REVIEW OF FILL MINING TECHNOLOGY IN CANADA

by

K.H. SINGH
Falconbridge Nickel Mines Limited

and

D.G.F. HEDLEY
CANMET
Dept. Energy, Mines & Resources

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SYNOPSIS

The Canadian mining industry has a long history of being in the forefront in developing new technology in underground hardrock mines. Examples include the development of hydraulic and cemented fills, undercut-and-fill, mechanized cut-and-fill, post pillar, vertical retreat and blasthole mining methods. The evolution of this technology is briefly described in an historical review.

Backfill serves many functions, although it is generally considered in terms of its support capabilities. These functions, mainly related to the mining method used, are evaluated in regard to regional support, pillar support, fill roof, working floor, dilution control and waste disposal.

With the advent of blasthole and vertical retreat methods for pillar recovery operations, the free-standing height of backfill walls has assumed greater importance. Consequently, more attention is being given to what fill properties are required to achieve fill wall exposures up to 25 m wide by 90 m high. With the large increases in energy costs, alternatives to partially replace Portland cement in fill are being examined.

The validation of mining concepts and the interaction of backfill is perhaps best evaluated by in-situ measurements. Examples are given of stress, deformation and fill pressure measurements in longitudinal cut-and-fill,
post pillar mining and blasthole stoping with delayed fill which were taken in several mines in Canada.

Finally, the overall design procedure used in deciding mining method, stope and pillar dimensions, sequence of extraction, fill properties and support systems at a new mine is described.
INTRODUCTION

Cut-and-fill mining is associated with Canada as sub-level caving is with Sweden and block caving with the United States. In many ways, over the last 50 years, Canadian mines have been in the forefront in developing cut-and-fill techniques, including undercut-and-fill, post pillar and vertical retreat mining methods. This development and innovation is still proceeding, to further improve the efficiency and productivity in mining especially as the mines go deeper. A recent trend has been the use of blasthole stoping methods with delayed bulk filling to replace, in certain applications, conventional cut-and-fill techniques.

Although research and development, including rock mechanics, contributed to these past developments it was mainly an engineering approach that was used, relying to a large extent on trial-and-error methods. Canadian hard rock mines are now extensively mechanized and there has been a large increase in productivity at the working face, but this has not resulted in any noticeable increase in productivity on a mine-wide basis. It is apparent that research and development as well as design and planning has to encompass the whole mining operation from development to hoisting. Consequently, rock mechanics research is being increasingly applied to the planning stage to investigate such aspects as stope and pillar dimensions and sequence of extraction rather than primarily as a corrective tool.

In 1978 the Canadian National Committee on Rock Mechanics organized a symposium on "Mining With Backfill" in Sudbury (1). State-of-the-art reviews were presented on fill mining practice in Canada and other countries as well as recent research and innovations in backfilling. We have drawn on the information presented at this symposium.
In this review of Canadian fill mining technology we have taken the broadest definition and included all mining methods which utilize fill. A brief historical review highlights the developments in Canada and how they are inter-related in new mining methods. The various functions of fill are described in terms of the mining methods used. Examples are given on research in Canada on fill properties, and in-situ measurements. Finally, possible future trends in mining technology are discussed.

HISTORICAL REVIEW

In the 1930's many of the major mining camps in Canada were already in production; Sudbury, Kirkland Lake and Porcupine areas in Ontario, Noranda in Quebec, Flin Flon in Manitoba and Kimberley in British Columbia. Major mining methods, at that time, were open and shrinkage stoping for primary extraction and square set mining for pillar recovery operations and for primary stoping in difficult ground conditions. Waste rock, surface sand and gravel were used as backfill material, but mainly as a supplement to timber support rather than as a primary support system. Backfill was usually transported in raises or in mine cars down the shaft, trammed along the levels and emptied into the stopes via raises.

In 1933 experiments started at the Horne Mine at Noranda using granulated reverberatory furnace slag and pyrrhotite tailings (2). It was found that the oxidation of the pyrrhotite consolidated the slag and produced a backfill very similar to the cemented fills used today. Drifts could be driven through this consolidated fill without support and pillars could be mined against the fill without significant dilution. Unfortunately, although other mining companies attempted to duplicate this type of pyrrhotite cemented fill, it appeared to be unique to the Horne and Quemont mines at Noranda.
Further development of backfill systems was limited during the Second World War and the next major development was the installation, in 1948, of an hydraulic system for transporting classified mill tailings via pipelines at INCO's Frood-Stobie Mine in Sudbury (3). This allowed fill to be placed tight in the stopes, resulting in better ground conditions and reducing the amount of subsidence that occurred around loose filled stopes. This technique was soon adopted by other mining companies and further developments included, using cyclones to classify mill tailings and a borehole transportation system at Madsen Red Lake in 1950 (4).

Also at INCO's Frood-Stobie Mine undercut-and-fill methods were developed in 1956 for mining broken-up rib pillars (5). In the early 1960's Falconbridge Nickel Mines Ltd adopted undercut-and-fill methods for recovering longitudinal pillars. These methods soon replaced the more expensive and less efficient square set mining.

One of the major developments in the 1950's was the introduction of rock bolts into Canadian hardrock mines, which rapidly superseded timber supports (6). The improved ground conditions with hydraulic fill in conjunction with rock bolts permitted the changeover from square set to cut-and-fill mining in certain applications. Also, the larger stope spans and unobstructed workplace allowed the introduction of larger pieces of equipment, such as slushers, at the stope face.

Although hydraulic backfill was an improvement over the loose fills used previously, it lacked cohesion. When exposed over limited vertical heights, sloughing occurred causing dilution. Also, during pillar filling operations, washouts of the stope fill caused major problems. In the late 1950's and early 1960's extensive research was undertaken by INCO Ltd., Falconbridge Nickel Mines Ltd. and Canada Cement Company Ltd., on stabilizing backfill by adding small quantities of Portland cement (7,8,9). From this research came the general practice of using 30:1 sand-cement ratio for bulk filling and 10:1 ratio for stope floors. These cemented sand fills could be
exposed over vertical heights of 60 metres without significant sloughing. Enriched cement floors improved slusher operations and allowed more efficient clean-up of metal fines. Its use continued with widespread introduction of rubber tired load-haul-dump (LHD) units in the late 1960's and 1970's. Cemented fill also improved ground conditions in both cut-and-fill and undercut-and-fill operations and the use of timber was further reduced.

In the mid-1960's a significant change took place in the layout and types of equipment used in cut-and-fill stopes. Conventional cut-and-fill mining used jacklegs for drilling and slushers for mucking to ore passes within each stope. The equipment was captive and the stopes were isolated from each other. In mechanized cut-and-fill mining, multi-drill jumbos were used for breasting or uppers drilling and LHD units for mucking. Stopes were now interconnected to take advantage of the mobility of this new equipment and some mines installed a spiral ramp system and a central ore pass to service a block of stopes. This resulted in a significant increase in stoping efficiency and productivity over the conventional cut-and-fill methods (10).

At the Geco Mine of Noranda Ltd. a unique filling system was developed in the 1960's (11). Quarried rock was introduced into the top of a stope as the broken ore was pulled down. On completion of mucking, the rockfill was consolidated with a cement-tailings slurry. The resultant cemented rockfill was self supporting when mining the adjacent pillar (12). Cable bolts at the Geco Mine and long bar bolts at the adjacent Willroy Mines Ltd were first used during the 1960's to reinforce the stope back and walls and the practice was adopted by other mines.

During the 1970's the development of improved mining methods and new equipment has proceeded at a more rapid pace. Falconbridge Nickel Mines Ltd developed post pillar mining at their Strathcona Mine which maximizes the extraction from primary stoping (13). Large hole drilling using either rotary or in-the-hole drills were used in blasthole stoping (14,15). A
survey in 1977 indicated that 1240 diesel-powered load-haul-dump units were being used in Canadian underground mines and at least 75% of the tonnage was being handled by these machines. Recently, electric (16,17) and remote controlled (18) load-haul-dump units have been introduced.

To replace undercut-and-fill methods for recovery of rib pillars between backfilled stopes, a vertical retreat method was developed by Canadian Industries Ltd. in conjunction with INCO Metals Company and Falconbridge Nickel Mines Ltd. (19,20). This method has also been used for primary stoping in a narrow steeply-dipping orebody at the Centennial Mine of Hudson Bay Mining and Smelting Co. Ltd. (21).

In the mid-1970's both Falconbridge Nickel Mines Ltd. and INCO Metals Company again turned their attention to slag, but this time as a substitute for expensive cement to stabilize sandfill (22,23).

This brief review clearly indicates that the development of more efficient mining methods was dependent on better types of backfill (i.e., cemented), improved support systems (i.e., rock bolts) and new mining equipment (i.e., load-haul-dump units, multi-drill jumbos, large hole drills). Also these developments were inter-dependent. The evolution of fill mining methods is illustrated in Fig. 1 which shows the tonnage mined by various methods in the Sudbury Basin by INCO Metals Company over the last 40 years (23). As can be seen, the introduction of cut-and-fill and then undercut-and-fill eliminated square set mining. Also in the late 1960's and 1970's mechanized cut-and-fill replaced conventional cut-and-fill. A specific example of the evolution in mining methods is in the recovery of rib pillars between backfilled stopes. In the 1960's undercut-and-fill replaced square set mining and by the late 1970's vertical retreat mining started to replace undercut-and-fill methods.
Fig. 1: Evolution of fill mining methods at the Ontario division of INCO Metals (after Barsotti, (23)).

Table 1 - Mining Methods in Canadian Metal Mines

<table>
<thead>
<tr>
<th>Mining Method</th>
<th>Cut &amp; Fill</th>
<th>Blasthole</th>
<th>Shrinkage</th>
<th>Room &amp; Pillar</th>
<th>Sub-Level Caving</th>
</tr>
</thead>
<tbody>
<tr>
<td>1978 Tonnage*</td>
<td>17,800,000</td>
<td>25,800,000</td>
<td>1,400,000</td>
<td>6,700,000</td>
<td>4,700,000</td>
</tr>
<tr>
<td>%</td>
<td>32</td>
<td>46</td>
<td>2</td>
<td>12</td>
<td>8</td>
</tr>
<tr>
<td>Number of Mines</td>
<td>41</td>
<td>41</td>
<td>15</td>
<td>9</td>
<td>6</td>
</tr>
</tbody>
</table>

* 1977 tonnage for INCO Metals Company and Falconbridge Nickel Mines Limited

In 1978, about 56,400,000 metric tonnes were extracted from Canadian underground metal mines. The distribution by mining method is given in Table 1. Blasthole stoping is the major mining method accounting for 46% of the total tonnage, however, 9,600,000 tonnes or 17% comes from blasthole with delayed bulk filling. Incremental cut-and-fill methods are next in importance and account for 32% of production. There has been a significant change in mining methods since and as recently as 1974, when cut-and-
fill methods accounted for about 50% of production and blasthole stoping about 30% (24).

Until recently, the general practice in Canadian mines was to convert to cut-and-fill mining below a depth of about 1000 m. Now blasthole stoping with delayed backfilling is being used to a depth of almost 2000 m at both INCO's Creighton Mine and Falconbridge's East Mine. Consequently, a future development could be the further replacement of incremental cut-and-fill methods by blasthole stoping.

ROLE OF FILL

The reasons for putting backfill underground in Canada range from providing regional support to disposal of a waste product. Consequently, fill serves many functions although it is generally considered in terms of its support capabilities. Other than its own body weight, backfill is a passive support system, in that it has to be compressed before exerting a restraining force. Many theoretical studies have shown that backfill has little effect on the stress distribution in the surrounding rock strata, due to the large difference in deformation modulus between rock and backfill. However, backfill can have a considerable effect on the strength of a rock structure even if it only prevents the rock from unravelling. This allows the rock itself, although fractured, to continue to support the major portion of the applied load. A common observation underground is that ground conditions in a stope with cemented fill are better than with hydraulic sand which in turn is better than a loose sand and gravel fill.

The various functions of fill as related to mining methods are briefly described below:
REGIONAL SUPPORT

An example of where fill provides regional support is in longitudinal cut-and-fill mining in narrow steeply-dipping orebodies. The fill prevents break up and sloughing of the hanging wall and footwall. Due to wall closure load is transmitted through the fill which reduces the stress concentration in the sill and rib pillars. This is perhaps the one case where fill can exert a reasonable support pressure, up to about 5 MPa. If the stope is considered to be a thin slot, then the closure of the hanging wall and footwall is mainly due to the release of the perpendicular stress. Hence closure is independent of stope thickness and the narrower the orebodies the greater the strain and consequently the stress in the fill. To maximize regional support the fill should be placed as soon as possible after mining to take advantage of any wall closure. Deformation modulus is the most important fill property in regional support and it increases with cement content. Alternatively a cemented rock fill would have a higher modulus than a cemented sandfill.

PILLAR SUPPORT

Rib pillars in transverse cut-and-fill and post pillars are generally fractured and have yielded (i.e., exceeded their peak strength) at a fairly early stage in the mining sequence. A general rule of thumb would be that initial pillar yield occurs when their height/width ratio is between 2 and 3 at a depth of about 1000 m. The role of fill in this situation is to prevent disintegration or unravelling so that the pillars can continue to support the immediate roof strata rather than the total overlying strata. In this case the constraint provided by the fill need not be as great as that required for regional support and up to 1 MPa is probably sufficient. Cement requirements are also reduced and 30:1 is commonly used in transverse cut-and-fill and post pillar mining in Canada. In fact, cement content could be more related to the
planned method of pillar recovery than to primary stoping operations.

FILL ROOF

In undercut-and-fill methods, backfill in conjunction with a wooden mat forms the roof of a stope. This method is used to recover rib pillars between previously backfilled stopes and occasionally as a primary stoping method in very friable ore where a stable roof cannot be maintained. When the wooden mat is undermined it deflects resulting in a redistribution of pressure to the adjacent fill. Consequently, the wooden mat only supports a fraction of the weight of the overlying fill. Using the concept of a timbered tunnel in sand (25) or drawing of caved ore (26) the load supported by the wooden mat can be estimated. The cohesive strength of the fill is the most important property and as it decreases the load increases. Cohesive strength of fill is dependent on cement content and normal practice in Canada is to use 20:1 in undercut-and-fill operations. In the era before cemented fills, failure and runs of unconsolidated hydraulic fill were fairly frequent in undercut-and-fill mining (23).

WORKING FLOOR

Before the consolidation of fill with cement, wooden planks and round lagging were normally placed on top of the fill to separate the broken ore during slushing operations in transverse and longitudinal cut-and-fill mining. Now practice in Canadian mines is to enrich the last 30 cm of the fill pour with 5:1 to 10:1 fill-cement mix. This allows the use of load-haul-dump units, which fully loaded weigh up to 40 tonnes, to operate on top of the fill. The important property of the fill for this function is its bearing strength which increases with cement content. However, cement affects the drainage characteristics of the fill which in turn affects how soon the equipment can operate on top of the fill.
The role of fill is especially important in pillar recovery between previously backfilled stopes either by blasthole, vertical retreat or undercut-and-fill methods. Besides supporting the hanging wall and footwall, it fills the void preventing scattering of the blasted ore. The backfill itself must be capable of standing unsupported over large vertical exposures which can be 100 m high in blasthole mining. The free-standing height of fill is mainly dependent on the cohesive strength, which is affected by the cement content, angle of internal friction, and the density of the fill. In blasthole mining a bulk sand fill of 15:1 is generally used, although one Canadian mine used a 20:1 cemented rockfill.

At some Canadian mines the mill tailings contain substances which are a potential health hazard (e.g., radium) and the volume of tailings may be such that surface disposal sites are inadequate. In these cases returning part of the tailings to underground worked-out stopes is being considered. The primary purpose of backfill is waste disposal and the support characteristics are secondary.

In the late 1950's, Canadian mining companies and a cement manufacturer embarked upon a major research program to investigate engineering properties of hydraulic fill (7,8,9). Most of the work was carried out in the laboratory and has not yet been published in detail. These studies enabled some practical conclusions to be drawn on the properties of fill. The most important of these were:
(a) Mill tailings need desliming to achieve acceptable percolation rates (10 cm/hr.) for drainage control.

(b) An addition of 8 to 12% cement to fill allows development of good mining floors.

(c) Lean cement-tailings mixes (3 to 4% cement) permit elimination of gob fences during pillar extraction and contribute towards improved ground conditions.

(d) Rich cement-tailings mixes (usually 16% cement) allow safe mining with undercut and fill methods.

Figure 2 presents a typical relationship between cement content and engineering properties such as cohesion, shear and uniaxial compressive strength. These results obtained in the laboratory are based on Falconbridge mill tailings (27).

More recent tests carried out at the Strathcona Mine of Falconbridge Nickel Mines are shown in Table 2 (28). These results are more representative of in-situ fill properties when the backfill is allowed to settle for a long period of time. This test work indicated that bulk density varied with depth. In addition the material was found to be heterogeneous with respect to compressive strength, gradation, moisture content and cement content.

The in-situ properties of fill and its behaviour under stress are receiving increasing attention as Canadian mines utilize mining methods which expose fill walls over considerable heights and widths without causing any significant dilution.

As mentioned in the historical review, both INCO Metals Company and Falconbridge Nickel Mines Ltd. have been conducting laboratory tests on the partial substitution of slag for Portland cement to stabilize backfill. INCO's research indicated that a mixture of 4% cement, 8% air-cooled nickel
Fig. 2: Compressive, shear and cohesive strengths versus cement contents.

Fig. 3: In-situ measuring procedures and fill percolation rates.
reverberatory slag ground at 3500 to 4000 Blaine and 88% tailing should produce a backfill capable of standing unsupported up to 60 m high (23). Falconbridge's research indicated that granulated nickel-copper electric furnace slag ground at 3000 Blaine could be mixed in a 1:1 ratio with cement in lean concentrations to produce a suitable bulk fill (22).

Table 2 - In-Situ Properties

| Fill Property                                           | Value
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Tailing:Cement Ratio</td>
<td>32:1</td>
</tr>
<tr>
<td>Water Content %</td>
<td>19</td>
</tr>
<tr>
<td>Void Ratio</td>
<td>0.51</td>
</tr>
<tr>
<td>Bulk Density</td>
<td>2.08</td>
</tr>
<tr>
<td>Uniaxial Compressure Strength MPa</td>
<td>0.60</td>
</tr>
<tr>
<td>Young's Modulus MPa</td>
<td>82</td>
</tr>
<tr>
<td>Poisson's Ratio</td>
<td>0.3</td>
</tr>
<tr>
<td>Shear Modulus MPa</td>
<td>16</td>
</tr>
<tr>
<td>Cohesion MPa</td>
<td>0.14</td>
</tr>
<tr>
<td>Friction Angle</td>
<td>40°</td>
</tr>
</tbody>
</table>

Laboratory testing is normally employed to determine the drainage properties of mine backfill. Recently, in-situ measurements have been taken in Canadian mines using three types of permeameters which utilize electrodes and an electrolytic solution (29). The measuring principles are illustrated in Figure 3 together with the percolation rates measured at two sites. Laboratory percolation rates on the same backfill are also listed. Standardized to 20°C and 100% saturation, the percolation rates from the three in-situ measuring procedures are compatible, whereas, the laboratory percolation rate is much lower. However, when adjusting for the difference
in porosity between the in-situ and laboratory backfill samples, the results are in much closer agreement.

Increasing use of large hole blasting methods and high exposure of fill walls has necessitated some attention to dynamic response of in-situ backfill, however, very little research has been done on the dynamic properties of backfill material. In test work in an operating mine, blast vibration monitoring instruments were employed in a backfilled stope (32:1 tailing cement ratio) subjected to vibrations caused by blasting of 160 mm diameter drill holes. The following general conclusions were drawn:

(a) Vibrations measured in the fill are subjected to higher rates of attenuation than those measured in massive rock formation. Peak particle velocity was found to be one fourth of that measured in rock.

(b) Frequencies in the fill ranged from 30-50 Hz as compared to 400-600 Hz commonly found in rock under similar blasting conditions.

(c) No failure was reported due to tensile spalling.

IN-SITU MEASUREMENTS

LONGITUDINAL CUT-AND-FILL

In narrow, steeply-dipping orebodies longitudinal cut-and-fill methods are normally employed, although recently vertical retreat methods have been tried out (21). As mentioned previously, the fill can be subjected to much higher pressures than in other cut-and-fill methods and the higher the pressure on the fill the less the stress concentration on the sill and crown pillars. Fig. 4(a) and (b) show the measurements in one longitudinal cut-and-fill stope at a depth of 900 m at INCO's Garson Mine (30).
Fig. 4: Measurements in a longitudinal cut-and-fill stope at the Garson Mine of INCO Metals Company.
Alluvial sand was used as backfill with the last 30 cm being 10:1 cemented fill to provide a working floor. Fig. 4(a) shows the relationship between fill pressure and stope closure measured in the cemented and uncemented fill. The pressure on the cemented fill, which also had a narrower stope width, is much higher than on uncemented fill. Also the deformation modulus is about 5 times higher, although still only about 0.5% that of the rock mass.

Fig. 4(b) shows the fill pressure measurements at four locations along the stope and the hanging wall deformation from borehole extensometers near the ends of the stope. In this case all the fill pressure cells were in uncemented fill. Near the centre of the stope fill pressure is fairly uniform at 600 to 650 kPa, also at one location where 3 cells were spaced at different positions across the width, the pressures were almost identical. Again the greatest fill pressure of 1500 kPa was measured at the narrowest part of the stope. Movement of the hanging wall towards the stope is plotted relative to the end anchor in the boreholes and a positive gradient indicates compression and a negative gradient expansion. As shown, the first 3-4 m of the hanging wall is being compressed against the fill, surrounded by an expansion zone to a depth of about 15 m. Probably, near the centre of the stope the compression and expansion zones will be somewhat larger.

Fig. 5 shows the relationship between depth of fill cover and the measured vertical and horizontal (hanging wall to footwall) pressure measured in backfill at INCO's Garson Mine and the adjacent Falconbridge Mine. The main parameters are listed beneath the Figure. There is reasonable agreement between the vertical fill pressures measured at both mines, but they are both higher than would be expected from the bulk density of the fills. However, in-situ sampling at one site has indicated that bulk density increases by about 13 kg/m$^3$ per metre depth (28). Horizontal fill pressures are site
Fig. 5: Vertical and horizontal fill pressures in longitudinal cut-and-fill stopes.
specific, being higher than the vertical pressure at the Garson Mine and lower at the Falconbridge Mine. The pressure cells at the Garson Mine were located near the mid-height of the stope and hence were subjected to the maximum convergence of the hanging wall and footwall. At the Falconbridge Mine the cells were located about 4 m above the sill pillar which would limit the amount of wall convergence.

POST PILLAR MINING

Measurements of pillar stress, deformation and fill pressure have been taken in post-pillar areas at INCO's Coleman Mine and Falconbridge's Strathcona Mine which are on adjoining properties. At both mines an extraction of about 87% was achieved in primary stoping, leaving post pillars 6 m square and stopes 13 m wide and 9 m wide rooms between pillars. Depth below surface ranged from 550 m to 700 m. The rock mechanics aspects of the mining method are that the pillars should yield and be in a post-failure condition with most of the superincumbent load being transferred to the abutments. However, the pillars, surrounded by backfill, must still have sufficient strength to support the immediate roof structure.

Fig. 6(a) shows the stresses measured, by overcoring techniques, in three post pillars at the Coleman Mine. The distribution is the opposite of what would be expected in conventional room-and-pillar mining: the highest stress is in the pillar near the abutment and the lowest stress is in the pillar near the centre of the workings. Stresses on pillars B and C, which were 20 m and 25 m high respectively, were below the pre-mining vertical stress, having already yielded. Fig. 6(b) shows the change in stress measured on pillar A as mining progressed upwards. It was observed that non-violent yielding occurred when the pillar was 18 m high, the stress reducing from 27 MPa to 10 MPa, which was almost identical to the stress measured in pillar B.
Fig. 6: Measurement in post pillars at the Coleman Mine of INCO Metals Company.
Consequently, in this case a post pillar of height/width ratio of 3 had a peak strength of 27 MPa and a post-failure strength of 10 MPa (when it is surrounded by fill). Fig. 6(b) also shows the lateral pillar expansion and lateral fill pressure for pillar A. Expansion is confined to the outside 1.5 m of the pillar with no significant deformation of the central core. Also mining 12 m above the extensometer resulted in no additional expansion. Lateral fill pressure increased by 20 kPa after the first pour, but additional pours of fill had little effect. Blasting down 3.5 m of roof onto the fill had a much greater effect, which decreased with increasing cover of fill. Final recorded pressure was about 100 kPa with the cell being covered with 10 m of fill.

Fig. 7(a) shows the vertical deformation measured in a post pillar at the Strathcona Mine. For the three curves drawn, a positive gradient indicates compression and a negative gradient, expansion. Beneath and above the pillar the rock was expanding indicating a reduction in the vertical compressive stress. This is consistent with the concept of a yielding pillar with load being transferred to the abutments. The pillar itself is being compressed and the peak in the downward movement progresses upwards as additional cuts are mined.

Also at the Strathcona Mine, measurements of vertical and horizontal fill pressure were taken in different stopes. Typical results are shown in Fig. 7(b) together with the mining activity (i.e. blasting and filling) and the depth of fill cover. At shallow depths of fill cover, blasting down and subsequent mucking of the broken ore causes up and down fluctuations in the fill pressure, similar to measurements at the Coleman Mine. The scale of the depth of fill cover is such that it also represents the expected vertical pressure for a fill with a density of 2000 kg/m$^3$. Similar to measurements in longitudinal cut-and-fill stopes the measured vertical pressure is greater
Fig. 7: Measurements in post pillars at the Strathcona Mine of Falconbridge Nickel Mines Limited.
than expected, which could be due to an increase in bulk density with depth. Although the horizontal pressure exerted on the post pillars is fairly low, it is still sufficient to maintain the integrity of these pillars which can be 75 m high with a height/width ratio of 12.5:1.

**BLASTHOLE STOPING**

This method, with delayed bulk backfilling, is being increasingly used in Canadian mines. The rock mechanics aspects of the mining method are in choosing the dimensions of the stopes and pillars and sequence of extraction so that sloughing or collapse of either the hanging wall, footwall, rib or crown pillars does not occur. Figs. 8(a) and (b) are two examples of measured wall deformation at two mines using blasthole stoping with cemented rockfill.

In Fig. 8(a) four horizontal borehole extensometers were installed in Noranda's Geco Mine from a haulage drift to an almost vertical orebody, to measure wall deformation resulting from blasting a pillar between an open stope and a cemented rockfilled stope (31). Up to 30 mm expansion was measured and the distribution, as shown by the deformation contours, was symmetrical about the centre of the combined stope span rather than the blasted pillar. This indicates that even cemented rockfill does not act as an abutment. Maximum strain, as indicated by the closeness of the contours, occurred about 25 m into the wall rather than next to the stope. Borehole 3, located at the edge of the open stope, indicated less than 1 mm expansion, whereas, in borehole 4, near the centre of the adjacent pillar, slight compression was measured over most of its length. Total wall deformation will be higher than that shown, which was measured relative to the borehole collars, and a semi-circular expansion dome is envisaged extending about half the slope span into the wall.

Lateral deformation of the haulage drift was also measured at this site and indicated symmetrical expansion opposite the stope and compression opposite
Fig. 8(a): Measured footwall deformation at the Geco Mine of Noranda Ltd.

Fig. 8(b): Measured footwall deformation at the Kidd Creek Mine of Texasgulf Canada Ltd.
the pillar. It is interesting to note that damage to the haulage drift was more severe in the compression zone.

The Kidd Creek Mine of Texasgulf Canada Ltd. uses a blasthole open stoping method with delayed bulk filling with a cemented rockfill. Open pit waste is crushed to attain 75% coarse material (-15cm to 1cm) and 25% fine material (-1cm to 100 mesh). This material is transported underground via raises and mixed with 5% Portland cement and 4.1% water by weight in a baffled culvert before placement in the stopes. Cemented sandfill is used to top-off the stopes. The cemented rockfill is designed to have a minimum in-situ compressive strength of 4.1 MPa after 28 days. To date, six large exposures of stope backfill have occurred due to mining adjacent pillars or stopes. These exposed faces range from 15 m wide by 60 m high to 24 m wide by 91 m high and are free standing with only minor dilution in one case where unconsolidated backfill was encountered. The skin of unblasted ore left against the backfill wall to protect it from blast damage has, in most cases, spalled into the stope and has been recovered.

Fig. 8(b) shows the deformation measured in the footwall of the first mined stope at the Kidd Creek Mine. An 8 m thick shear zone adjoins the footwall contact and the two longest wires were anchored in this shear zone. There was a distinct cause-and-effect relationship between blasting and resultant deformation. Blasting on the same sub-level produced the major effect, indicating that stope span is the controlling parameter. Between blasts no significant deformation occurred confirming the elastic nature of the rock mass; an exception is the sudden 4 cm displacement measured on the longest wire which is probably due to breakdown in the shear zone. During the final stages of ore drawdown in the stope, additional expansion occurred on the two longest wires, indicating that broken ore can provide some constraint to the
stope walls similar in effect to a shrinkage stoping operation. Introduction
of the cemented rockfill produced a small but measurable compaction of the
footwall shear zone.

At Falconbridge's Strathcona Mine transverse rib pillars between
backfilled stopes are being mined with large hole bench blasting. Fig. 9
shows the mining layout, in one area of the mine, where a borehole extenso-

meter was installed to measure the deformation in the backfill. The stopes
on either side of the pillar had been filled with 16:1 and 32:1 tailing/cement,

with the borehole extensometer going through the latter stope. Three

stages of pillar mining are shown and after stage 1 that area was filled
mainly with 32:1 backfill prior to mining stages 2 and 3. Very little
deformation was measured in the backfill during stages 1 and 2. Stage 3
removed the pillar opposite the extensometer and up to 31 mm of expansion was
measured. The deformation profile shown in Fig. 9 indicates a slumping of
about 15 mm at the rock/fill interface (it is assumed that the rock above
the fill is not expanding). There is a uniform deformation gradient extending
from the top of the fill to within about 10 m from the stope wall. This 10 m
outside zone appears to have been bodily moved into the mined area without
differential displacement. The movements measured in the fill have not
resulted in any significant sloughing of the fill wall, in fact, all three
fill walls of 16:1 and 32:1 are self supporting over exposures 12 to 27 m
wide by 70 m high.

NEW DEVELOPMENTS

Several new methods have been introduced in recent years with the
aim of decreasing the cost of production. Large hole bench blasting (14,15),
vertical crater retreat (20) and inverted bench blasting (34) methods are gradually
replacing, in certain situations, less efficient cut-and-fill methods in
Fig. 9: Measured fill displacement at the Strathcona Mine of Falconbridge Nickel Mines Ltd.
Canadian base metal mines. In addition to the role of mine equipment, fill technology and rock mechanics have made significant contributions towards successful evolution of these concepts. The principal advantages of these methods are:

(a) stope layouts are simpler, reducing engineering time;
(b) reduction in the amount of development;
(c) faster mining cycle with increased speed and efficiency of standard drilling, blasting and mucking cycle;
(d) many of the time consuming operations, such as secondary breaking, scaling, equipment set-up and bolting are either eliminated or reduced during the production cycle;
(e) better fragmentation with improved drawpoint efficiency;
(f) avoidance of ore dilution with backfill material during pillar extraction;
(g) working environment is better and safer, men work in well supported areas, large hole drills are much quieter, produce less dust and require less physical effort to operate.

There are many cases in Canadian literature on the design of more efficient mining methods and as an example the design procedure used at the Lockerby Mine of Falconbridge Nickel Mines Ltd. is given. This mine is the newest of five producing mines of the company in the Sudbury area of Ontario. The nickel sulphide orebody is steeply dipping and conforms to the shape of a bow extending from 250 m to 1300 m in depth. Ore width varies from a few metres to 55 m with an average width of 22 m.

Initial plans were based upon an incremental cut-and-fill approach. However, during the development phase at the mine, extensive studies were
carried out to assess the feasibility of bulk mining methods. These investigations, including finite element techniques, indicated that large hole bench blasting methods could be introduced provided rock mechanics guidelines on location of permanent openings, stoping sequence, dimensions of stopes and pillars, support systems, blast vibrations and fill properties were followed. This mine now uses bench blasting methods in both primary stoping and pillar recovery and accounts for most of its production.

Fig. 10(a) and (b) show a general level plan of a blasthole mining block and a section through a blasthole stope. Levels are at 45 m intervals with stopes and pillars 11 m wide. The cross-cuts in the primary stopes are extended into the hanging wall and from the cross-cuts cable bolts are installed in a cone type configuration to stabilize the hanging wall and footwall, as shown in Fig. 10(b). A four stage mining sequence is used as shown in Fig. 10(a). In stage 1, primary stopes are separated by pillars 33 m wide (i.e. 3 stope widths). These stopes are backfilled with 12:1 tailing/cement prior to mining primary stopes in stage 2. These stopes are also filled with 12:1 backfill prior to mining, in sequence, the pillars in stages 3 and 4. The pillar openings are filled with straight tailings with no cement added so that the average tailing/cement ratio in the block is about 24:1. This sequence of mining eliminates simultaneous extraction on both sides of a 11 m wide ore pillar or fill pillar which is the weakest configuration.

During pillar extraction, fill heights of up to 45 m are exposed. Finite element models were run to determine the effect of different tailings/cement ratios on the stress and deformation distribution in and around the stopes. Ratios of the modulus of elasticity of the fill to wall rock were chosen at 1:40 and 1:70 which represents fills of 12:1 and 30:1 tailings/
Fig. 10: Mining layout at the Lockerby Mine of Falconbridge Nickel Mines Ltd.
cement ratios respectively. In the model the fill was placed immediately after primary stoping (i.e. stages 1 and 2). Figure 11 shows typical results of horizontal closure and stresses exerted on the fill due to mining of stopes in stage 2. For the 12:1 tailing/cement fill, the results indicate that the maximum stress on the fill should be about 75% greater and the horizontal closure 12.5% less than on fill with a 30:1 ratio. These studies provided guidelines for determining the fill properties required and it was decided to pour 12:1 tailings/cement mixes in primary stope.

CONCLUDING STATEMENT

In this review we have tried to demonstrate that fill mining technology in Canadian mines is in a continuous process of change. Mining methods used today will be modified and improved in the future. It is interesting to note that in the 1950's the question was raised whether cut-and-fill methods could be used at depths hitherto mined by square setting (34). In fact, with the introduction of cemented fills, cut-and-fill mining became the standard, if not exclusive, method for mining below 1000 m depth, until in the 1970's when blasthole stoping started to replace it in certain applications.

Research and development in Canada is being undertaken by many mining companies, equipment manufacturers, service industries, consultants, universities, and government agencies, as attested by the many published articles. Recently the Canada Centre for Mineral and Energy Technology (CANMET), Department of Energy, Mines and Resources initiated a joint project with industry to develop bulk mining methods at depth. Research contracts are placed with mining companies and include:

(a) design of a bulk mining trial in a cost-shared agreement with Falconbridge Nickel Mines Ltd.;
Fig. 11: Closure and stress distribution in fill from finite element models.
(b) compilation of case histories on design of production
blasts and monitoring of blast vibrations for bulk
mining methods by INCO Tech., a division of INCO Ltd.;
(c) development of a computer program for mining layout
selection by Norcomp, a division of Noranda Mines Ltd.

These type of studies, in conjunction with other projects being
undertaken by the mining industry, should assist in further development of
Canadian mining technology.

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