lambda_w \frac{v^2}{2D} \rho_w \]

\[ + \frac{c}{v} \frac{d_s}{D} \rho_s g \]  \[ \text{[4]} \]

(b) For a slurry containing sand, crushed waste rock, furnace slag, and fly ash in a vertical pipeline:

\[ \Delta p = \lambda_w \frac{v^2}{2D} \rho_w + \lambda_{fs} \frac{v^2}{2D} \rho_{fs} C_v \]

\[ + a_{c,s} \frac{d_s}{D} \rho_s g \]  \[ \text{[5a]} \]

(c) For slurry (b) but in a horizontal pipeline,
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\[ \Delta p = \frac{\lambda_w}{2D} \rho_w + \frac{\lambda_t}{2D} \rho_w x_t C_v \]

\[ a, c, s, w, d_s, D, f_t, f_w, \rho_w, \gamma \]

(d) For a slurry containing flotation tailings, fly ash, and different binders:

\[ \Delta p = \frac{\lambda_w}{2D} \rho_w (1 - C_v) \]

\[ + \frac{\lambda_t}{2D} \rho_t C_v (\text{Pa/m}) \]

The parameters used in equations [4] to [6] are as follows:

- \( a_i = \rho_i - \rho_w \),
- \( a_s = \rho_s - \rho_w \),
- \( a_d = \rho_d - \rho_w \),
- \( C_v = \frac{\rho_d - \rho_w}{\rho_s - \rho_w} \) — concentration by volume

\( d_s \) = diameter of particles
\( D \) = diameter of pipeline
\( f_t \) = coefficient of friction for waste rock (0.34 to 0.38)
\( f_w \) = coefficient of friction for furnace slag (0.40 to 0.43)
\( k \) = coefficient (0.45 to 0.50)
\( x_{fa} \) = quantity of fly ash
\( x_r \) = quantity of waste rock in dry mixture
\( x_s \) = quantity of sand in dry mixture
\( x_{fa} \) = quantity of furnace slag in dry mixture
\( w_s \) = sedimentation velocity
\( \lambda_f \) = coefficient of resistance for the mixture of flotation tailings, fly ash, and additives (0.022 to 0.026)
\( \lambda_w \) = coefficient of resistance for fly ash (0.020 to 0.023)
\( \nu_w \) = coefficient of resistance for water (0.012 to 0.017)
\( \rho_t \) = density of the mixture of flotation tailings, fly ash, and additives
\( \rho_w \) = density of water.

From these equations, the velocity of flow can be calculated. The intensity of flow and other effective parameters are calculated as follows:

Intensity of a flow of slurry, \( Q_m \):

\[ Q_m = \frac{nD^4 v}{\gamma} \] (m³/s) [7]

Efficiency of transportation of solid bodies, \( Q_s \):

\[ Q_s = C_v Q_m \] (m³/s) [8]

Efficiency of filling the voids, \( Q_p \):

\[ Q_p = \rho_t Q_m \] (m³/s) [9]

where: \( \rho_t \) (the empirical efficiency coefficient) = 0.859 (\( \rho_w - 1 \)).

The following two conditions are the deciding criteria for the design parameters of the mixture and the pipeline.

\[ M v_c < v < v_{fo} \text{ and } P_{vp} < P < P_{te}, \]

where: \( M \) = flow index (1.1 to 1.3)
\( v_c \) = critical velocity of flow
\( v_{fo} \) = maximum velocity (8 to 10 m/s), i.e. maximum acceptable velocity on account of wear of the pipe wall and reliability of the flow
\( P_{vp} \) = saturated vapour pressure
\( P_{te} \) = tear strength for established thickness of pipe wall.
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References


Figure 15 shows the efficiency of filling from a gravitational pipeline in relation to the concentration of slurry and type of fill material. It is apparent that the filling efficiency increases with an increase in the concentration of fine-grained materials. For coarse-grained materials, the efficiency increases only up to a certain concentration. Figure 15 shows curves for both the capacity of the mixture flow and for critical values of the mixture flow. Flow is impossible for concentrations greater than the point of intersection of the flow curves.

The distribution of pressure in a pipeline depends mainly on the spatial arrangement of the pipeline, its diameter, and on the parameters of the mixture being transported.

Flushing of a Pipeline

The gravitational transportation of mixtures begins and ends with the flushing of the pipeline. Flushing is unnecessary only when the mixture being transported does not settle in the pipeline after the flow ceases. Flushing means that water is delivered to a fill area, and it must be drained back to the surface.

Measurement, Control, and Optimization in Mineral Processing

The various facets of automated control have collectively constituted the most rapidly advancing aspect of mineral processing in recent years. The advent of more affordable, robust process-control computers has changed advanced instrumentation from being an esoteric 'add-on' to being an essential part of any modern mineral-processing complex. The development of modelling, control, and optimization techniques has also gained tremendous impetus from the more ready availability of plant monitoring and control systems.

Although various aspects of process measurement, control, and optimization are addressed in many mineral-processing publications, it was the opinion of The South African Institute of Mining and Metallurgy that there was no single publication that addressed this field on a practical plant-design and operational level. It was thus decided to compile a suitable publication comprising selected edited papers from the SAIMM Schools held during October 1987 and August 1993. For this publication to have practical benefit to the design and operation of mineral-processing plants, it was considered essential not only that principles be addressed, but that specific techniques and equipment be described.

Although the disadvantage of techniques and equipment becoming dated is acknowledged, as is the case in any fast-changing technology, it is felt that the insight provided by this publication will remain applicable for many years to come—provided that topical assistance is requested form the relevant technology supplier. Every attempt has been made to maintain the contents at a technical level below that of a professional control engineer, but commensurate with the requirements of engineers of other disciplines.

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