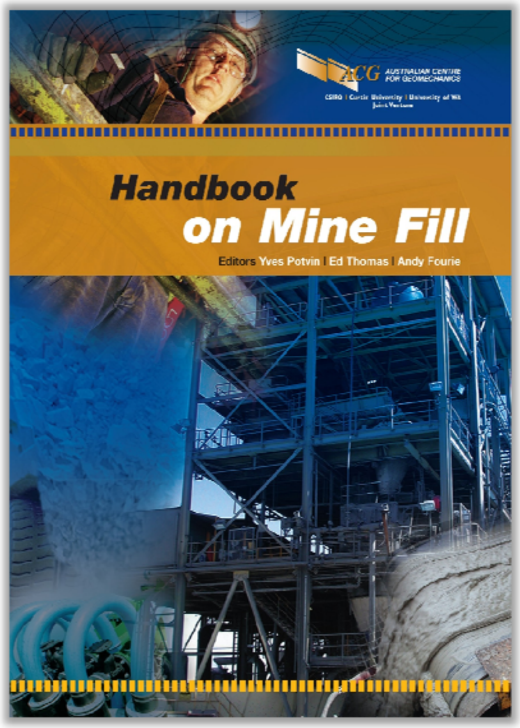
For further information:

Publication Officer
Australian Centre for Geomechanics
The University of Western Australia
35 Stirling Highway (M600)
CRAWLEY 6009 WESTERN AUSTRALIA
AUSTRALIA
Ph: +61 8 6488 3300
publications-acg@uwa.edu.au
www.acg.uwa.edu.au



CHAPTER 1

Introduction

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Yves Potvin Australian Centre for Geomechanics

1.1 PREAMBLE

There is no shortage of scientific literature on mine fill. The subject has been periodically documented through a series of dedicated international conferences, in addition to isolated publications and various articles published in mine engineering meetings held around the world. Thomas (1976) wrote comprehensive workshop notes published by the Australian Minerals Foundation Inc. This was followed by "Fill Technology in Underground Metalliferous Mines" by Thomas et al. (1979), which to date remains the only textbook devoted to the topic of mine fill, at least in the English language. The technology behind mine fill and its application has continued to evolve over the past two decades and this book responds to the need for updating these landmark publications. It is presented as a practical Handbook, examining the main engineering principles behind mine fill with relevant examples of mine site applications.

This Handbook on Mine Fill is the result of a collective effort by specialists from Australia with further contributions from abroad. It relies extensively on the past 25 years of, in the main, published literature and aims to provide engineers working at mine sites with a one-stop reference covering the most significant aspects of mine fill.

For the purpose of this Handbook, mine fill has been divided into three types:

- *hydraulic fill,*
- *paste fill, and*
- *rock fill.*

Each type of fill is discussed in appropriate detail in dedicated chapters of the book.

A wide variety of professional disciplines are involved in the conception, design, construction and operation of mine fill systems. These may include (after Thomas, 1976):

- *mining engineering,*
- *operating,*
- *planning,*
- *mineral processing,*
- *rock mechanics,*
- *soil mechanics,*
- *environmental engineering,*
- *cement technology,*
- *pozzolan chemistry,*
- *mineral chemistry,*
- *industrial engineering, and*
- *geology.*

As it is not practical for the Handbook to cover this wide spectrum of disciplines in detail, some of the governing principles from most disciplines have been regrouped and discussed within the following chapters:

- *chapter 2 — Basic mine fill materials*
- *chapter 3 — Geomechanics of mine fill*
- *chapter 4 — Fluid mechanics of mine fill*

The Handbook is also concerned with the environmental and safety aspects of filling operations. The increasing reliance of the mining industry on risk management techniques as a framework to mine safety is reflected in chapter 9 — Hazards, Risk and Environment.

1.2 WHY MINE FILL?

The essence of mining is to extract valuable minerals from the earth's crust. This generally results in the creation of voids. The main function of fill materials in mines is to assist in managing the stability of mining related voids. There are many ways of managing mine voids. Fill is one of the tools that is used to increase the flexibility of ore extraction strategies and often allows for an improved recovery of orebodies. The use of different types of fill and their specific functions and engineering requirements are intimately related to mining methods, mining strategies and mining sequences.

Brady and Brown (1985) proposed a subdivision of mining methods according to three main categories (Figure 1.1):

- *Unsupported (caving) methods:* where the voids created are meant to be continuously self-filling with caving material, as mining progresses. These include block caving methods where orebodies are undercut to induce caving of the ore. They also include sublevel caving methods where the hangingwall progressively caves to fill voids produced by ore extraction. Caving methods do not require fill, the mining void being filled by caved country rock. It is generally accepted that the cave may eventually propagate to surface and surface subsidence will occur.
- *Naturally supported methods*: where pillars are left in place as the main way of controlling the stability of extracted areas (voids). This includes room-and-pillar methods, often employed in low dipping orebodies. Fill is generally less effective and efficient in mining shallow dipping orebodies as gravity based fill transport systems are less feasible. Also, the active support of hangingwalls provided by fill in steeply

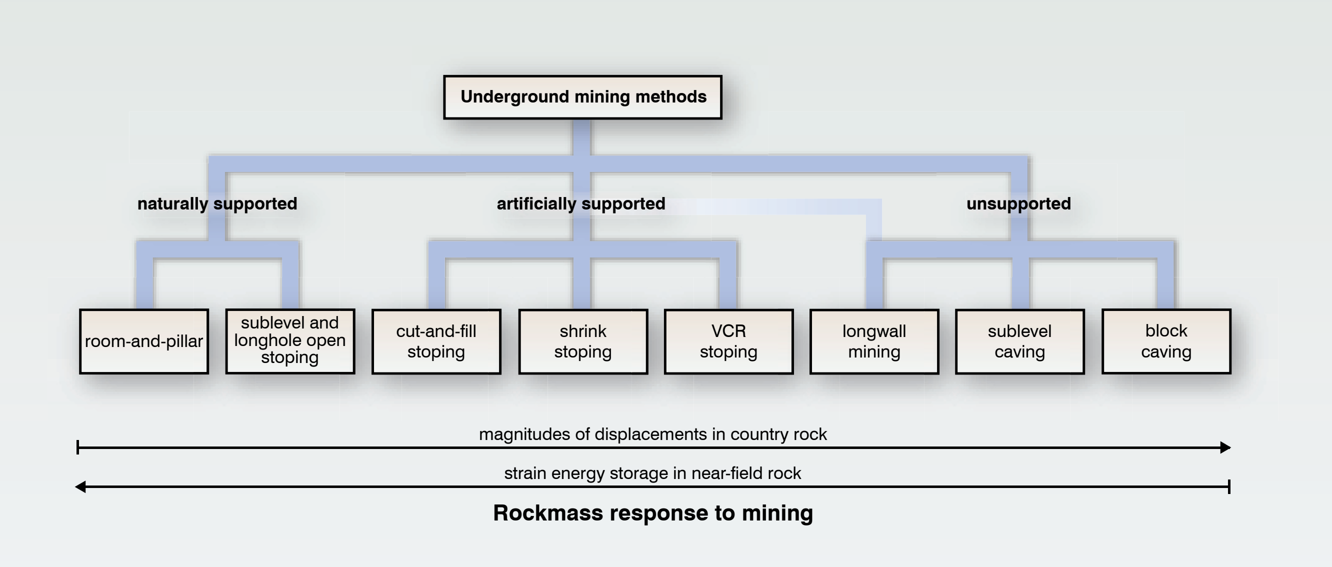

FIGURE 1.1 Diagram showing a breakdown of mining methods according to the mine regional support system used (after Brady and Brown, 1985)

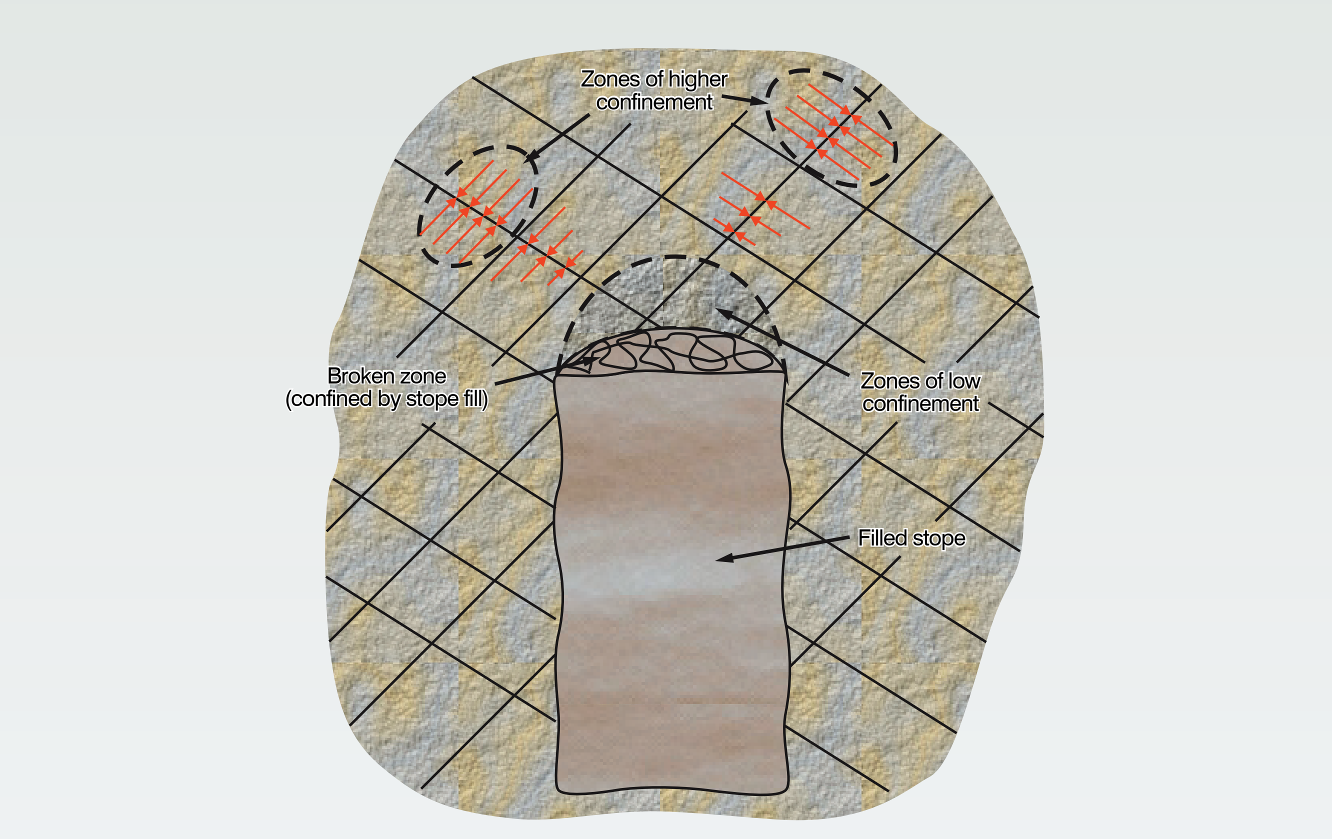

FIGURE 1.2 Schematic diagram showing how fill preserves confinement at the boundary of an excavation and assists in mobilising the shear strength along existing joints and arresting potential failure propagation

dipping mines is not easily achieved in subhorizontal conditions. Shrinkage and some variations of open stope mining can also rely on naturally supported methods, using crown and rib pillars to separate stoping blocks. This approach generally yields lower ore recovery and is often practised in low-grade orebodies, where the increase in ore recovery does not justify the cost of fill.

- *Artificially supported methods:* where fill is used (often in combination with temporary or/and permanent pillars) to limit voids exposure so as not to exceed critical stable dimensions. This includes variations of cut-and-fill and open stope mining methods. As underground extraction reaches deeper levels, stable void exposure becomes smaller, and reliance on an efficient fill delivery system is emphasised.

Mine fill is generally applicable to artificially supported methods. Within these methods, a wide range of fill applications has been used to satisfy a variety of engineering goals. The main functions of fill in artificially supported mining methods are summarised in the following sections.

1.2.1 Ensuring long-term regional stability

The stability of underground excavations is a function of a number of variables including ground conditions, spans and time. Therefore, large excavations that are left open indefinitely have an increasing risk of collapsing with time. There have been a number of well documented cases where fill was not used and crown pillars have failed up to surface, with a variety of consequences.

Fill can be used to mitigate this risk. In this application, fill acts as a bulking agent and its function is simply to occupy the mining void. If the excavation becomes unstable, loosening material from the excavation boundary is kept in place by fill, and the rockmass failure process is arrested.

This is done by preserving some of the confining forces within the rockmass, which generally increase with the distance away from the excavation boundary. This increased confinement may promote stability in two ways. First, in jointed rockmasses it prevents the opening of joints and discontinuities, allowing the friction along these planes of weakness to mobilise. As a result, the rockmass shear strength is preserved and the propagation of failure is arrested (Figure 1.2).

Second, bulk filling may also assist in limiting the amount of wall convergence, which may in turn have a positive impact on the regional stability of the mine.

If the bulk filling will not be exposed to other voids, there is generally no specific material properties requirement for the filling material in terms of cohesion and strength. However, other risks such as potential chemical reactions with the environment or the potential for liquefaction in the case of an unforeseen collapse breaking through the filled excavation must be considered when selecting bulk filling material and when determining whether a binding agent needs to be added. For example, even when paste fill is used as a bulking material and is not to be exposed to adjacent voids, a minimum target uniaxial compressive strength (UCS) of 100 kPa is maintained to prevent liquefaction of the fill mass. This is based on the Clough et al. (1989) work on cemented natural sands. The risk of liquefaction of the paste fill used at Golden Giant mine in Canada was reported by le Roux et al. (2004) to be minimal after 24 hours, as the UCS had exceeded 30 kPa under static and dynamic loads.

Loose waste rock material from development mining or classified tailings is often disposed of as cost efficient bulk filling material. Mines where waste and tailings are not available in sufficient quantities have used alternative sources such as quarried rock, alluvial sand and crushed/ground slag.

1.2.2 Limiting excavation exposure

In a given set of ground and operating conditions, underground excavations smaller than a critical dimension will remain stable for the duration of mining activities. Unless an orebody can be entirely mined as an open, single excavation without exceeding the critical stable dimension, fill can be used to limit the exposure of walls and/or backs of excavations. In this context, orebodies will often be divided into stopes and/or lifts to be sequentially extracted and filled. For extraction sequences progressing upwards (bottom-up mining), the fill mass in the stope (or lift) below will serve as a working floor for the stope (or lift) above. Conversely, extraction sequences such as undercut-and-fill progressing downward may use fill as a replacement roof. This is particularly appropriate when the ground conditions in the back are deemed unsafe and the option of leaving crown pillars separating stopes is rejected.

A number of strategies can be adopted to limit stope wall exposure, some involving uncemented fill and others cemented fill. Uncemented fill (waste rock or tailings) is sometimes introduced after the stope extraction is completed (delayed filling). The uncemented fill can be contained with thin permanent retaining pillars to avoid the incremental cost of a binder, which is often a significant percentage of the total fill cost. Such practice is sometimes justified in lower grade ore.

Alternatively, in bench mining, waste rock can be placed in a pseudo-continuous way, advancing behind the mining front. An example is the Avoca method, which involves choked blasting of a slice of ore against uncemented fill material. As soon as the ore is blasted and removed, the void is filled and the cycle of ore slicing and filling continues. Bench mining is a common variation of the Avoca mining method and it involves delaying the filling step, leaving a moving triangular shape of wall exposure (Figure 1.3). The extent of wall exposure or distance between the bench face and the fill slope is a function of wall stability. The closer the toe of the fill slope comes to the bench face, the more likely fill dilution will be introduced during the mucking operation.

The use of cemented fill to limit excavation exposure can lead to very productive and flexible mining sequences, especially when mining massive orebodies of reasonable grade. The entire orebody can be divided into stopes. Whilst a number of them can be exploited simultaneously (subject to

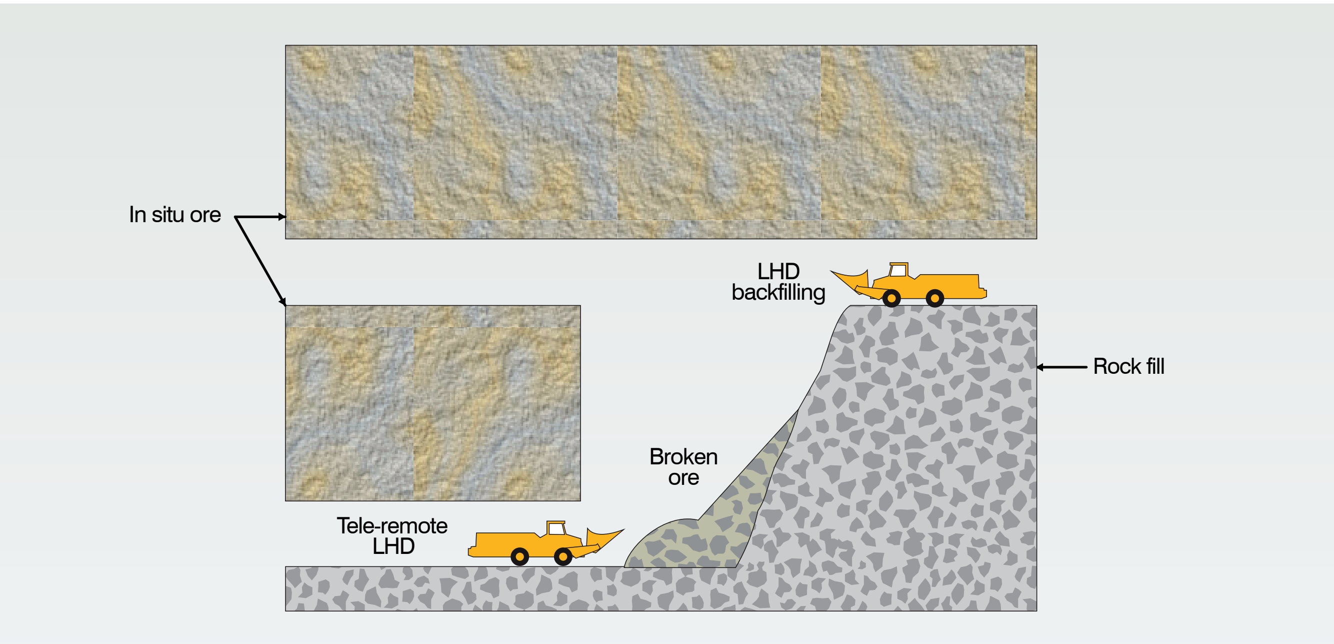

FIGURE 1.3 Schematic longitudinal view of a benching operation showing the introduction of delayed uncemented fill

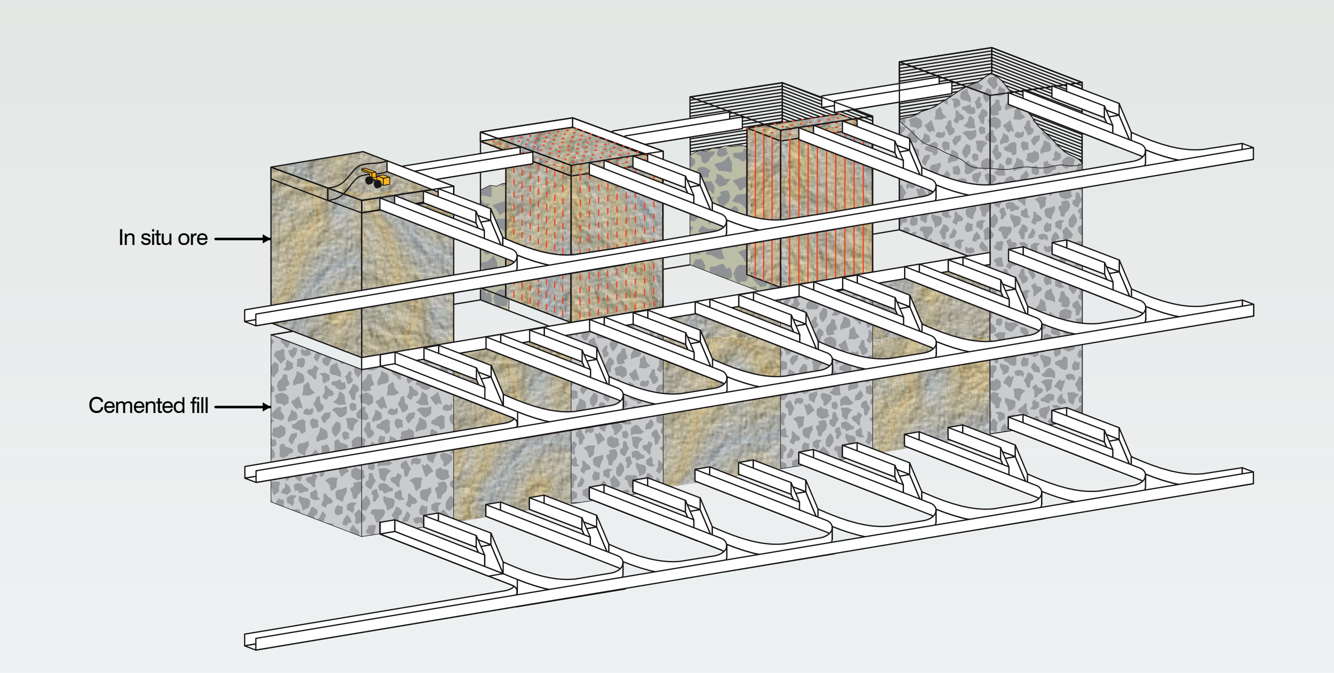

FIGURE 1.4 Schematic three dimensional view of open stope mining showing the different stages of mining, starting from left to right with pre-production, development of overcut, drilling, blasting and filling (after Potvin, 1988)

logistics and other constraints), another series of stopes can be in pre-production (drilling) and yet another series can be in post-production (filling), as shown in Figure 1.4.

An essential engineering consideration for the fill material in this application is its capacity to become self supporting when exposed, as a fill wall, to the void created by the extraction of adjacent stopes. For example, vertical fill walls in excess of 200 m have been successfully exposed in Mount Isa Mines. A binding agent (most commonly Portland cement or slag cement and/or fly ash or a combination of them) is used to cement the bulk filling material (often tailings or waste rock). In some situations, where the sequence of extraction relies on a rapid production cycle, the curing time of fill becomes critical.

In deep mining conditions, where the in situ stress field is high, the sequence of stope extraction becomes one of the main strategic control measures for managing the effect of mine induced stress concentration.

Deep mining experience has clearly demonstrated that avoiding significant size pillars is the best way of minimising areas of high stress concentration and lowering the potential for rockburst and rockfall related problems. This can also be demonstrated using numerical modelling and stress analysis software.

In a deep and steep tabular orebody, ideal sequences will have no pillars and may involve starting with one stope centrally located whilst progressing to adjacent stopes in a way that allows the total excavation to expand outwards in an inverted "V" shape. Alternatively, the initial stope could be located at one of the corners of the orebody, progressing the extraction by mining adjacent stopes diagonally, which also results in the elimination of in situ pillars.

Pillarless mining using a continuous retreat sequence has the added advantage of pushing stress concentrations out towards the stope abutments. This results in the stress distribution being relatively uniform across most excavations and reduces the possibility of high stress concentrations. However, pillarless mining has limited production flexibility as only stopes adjacent to filled areas can be mined to avoid the creation of pillars. This is particularly constraining at the early stages of orebody extraction. Another characteristic of this approach is its reliance on an efficient and effective fill system.

The general principle of a pillarless mining and continuous retreat mining sequence is often applied with some modifications because, in many situations (especially at the early stages of mining), pillars cannot be totally avoided.

It is possible to use a combination of primary stopes, and pillars that become secondary stopes, that expands in a triangular shape (Figure 1.5). This allows the production rate to be increased whilst managing high stress conditions. The secondary pillars are recovered as early as possible in a continuous retreat from the centre outwards. The stresses concentrated in the secondary pillars are then progressively shed towards the abutment. The Williams mine in Canada is an example of a primary/secondary continuous retreat sequence (Figure 1.6) where two overlying continuous retreat fronts have been developed.

In many Canadian mines, cemented slurry rock fill (with no tailings) has been the fill of choice, in continuous retreat and pillarless mining. This is due to its reliability and fast curing characteristics. In recent years, paste fill has gained increasing popularity with this mining approach because of its fast curing characteristics and efficient mode of delivery (pipe network instead of mobile equipment).

1.2.3 Waste disposal

The cost and liability of surface storage facilities for mine waste, whether rock or tailings, have increased significantly in recent years. Environmental standards and mine closure requirements are gradually transforming the economics of mine waste disposal. Filling of underground voids is increasingly perceived as an environmentally friendly vehicle as well as a cost saving option to permanent disposal of mine waste.

1.3 CONSIDERATIONS FOR SELECTION OF MINE FILL SYSTEMS

Mine fill systems are designated according to the fill material and binder used to fulfil specific applications. Each application will dictate the engineering characteristics of the fill mass and the production rate requirements, which in turn will define the system that can produce and deliver it to underground voids.

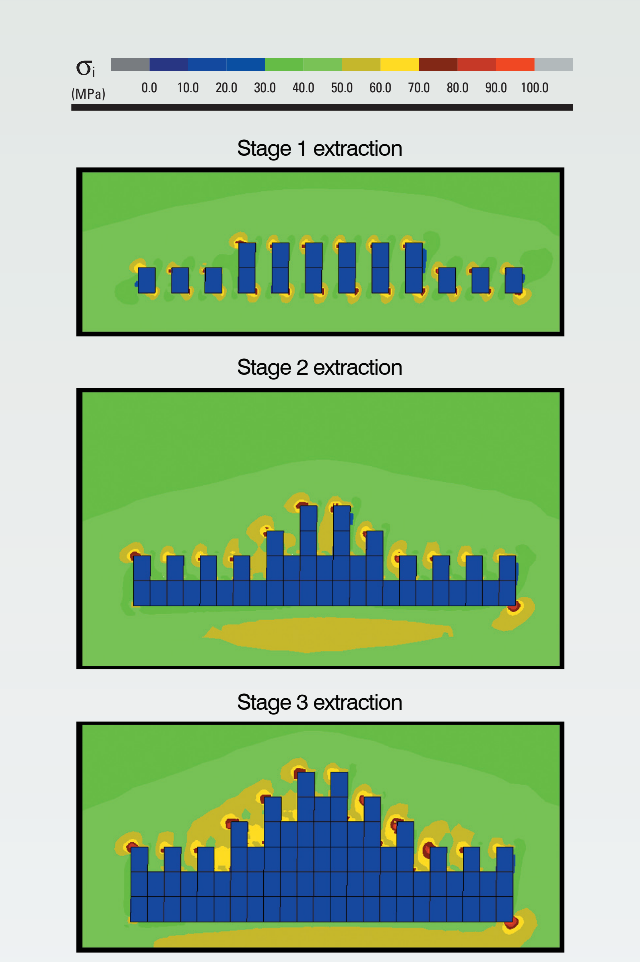

FIGURE 1.5 Series of computer model outputs showing stress redistribution around a triangular stope and pillar retreat (after Potvin and Nedin, 2003)

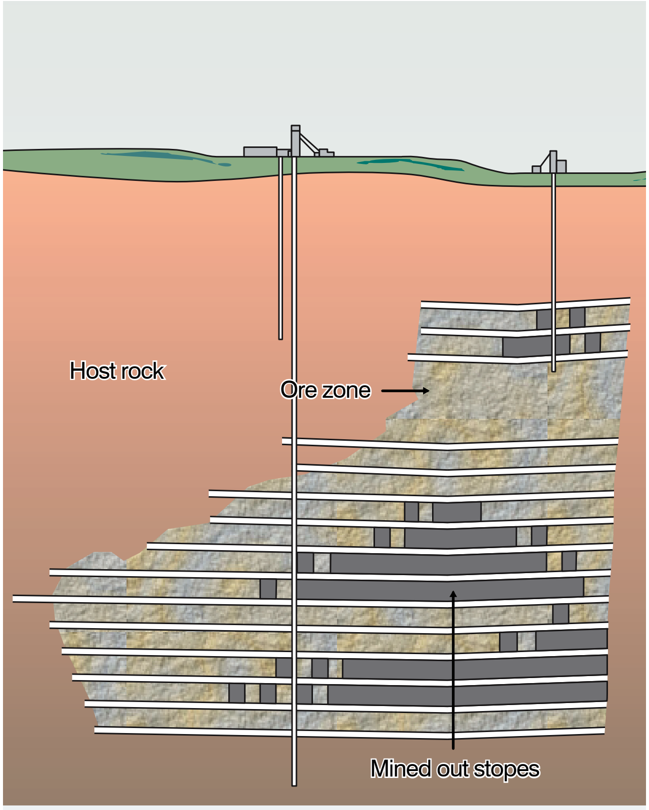

FIGURE 1.6 Diagram of the mining sequence using a triangular stope and pillar retreat (modified from Bronkhorst et al., 1993)

There could be many ways of classifying mine fill systems. It is convenient to classify mine fill based on the raw material used and the processes of producing and delivering the fill, namely, hydraulic fill, paste fill and rock fill.

Hydraulic fill generally refers to free-draining, classified mill tailings transported through a network of pipes and boreholes at a percentage solids that will result in turbulent flow. Although the percentage solids may vary dramatically in different applications (from over 50% to more than 70% by mass), there is a clear advantage in minimising the water component in hydraulic fill, in terms of drainage and underground water handling costs. Bulkheads erected to contain hydraulic fill within the stopes must allow fill drainage and be designed and constructed to sustain the anticipated hydraulic head of pressure at an appropriate factor of safety.

Without binder, hydraulic fill can be used as a bulk filling material. If the fill mass is to be exposed to voids, a binder (usually Portland cement and/or fly ash and/or crushed slag) is required to cement the fill mass. Hydraulic fill systems are described in appropriate detail in chapter 5 of this Handbook, Introduction to Hydraulic Fill.

Paste fill is also a mine tailings product, often (but not always) using the total mill tailings, with no removal of fine material by classification. In comparison with hydraulic fill, paste fill has lower water content, approximately between 78 and 87% solids by mass, and is a non-draining material. It uses some water for cement hydration and retains the rest within its matrix. Real paste fill will not create any bleed water. A number of so-called paste fills would be better characterised as high-density fill as they show some bleed water at placement. Another important characteristic of paste fill is the laminar flow produced during pipe transportation. The paste material will generally not segregate inside pipelines, even if the flow is interrupted for a period of several hours. If the stoppage continues beyond that, cement curing is likely to result in blockage of the pipeline.

Paste fill offers a number of advantages over hydraulic fill. The particle size distribution and water content of the paste are generally designed to optimise the cement hydration process, minimising the cement requirement while achieving the target strength. Being non-draining, paste fill cures relatively fast, does not generate high pressure on bulkheads and does not create underground mine water handling related problems. The laminar flow brings increased flexibility into the delivery system.

Amongst its disadvantages, paste fill has a more difficult rheology than lower density hydraulic fill due to a higher pressure loss per unit length of pipe. It also has a higher angle of repose when placed in a stope (Spearing, A.J.S., 2003, pers. comm.). Paste fill plants and distribution systems are generally capital intensive. However, they are not labour intensive and often yield relatively low operating costs (depending on cement content). Paste fill systems are described in appropriate detail in chapter 6 of this Handbook, Paste Fill.

Rock fill uses waste rock, quarried rock or aggregate as a bulking material. Depending on the engineering purpose of the fill, a hydraulic component (cement slurry or cemented tailings) can be combined with the bulking material to produce a cemented rock fill mass.

Transportation of the bulking material often relies on a combination of fill passes, mobile equipment and/or conveyors, whereas the hydraulic component is usually transported to the area where it is to be mixed with rock, using a network of pipelines and boreholes. A number of variations exist on how and where it may be combined with the dry component. Most cemented rock fill systems are relatively simple and generally involve low capital expenditure unless an underground conveyor for coarse rock is required. Fill production and transport can be labour intensive and the operating cost can be relatively high as the binder requirement is often higher than for hydraulic and paste fill. Rock fill systems are described in appropriate detail in chapter 7 of this Handbook, Rock Fill in Mine Filling.

The selection of an appropriate fill system is often related to the availability of filling material. For example, mining operations that have neither a mineral processing plant on site nor pre-existing tailings cannot use a tailings based fill. Moreover, some tailings material may not be suitable for producing a cost efficient paste fill, because of prohibitive cement content requirements. Other operations may not have access to a supply of waste rock or aggregate.

It is imperative that the fill system selected reliably delivers the fill properties and production rates required to achieve the mining plan.

The capital and operating cost of producing and distributing fill is indeed another critical consideration. A holistic approach to estimating fill costs is essential. This implies not only that direct costs are considered but also that hidden incremental costs are accounted for. For example, the long-term waste disposal of material not used in fill can be significant. Particularly when using material with high environmental liability and also, when mobile equipment is used for delivering rock fill, potential delays in production may occur due to increased traffic, or if the equipment used to deliver fill has to be sourced from the production fleet as a result of breakdowns or maintenance requirements. The use of hydraulic fill will likely exacerbate water handling issues and promote local corrosion of ground support. The cost of binder, cement in most cases, can also be a major issue, especially for remote mine sites.

The process for selecting a backfill system should therefore look at feasible options considering material availability, the required fill production rate and fill mass properties. Each feasible option should be costed holistically. When options are narrowed down, the following risk identification tools can assist in making the final decision (after MISHC, 2002).

Risk identification tools that can assist with option review include:

- Preliminary Hazard Analysis (PHA), Hazard Analysis (HAZAN) or Workplace Risk Assessment and Control (WRAC)
- Fault Tree Analysis (FTA)
- Event Tree Analysis (ETA)
- Human Error Analysis (HEA)

1.4 BRIEF HISTORY OF MINE FILL

In his keynote address to the Brisbane Minefill 1998 International Symposium, Cowling (1998) noted that: *"…there are no comprehensive histories of filling practices in any of the major mining countries"*. This section is not intended to redress this lack of information in the documented history of mining but rather to provide a brief perspective on the origin of mine fill. Discussion will be brief and biased towards North American and Australian experiences.

The practice of filling the voids created by mining is probably as old as mining itself. In previous centuries, filling may have occurred naturally through caving of overlying strata or as a part of a mining process to conveniently dispose of waste rock.

1.4.1 Australian mine fill history

According to McLeod (1992), the engineered preparation of fill material to be placed in depleted underground stopes as part of an extraction strategy or to stabilise an area was not common until late in the 19th century. He reported that in the early 1900s, the placement of dry fill gained popularity in Australia with one of the main drivers (at Broken Hill) being the arresting or minimisation of subsidence to protect the mine infrastructure and surface plants.

"As a result of massive subsidence the Central Mine had lost two shafts, its offices, store and boiler plant in 1905 and the concentrating mill in 1906."

However, the effectiveness of dry fill was limited owing to difficulties in transporting and placing it where it was most needed. Since the very early days, fill transport and fill placement have been identified as major challenges.

According to Murray (1915), dry fill practices at Mount Lyell in Tasmania tried to overcome the problem of placement by introducing quarried rock (or sometimes mine waste rock) mixed with water, from a pass at the top of large, domed shape stope backs. This facilitated fill distribution within the stope, and contributed to limiting wall exposure. This solution was, however, unique to Mount Lyell practices and conditions.

Problems associated with dry fill transportation and placement were largely overcome with the introduction of mill tailings transported hydraulically as it became popular in Australia in the 1940s. McLeod (1992) stated that: *"Prior to 1940, reference in literature to the stowage of sands by hydraulic means is infrequent and often sketchy"*. *"Broken Hill South first used hydraulic stowing in 1929."* McLeod (1992) noted that according to Arnold Black in 1944, the experiment was successful but the practice had to be discontinued until the underground pumping capacity of used water available at the mine was increased.

Mount Lyell started using mill tailings for filling stopes as early as 1933. Pumping excess water was not a critical issue at Mount Lyell as the mine benefited from mining inside a mountain, accesses having been added to facilitate the drainage of filled stopes.

In 1940, Broken Hill began to use hydraulic fill as the main filling material in their South Mine. Although some of the pumping capacity issues had been resolved with the commissioning of No. 7 shaft (McLeod, 1992), Cowling (1998) noted that the negative impact of excess water was already well recognised and that the slurry density was kept within a range of 70 to 75% solids by mass. Even in terms of current standards, this would be considered good practice.

During the two decades following the introduction of hydraulic fill at Mount Lyell and Broken Hill, the practice was refined and spread throughout Australia, wherever filling was required.

1.4.2 North American mine fill history

The early days of fill in North America were not dissimilar to the Australian experience. Landriault (2001) describes how, in the first few decades of the last century, dry fill was introduced into depleted shrinkage stopes in subvertical orebodies and in square-set stopes in shallow dipping orebodies. Local collapse of these stopes often occurred when the critical span was exceeded, forcing the abandonment of a whole area and the loss of ore reserves. Although dry fill did not totally eliminate unstable stopes, it did limit the extent of the collapses and allow for mining to continue in the periphery of failure.

Landriault (2001) describes the early dry fill practices as follows:

"Initially unconsolidated material such as development waste rock or surface sand and gravel was supplied through raises and mine cars to the stopes. It was fed into the stopes through raises and then was moved into place manually. This backfill method greatly increased labour requirements, backfill cycle time and ore recovery."

The economic benefits of mine fill provided the incentive to address those issues of transport and placement also described in the Australian historical context. It was at Noranda's Horne mine in 1933 that a first version of cemented fill was discovered by mixing granulated furnace slag with pyrrhotite tailings. The oxidation process of this fill mixture produced a cemented material capable of self supporting. This new cemented fill opened the way to new extraction strategies, as mining could now proceed directly against filled stopes. However, since the production of this particular fill was material specific, few operations, if any, could duplicate this process.

The introduction of hydraulic fill in North American metal mines came in the late 1940s according to Landriault (2001), which is approximately a decade later than in Australia.

"In this case, the system supplied classified tailings (fine size fraction removed) to square set stoping areas. Because the volume of fill height was small, the risk of the fill becoming mobilised through liquefaction was minimal, although the rate of drainage was slow by today's standards. However, even allowing for the slow rate of drainage through material with a relatively high fines content, the speed at which stopes could be filled was much greater than the rate of supplying rock fill. Another advantage was that the fill could be directed to different areas of the stope relatively easily by setting up a series of pipes to direct the flow of material."

Clearly, hydraulic fill owed its success to the fact that it alleviates some of the problems associated with dry fill transport and placement. The next phase of mine fill evolution occurred during the 1950s and 1960s, with a focus on cementing hydraulic fill by adding a binder, which in most cases was Portland cement. This essentially permitted the mining of large stopes against fill walls. As previously experienced at the Horne mine, such practice increases the flexibility of applying different extraction strategies and mining methods.

In the 1960s, cut-and-fill methods gained popularity in North America. Whilst the cement content of the cemented hydraulic fill was around 3 to 4% by mass, a layer of 10% was often poured to produce a good quality mucking floor. A similar layer rich in cement content was used in undercut-and-fill methods to reduce the timbering requirements and to improve the cycle time. Also in the 1960s, rock fill and cemented rock fill were first introduced in North American mines.

1.4.3 Recent mine fill history

In the 1970s, the emphasis in industrialised countries was on mechanisation and high production rates, leading the way to bulk mining methods and large open stopes. This meant that fill exposures were becoming larger and that pressure for producing structurally stable fill masses at a low cost was increasing. Fill research initiatives pursued at operations such as Kidd Creek mine in Ontario, Canada and Mount Isa Mines in Queensland, Australia led to the development of varieties of cemented fills having relatively strong structural properties and low overall cement content.

For example, the addition of waste rock, quarried rock or aggregate to cemented tailings at a 2 to 1 ratio allowed fill walls exceeding 200 m in height by 40 m wide to be routinely exposed without significant dilution at Mount Isa Mines. This fill effectively contained less than 1% cement content (Cowling, 1998).

Sustained research led to further reductions in cemented fill cost by substituting cement, an effective but often expensive binder, with cheaper substitutes such as ground furnace slag and fly ash. It was found that although such substitution can produce fill with low early strength, in the long term, the fill strength was adequate for most applications.

The North American mining industry in the 1980s and 90s faced new challenges, including the extraction of deep reserves and the recovery of highly stressed sill pillars. The trend towards mechanisation and high productivity was accelerating. Whilst stope dimensions gradually became smaller to accommodate high stress conditions, the requirement for the rapid turn around of stopes and high tonnage output called for mine fills of different characteristics. Rapid curing time became an essential specification to achieve the compressed production cycles.

The practice of mixing waste rock with thin cement slurry (with no tailings) eliminates the need for drainage and bulkheads. It produces a matrix of rocks bound at contact points with cement offering not only a fast curing and structurally strong fill mass but also a very simple, flexible and low capital cost filling system, with overall cost depending mainly on that of delivering the waste rock. Delivery of cemented slurry rock fill to stopes is done with mobile equipment gathering rock fill from passes or chutes, then driving through underground cement stations where cement slurry is poured over the rock. Mixing of the rock and cement slurry occurs as it is tipped from the edge of the stope.

Research into paste fill technology undertaken in the 1980s, followed by its implementation in many North American mines in the 90s, was also driven by the elimination of drainage, fast curing and fast turn-around time to fill stopes. The paste fill was promising another strategic advantage over cemented slurry rock fill in that it disposed of large volumes of mine tailings underground. This increasingly became the main driver for the development and implementation of paste fill, as the mining industry wanted to improve its image towards impact on the environment, and realised the true cost of tailings surface disposal.

The first operational paste fill system was built in the early 1980s at the Bad Grund mine in Germany (Lerch and Renezeder, 1984). Paste fill reticulation was assisted with piston pumps to overcome the long horizontal distance from the shaft to the delivery points in the active stoping area. The pipelines were run full to minimise wear and the cement was added in the pipeline only near the delivery point.

Not long after, the first North American paste fill application followed at the Lucky Friday mine in Idaho in the US (Brackebusch, 1994). The plant was designed like a concrete batching plant. Although the paste was pumped to the shaft, the underground reticulation was gravity driven. The pipelines were not kept full and excessive wear issues had to be overcome.

Also in the early 1980s, Australia attempted to implement a paste fill system at the Elura mine in New South Wales (Barrett, 2000). However, a number of technical issues were never overcome and the plant was later transformed to produce hydraulic fill. This unsuccessful trial may have contributed to the relatively slow implementation of this technology in Australia, with the first fully producing plant commissioned in 1997 at Henty Gold mine (Henderson et al., 1998), followed by Cannington in 1998 (Skeeles, 1998) and Mount Isa mines (Kuganathan, 2001), and Junction mine (Coxon, 2003).

Inco Limited has led the paste fill R&D effort in Canada and from the early 1980s to the commissioning of the first Canadian paste fill plant at the Garson mine in 1994. Following the success at Garson, paste fill was rapidly adopted by a number of other Canadian operations such as Chimo (Vallieres and Greiner, 1995), Lupin (Hinton, 1996), and Louvicourt (Lavoie, 1995).

YVES POTVIN

Australian Centre for Geomechanics

Yves joined the Australian Centre for Geomechanics in 1998 and was appointed Centre director in 2000. In close collaboration with industry, Yves seeks to advance mine safety through the development of geotechnical research projects, further education and training courses, and training material. He is lead author of the Minerals Council of Australia's Management of Rockfall Risks in Underground Metalliferous Mines – Guideline and Reference Manual and lead editor of the ACG's Surface Support in Mining publication. Professor Potvin has also published more than 60 papers. He has more than 20 years experience in rock mechanics and mine design and has previously held positions at Mount Isa Mines, Noranda Technology Centre and Noranda Mines, Gaspe Division.

REFERENCES

Barrett, J. (2000) Paste Fill Development in Australia – Lessons and Surprises. Australian Centre for Geomechanics, Paste Technology 2000 Seminar, Perth, Australia, 13–14 April, 2000.

Brackebusch, F.W. (1994) Basics of Paste Backfill Systems. Mining Engineering, October 1994: pp. 1175–1178.

Brady, B.H.G. and Brown, E.T. (1985) Rock Mechanics for Underground Mining. George Allen and Unwin, pp. 214–218.

Bronkhorst, D., Rheault, J. and Ley, G.M.M. (1993) Geotechnical Principles Governing Mine Design at Williams Mine. Innovative Mine Design for the 21st Century. W. Bawden and J. Archibald (eds), Balkema, Rotterdam.

Clough, G.W., Iwabuchi, J., Rad, N.S. and Kuppusamy, T. (1989) Influence of cementation on liquefaction of natural sands. Journal of Geotechnical Engineering, 15, No. 8, pp. 1102–1117.

Cowling, R. (1998) 25 Years of Mine Filling: Developments and Directions. MINEFILL '98, Sixth International Symposium on Mining with Backfill, Brisbane, Australia.

Coxon, J. (2003) Development and Application of Paste Fill Using Dried Tailings. Australian Centre for Geomechanics, Paste and Thickened Tailings Seminar, Melbourne, Australia, 14–16 May, 2003.

Henderson, A., Jardine, G. and Woodall, C. (1998) The Implementation of Paste Fill at the Henty Gold Mine. MINEFILL '98, Sixth International Symposium on Mining with Backfill, Brisbane, Australia.

Hinton, M. (1996). Pastefill Operations at Echo Bay's Lupin Mine. 98th Annual General Meeting of C.I.M. Rock Mechanics and Strata Control Session, Edmonton, Alberta, Canada, April, 1996.

Kuganathan, K. (2001) Pastefill can reduce stope-fill risks. Australia's Mining Monthly, July, 2001, pp. 36–39.

Landriault, D. (2001) Backfill in Underground Mining. Underground Mining Methods, Engineering Fundamentals and International Case Studies, W.A. Hustrulid and R.L. Bullock (eds) SME, ISBN 0-87335-193-2, chapter 69, pp. 601–614.

Lavoie, C. (1995) Paste Fill at Louvicourt. Mine publication, presented at Chief Mine Engineers Conference, Noranda Technology Centre, Montreal, 1995.

Lerch, R. and Renezeder, H.G. (1984) The Development of Pumped Fill at Grund Mine. 9th International Conference on the Hydraulic Transportation of Solids in Pipes, Italy.

Le Roux, K., Bawden, W. and Grabinsky, M. (2004) Liquefaction Analysis of Early Age Cemented Paste Backfill. Proceedings Minefill 2004. 8th International Symposium on Mining with Backfill, Beijing, China, pp. 233–241.

McLeod, (1992) Arnold Black and Hydraulic Fill. Mineral Heritage Oration, AusIMM Annual Conference, Broken Hill, Australia, pp. 17–21.

MISHC (2002) National Minerals Industry Safety and Health Risk Assessment Guideline. http://misch.uq.edu.au, 2001. Minerals Industry Safety and Health Centre, The University of Queensland, Brisbane, Queensland, Australia.

Murray, R.M. (1915) Mining Methods at Mount Lyell. Proceedings of The AusIMM, 19:125, pp. 125–139.

Potvin, Y. (1988) Empirical Open Stope Design in Canada. PhD. Thesis, Dept. Mining and Mineral Processing, University of British Columbia, Canada, p. 343.

Potvin, Y. and Nedin, P. (2003) Industry Guideline for Rockfall Risks Management in Underground Metalliferous Mines. Minerals Council of Australia, ISBN 9092776935.

Skeeles, B.E.J. (1998) Design of Paste Backfill Plant and Distribution System for the Cannington Project. MINEFILL '98, Sixth International Symposium on Mining with Backfill, Brisbane, Australia.

Thomas, E.G. (1976) Fill Technology in Underground Metalliferous Mines. Australian Minerals Foundation Inc. Workshop notes (unpublished).

Thomas, E.G., Nantel, J.H. and Notley, K.R. (1979) Fill Technology in Underground Metalliferous Mines. International Academic Services Limited, Kingston, Ontario, Canada.

Vallieres, D. and Greiner, A. (1995) Paste Backfill Implementation at the Chimo Mine, 12th Underground Operators Conference, CIM, Timmins, Canada, February, 1995.