State of the art of backfill technology in underground mining excavations

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Date(18/12/2014)

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Declaration of authorship

„I declare in lieu of oath that this thesis is entirely my own work except where otherwise indicated. The presence of quoted or paraphrased material has been clearly signaled and all sources have been referred. The thesis has not been submitted for a degree at any other institution and has not been published yet."

Preface, Dedication, Acknowledgement

I would like to thank Prof. Horst Wagner for the excellent supervision and support during my work. His pointed questions and constructive criticism helped me to advance. Further on I really appreciated the interesting discussions which helped me to understand difficult questions and problems. I benefited from each conversation and could learn a lot, thank you very much for that. I really enjoyed working together with you.

Further on I would like to thank all my office colleagues who made my daily life very pleasant. Especially Tim Fröhlich supported me during my very long working evenings and encouraged me to continue when my motivation was disappearing.

Thank you, Eva Brodtrager for your nearly daily visits, for our coffee breaks and your motivation – you are adorable.

I also want to say thank you to all my other friends who helped me to keep the necessary balance between work and leisure time.

Last but not least, my parents, who gave me all their support, as always; I could not have reached my goals without you.

Abstract

When the in-situ stress state far exceeds the compressive strength of the rock mass, the maximum extraction possible from a deposit may be unacceptably low. Therefore artificial support has to be applied, which has to control mine-near displacements and local stability as well. The most widely used artificial support is backfill, which is placed in openings underground. Today in most underground mines in Austria backfill represents an important part of the mining activities. The main reason for implementation of backfill in these mines is the need to stabilize the underground openings. Backfill should though not be considered as disposal of waste, but as reutilization, as benefits are taken from its application. So the overall goal of this work is to describe the role of backfill as essential part of mining activities and to conduct a review over different application fields of backfill. As several authors highlighted the importance of the influence of binding agent addition, a certain attention is contributed to cemented fill masses. Binding agents can be added to the fill product to reach the required physical and chemical properties, whereby the most used binding agent is Portland cement.

During this work a look at different backfill classification systems is taken which are predominantly based on the material for backfill production, the backfill production and delivery methods. According to the literature, generally rock fill, hydraulic fill and paste fill are the most frequently used types of backfill. Further on the review of different bibliography about backfill showed the importance of backfill not just for stability reasons, but also for mine ventilation, climatization, as waste disposal, for higher selectivity and to avoid transportation.

In underground mining, backfill is applied in supported mining methods, generally called cut and fill mining methods. Filling of underground openings in combination with pillars plays an important role as well, as the strength of pillars, their post-failure strength and their failure behavior are positively influenced. During this work it was observed that several properties can influence the performance of backfill, according to its specified application purpose. The main influencing parameters are the mineralogical composition, the particle size distribution and uniformity index, the addition of binding agents and additives, the addition of water and accordingly the water:cement ratio. Different laboratory testing methods to control the important properties were investigated and discussed during this work.

Over all it was concluded that backfill represents an important part of mining activities. Several deposits could not be mined without backfill application and safety of works is increased as well. An important point to consider is that every backfill system is different due to different application purposes of backfill, which makes it difficult to design a general regulatory for backfill.

However, some important questions like backfill application in high-productive mining methods and backfilling and mining activities as concurrent activities were not discussed in this work and require further investigations.

Zusammenfassung

Mit zunehmender Teufe der Abbautätigkeiten im untertägigen Bergbau steigen die Spannungen auf die Hohlräume stark an. Dies kann die maximale Abbaumenge einer Lagerstätte stark begrenzen, was sehr oft den Grund für die Anwendung von Versatz darstellt. Versatz wird eingebracht um die lokale Stabilität der Hohlräume zu gewährleisten und um Gebirgsbewegungen aufgrund der Abbautätigkeiten zu kontrollieren. In Österreich stellt die Versatztechnologie in vielen untertägigen Bergwerken einen wichtigen Teil der Abbautätigkeiten dar. Der Hauptgrund für das Versatzeinbringen in diesen Bergwerken ist der Grund die Abbaue zu stabilisieren. Daher sollte Versatz nicht als Ablagerung von Taubmaterial, sondern als Wiederverwendung von Material betrachtet werden, da Vorteile aus dem Materialeinbringen gewonnen werden können. Das Ziel dieser Arbeit war somit die Rolle des Versatzes als essentiellen Teil der Bergbautätigkeiten zu beschreiben und einen Überblick über verschiedene Anwendungsbereiche von Versatz zu erstellen.

Da einige Autoren die Bedeutung von Bindemittelzugabe zu Versatzmischungen hervorhoben, wurde diesem Bereich ein eigenes Kapitel gewidmet. Bindemittel werden generell zugesetzt um die benötigten physikalischen Eigenschaften zu erreichen, wobei Portland Zement das in der Versatztechnologie am häufigsten verwendete Bindemittel darstellt. Während dieser Arbeit wurden verschiedene Versatz-Klassifikationsschemen betrachtet, welche sich üblicherweise auf das Material, die Produktion und den Transport von Versatz beziehen. In verschiedenen Literaturquellen werden „rock fill“ „hydraulic fill“ und „paste fill“ als die wichtigsten Versatztypen unterschieden. Weiterhin wurden verschiedene Anwendungsgründe für Versatz untersucht. Versatz wird nicht nur aufgrund seiner gebirgsmechanischen Wirkung, sondern auch aufgrund von Wetterführung, Taubmaterial-Ablagerung, Selektivität und Vermeidung von Transport angewandt. In untertägigen Abbaumethoden wird Versatz in „Supported mining methods“ angewandt, die man generell als „cut and fill mining methods“ bezeichnet. Ein besonders interessanter Faktor ist die Anwendung von Versatz in Kombination mit Bergfesten, da Versatz die Festenfestigkeit, die Nachversagens-Festigkeit und das Versagensverhalten positiv beeinflusst. Während dieser Arbeit wurden auch einige Versatzeigenschaften diskutiert, die die Leistungsfähigkeit von Versatz bezugnehmend auf die jeweiligen Einsatzgründe beeinflussen. Die Haupteinflussgrößen sind die mineralogische Zusammensetzung, die Korngrößenverteilung und der Gleichförmigkeitsindex, die Bindemittelzugabe und die Zugabe von Zementzusätzen, die Wasserzugabe und das jeweilige Wasser:Zement Verhältnis. In weiterer Folge wurden verschiedene Laborversuche diskutiert, mit denen man die genannten Eigenschaften überprüfen kann.

Aus der Arbeit wurde geschlossen, dass Versatz einen erheblichen Beitrag im untertägigen Bergbau leistet, dass einige Lagerstätten ohne Versatz nicht abgebaut werden könnten und dass die Sicherheit der Abbautätigkeiten erhöht wird. Es ist jedoch wichtig zu erwähnen, dass es nicht einfach ist ein generelles Regelwerk für Versatzsysteme zu erstellen, da jedes Bergwerk und somit auch jedes Versatzsystem unterschiedlich sind und auch die Gründe für die Anwendung von Versatz stark variieren. Wichtige Punkte, die in dieser Arbeit nicht im Detail diskutiert wurden sind die Anwendung von Versatz in hoch produktiven Abbaumethoden und die Betrachtung von Abbau und Versatz als Konkurrenz-Aktivitäten.

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1 Introduction

When the in-situ stress state far exceeds the compressive strength of the rock mass, the maximum extraction possible from a deposit may be unacceptably low. Therefore artificial support has to be applied, which has to control mine-near displacements and local stability as well. The most widely used artificial support is backfill, which is placed in openings underground. [1] Filling in underground openings has been practiced as long as human have been mining minerals from the earth. The earliest forms of backfill were implemented in order to reduce transport of waste material or low grade ore from underground.[2]

Today in most underground mines in Austria backfill represents an important part of the mining activities. The main reason for implementation of backfill in these mines is the need to stabilize the underground openings. [3]

Different types of backfill are distinguished, according to the material used, the use of binding agents and the delivery method. The application of different backfill types depends on the properties required for the particular application reason. For higher strength of a backfill body, binding agents, mostly Portland cement are added to the mixture, increasing the cohesion between the particles. However, not only the addition of binding agents, but also the range of properties influencing backfill performance like fines content or water content is widespread.

Several authors (Yao et al. 2012) however highlighted the importance of the properties of the binder on the mechanical properties of the backfill. That is why in the following the influence of the binding agent is discussed in detail.

The overall goal of this work is to describe the role of backfill as fill material for underground openings and to conduct a review over different application fields of backfill, the different types of backfill, its properties and laboratory tests and experimental investigations in order to get more insight into this integral part of mining activities.

2 Notation and units

Parameter	Symbol	Unit
Force	F	$\frac{kgm}{s^2} = 1N$
Compressive Strength	σ	$1\frac{N}{mm^2} = 1MPa$ $= 145{,}04\ \mathrm{PSI}$
Strain	ε	%
Young's modulus	E	$1\frac{kN}{mm^2} = 1GPa$
Density	ρ	$\frac{kg}{m^3}$
Unit weight	γ	$\frac{N}{m^3}$
Capacity	Q_c	$\frac{J}{s} = 1W = \frac{kgm^2}{s^3}$
Water content	w	%
Cohesion	c'	*Pa*
Flow quantity	Q	m^3
Flow ratio	Q	m^3/s
Uniformity index	C_u	-

3 Basic information about backfill

Yao et al. (2012) define backfill as any waste material that is placed into underground openings as disposal or for engineering purposes. According to the BVÖ-Versatzrichtlinie backfill is considered as all activities, which are included in the framework of mining activities concerning the partial or total filling of openings. [3] Backfill should though not be considered as disposal of waste, but as reutilization, as benefits are taken from its application. The reason to consider backfill as "waste" is the fact, that primarily material, containing precious substances with so low grade that their recovery is deemed uneconomical is used to fill underground openings. Contrarily to the definition by Yao et al. (2012), backfill is not worthless material, as by introduction into underground openings economic advantages can be achieved.

The reasons for using waste material as fill are its availability and its cost effectiveness. Different sources of waste fill material are mine dumps, smelter slag, fly ash, naturally broken fault material, glacial till, dune sand, river gravel and mine wastes from underground and surface mines. These source materials can be ungraded or graded and sized containing also cement and water for greater strength.

Waste material for backfill is normally less expensive than other backfill materials and by adding binding agents, a strong consolidated fill is generated. If the material is not sized and no binding agents are used, a loose, uncompacted fill is formed, which can lead to unsafe conditions in a mine. [4]

Mine backfill consists, like soil, of three different phases (liquid, gas and solid) and depending on the type of backfill and its composition, a difference in mass fraction of these three phases appears. [5]

For transport of backfill, often water is used which normally causes an excess of water, and therefore effective de-watering and drainage mechanisms to keep a low level of pore pressure are crucial factors in backfill technology. [1]

In the following work the terms backfill material, backfill product or backfill mixture and backfill body are frequently used, which requires a definition of these expressions.

Backfill material refers to the ingredients of backfill like water, binding agents and tailings which, after mixing together, represent the *backfill product*, which is ready to be transported to the underground openings to be placed.

Backfill body refers to the placed backfill product, which is compacted and hydrated (when binding agents are used) and forms a homogeneous compound.

3.1 Backfill material

Five types of backfill materials exist according to Brady and Brown (2005):

- Run-of-mill concentrator tailings (with cementing agent to form paste fill)
- Deslimed mill or concentrator tailings (sand fill)
- Natural sands
- Aggregates (coarse cohesion less media)
- Cementing agents

1

A very similar distinction is presented by the Handbook on Minefill [6] into:

- Tailings
- Natural sands
- Rock and aggregate
- Water
- Cement and Pozzolans (Binding agents)

A typical mix of backfill materials by weight contains 73% waste material from mine development, 25% water and 2% cement.[6]

The amount of necessary backfill material depends on:

- The geometry of the opening
- Type of backfill
- Backfill placement method

The underground opening, which is supposed to be filled, might be significantly smaller than the mined volume as a result of room convergence. The amount of convergence mainly depends on the dipping of the deposit, on the flexibility of the short-term support and on the time period before backfill introduction.[7]

For the amount of necessary backfill material, the fill parameter plays a major role as well. It describes the fact, that dumped material requires a larger volume than in-situ rock mass. The fill parameter is defined as:

$$Fill\ parameter\ (Schüttzungszahl) = \frac{m^3\ of\ available\ volume}{m^3\ of\ backfill\ product}$$

The fill parameter depends on the material properties, but is always >1. If multiplying the volume of the in-situ rock mass with the fill parameter, the required space for the excavated material is calculated. [7]

Excavated material	In-situ density [t/m³$_s$]	Fill density [t/m³$_r$]	Fill parameter [m³$_r$/m³$_s$]
Coal	1,3-1,6	0,8-1,2	1,4-1,6
Sandstone	1,9-2,7	0,8-1,2	2,0-2,5
Arenaceous Shale	2,6-2,8	1,3-1,8	1,5-2,0
Cross heading tailings	2,3-2,7	1,2-1,7	1,7-2,3
Comminution tailings	-	1,5-1,9	1,4-1,7
Wash tailings	-	1,4-1,6	1,5-1,7
Wash tailings (dried)	-	1,8-2,0	1,4-1,6

Table 1: Fill parameters of different rock types[7]

Several placement methods like drop fill, hydraulic fill or paste fill reach a much denser packing than pneumatic fill, which is considered in the amount of fill material as well. [7]

3.1.1 Tailings

During mineral processing of the extracted material, two main material streams are produced: the ore stream and the waste stream. To separate the worthless material from the valuable material, it has to be processed by crushing, grinding, flotation, leaching etc. The waste portion of the mineral processing is called "tailings" and contains particles from clay through silt to fine sand in particle sizing. In Figure 1 a flow sheet for the material streams in an underground mining activity is presented. From the valuable ore stream, money can be generated whereas the waste stream produces cost as it has to be disposed, mostly in storage facilities on the surface.

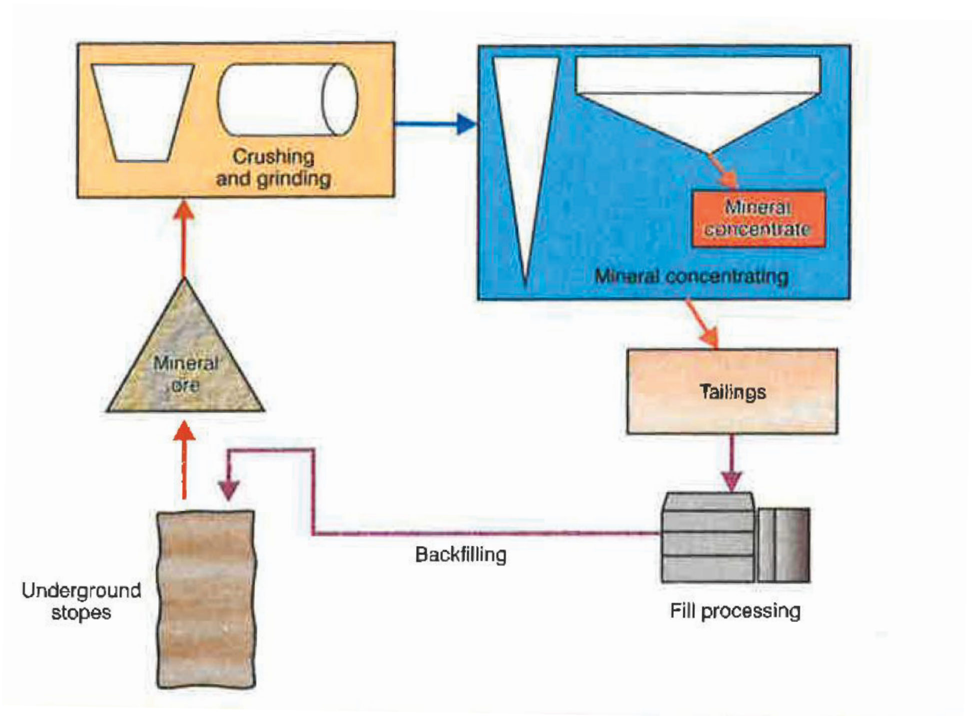

Figure 1: Mineral processing and tailings generation [6] p.26

The most efficient way to store tailings is the filling of underground openings. By the rock fragmentation process and mineral processing of the material, the rock is broken into smaller pieces which don't fit properly together. This leads to an increase in the occupied volume, which depends on the type of rock and the breaking characteristics and varies from 1,3 to about 1,8. That is why not all tailings can be used as fill and some tailings hence require storage on the surface. As part of the processing process, other materials like cyanide, lime or acid may be added to the waste stream. [6]

The final sizing of tailings depends on the nature of ore which determines the degree of comminution which is required to liberate the metal. The grinding conducted during processing and typical particle size distributions of tailings can be seen in Figure 2.

Also the particle shape and fineness of tailings represents an important factor, affecting thickening performance, consolidation properties and drainage times, when using hydraulic backfill. Mineral processing normally produces angular particle shapes, which produce a dense and competent backfill body. However, some minerals produce flat or rounded particles.

The mineralogy of tailings is an important parameter as well, because it influences the performance and properties of backfill concerning water retention, strength, settling characteristics and abrasive action. It can also influence the final strength of a backfill product by influencing chemical reactions. Silica minerals for example are very abrasive and usually cause excessive wear in backfill transport facilities. Sulfides may cause a breakdown of hydrated cement in the fill over time. [6]

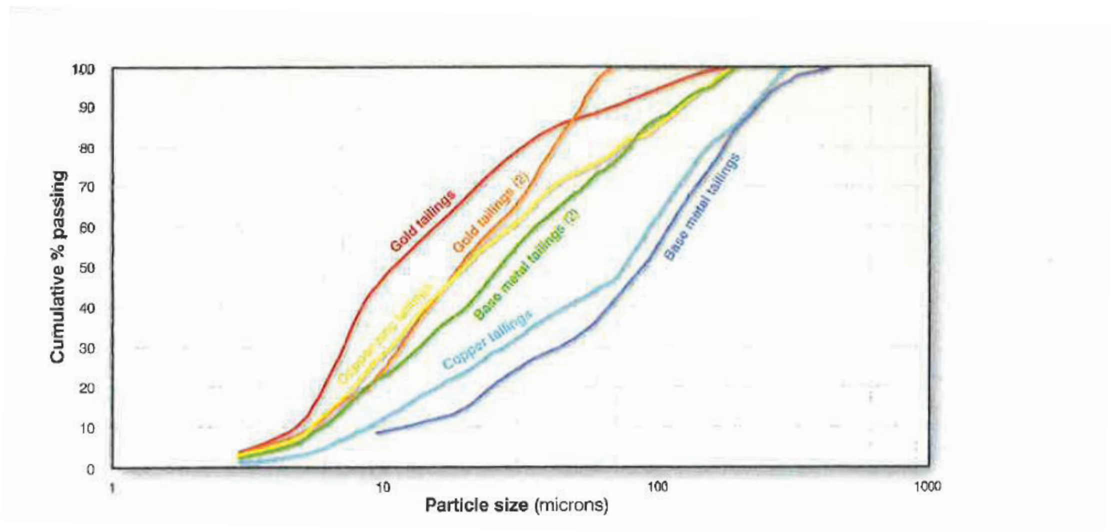

Figure 2: Typical particle size distributions for tailings [6]

3.1.2 Natural sands and rock and aggregate

Natural sands can be used as main source for hydraulic fill or as supplement material for paste fills. These natural sands are generally formed by fluvial, glacial or aeolian processes and are often high in silica with round particles. [6]

Sources of rock and aggregate can be waste rock from open cut operations, waste rock from underground development or quarried rocks and coarse gravels. Quarried rock is only used, when other cheaper suitable materials are not available. Alluvial sands can also be used if found close to the underground mine, but ecological damage can occur, when recovering these sands from the river system.

With all aggregates, moisture content is a critical parameter as water content mainly influences the performance in terms of transportation, drainage and fill stability. Aggregates also mainly contribute to the uniaxial compressive strength of the backfill material. [6]

When using aggregates or rock, grading is an extremely important factor. An efficient backfill system should contain fine and coarse rock, whereby fine rock is considered as all material < 10mm and coarse rock >10mm and <200mm. Investigations showed that the optimal fill strength was reached at 25% fine rock content. [6] Details on the influence of the particle size distribution on fill performance can be found in chapter 6.2.

Regarding rocks and aggregates, the attrition is important as it occurs through grinding in fill passes and impact after dropping. Rock attrition increases the fines content, reduces the maximum particle size and therefore influences backfill properties. Figure 3 shows the influence of a transport distance of 275m through a rise from surface to underground on the particle size distribution. A displacement of the curve to higher fines content can be observed. [6]

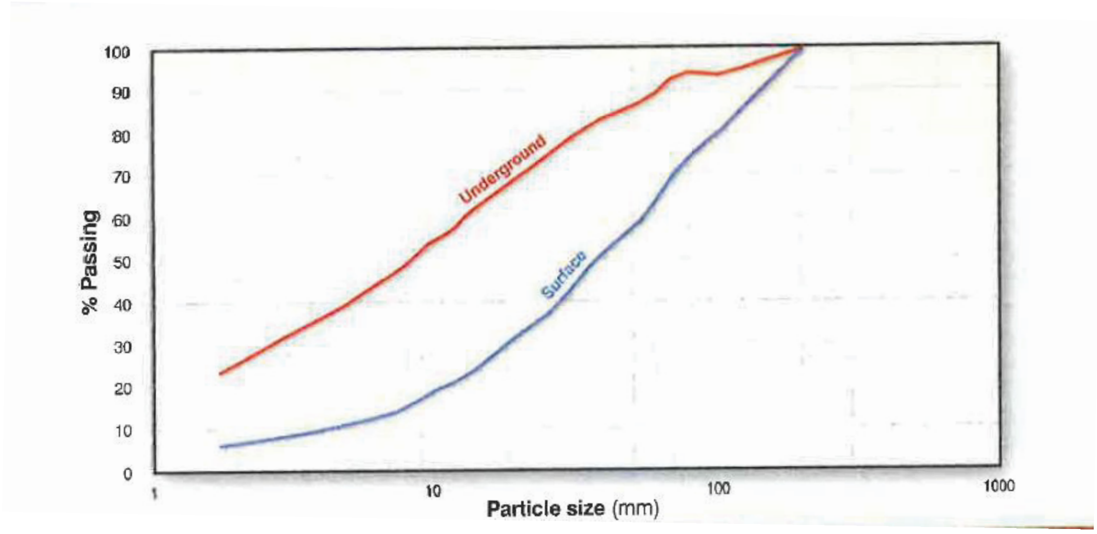

Figure 3: Influence of rock and aggregate transport on particle size distribution [6]

3.1.3 Water

Water is added to the backfill material, especially for hydraulic backfill to guarantee suitable transportation properties and provide water for hydration reaction when using cemented backfill. The presence of salt in the water can affect the strength development of cemented backfill, as during the curing process, a large amount of the salt crystallizes which might reduce the dispersion of cement and therefore the strength of the backfill body. Studies by Li et al. (2003) however showed that the strength of backfill bodies can increase due to saline water. [6] Also the pH value of water has to be considered, as it can have a negative influence on the strength development of a fill mass. Especially when sulfide tailings are used, the strength development might be affected.[6]

3.1.4 Binding agents

To reach the required physical and chemical properties of the backfill product, binding agents can be added to the backfill product. Common binders are all substances with hydraulic or latent hydraulic properties, from natural or artificial source. Natural examples are sulfide minerals like pyrite and pyrrhotite [2] or fly-ash. [2,3]

However the most used binding agent remains Portland cement, but also iron blast furnace cement, REA-gypsum, a mixture of these materials [3], clinker ash and smelter slags are used [2] Replacing some of the Portland cement in the binding agent mix by quenched slags, ground to a fineness of 300 m²/kg or greater, may heal any damage caused due to disturbance of the rapid curing Portland cement, because of their slow reactivity. [1]

The main purpose of binding agents is to increase the cohesive component of strength of the backfill material at a low addition of the substance. [1]

A detailed discussion of the influence of binding agents on backfill performance can be found in Chapter 7.

According to a Canadian survey (De Souza, Archibald, Dirige 2004) 60% of Canadian mines use Normal Portland Cement (NPC) as binding agent. The most popular alternative binder is fly ash combined with NPC (25,7%) followed by slag (11,4%). [8]

3.2 Types of backfill

Many ways to classify mine fill material exist. The most convenient method to classify backfill types is based on the raw material used and the processes of producing and delivering the fill.

3.2.1 Considerations for backfill classification

When approaching a classification, different properties of backfill can be considered:

- Addition of binding agents
- Material used
- Transportation

An important characteristic of backfill systems is the fact that binding agents can be added to every kind of backfill material. Therefore the first step of this classification approach is to distinguish between cemented and uncemented fill material. The basic backfill material is more or less the same for cemented and uncemented fill, so generally tailings from processing plants, aggregates and rocks and natural sands are used. The only difference among these materials is the different particle size but any kind of material can be cemented. Therefore a distinction among different backfill material sources is not productive. Important factors of backfill technology are backfill delivery and placement techniques. These techniques can be used for every kind of material and for cemented and uncemented backfill. Therefore the most reasonable backfill classification is based on different backfill placement methods:

- Gravity transport
- Pumping
- Pneumatic stowing
- Slinger stowing

Also called backfill using machines

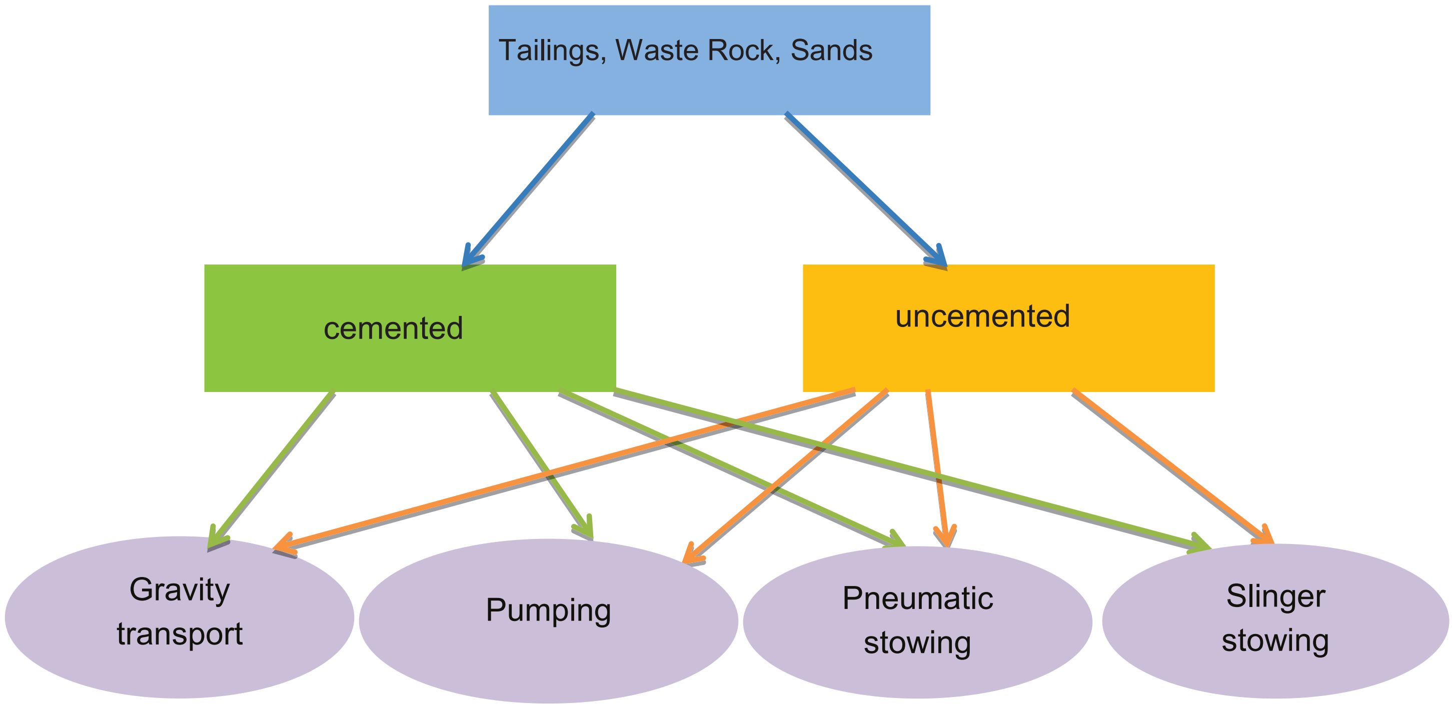

Figure 4: Backfill classification considerations

When considering literature for backfill types, different classification systems can be found. Grice (1998) distinguishes three major types of backfill: Rock backfill, hydraulic backfill and paste backfill.

"Rock backfill is a technology, which transports backfill materials such as stone, gravel, soil, industrial solid waste through manpower, gravity or mechanical equipment in order to form a compressible backfill body." (Yao et al. 2012)

"Hydraulic backfill is a technology which takes water as transport medium to carry the backfill materials, such as mountain sand, river sand, crushing sand, tailings or water quenching slag." (Yao et al. 2012)

"Paste backfill is (cemented) slurry that is prepared by mixing and stirring water with aggregate materials." (Yao et al. 2012)

Brady and Brown (2005) distinguish the same types of backfill, with an additional distinction between hydraulic fill and cemented hydraulic fill. For many years, hydraulic fill represented the most common type of backfill, but was replaced by paste fill as most applied backfill type because of the more economic use of binding agents, the disposal of a higher proportion of mine waste underground and the more homogeneous properties of the fill material. [1]

Patchet (1972) divides backfill methods into two groups: fill methods for metal mining and fill methods for coal mining. Generally fill and backfill are associated in literature with metal mining and the use of backfill material with a higher density, which is placed hydraulically in the mine. Stowing is linked with lower-density backfill materials, which are placed in coal mines for ground support and which are transported pneumatically.

In metal mining some of the most common mining methods are cut and fill (usually horizontal, mechanized and progresses upwards) and pillar recovery between stabilized fill. For these mining methods hydraulic fill is generally used. The main objectives of the introduction of backfill in metal mines are stabilization of the mine, creation of a working floor, underground filling, tailings disposal and subsidence and fire control. [2]

Coal seams are developed by means of sedimentation. When the cycle of sedimentation is short, the height of the coal seam is limited which leads to low working heights. Therefore pneumatic backfill transport methods are generally used as backfill technique in coal mines. The low value of coal per ton makes it difficult to justify expensive backfill systems, so a reduction in backfill costs is even more important in coal mines. [2] The main reason for limited use of backfill in coal mines however is the low rate of backfill placement compared to the high rate of coal extraction. As a result coal output in coal mines using backfill tends to be very low.

Different placement characteristics form an additional distinction of backfill types. Drop fill, slinger stowing, pneumatic stowing, hydraulic fill and paste fill represent the most used placement techniques. [9]

The SME Mining Handbook distinguishes four different types of backfill:

- Waste fill
- Pneumatic fill
- Hydraulic fill with dilute slurry
- High-density backfill

The distinctions by Patchet (1972) and the SME Mining handbook represent a mixture of different backfill materials and transport types, as waste fill is concerning the material “waste” and pneumatic filling refers to a transporting method. However the descriptions of “hydraulic fill with dilute slurry” and “high-density backfill” from the SME Mining Handbook match “hydraulic fill” and “paste fill”. The description of “pneumatic filling”, as it represents a fill transport method, can be found in chapter 3.4 “Backfill Transportation Systems”.

Therefore in the following the backfill types according to the distinction regarding the definition of Grice (1998) are explained.

3.2.2 Rock backfill

Rock fill consists of rock fill materials like underground mine development waste rock, overburden rocks from surface mining operations, river gravels etc.

In its simplest type, this material is dumped into the underground openings without further treatment to fill them. By filling of the openings, deterioration of the rock mass is delayed or arrested and additionally surface waste disposal is prevented. [6]

The available supply for rock fill is therefore limited by the availability of these materials. [1] For rock backfill the raw material can either be prepared before being introduced into the underground opening by crushing, sieving and mixing according to the particle size distribution or it can be unmodified[6]. In Canadian mines rock fill is the primary backfill. [8]

In its natural state, rock fill is a loose, granular medium which cannot form a vertical face when exposed. This loose material needs to be confined by stope walls. When the material is dumped in an underground opening, loose rock rills down forming a sloping face. The angle of this rock slope is called the angle of repose of a material, which is depending on physical factors like maximum particle size, particle grading, moisture content, height of dumping etc. The typical angle of repose for rock fill varies between 35-55°.

Used in its unmodified form, the material has not undergone particle grading and does not contain binding agents, which means that the strength of the fill mass is not an important criterion for the application. When unmodified fill is used and mining activities should be conducted in adjacent openings, parts of the deposit, called diaphragm walls have to be left, to prevent the rock fill to rush into the opening (Figure 6). These diaphragm walls have to be designed with sufficient thickness to withstand the lateral pressure of the fill mass. When the thickness of these walls has to be too great, binding agents might be used so not too much valuable ore is lost in diaphragm walls. [6] In Figure 5 the application rock fill in open stoping and in bench stoping is illustrated. The left figure also indicates the segregation phenomenon as a result of different particle sizes during dumping.

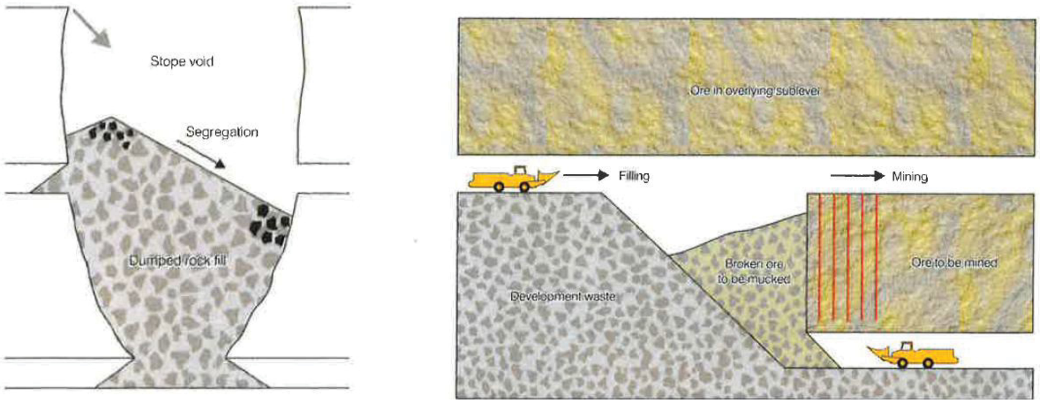

Figure 5: Application of rock fill in open stoping (left) and bench stoping (right)[6] p.102

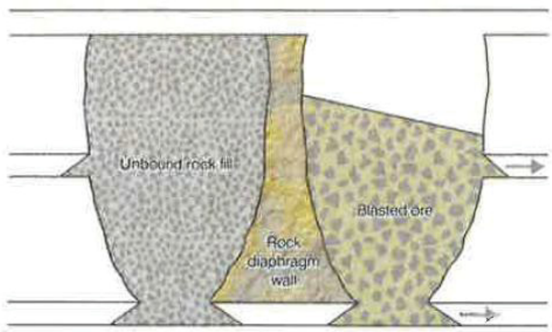

Figure 6: Rock diaphragm wall for rock fill support [6] p.10

When backfill material of higher stability or cohesion is required, the rock fill material can be modified by optimizing the particle size distribution or by adding binding agents to the fill material. [6]

Due to this modification, different rock fill types like dilating fills or contracting fills are produced. Dilating fills are rocky paste fills, which is a densely packed dilating medium. Dilating fills are self-stabilizing due to the effect of negative pore water pressured which is induced during dilation. In a confined environment, after initial failure dilating fills show a strain hardening effect, which helps to stabilize the fill mass through an increased arching effect. [6]

Contracting fills are rock fills with a very uniform size distribution. These tend to be unstable, whether they are dry, moist or saturated. A contracting fill usually has a high porosity which has a negative influence on its stability. [6]

The purpose of particle size distribution optimization is to achieve a well-graded material with a high coefficient of uniformity to form a dense packing during placement. Details about this issue can be found in Chapter 6.2. [6]

It might be necessary to blend different rock fill materials, to reach dense packing of the backfill body. [6]

Binding agents can be added to rock fill material, to achieve a self-standing backfill body when exposed. A minimum amount of binder should be used as it increases costs for backfill activities. With a correct grading and appropriate mixing, a small amount of binding agents can be sufficient to mainly increase the stability of the backfill body. [6] The combination of unsized or sized aggregates coated with binding agent slurry, especially Portland cement, with a binding agent content of 1-6% is called Cemented rock fill (CRF). A binder content below 1,5% is not sufficient for coating the dry rock material, and a content above 6% affects the economics of the backfill product. Cemented rock fill is able to carry active pressures, improves the wall rock stability and provides ground support. The water content of this backfill product is very low, so it does not seep out and additional drainage is not required. Quality problems can arise from segregation of the material, which is difficult to control. CRF is generally employed for small mining operations with a low mining rate and narrow vein ore bodies, using delayed backfill. [10]

Brady and Brown (2005) propose the simultaneous placement of dry rock fill with cemented sand fill as efficient backfill method to reduce unit costs of filling, which results out of the reduction of the total amount of the binding agent content. This material is called rocky paste fill, where the pores are filled with a mixture of tailings and binder. [6] In this case, the location of the discharge points represents a crucial parameter, as the fill mass is extremely heterogeneous and as its structure is controlled by placement conditions. [1]

Preparation of rock fill

For rock fill preparation, the mentioned waste material from surface or underground development or quarry waste can be used. This material follows a pre-determined flow path, as rock fill production plants should avoid re-handling of the material which would increase filling operations costs. The type of fill preparation depends whether the rock fill remains unmodified or modified with binding agent addition or not. Modified rock fill is generally crushed to reduce the maximum size by primary crushing, generally using a compression type jaw crusher or an impact type crusher. The amount of secondary crushing necessary depends on the degree of primary crushing achieved and on the requirements on the backfill product. Jaw crushers can reach a size reduction of 6 to 1 and impact crushers can achieve a 20 to 1 ratio.[6]

Rock fill is generally transported by gravity to the underground opening. This can be conducted by boreholes to the underground mine or trucks transporting the fill material to the underground mine or within the mine itself. Binding agents and water can be added before the transport through boreholes or on the truck before dumping.

3.2.3 Hydraulic backfill

Hydraulic fill represents the most commonly used backfill type in underground mining, especially because of its low preparation and delivery costs, depending also on the use of binding agents. Uncemented hydraulic fill is one of the cheapest bulk fill systems[6], which is predominantly used where big openings have to be filled at once, as it is very efficient (up to 500 m³/h).[7]

Conditions for the use of hydraulic fill include:

- Mining method favors sealing of the openings for dewatering of the backfill body
- Water does not have a deleterious effect on the mine climate (too low temperature) and on the surrounding rock mass
- Fine-grained tailings, which cause little pipe wear, are available at low prices[7]

Generally, hydraulic backfill is composed of fine grained hard backfill product (normally <1mm), which is transformed into a suspension by addition of water with regards to the transport of the backfill by pipelines or boreholes. The solids content of the suspension represents normally more than 70% by weight[6] (40-50% solids by volume) and flow velocities vary between 1,5 and 2 m/s (greater than critical flow settling velocity) to prevent settlement of the material from the slurry and plugging of the pipelines [1,3] (details see Chapter 3.4.2 – Hydraulic fill placement). However a too high density of the fill may cause plugged pipelines, so a high attention has to be accorded to density adjustment. [4] Typical relations for water:tailings ratio are 1:1, 1:2,5 and 5:1.[7]

The slurry transport regime is normally heterogeneous and turbulent.[6] Typical materials for hydraulic fill are concentrator tailings treated in a hydro-cyclone to remove slimes or clay-fraction size. The highest proportion of this classified product is represented by the range between 40-150 microns, whereas the <10 microns proportion normally represents less than 4%[1] (not more than 10% by weight[6]).

A typical grain size distribution of a well-graded fill product for hydraulic fill is shown in Figure 7.

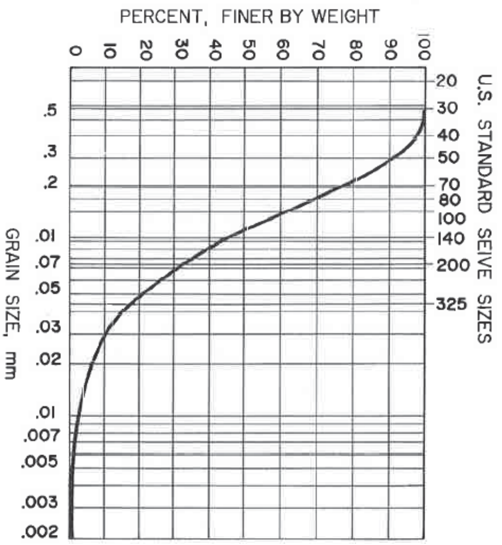

Fig. 19.3.10. Grain size distribution of a moderately well-graded fill (Dickhout, 1973).

Figure 7: Grain size distribution of a well-graded fill [4] p. 1765

In a hydraulic backfill body, internal stresses are developed by the self-weight of the fill particles. This backfill body can then resist stresses coming from rock mass displacements because of inter-particle friction. When the backfill body is fully drained and the hydraulic fill body is not planned to be exposed, binding agents are not required. However, when the fill is exposed or liquefaction risk occurs, additional strength is required and the addition of binding agents is necessary to provide cohesive strength. [6] The compressive strength of a cemented fill increases with decreased moisture content, increased cement content (Figure 8), confining pressure, fines content, decreased void ratio, when dry cured, with increase in pulp density and with age. However strength is not affected by acid mine water, curing temperature and variations in Portland cement fineness. [4]

Brady and Brown (2005) define cemented sand fill as hydraulic fill with binding agents. The function of a cemented sand fill is to increase the cohesion of the material to broaden the application fields of sand fill in mining. [1]

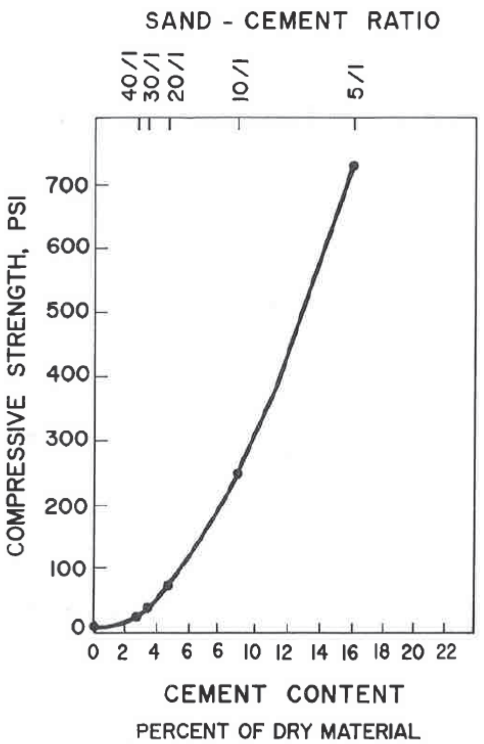

Figure 8: Unconfined compressive strength as a function of cement content [4] p.1765

Water management

When water is added to the fill material for transportation purpose, all the excess water has to be removed after placement. Especially when no binding agents are added to the fill mixture, no excess water should remain in the fill body as it reduces the performance of the fill body and causes the risk of liquefaction. Further on it has to be recovered to maximize the density of the fill material.

Generally water for backfill is recycled water from the processing plant and from backfill operations. The water is added to the backfill material on the surface and is then transported to underground in the mixture. Drainage is of special importance for hydraulic fill, because this type of fill possesses the largest amount of water.

The fill is either naturally drained or drained by pumps. Either the water drains quite quickly out of the backfill body by surface flow processes during deposition, or slowly by post-deposition drainage. Post-deposition drainage may be downward drainage into purpose-built drainage systems or upward drainage, where the water on the surface runs down towards a decant sump. [4]

Drainage systems must cope with the suspended solids load which drains out of the backfill body together with the excess water. Such a drainage system should consist of pipes for direct drainage of the backfill body, boreholes and designated storage areas. Storage areas are generally local sumps. Sumps are used to equalize the changing rate of water inflow, for storage purpose during power interruptions and periods of an increased amount of drainage water and for settlement of the suspended solids. After settlement of the solid particles in sumps, the water can be pumped to the surface to be reused. [4]

A free flow in the drainage pipes or boreholes is favored to minimize settlement of the solid particles before the main sump in the mine. Frequent cleaning of pipes and boreholes to assure a continuous drainage operation is required as well. [4]

The excess water should not be directed into old stopes or abandoned workings to avoid the risk of water inrush. The expected amount of water draining out of the backfill body has to be calculated and included in an underground mine water contingency plan.[4]

Drainage of hydraulic fill

A very important factor concerning hydraulic backfill is the management of the placement medium, which escapes from the backfill body after placement of the backfill, and which has to be discharged afterwards.[3] For this purpose the hydraulic fill has to possess an in situ permeability in the range of 10^{-5}-10^{-6} m/s and an in situ placement porosity of 50%[6]. At 50% porosity, the bulk dry density is one half of the dry solids density. However some investigations from the literature propose a permeability coefficient between $7x10^{-8}$ and $7,8x10^{-5}$m/s to guarantee a functioning dewatering of the backfill body.[1]

Fill barricades

Fill barricades are constructed to retain the solids in the fill material while allowing the excess water to drain out of the filled opening. The barricade has to be constructed in such a manner, that it can withstand the lateral pressure that the fill will impose. This means that the barricade has to possess a higher permeability than the fill mass or must have drainage fittings to allow the water to drain out of the fill body. Fill barricades are predominantly used for hydraulic fill but might be applied for rock fill and paste fill as well. Fill barricades for paste fill are called bulk heads and are generally impermeable. Bulkheads imply higher loading conditions and non-draining structures, whereas barricades describe low loading conditions and porous structures. For rock fill diaphragm walls are used as barricades to retain the material when no binding agents are used. [6]

Fill barricades for drainage of hydraulic fill represent a very important issue as poor drainage could cause very dangerous conditions. Therefore in the following, fill barricades for hydraulic fill are described in detail.

The following barricade systems were developed for fill drainage:

- arched concrete block works (porous or impermeable with drainage pipes)
- planar porous concrete blocks
- planar and impermeable reinforced fibrecrete walls
- timber barricades
- steel and mesh formworks with geotextile onto which fibrecrete is sprayed leaving permeable windows for drainage
- barricades using waste rocks

6

The design of fill barricades represents a critical parameter in a backfill system. Regarding the dimensions of openings that are supposed to be filled with hydraulic fill, in general very strong structures are required. Strong barricade structures are more easily built in small openings. The distance of the barricade to the brow of the opening should not exceed one drive width away from the brow, to balance out loading conditions and drainage conditions. Drainage is reduced with longer distances from the opening, whereas close barricades lead to higher loading conditions. [6]

The quality and condition of the surrounding rock are significant as the rock-barricade interface has a large effect on the capacity of the barricade. [6]

The time between building the barricade and filling the opening influences the curing time of the barricade. The curing time is only important when using mortar or fibrecrete for the barricades. [6]

It has to be pointed out that the construction of fill barricades is a labor intensive, time consuming and costly activity. Careful attention has therefore to be given to this aspect of backfilling.

Liquefaction

If the jetting medium is not discharged properly, excessive water in the backfill body can create pore-water pressure, which could cause sudden shear failure or liquefaction [2].

"Liquefaction may occur when the pore water in a saturated and loose granular fill medium is suddenly pressurized to shearing or shock or vibration to the extent that the intergranular contact stresses are reduced to zero, and the fill mass starts moving like a thick fluid or paste. " [6]

Liquefaction can occur if a fine grained fill, which is open structured, saturated and uncemented is subjected to a sudden shock. This sudden shock could be induced by blasting near the backfill placement or the fall of large rock blocks from the roof.[6]

Therefore a crucial factor concerning hydraulic fill represents the drainage of the backfill body, which is determined by its permeability and therefore by the amount of finest grain in the backfill product. [1,3]. Especially when rock burst or blasting vibrations occur (dynamic loads), this is of extreme importance [2]. In addition to that the transport water should be removed to reuse it afterwards [2].

Water can be removed from hydraulic fill by two mechanisms. Firstly, excess water which is collected on the fill surface can be removed by vertical drainage through perforated pipes, drainage towers and timbered raises. Otherwise, surface water may flow through the porous fill bed and can be discharged at the stope base through horizontal drains in the bulkhead in the backfilled draw point. Generally, drainage properties are particularly influenced by the permeability of the backfill body. [1]

For drainage purposes hydraulic fill barricades have to be disposed; their main function is to retain the fill solids while permitting the excess transport water to drain out of the underground opening. Therefore the wall must be more permeable than the hydraulic fill or must have drainage fittings to allow the excess water to escape from the backfill body. Additionally the barricade must possess the capacity to withstand the lateral pressure which is imposed by the hydraulic fill. [6] Therefore precautions for the long term control of the water pressure behind the backfill barricades have to be taken as well.[11]

Brady and Brown (2005) refer to hydraulic backfill as “sand fill” and distinguish cemented sand fill and ordinary sand fill. Making this distinction, sand fill is always represented by a cohesionless material with a resistance to deformation due to friction. The angle of friction of a material depends on the angularity of the particles and the packing density of the material. In hydraulic backfill, the transport water produces a loose fill structure with a void ratio of 0,7, which corresponds to an in situ dry unit weight of γ_d=15,7 kN/m³ or a dry density of 1,6 [t/m³] The peak angle of friction is about 37° in this condition. Reducing the water content also leads to a significant cohesion, because of suction developed in the pores of the dilatant medium, when subjected to a load change. These conditions would allow free-standing vertical walls of limited height to be temporarily stable. [1]

Preparation of hydraulic fill

The tailings used for hydraulic backfill are very often a product from milling operations. This slurry has to be passed through a thickener to recover some of the processing water. By the thickening process the slurry density is increased to around 50% solids by weight. Hydraulic fill plants then take mill tailings slurries from the final tailings discharge circuit to perform dewatering of the slurry and to remove the finest fraction of the tailings material. Optionally the fill plant has a binder addition facility. By dewatering the water content of the slurry is minimized, to reduce the amount of water that has to be drained after fill placement. By this dewatering process the solids content by weight rises up over 70% (45-50% solids by volume). Commonly hydrocyclones are used for dewatering in backfill plants, which use a vortex mechanism to achieve the classification process. In Figure 9 the particle size distribution curves from cycloned material can be observed. Further on reference points for hydraulic fill and paste fill are plotted. The sizing limit for hydraulic fill is not more than 10% passing <10 microns, whereas the sizing limit for paste fill can be found at maximum 15% passing <20 microns. [6]

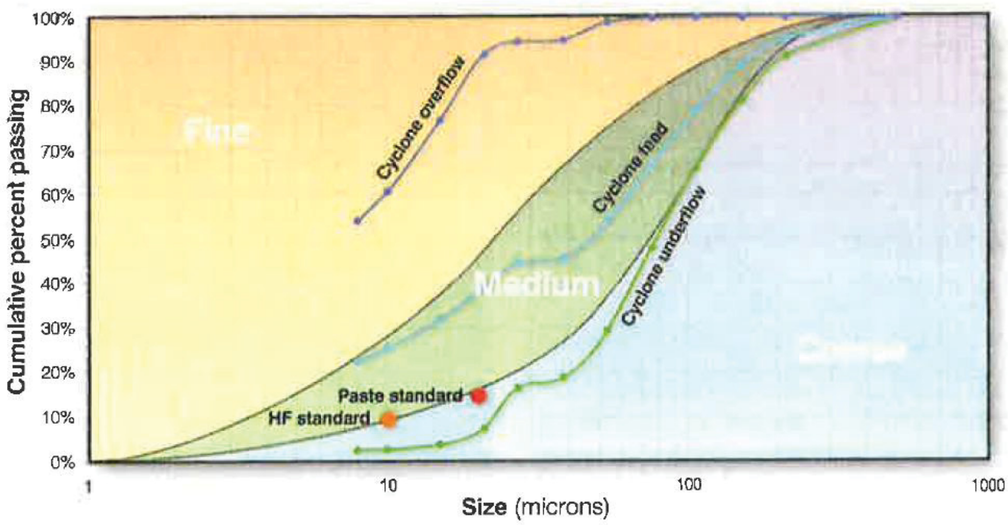

Figure 9: Particle size distribution curves [6] p.71

To reach permeability targets, the finest fractions are removed from the slurry. [6]

3.2.4 Paste backfill

Mill tailings tend to become finer and finer to increase metallurgical recoveries and so to increase processing of previously uneconomic ore. The finer mill tailings become, the smaller the portion of these tailings that can be used for hydraulic fill can be.

In addition to this factor, deep mines are subjected to high stresses and therefore require a strong, dense, consolidated fill to resist closure and rock bursts.[4] To meet these demands and to overcome some of the disadvantages of other backfill types, paste fill was developed.

Paste backfill is mostly composed of the complete solids content of de-watered run-of-mill tailings and cement and possesses a high fines content (>15% by weight <20 microns) with a sufficient water content to form a high viscosity paste[1], reaching uniaxial compressive strengths of 1-2 MPa, which is sufficient for the support of the surrounding rock strata.[7]

Coarse rock, gravel or waste can be added to the mixture to increase the strength of the fill material, which is then called paste aggregate fill or paste rock fill.[12] The critical aspect of coarse material added to paste fill is the interference with the flow behavior and therefore the size of the largest particles depends on the placement method. The SME Mining Handbook proposes particle sizes of maximum 25mm if pumps are used and maximum 1/3 of the pipe or borehole diameter for gravity transport.[4] According to the Australian Center of Geosciences, the maximum particle size should be limited to 1/5 of the minimum pipe or borehole diameter, depending on material properties, particle size distribution and particle shape as well.[6]

To form a high density non-settling slurry, the material should contain more than 65% solids, typically between 78 and 85% (solids by weight) which then can be transported underground, either by gravity or by pumps [1,3]. The paste can be considered as non-segregating slurry, which means that it has negligible excess water and has a homogenous appearance. [1,6]

The control of the fines content represents a critical parameter, as the fines fraction forms the transport medium for the coarser fractions. [4] These fine particles retain the water and even if the paste comes to rest, the flow characteristics of the paste remain the same and therefore no critical velocity exists for paste fill.[6] However to reinitiate the flow of paste fill, sufficient shear yield stress is required. The longer the paste is left idle, the greater the required yield stress to reinitiate the flow will be (Figure 10). The yield stress is a function of the tailings properties and the water content, which when increased, reduces the yield stress.[6]

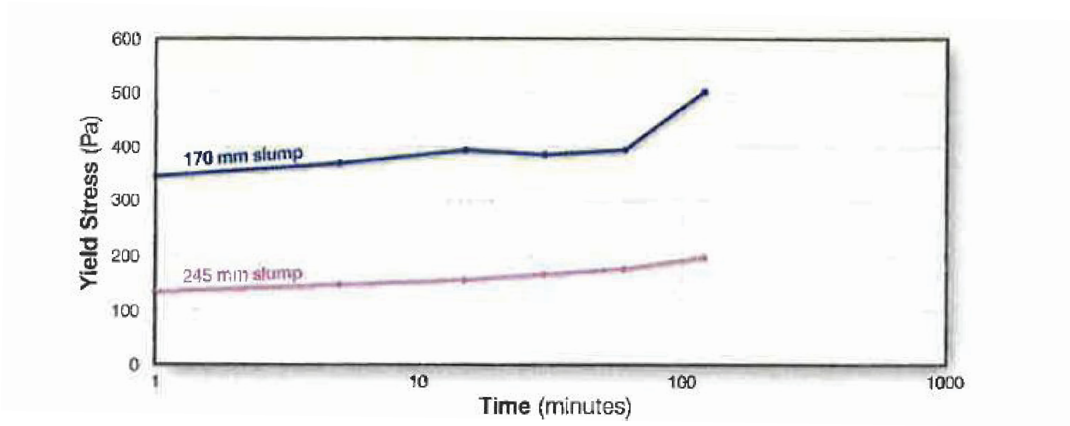

Figure 10: Increasing yield stress with time [6] p.84

The need for at least 15% passing 20 microns is required to retain the water, transport the coarser fractions and to maintain plug flow, which is a characteristic flow type for paste fill. Plug flow is characterized by a slow moving annulus of fines that coat the pipe walls and a central plug which moves at a higher velocity (more details in Chapter 3.4.3 "Fluid mechanics of paste fill"). As no critical settling velocity exists for paste fill, the flow velocities can be reduced, which results in lower pipe wear. [6]

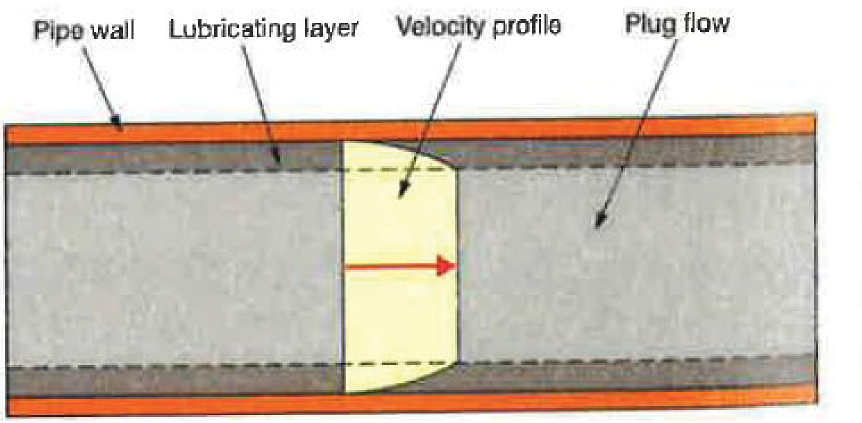

Figure 11: Flow profile of paste fill [6] p.84

However, because of the high fines fraction, an elevated risk of liquefaction arises, which requires the absolute need of cementing agents. For paste fill binding agent contents vary between 1-5%[1] and when cement is used with a low addition of water a maximum strength of the backfill can be achieved.[4] Higher cement contents can be added as well, when the fill body will be exposed as a result of future mining activities. The Australian Center for Geosciences even report about cement contents up to 10% which has an extreme influence on the paste fill production costs. [6]

The two key aspects of paste fill are reliable transportability and fill stability. To improve transportability, more water can be added to the paste, however which reduces the efficiency of binders and increases the risk of water separation from the paste. Therefore the water content has to be optimized, to maximize fill stability (the water must be sufficient to hydrolyze the cement: w:c ratio of 0,4-0,5 which generates the greatest strength[1]) but also to guarantee a reliable paste transport to underground openings.[6].

Trafficability of paste fill is generally poor, as water can make the surface slippery and mobile equipment can cause deep ruts in the fill surface. To improve trafficability of paste fill, a meter of waste rock can be placed on its top, which increases fill handling costs.[6]

The rheology of the paste is not only a function of the fines content and the water content but of the chemistry and mineralogy.[3] The mineralogy influences the water holding capability of the tailings and solids content required forming a paste. The mineralogy can also influence the final strength of the paste fill body.[6]

Preparation of paste fill

Depending on the degree of desired support, the preparation of paste fill varies. The first step of the fill preparation is the processing of the components. Tailings coming from mill processing generally possess a very high water content, which has to be removed before mixing with other components. For this purpose mechanical and natural dewatering can be used. For mechanical dewatering thickeners and filters are the main dewatering facilities. [6]

Thickening relies on gravity settlement of the solids of the mill tailings, which settle to the bottom of the thickener and can there be removed as underflow. The remaining water overflows at the top. A thickener can produce slurries with a solids content of 60-70% by weight. [6]

Filtration is often the second step of the dewatering operation. Filters possess a porous surface, which is used to retain the solid particles from the slurry, but which allow the water to pass. The solids which remain in the filter are called the filter cake and the passing liquid is called the filtrate. For filtration of paste fill, generally disc filters, belt filters and drum filters are used. [6]

After dewatering of the tailings, the components have to be mixed, which is the most important step in paste fill preparation. The filter cake, binding agents and water are mixed together to form a homogeneous slurry. Binding agents are added to the mixer via a screw conveyor. [6] The cement can be added near the end of the transporting line as well. If the cement is added near the end of the transporting system, problems with plugs are prevented and the density of the slurry can be higher. [4]

For fill placement a positive displacement concrete or mud pump or gravity transport is used when the preparation plant is located on the surface. If horizontal distances are too great, secondary pumps have to be installed underground.[4]

3.3 Filling process

Four stages in a filling process exist:

- Stope preparation
- Filling the stope
- Curing
- Water removal/recovery

The stope preparation is mainly linked to the construction of fill barricades or draw points. For paste fill simple barricades are required but hydraulic fill on the other hand requires robust and stable barricades, which withstand the lateral pressure of the fill mass. These barricades have to be placed in draw points to retain the fill masses and have to permit drainage of water when hydraulic fill or paste fill are used.[6]

The installation of the transporting system can be conducted during mining activities.

A continuous filling process of the stope is desirable; however the level of the fill mass should not rise too quickly not to overload barricades or the underlying fill. If cemented fill is used, also a certain time for curing is required. The final strength is achieved after 28 days, but 50% percent of the fill's final strength are achieved within 3 days. [2,6]

Water removal systems normally consist of a clear surface water recovery system and/or an underdrainage system. Surface water is removed by a pontoon-mounted pump or a submersible pump, which is raised up when the backfill level rises. Underdrainage water recovery systems can be installed in the base of the mine.[6]

3.3.1 Fill rate

Generally a quasi-linear relationship can be found between backfill pour rates and mining rates[6] but also the supply of tailings and waste material is of extreme importance.[6] Usually underground openings should be filled as fast as possible, to preserve the stability of the excavation or to be able to continue mining activities in secondary stopes. [6]

According to a survey in Canada [8] the backfill rate in surveyed mines ranged from 500 t/day up to over 5000t/day, with 78% of operations at pour rates between 2000 and 5000 t/day with 36% of the mines working at a backfill rate between 1000 and 2000 t/day. [8]

3.3.2 Fill ratio

The fill ratio is an important parameter for the description of how much of the excavated opening is filled with backfill material. It is defined as:

$$Fill\ ratio = \frac{Introduced\ material\ [m^3{}_r]}{Volume\ of\ excavation\ [m^3{}_s]}$$ [7]

In general for backfill practices, the fill ratio can be found between 0,5-0,8. The fill ratio becomes higher with an increasing amount of fine material.

However the fill ratio is no measurement parameter for the resistance of the backfill against convergence.[7]

3.4 Backfill Transportation Systems

The choice of the backfill transportation system mainly depends on the backfill type, amount of material used and the transporting distance. In general it can be distinguished between transportation to the mine and transportation within the mine. Equipment for delivery into the mine can be:

- Containers or bins
- Rail haulage
- Haulage by trucks
- Pipelines
- Draw point raises
- Underground silo (no delivery to underground)

For delivery within the mine predominantly pipelines and trucks are employed.

According to Reuther (1989) the classification of backfill placement types can be done as follows:

	Backfill type	**Placement method**
	Hand fill	By hand or with shovel
Gravity	Drop fill	By gravity on slopes or chutes
	Hydraulic fill	By gravity through pipes
Hydraulic fill	Hydraulic fill	By gravity or by pumping through pipes with a considerable amount of water

	Paste fill	By a pump through a pipe with low water content
Machine fill	Pneumatic stowing	By pneumatic pressure through pipes
	Slinger stowing	By motor-driven short-belts

Table 2: Classification of backfill placement types[7]

3.4.1 Gravity placement or drop fill

Hydraulic fill, paste fill and dry rock fill can be placed by gravity in underground openings. Gravity placement is generally applied for steeply inclined or steeply dipping deposits. Rock fill is generally transported by trucks, dumping the material into an underground opening or by chutes. Hydraulic fill and paste fill can be delivered through pipes with consideration of friction losses, critical settling velocity and static head influencing the pressure distribution in a backfill system (Figure 12).

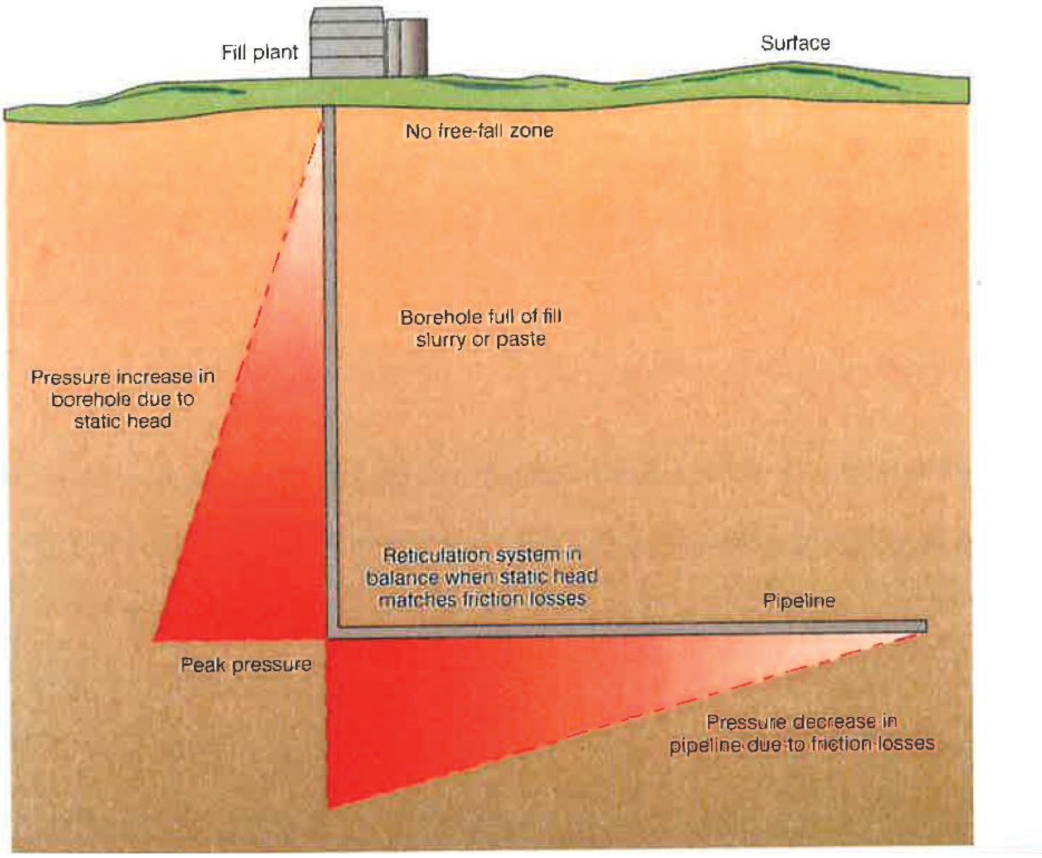

Figure 12: Pressure distribution in a backfill transporting system[6] p.55

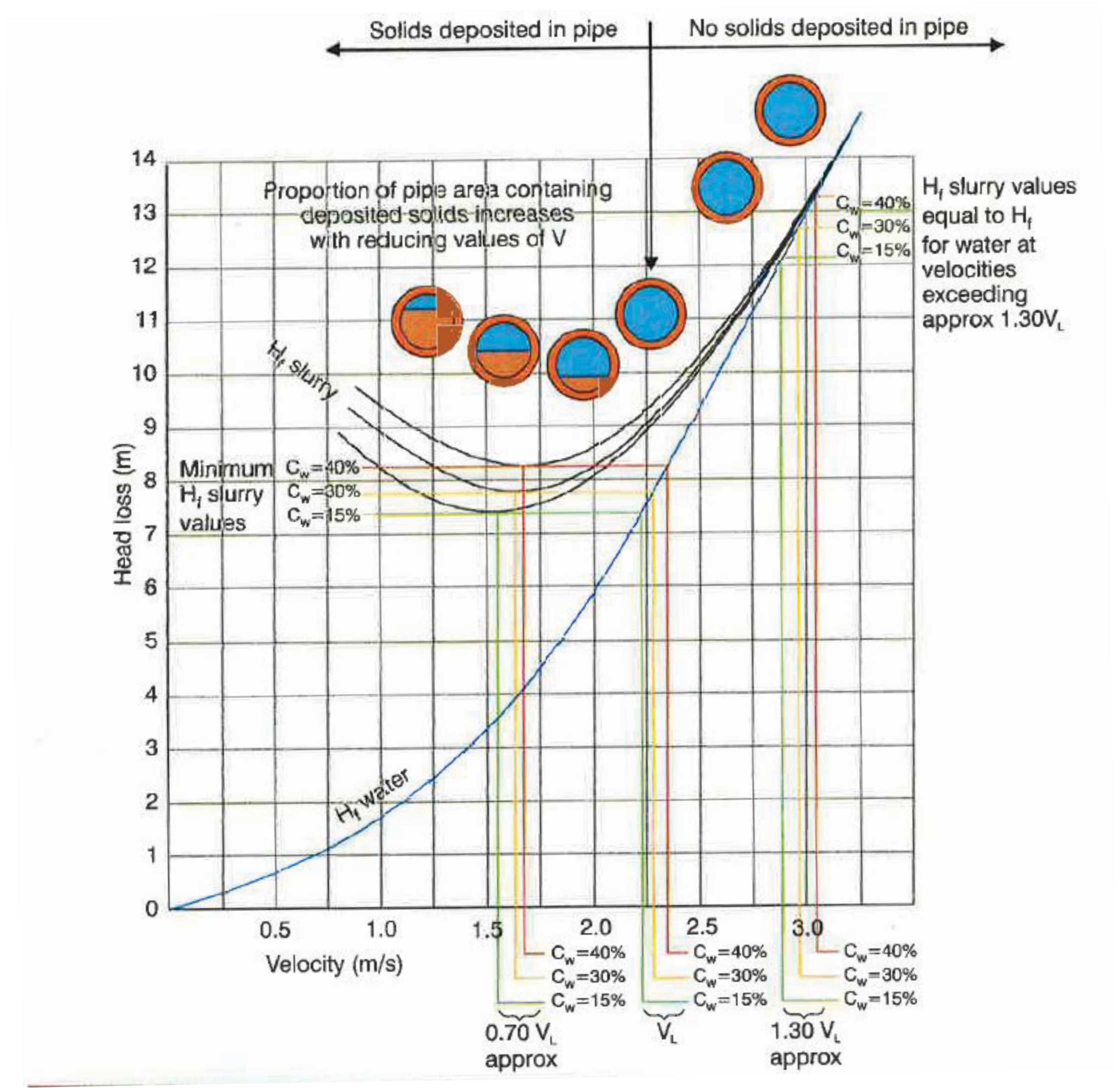

Figure 13: Typical head loss curves for Hydraulic fill[6] p.56

When using gravity transport for hydraulic fill the head loss in the horizontal pipes predominantly depends on the flow velocity and the solids content in the mixture. The head loss in the horizontal pipes as a function of the flow velocity and of the solids content is described by Figure 13. The lowest head loss can be found at approximately 0,7 x V_L for solids contents of 15% solids by weight. However this means that a significant amount of solids settles in the pipe during transport. When no solids are deposited in the pipe and the velocity of the mixture exceeds 1,3x V_L the head loss of the slurry equals the head loss of water.[6]

Considerations for hydraulic fill placement concerning critical settling velocity, friction losses, static head, discharge points and dewatering are the same for gravity placement and hydraulic fill placement by pumps and will therefore be described in the next chapter. Paste fill placement by pumps and gravity is discussed in chapter 3.4.3.

3.4.2 Hydraulic fill placement (by pumping)

The placement of hydraulic fill has to be designed so that the excess transport water is able to drain out to leave a porous fill mass with residual moisture content. Therefore hydraulic backfill is placed in underground openings by a fill hole in conventional steel and rubber pipes[2] entering the open stope through the crown of the stope. As the filling progresses, water accumulates at the surface of the fill. All the water entering the stope with the hydraulic fill, has to be removed through the bulkheads as seepage water or as decanted water (see scheme Figure 14). [6]

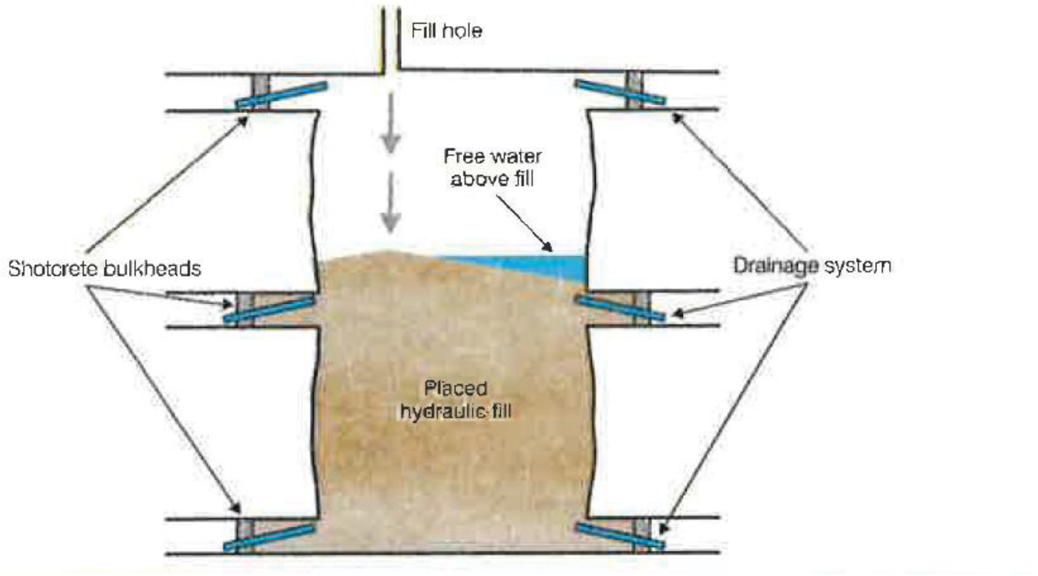

Figure 14: Scheme of hydraulic fill placement [6] p.24

Sometimes pumps are required to transport the hydraulic fill. Therefore generally centrifugal pumps are chosen when high volumes and low pressure heads are present. For long-distance pumping sometimes high pressure positive displacement pumps are required.[6]

When introducing hydraulic backfill, the material is discharged into the opening at different points with the objective to reach a certain distribution of the fill in the opening. After discharge, segregation occurs and coarser particles will settle close to the discharge point whereas fine particles stay in the flow and are transported further on. As binding agents are part of the fine particles, the binding agent content is irregularly distributed which leads to cement-lean and cement-rich zones in any horizontal plane through the fill mass (Figure 15). The different settling rates of coarse and fine material also lead to the development of a sedimentary structure in the mass, with high cement content at the top and low cement content at the bottom of the material. [1]

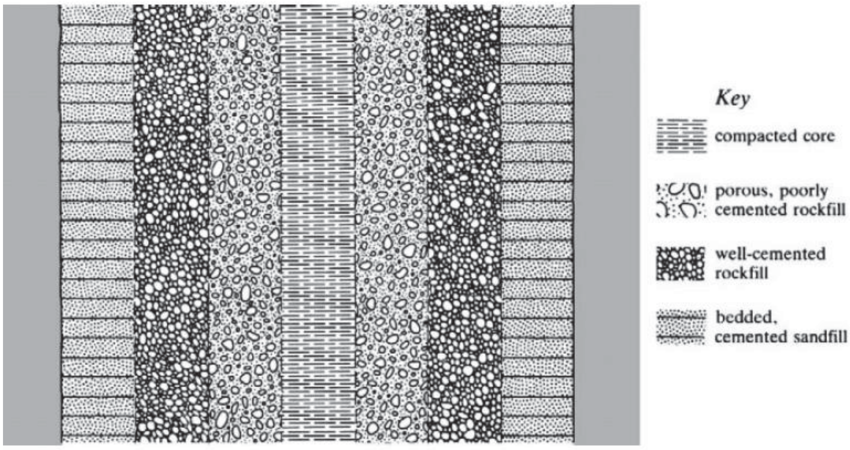

Figure 15: Structure of a composite cemented sand fill [1] p.414

Fluid mechanics of hydraulic fill

When transporting hydraulic fill, the goal is to maximize the density of the hydraulic fill slurry so that it can be transported to the limits of the underground mine, but at the same time preventing blockages or line breakages in the pipes. Modern hydraulic fill slurries typically have fill densities between 45 and 50% of solids by volume. These slurries possess a critical velocity, under which settling of the solid particles occurs. The critical settling velocity C_V is defined by Durand (1953):

$$C_V = F_L[2gD(s-1)]^{0,5}$$ [6]

g…gravitational constant: 9,81[m/s²]

D…internal pipe diameter [m]

s…specific gravity of particles

F_L…Durand's settling velocity parameter [%]

In literature sometimes V_L is used to refer to the critical settling velocity. Figure 16 shows a graph, which can be used to estimate the critical velocity of a material as a function of the particle diameters.

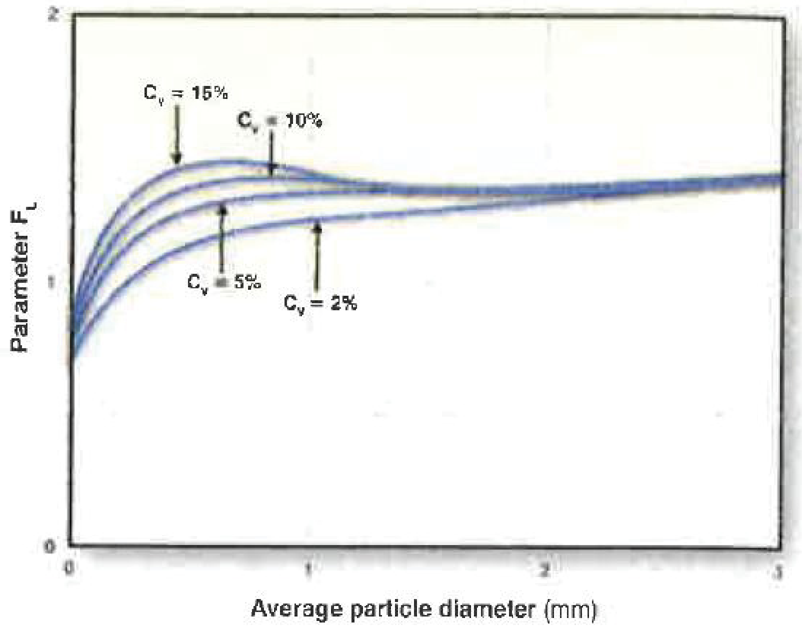

Figure 16: Critical fill velocity parameters [6] p.52

To describe the behavior of hydraulic fill during transportation through horizontal pipes, four flow regimes (Figure 18) were distinguished:

- Homogeneous flow: constant particle concentration across pipe cross-section
- Heterogeneous flow: no constant concentration of particle across pipe cross-section, particles are suspended by turbulence within the flow
- Moving bed: particles move along pipe invert as a dispersed bed
- Stationary bed: a stationary bed of particles remains in contact to the pipe invert

The type of the flow regime mainly depends on the average particle diameter, the flow velocity and the density of the slurry. In Figure 17 the four flow regimes are displayed as a function of the average particle diameter and the flow velocity. [6]

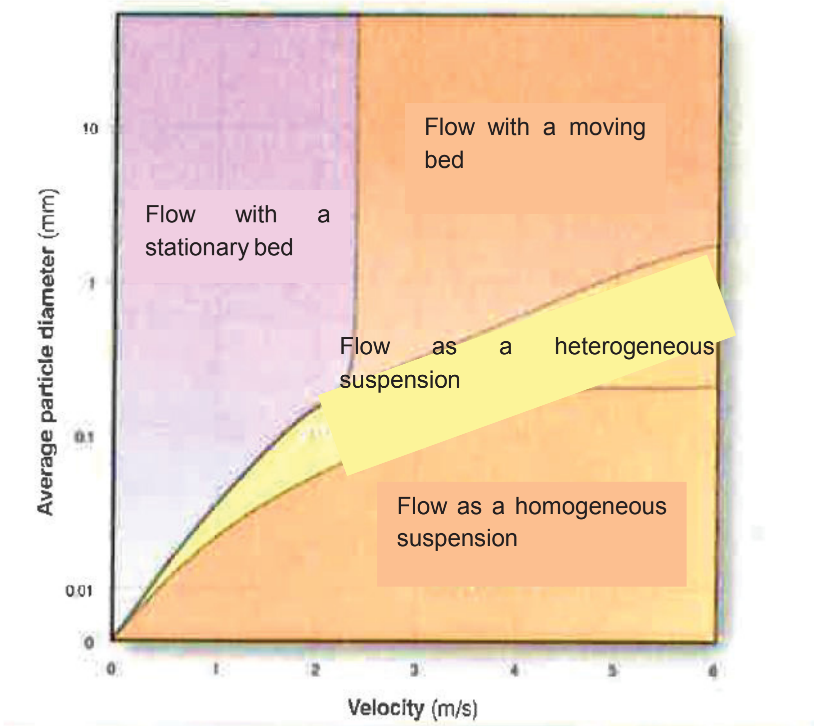

Figure 17: Flow regimes as a function of velocity and particle diameter [6] p.52

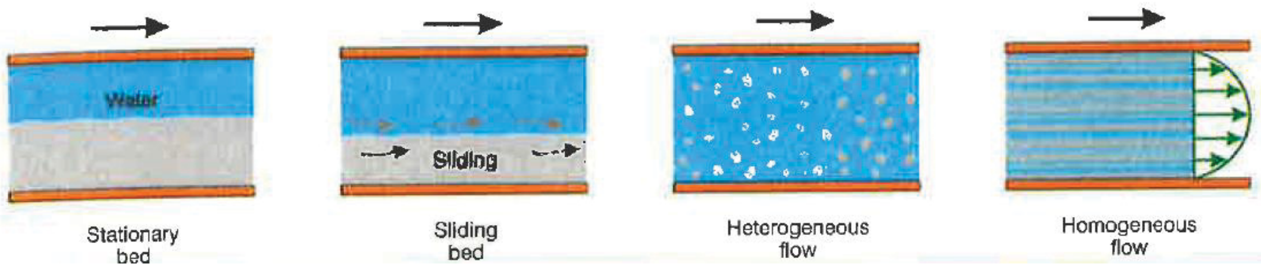

Figure 18: Flow regimes [6] p.53

3.4.3 Paste fill placement

For the transport of paste fill, gravity delivery systems are preferred when the static head is sufficient to overcome the total friction losses. When pumps are required, high pressure positive displacement pumps are used. [6] As paste fill has no critical flow velocity, it can be transported at any flow rate as long as the pressure is sufficient to overcome the borehole/pipeline pressure losses. So the main concerns of paste fill transporting design are the system operating pressure and also the placement rate. The transport ratio is a function of the friction losses and of the placement rates. The range can be found between 5-15kPa/m, with a typical value of 8kPa/m for gravity transport. For gravity transport the vertical head should be maximized and therefore paste fill should be placed always at least 100m from the surface. [6]

To calculate the horizontal transporting distance, the peak pressure due to the static head must be divided by the friction losses per meter in the pipeline. The friction losses strongly depend on the solids content of the mixture and on the mineralogy and geometric properties of the tailings and waste material. An average density of 1900 kg/m³ (2400 kg/m³ would be typical for concrete) is supposed for the backfill material.

Vertical transporting distance [m]	100	200	500
Horizontal distance [m]	230	450	1000

Table 3: Horizontal transporting distance for 100m, 200m and 500m vertical transporting distance

The presented distances assume a completely linear transport in a pipeline without turns or transitions. Therefore the results might be strongly overstated.

Either boreholes or pipes are used for paste fill transport. Typically, two boreholes are drilled from the paste fill processing plant to the underground mine, to have a backup hole if one becomes blocked. Vertical holes are more susceptible to deadlock because of linear damage as a result of wear. Inclined boreholes (60-70°) show decreased wear rates and therefore the blocking hazard is reduced.

If pipes are used for transport, they have to be made of steel as a result of high operating pressures. The size of the pipes is a function of the geometry of the paste fill system and of the fill rates.

Where the circumstances don't allow gravity transport, pumps have to be installed to deliver enough pressure for fill transport. All delivery pumps are high pressure positive displacement systems, which are hydraulically powered. Piston or reciprocating pumps possess the highest flexibility with maximum pressures up to 15 MPa and a maximum particle size up to maximum 100mm, which is a very unusual particle size for fill material. The diameter of the boreholes and pipes is a function of the particle size and required delivery and varies normally between 150 and 250mm. [6]

After the fill operations are completed or before a shutdown of the filling system, a volume of water and compressed air is introduced in boreholes or pipes for cleaning of the line.[6]

Fluid mechanics of paste fill

Paste fill behaves as a non-settling slurry, which means that solids don't tend to settle on the pipe invert as a result of too low flow velocity. Therefore no critical flow velocity for paste fill slurries exists. Paste fills generally possess a solids content between 75 and 80% solids by weight. The critical parameter concerning paste fill flow is the pulp density and the wall shear stress. Flow will occur when the wall shear stress is exceeded. The shear stress in the fluid is maximum at the pipe walls and zero in the center line, distributed linearly across the cross-section as the flow regime for paste fill is laminar. At the pipe wall the effective velocity is zero but it increases until the critical radius r_c, where the shear stress equals the maximum shear stress and the paste will flow in a laminar way. Outside the critical radius, the shear stress is insufficient to shear the fluid and therefore this part of the paste fill flows as a coherent plug (Figure 19). The yield shear stress is of extreme importance as it mainly influences the flow behavior of the paste fill. For paste fill, the maximum shear stress is exponentially proportional to the pulp density of the fluid (Figure 20).

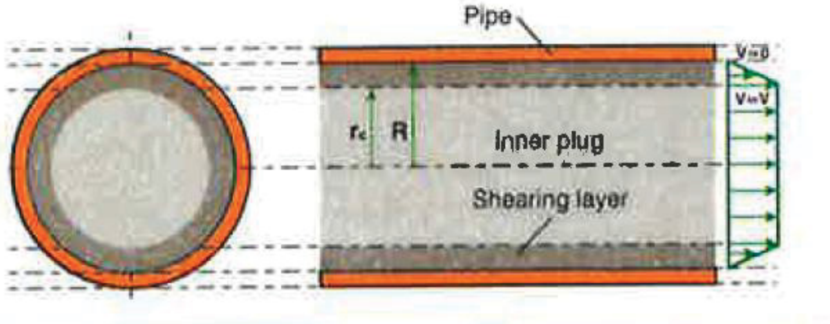

Figure 19: Paste fill flow behavior in a pipe [6] p.53

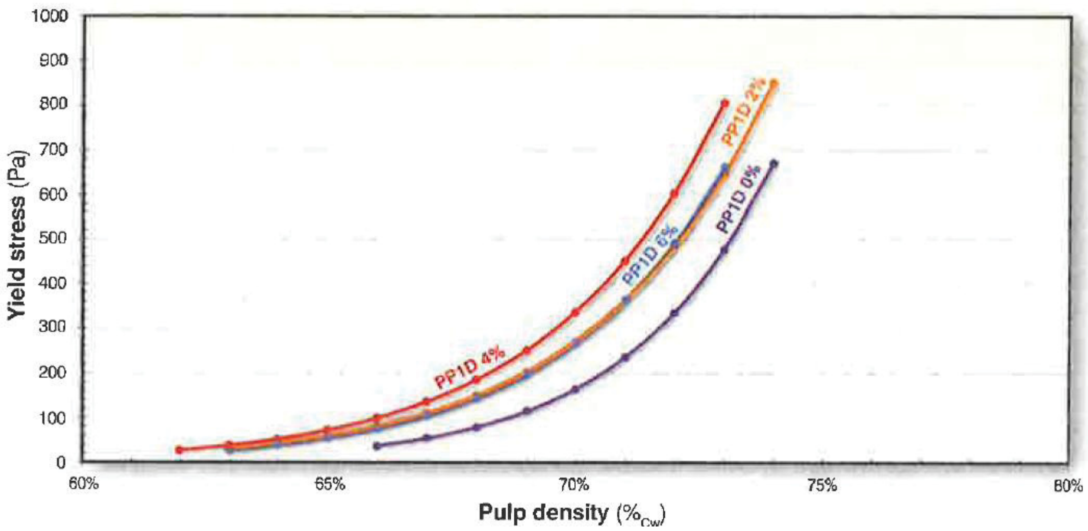

Figure 20: Maximum shear stress as a function of pulp density

The low velocity at the pipe walls is desirable as it leads to minimal wear rates on the pipe.

Turbulent flow is avoided because of higher friction losses and increasing pipe wear. Turbulent flow only exists in the free-fall zones between the surface borehole collar and the top of the paste column of the borehole. These free-fall zones have to be reduced to a minimum to keep friction losses and pipe wear as low as possible.

Abrasion in pipelines for transportation of paste fill

The deeper mining activities become, the longer transporting distances for backfill will be. In general a gravity pipeline transportation system consists of vertical drill-holes and horizontal pipelines. The driving force of the slurry comes from the static pressure due to the height of the slurry column. In such a system the slurry transportation can occur in three different phases (Figure 21):

1. When the slurry does not have enough pressure to overcome the pressure loss along the way (system does not flow or becomes clogged)
2. When the slurry pressure balances out the resistance (system is in a state of flow)
3. When the natural pressure is too large (upper section of vertical pipeline is in free-fall flow)

In many mines the slurry is in a free-fall state in the vertical pipeline sections, to ensure sufficient pressure for the horizontal transport which on the other hand causes high wear of the pipes. The slurry is in a free-flow phase until it reaches the air-slurry interface which results in free-fall flow in the upper part of the vertical pipeline and in full-flow in the lower part (Figure 21).

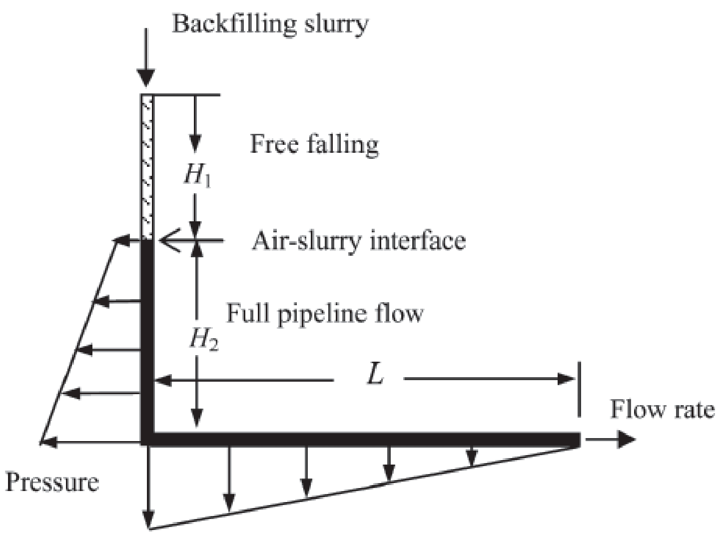

Figure 21: Flow phases in vertical pipelines[13] p.213

In the free-fall section the maximum velocity might reach 50m/s or higher, which causes rapid abrasion of the pipeline. If the vertical pipeline is inclined, the abrasion will be even more serious in this section. The pressure at the air/slurry interface is tremendous and might even cause pipeline rupturing (Figure 22 b).

Usually free-fall sections should be avoided as they cause high wear and compaction of the fill product. Therefore a useful practice is to fill the pipe with water and introduce the slurry.[14]

In the full-flow section, the velocity of the paste is uniform and there is almost no impact abrasion. The only abrasion is this section occurs where the paste fill is in contact with the pipeline's inner wall (Figure 22 a).

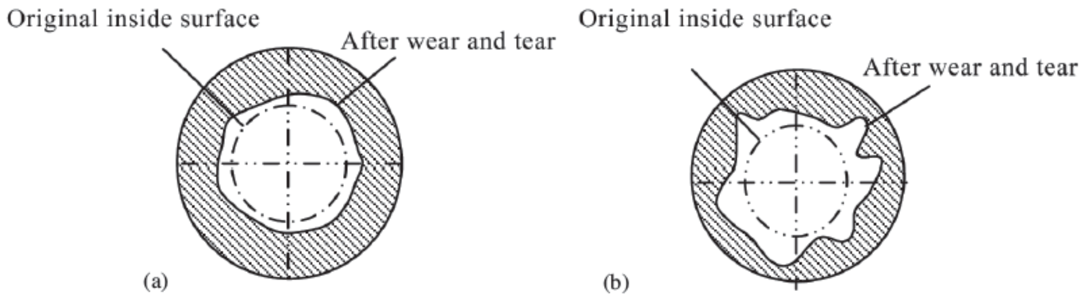

Figure 22: Pipeline abrasion in full-flow conditions (a) and free-fall conditions (b)

The main form of abrasion in deep mines is partial impact abrasion, which results from the slurry moving at high speed in the free-fall section. However, the highest rate of abrasion can be found in the full-flow section. In general Wang (2014) states that the higher the free-fall section the more serious the pipeline abrasion will be.[13]

3.4.4 Machine placement methods

Pneumatic stowing

Pneumatic filling means the disposal of mine tailings by using pneumatic conveying techniques at a maximum introduction performance of 250 m³/h. The sources of pneumatic fill are more or less the same as for waste fill: mine dumps, mill tailings, smelter slag, mine waste etc. Like for waste fill, cement can also be added to pneumatic fill with an appropriate moisture content to increase strength and stability. [4] Pneumatic stowing is often applied in flat or slightly dipping deposits like in hard coal underground mines using longwall mining, however it is applied in metal mining as well.[7] A pneumatic conveying system consists of an air supply, an in-feed arrangement, a pipeline (diameter: 175-225mm[7]) with necessary elbows and a discharge. Air at high pressure, provided by a reciprocating compressor, is used to transport fine-grained materials that are readily flushed in pneumatic conveyors. This results in an excessive power requirement, especially for long distances. The transport pressure of the air is low and varies between 34 and 138 kPa. This airstream forms a suspension with the particles and carries them through pipelines. Larger particles bounce along the bottom, whereas fine particles are carried in the airstream, and intermediate particles in between. Consequentially saltation occurs and slugs are generated, which results in an inefficient flow. Therefore kickers are installed, which direct the material towards the center of the airstream in the pipeline. The pipeline diameter should be larger than 3 times the largest particle size. (pipe diameter - usually 203 mm) according to the SME Handbook[4] and according to Reuther (1989) twice the largest particle size[7]. Due to the high friction factor of the sharp particles, abrasion-resistant steel pipes with a hardened inner surface are recommended for pneumatic transport.[4] When the backfill material consists of sandstone tailings, the wear of the pipes increases and the lifetime of the pipes is reduced. Usual wash tailings from the processing plant are the most suitable material for pneumatic stowing according to fill performance and pipe wear.[7]

Heavy pipes increase labor cost for laying and recovering, which results in a cost increase, but other materials like sand, plastic or fiberglass pipes wear more rapidly than steel pipes. The pipes should be straight and turned regularly to even out the wear that occurs at the bottom of the pipeline. To guarantee an early-bearing compact fill, sized material with a particle size distribution curve similar to the Fuller curve for concrete should be used.

A problem with pneumatic stowing is the dust development, which is countered by adding water to reduce the dust. Especially when cement is used, water is added; normally through a wetting ring near the discharge nozzle, resulting in an average moisture content of 8-10%. The air stream is not only a transport medium, but represents an instrument for ventilation purposes. Pneumatic conveying can overcome many problems with hydraulic filling, as with pneumatic filling stopes can be filled completely, with a high degree of compaction.[4] The degree of compaction can be increased, when introducing the backfill material in 1-3 layers.[7] A denser backfill body also improves mine gas conditions, as for example methane does not leak through a tightly packed fill. This compact fill reduces also surface subsidence and convergence and also problems with water excess are eliminated.[4] Surface subsidence with pneumatic fill generally represents 50% of the thickness of the mined deposit, if convergence is reduced and the opening is completely filled. Therefore surface subsidence not only depends on the depth of mining and nature of roof strata, but also on the properties of the backfill body.[7]

Regarding the disadvantages of pneumatic stowing, high operational costs, high noise levels and excessive power consumption stand out. As a result of the use of abrasion-resistant steel pipes, the manual handling of these heavy pipes is more difficult and therefore labor-intensive which increases operational costs with pneumatic stowing. Further on high noise levels are generated because of the compressor.

Slinger stowing

Placing the backfill product by a short rotating rubber belt into the underground opening, is called slinger stowing. The largest application field of slinger stowing can be found in metal mining with its maximum placement capacity of 40-90 m³/h.

The backfill sling consists of a belt which rotates with 20 m/s to distribute the backfill product in a dense stream up to 14 m far and 8 m in height. By slinger stowing a dense placement of the backfill right up to the roof of the opening is possible, which is very important to avoid mining damage on the surface.

The backfill sling is charged by a loader or a mobile backfill sling machine with a 6 m³ skip is utilized (Figure 23).

Like for all other backfill types, the material for slinger stowing consists of mine internal waste material or external waste material from other mines, open pit mines or tailings from processing. The maximum particle size is about 50mm and the addition of binding agents is possible as well. Cemented backfill placed by slinger stowing can reach uniaxial compressive strengths of 2-4 MPa.[7]

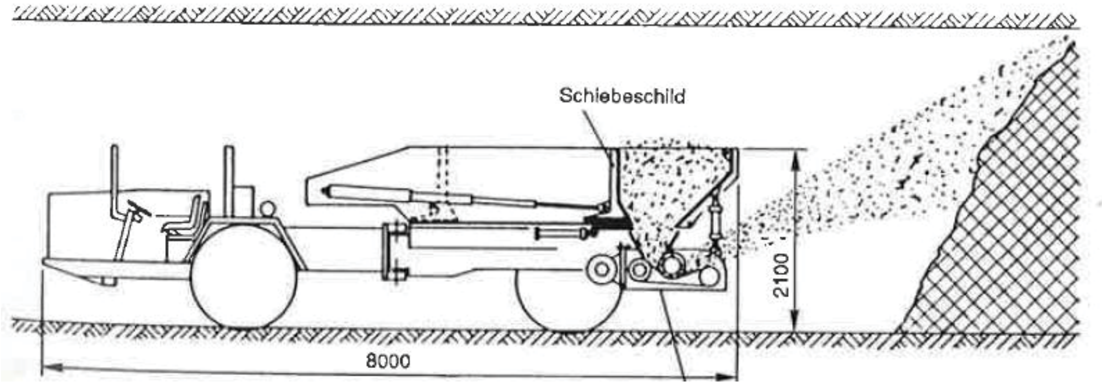

Figure 23: Mobile backfill sling machine[7] p.561

3.4.5 Reticulation design

The objective of the reticulation design is to combine the demands of delivery volumes and distances, slurry densities, pipeline diameters, borehole diameters and friction losses with the static head required for delivery or with the pumping facility.

Both hydraulic fill and paste fill require a minimization of free-fall zones where high velocity arises and therefore extreme wear of the transporting pipes and friction losses occur. [6]

Regarding the reticulation design for hydraulic backfill, special attention has to be contributed to the transport velocity, to prevent settling of the solids in the pipe, but it must be kept as low as possible to minimize friction losses and pipe wear. [6]

Reticulation design for paste fill is similar to the design for hydraulic fill. Regarding paste fill, no critical velocity exists as solids don’t tend to settle on the pipe invert. But one of the key design drivers for paste and hydraulic backfill systems is the elimination of free-fall zones within vertical or inclined borehole sections. The solution is the design of a system, where the fill material has short free-fall zones and therefore cannot accelerate to damaging speeds. The design so seeks a balance between the static driving head with the rheological paste properties, especially with the yield shear stress.[6]

Among the surveyed Canadian mines, 83% use gravity as transport system, followed by gravity and pumping as a combined option.[8]

3.4.6 Transportation failure hazards

During transportation through pipelines and boreholes several failure modes can occur. According to De Souza et al. (2004) 35% of all pipeline or borehole failures in Canadian mines are due to plugging (Figure 24).

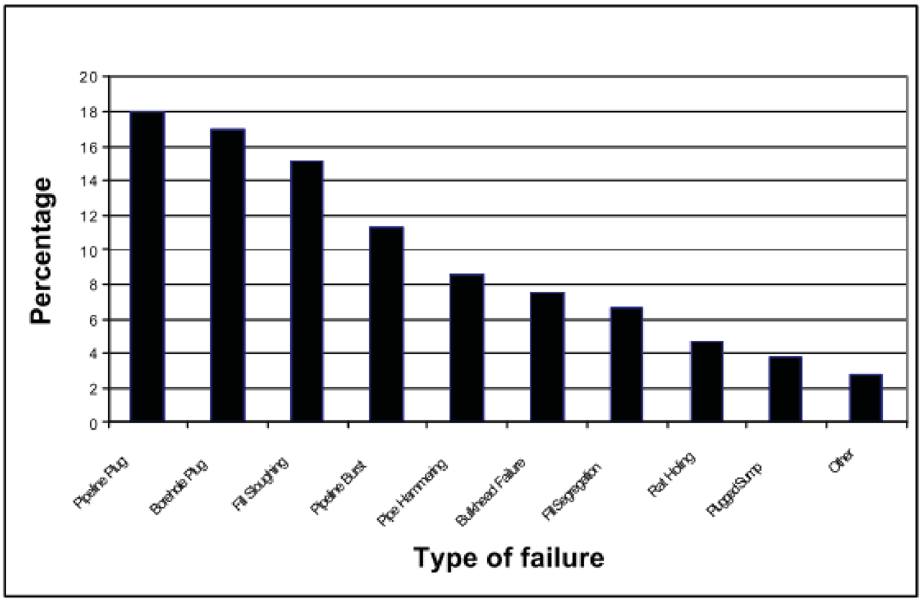

Figure 24: Backfill system failures[8]

Among pipeline plugging the most usual causes are the settlement of solids within horizontal sections of the pipeline system and setting of the Portland cement. Other common causes of blockages in pipelines and boreholes are foreign objects and too large solid particles. [15] Further on frequent failures in the backfill system are due to pipe ruptures because of excessive pipe wear or bulkhead or fill barricade failure.[8]

A problem when adding binding agents to the mixture without any retarders is the hazard of curing during pipe flow and any uncontrolled stoppage of the flow will increase the yield stress to reinitiate the flow, which finally will plug the pipe or the borehole. In general pipelines and boreholes should be designed according to the ANSI/ASME Code B31.11 for slurry transportation piping systems, which is a guideline for pipe design[16], but when flow properties vary from the planned properties, pipe plugging might occur. To assure a continuous flow of the fill mass, flow properties have to be surveyed constantly. When the flow is interrupted, the first approach is to fill up the vertical borehole or pipe to provide a maximum static head. If the static head is not sufficient normally backfill systems possess redundant pumping energy to increase the pumping pressure. If these measures are not working, an emergency high-pressure pump should be available to be able to recommence the flow when the pipeline gets plugged. [17]

High pressure air might be used to dissolve blockages as well. However this operation is very critical as it can lead to very high pressures in the pipe and finally can cause fatal pipe ruptures. This method is preferred for plugged boreholes.

A very practical way to open up blocked boreholes or pipes is the construction of sacrificial pipe pieces at the bottom of vertical boreholes or pipes. In these pipe pieces calculated charges of explosives can be used to release the blockage. Although the explosion will send a high shock wave through the transportation system, the sudden loading and unloading of the paste might release it from the inner wall to recommence the flow. These measures are quite drastic and should only be used in exceptional cases, but they are very effective in opening up blocked pipelines and boreholes. [17]

Figure 25: Sacrificial part of a pipeline system[17] p.184

3.5 Comparison of backfill types and conclusion

Comparing the three explained backfill types and the methods of transport, certain advantages and disadvantages could be found. Comparing paste fill with hydraulic fill, the advantage of the early removal of water stands out. With a paste fill system, the separation of solids from processing water takes place in the processing plant, resulting in a low liquid content sufficient for transport and hydration reaction. Consequently, more processing water can be recovered (up to 90% compared to 20% for hydraulic fill) and mill reagents like lime or cyanide can be reused and reduce environmental impact. [6] Comparing dry rock fill with paste fill and hydraulic fill the removal of water and the construction of fill barricades is dispensed with dry rock fill. In the case of dry rock fill, diaphragm walls might be left, which means that small parts of the deposit have to be left behind. Dry rock fill can be transported by gravity through pipes or boreholes or by trucks, dumping the material. It represents the cheapest type of backfill (especially when no binding agents are used), but material preparation costs (crushing, grading) and a low final strength of the backfill body have to considered.

Regarding the rheology and flow properties of paste fill, it is considered as a non-segregating material, as a result of fine particles retaining the water. Consequently no critical settling velocity exists and therefore transporting properties of paste fill are uncomplicated, as the flow can be stopped and reinitiated easily. Furthermore the preparation of underground openings for filling operations are less time-consuming as fill barricades can be less robust and as the paste can be placed at higher angles (between 3 and 8°). Regarding transporting facilities, higher operating and capital costs can be found for paste fill. As pressure losses are higher for paste fill and therefore transporting pressures are considerable high, robust steel pipes are required for transport of paste fill. When gravity flow is not adequate, positive displacement pumps are required which increases capital and operating costs. [6]

Over all it can be concluded that the choice of the backfill type predominantly depends on its application purpose. If backfill is placed for waste disposal reasons, the priority is attributed to low costs and not to high fill body stability. The choice of the transport system as well depends on the local conditions.

3.6 Modes of action of backfill

Backfill as mine stability support medium acts in three different ways on the rock mass. By imposing a kinematic constraint on the displacement of pieces of the rock mass, backfill prevents the disintegration of the near-field rock mass in low stress conditions (Figure 26 a). Through mining activities, pseudo-continuous and rigid body displacements of the wall rock are induced, which mobilizes the passive fill resistance (Figure 26 b). The third mechanism of well confined fill underground is the effect of a global support element for the underground mine structure. As such a support element, global displacements in the mine structure can be induced into the backfill body and cause deformations, which results in a global stress reduction (Figure 26 c). [1]

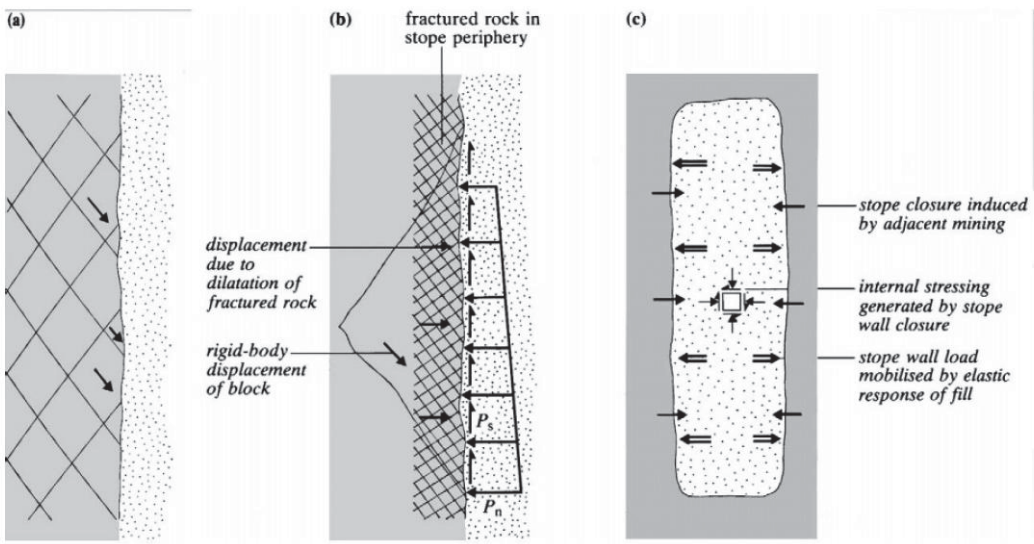

Figure 26: Modes of support of mine fill [1] p.409

The shear strength of backfill, as it is generally a granular medium, depends on the friction properties and grain size distribution of aggregates, cohesion provided by binding material and on the pore-water pressure as well. That is why the development of pore pressure has to be prevented in the backfill body. High pore pressure can lead to the loss of shear resistance and subsequent liquefaction of the medium. That is why any static or dynamic loading of backfill has to be conducted under drained conditions. [1]

3.7 Planning of a backfill system

When choosing a backfill system, the collectivity of the mining activities has to be considered, as mine filling represents a part of mining activities in an underground mine. Hence the decision for a suitable backfill system is an individual issue, as each mine has different characteristics and demands and so the motivation for the choice of a backfill system is diverse. Therefore the system environment-rock mass-backfill has to be analyzed to find a suitable solution for an underground mine. Different steps in the planning of a backfill system exist, which can be applied in all kinds of mining environments (Figure 27):

- Determination of specific backfill purposes
- Determination of the demands to the backfill body
- Determination of the characteristics of the backfill product
- Evaluation of the available raw material for backfill
- Choice of the backfill method
- Implementation of a quality management

3

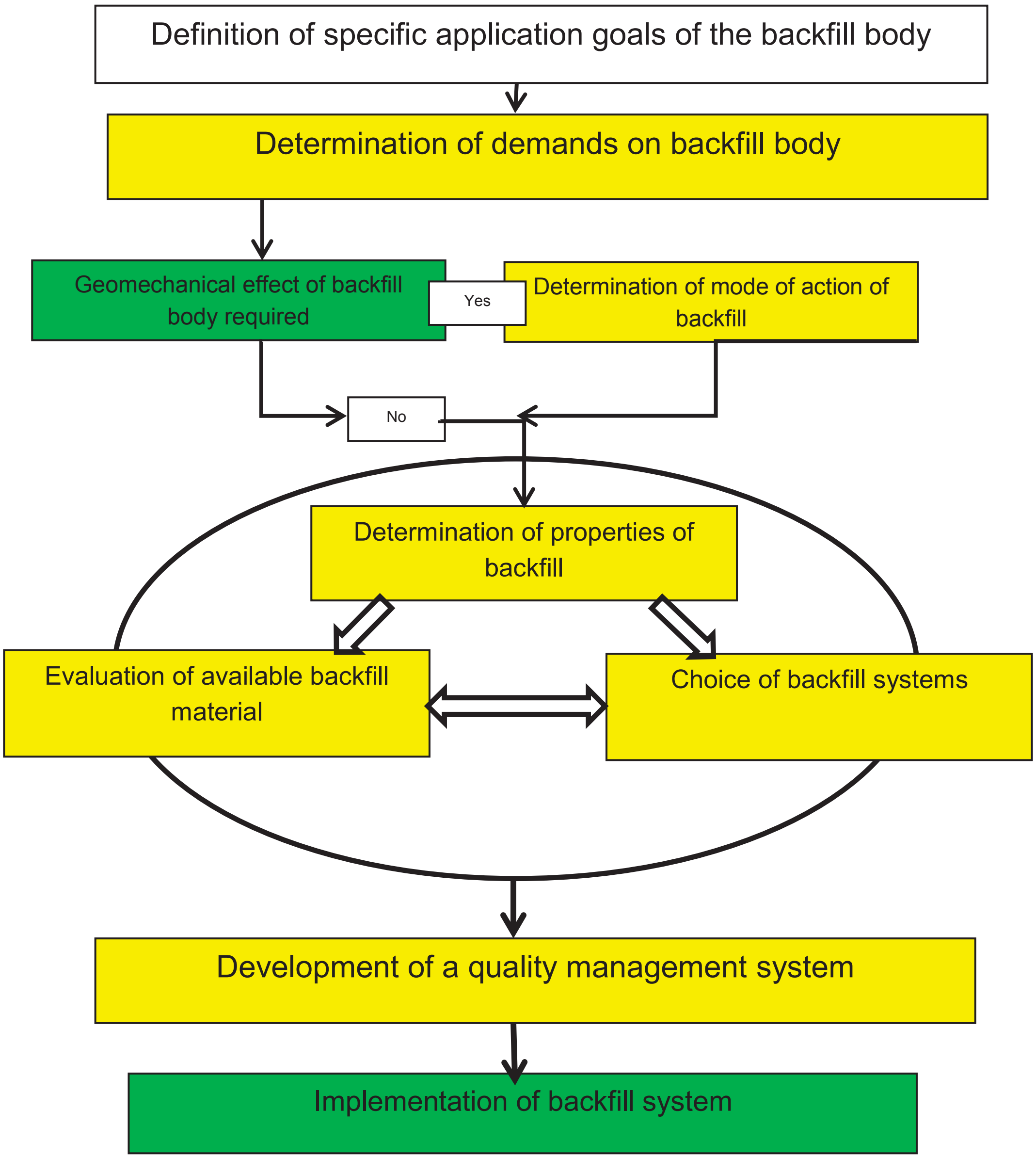

Figure 27: Planning cycle of a backfill system[3]

As each mining operation is different, the demands on backfill vary markedly. A certain importance is accorded to the characteristics of each mining operation which determine the demands on backfill. [3]

Determination of the demands to the backfill body

The demands on the backfill body derive directly from the objectives that should be achieved by the application of a backfill system. Possible demands to the backfill body are:

- ✓ Geotechnical impact for local or regional stabilization of the underground opening (description of the geotechnical interaction of rock mass and backfill body)

- ✓ Fast drainage of the backfill body (if hydraulic backfill is used)
- ✓ Load-bearing capacity of the backfill surface
- ✓ Low gas permeability
- ✓ Environmental sustainability

On the basis of the determined demands on the backfill body, the required material parameters are defined with regards to the available raw materials. According to Hohl & Frömmer (2013) the phase “after-mine” should be considered as well when determining the demands on the backfill body. [3]

Concerning the geotechnical demands on backfill, an analysis of stress and displacement in the backfill body has to be executed. According to Barrett et al. (1978) it is necessary to simulate the stope filling process as well as the extraction process to identify weakness zones in the backfill body. [1]

Determination of the backfill product characteristics

According to the defined characteristics of the backfill body, as a result necessary specific physical (uniaxial compressive strength, deformation parameters, particle size distribution, permeability etc.) and chemical properties (if external material is used, environmental harmlessness has to be proven) are required. Both types of properties have to be guaranteed by examination. [3]

Evaluation of available raw material

The availability with regards to time, quality and amount of suitable backfill raw material has to be considered during the planning phase of a backfill system, to avoid supply shortages. [3]

Choice of the backfill method

The choice of the backfill method generally depends on three factors: the physical properties of the backfill product, the type of transport and the introduction of the backfill into the mine. Further on cemented and not cemented backfill products have to be differentiated. Sometimes it is necessary to add binders, to reach the required physical and chemical properties. A choice between the above mentioned backfill methods has to be made with regards to the whole backfill system, respectively the backfill production, the transport, backfill barricades, introduction of the backfill into the mine as well as the storage of the backfill raw materials.

These factors mainly influence a constant quality, a constant filling rate and the overall costs of the backfill system. [3]

Quality management

To reach the goals which were defined in the planning phase, a quality management system has to be implemented. Further on this quality management guarantees the abidance of the environment protection. The quality management generally comprises the control of the backfill raw materials, the monitoring of the production process and the control of the properties of the final backfill product. [3]

3.8 Cost aspects of backfill operations

When calculating the costs for a backfill system, a distinction between possible and certain costs is done by Reuther (1989). Possible costs are:

- Operating costs for comminution and classification of the backfill material
- Costs for external backfill material
- Costs for dumping of excessive internal waste material (which is not suitable as fill material)
- Costs for mining damages (internal and external)
- Costs for extraction of backfill material (sand and gravel)

Certain costs are:

- Labor costs
- Machine costs
- Energy costs
- Material costs
- Placement costs[7]

7

For the development of large underground openings, full confidence in the stability of backfilled areas has to be attributed. That is the reason why not only costs for the backfill material itself, but also for handling and placement of the backfill material have to be considered. Further on the influence of backfill on mining costs and productivity is significant. [12]

Regarding the costs of a backfill system, differences among different backfill types can be observed. Concerning the overall costs, dry rock fill has the lowest capital and operational costs, whereas pneumatic stowing as placement method is the most costly one. When the local conditions are suitable for hydraulic fill, adequate material is available and easy sealing of the openings is possible, hydraulic fill represents a very favorable possibility as well.[7]

In the following, costs for Paste fill (PF), Paste aggregate fill (variation of Paste fill - combination with dry rock aggregates - PAF), cemented rock fill (CRF) and cemented hydraulic fill (blended with aggregates - CHF) are compared. [12]

The data shown in Table 4 is based on a case example, mining with a combination of cut and fill mining and open stoping at 6000t/day.

Binding agent requirements have been estimated based on laboratory testing experience. [12]

Backfill type	Annual fill tons	Binder requirement [%]	Annual binder [t]	Annual aggregate [t]	Labor required [h/day]
CRF	**1 227 400**	**7**	**89 400**	**1 188 000**	**290**
PF	**1 064 500**	**5**	**53 200**	**0**	**150**
CHF	**871 000**	**7**	**61 000**	**0**	**265**
PAF	**1 176 500**	**3**	**35 300**	**570 600**	**150**

Table 4: Backfill options and requirements[12] p.139

Capital expenditure

The capital expenditure of a backfill infrastructure mainly depends on the type of backfill. Depending on the backfill type, different equipment for backfill preparation and handling is necessary. Capital costs for tailings handling for PAF are less compared to a standard paste system, as the tailings handling is reduced due to the substitution of some of the tailings with aggregates. Using a standard 50:50 paste to aggregate blend, the tailings plant can be almost halved in size. However, comparing the whole PF and PAF systems, additional capital expenditure is necessary for a PAF system for crushing and additional infrastructure. The CHF system is the simplest and cheapest system, and therefore has the lowest capital costs. [12]

Backfill type	Capital cost [million $]
CRF	19
PF	25
CHF	15
PAF	27

Table 5: Comparison of capital costs for different backfill types[12] p.139

Operating expenditure

The cost of backfill can represent 25% of the overall mining costs (Slade, 2010). Among overall backfill costs, 75% can be as a result of binding agent addition. To reduce costs for binding agents, systems like PAF or CRF are advantageous, when suitable aggregates are available. Additionally Portland cement, which is very expensive, can be replaced with other binding agents like fly ash or metallurgical slag. [12]

To assess the potential savings with a PAF system compared to a common PF system, the following formula can be used[12]:

$$A < (CxD)/B$$ [12]

$A\ldots$ Cost of delivered aggregate [$/t]

$B\ldots$ Aggregate percentage in PAF [%]

C…Cement costs per ton [$/t]

D…Difference in binder percentage for required strength between PAF and PF [%]

For all four systems, the operation costs have been compared, using the following assumptions:

- Binder costs – 200$/t
- Aggregate cost – 4$/t
- Rock cost – 2$/t
- Labor cost – 50$/h

The additional mining and milling costs for the dilution material (rock and aggregates) are not considered in this comparison. [12]

Net present cost

The following table shows the comparison of net present costs for the 4 different backfill systems.

Backfill type	Backfill cost [$/t]	Net present cost 10 years [million $]	Net present cost 15 years [million $]
CRF	20,04	704,881	1063,447
PF	19,55	482,887	723,841
CHF	12,57	457,961	660,995
PAF	10,28	440,063	628,158

Table 6: Net present costs for 4 backfill systems[12] p.140

Sensitivity analysis

During the period of planning of a backfill system, finally implementing it and achieving the return on investment, major variations in the price of materials, equipment and products might occur. Therefore the different cost areas have been analyzed:

- Binding agent costs: highest costs, small changes in the amount can have a big impact on operating costs

- Capital costs: forms a significant part of the overall mine capital expenditure
- Aggregate: Costs for transport, crushing and screening
- Labor: labor costs for surface and underground works

A sensitivity analysis showed that the greatest sensitivity can be found in binder cost variations. 35% of the cement production costs are due to energy requirements for production. Energy costs are subject to significant fluctuations with a tendency to higher energy prices. If energy prices continue to grow, cement costs/t might rise about 8% in 5 years.[12]

Regarding labor costs, CRF requires higher labor force than the other systems for placement of the material. Hydraulic fill is quite labor-intensive as well as a result of fill barricade design and construction.[12]

4 Duties and demands on backfill

4.1 Duties on backfill

Some deposits can be mined entirely only because of backfill. This is particularly so in the case of difficult and poor weak rock conditions.[11] Hence the primary purpose of backfill is to support the openings, prevent them from caving, avoid surface subsidence, pressure on the work place and rock bursting. By filling the openings, the structural integrity of the mine is improved, which guarantees a lifelong stability. [18] This can be performed in an active manner by a direct support of the roof strata or in a passive way by indirect support through strengthening of existing pillars through confining stresses.[5] In Canadian mines, the primary purpose of backfill is to improve the hanging wall stability. [8]

Beside this application field, backfill has many different functions: technical, economical, safety and for environmental purposes. [11] The main safety purpose is to stabilize the mine and to reduce rock falls. Beside that also the water inflow can be reduced by backfill. Furthermore also the rock burst and underground mine fire hazards can be reduced by introducing backfill.

Using backfill, also mining damage and waste disposal facilities on the surface can be reduced. This means reduction of air contamination and preservation of the nature and landscape from an environmental point of view. [11]

If development waste is used as backfill material, less material has to be transported out of the mine thereby increasing ore hoisting capacity of the shafts. Through the placement of backfill material, for example in overhand cut and fill mining, a new working platform is created. Furthermore shafts are stabilized by backfill application. The mine climate is also improved, as the flow direction of air streams can be better controlled when openings are filled with backfill.

By introducing backfill, also money can be saved by reducing rehabilitation, reclamation, watering and mine ventilation costs. Further on surface areas use is extended and less material is lost by leaving precious material to stabilize the mine. Also haulage costs are reduced and material dilution is prevented.[11] In Canadian mines the increased ore extraction is considered to be the second main purpose of introducing backfill. [8]

Backfill has become an integral component of many mining methods, especially the cut and fill methods which are based on the placement of backfill. In the following some of the mentioned backfill application fields are discussed in detail.

4.1.1 Ensuring long-term regional stability

When large underground excavations are left open over a long period of time or when the rock mass stability is low, a high risk of collapsing arises. By the placement of backfill this risk can be reduced. When the opening becomes unstable, the fill material supports loosening material from the excavation boundary which is so kept in place by preserving the confining forces within the rock mass. Through backfill application, confinement on the roof and walls of the opening is increased, which prevents the opening of joints and fractures, as it mobilizes friction along the surfaces. This preserves rock mass shear strength. Further on backfill limits the amount of wall convergence, which increases the regional stability of a mine. [6] Backfill so prevents the creation of internal mine damage (on close mining excavations) and external mining damage (on the surface).[7]

According to the raw materials law in Austria (Mineralrohstoffgesetz, “MinroG”) *III. Abschnitt „Besondere Pflichten des Bergbauberechtigten“ - Sicherungspflicht des Bergbauberechtigten § 109. (1)* and *IV. Abschnitt “Sicherung der Oberflächennutzung nach Beendigung der Bergbautätigkeit”§ 159. (1),* the holder of the mining license has to take measures after termination of mining activities for the assurance of the surface reutilization. These measures generally include the filling of the underground openings, as it represents the best option to guarantee stability of the mine over a long period of time. [19]

4.1.2 Limiting excavation volume

Without support only few excavations, undercutting a critical dimension, will remain open during mining activities. When excavation activities proceed and the openings become larger, backfill can be introduced to limit the exposure of walls and/or backs of excavations. When for example in cut and fill mining adjacent stopes are mined, the backfill is exposed and therefore its self-supporting capacity has to be considered. To increase the stability of the backfill body binding agents (mostly Portland cement) have to be used as they increase cohesion and the self-supporting capacity of the backfill body. [6]

4.1.3 Backfill as working platform or roof

For extraction methods proceeding in upward direction (overhand cut and fill mining) the backfill body serves as working platform (floor). Extraction sequences progressing in downward direction use backfill as replacement of the roof strata. This application is especially useful in unstable ground conditions, where the roof is deemed unstable or when leaving crown pillars is not an option. [6]

4.1.4 Reduction of subsidence damage

Surface subsidence represents a major problem in mining activities. Surface subsidence can be caused by caving methods ore long-term failure of pillars. By the introduction of backfill the amount of surface subsidence can be minimized. [4] The hazard caused by surface subsidence affects employees, the infrastructure at the mine site and the community living in the neighborhood of the mining activities and might have serious consequences. Therefore the application of backfill is a suitable method to prevent these risks.[6]

4.1.5 Waste disposal

Backfilling can be applied, where environmental regulations restrict the surface waste disposal or simply no space for mine tailings is available on surface. [18] Additionally costs for surface storage facilities for mine waste (rock or tailings) have increased in recent years. The additional costs for surface storage include treatment of the material (to remove environmentally harmful substances), water inflow or drainage control or the construction of storage facilities like storage platforms or storage cells for tailings. Especially tailings increase the costs incrementally as they have to be dewatered when stored in cells. Furthermore the handling of the transport material and the transport out of the mine to the disposal facilities are significant cost factors. When backfilling underground openings with waste material in Austria, the Austrian law regarding the rehabilitation of inherited waste is considered. In *§3 (1)* the application of this law for backfill with mine waste material is cited. However, in *§3 (1a)* illustrates the release of fees for waste disposal (waste from mining activities according to the MinroG). This shows that when storing mine waste in underground openings, fees from the state for waste storage are not applicable. [20]

Filling of underground mines with this waste material is considered as an environmentally friendly opportunity, which is also a cost saving option for the permanent disposal of mine waste. [6] Further on the reuse of mine sites after mine closure has to be considered. Future land use requirements put increasing pressure on the re-use of former mine sites, which necessitates long-term stability in these areas. This can often be achieved only by backfilling shallow mining excavations. Also municipal waste can be used as filling material. The co-disposal of municipal waste with mine waste and tailings represents a possibility of filling underground openings, depending on the material properties required for stability of the mine.[6] If material, which has been declared as "waste", is used to fill underground openings, the instructions concerning waste from the Austrian waste management act (Abfallwirtschaftsgesetz – AWG) from *§§1,2 (2) or (3) AWG* as well as *§17 AWG* have to be considered. Therefore according to the Austrian waste management act, the application of municipal waste for mine fill purpose is feasible, if this is necessary from a mining and safety point of view (§§ 109,159 MinroG). [2119]

The question of reutilization of municipal waste is regulated in *§2 (3) AWG.*[21]

In general the possibility to fill underground openings with waste (waste as defined by the Austrian waste management act) is controlled by the Austrian waste management act. If for stability reasons it is necessary to fill the openings (also with waste) it is not declared as “waste disposal”, when the placement is done by mining activities. Therefore the application of the Austrian waste management act is not necessary in this case. [21,22]

4.1.6 Avoidance of transport

The first uses of backfill did not consider the stabilization of underground openings, but rather the avoidance of transporting the waste material out of the mine. As transport of material represents an important cost factor, an underground mine is more economic, when an important portion of the waste material does not require transport to the surface. Further on in former time periods, transporting equipment was not available for the transport of ore and waste material and therefore the waste was left in mined-out underground areas.

4.1.7 Mine ventilation/climatization

The most important environmental problem in deep underground mines is the question of ventilation and climatization. As with the depth, the virgin rock temperature increases the heat flow into the underground openings increases as well. The objective of mine climatization is to ensure a wet-bulb temperature of below 28°C in all working areas, which requires cooling of the underground openings (Figure 28).

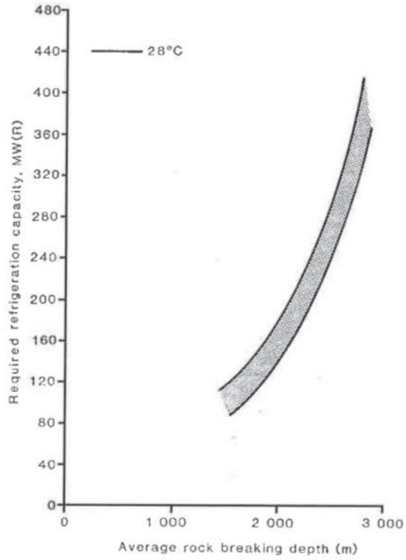

Figure 28: Effect of depth on refrigeration requirement (to ensure <28°C)[23] p.257

The thermal problem in deep underground mines is closely related to the total length of working faces and to the heat-flow into the openings (Figure 29). When reducing the total length of the working face and by backfilling the areas, the thermal environment in the underground opening can be improved significantly. By increasing the rate of face advance from 5 to 20m/month in combination with backfill, a heat/ton reduction of 50% can be achieved (Figure 29).[23]

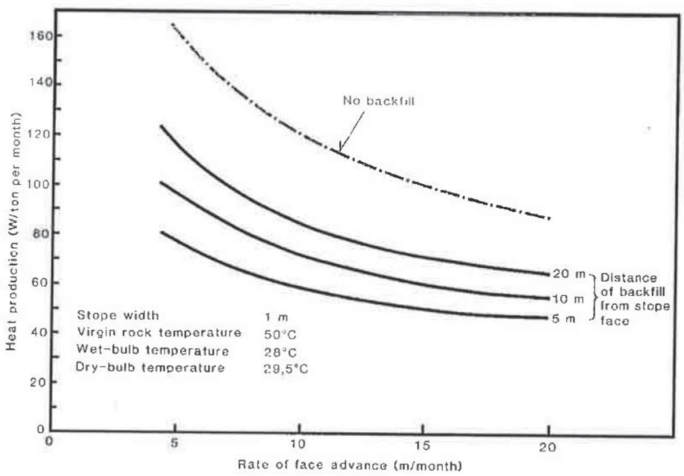

Figure 29: Relation between rate of face advance and heat production with regards to backfill use[23] p.258

As heat in deep underground mines is a main contributor to fatalities (Figure 30), the topic of mine climatization is of extreme importance.

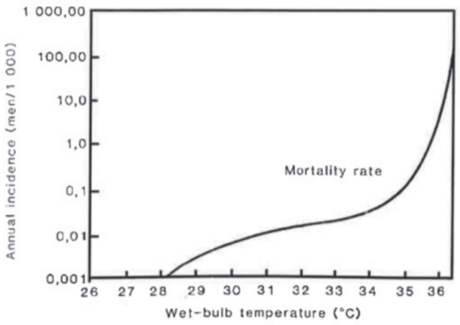

Figure 30: Predicted incidence of heat stroke fatalities as a function of the wet-bulb temperature[23] p.259

Additionally the implementation of backfill reduces the open space in underground openings, which leads to a reduction in heat production and also ventilation (quantity) requirements. Dangerous gas emissions can be reduced by filling mined-out areas and so combustible materials (wood) might be used when backfilling.[4] Further on the development of spontaneous combustion can be limited by sealing old workings with fill material. When old workings of coal mines for example are uncovered and exposed to the atmosphere, spontaneous combustion might occur. So backfill avoids gases escaping from these old workings and air entering which can cause combustion hazards.[6]

4.1.8 Mining selectivity and ore recovery

By the application of backfill, very irregular ore bodies can be extracted.[4] When filling the developed openings, the collapsing of surrounding rock masses, which leads to dilution of the ore, can be avoided and this results in greater selectivity.[7] So a higher-grade ore product and economic advantages are generated. [4]

Further on the recovery of the ore in a mine can be maximized by the application of backfill, which results in additional material processed and sold, offsetting also the costs of backfill. This leads to an increase in the ore reserve and to an extension of the mine life.[12]

4.1.9 Backfill for pillar recovery

In underground mines the location of pillars in the ore body depends on the maximum stable stope span which the rock mass can tolerate without the risk of collapse. Leaving these pillars as support medium in an underground mine represents an economical loss of precious material. Therefore the pressure to recover these pillars increases. If backfill is used as artificial support for this the recovery of some of the mine pillars becomes a possibility. The objective of the backfill placement in these underground openings is to achieve a high rate of exploitation of the remaining ore reserves. The main purpose of backfill for this application is to support and stabilize the hanging wall of excavated areas. Although the mechanical properties of backfill compared to the adjacent rock mass are quite poor, experience shows that comparatively small resisting forces can mobilize significant frictional resistance within the mass of wall rock.[1]

When recovering pillars using backfill, the backfill process is generally not as closely integrated in mine production activity as it is in cut and fill stoping methods.[1]

4.2 Demands on backfill

Depending on the reason for backfill application, the mining method and local circumstances, demands on backfill vary and are dominated by the following aspects:

- Health and safety (H&S)
- Environment
- Technical aspects
- Organization aspects
- Economic aspects
- Geomechanical aspects

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4.2.1 Health, safety and environment related demands

For H&S reasons, no harmful components (harmful for health or environment) must be used as backfill material. This is of special importance when external material is used for backfill.

A significant safety hazard is the risk of liquefaction when using hydraulic fill, so drainage properties have to be considered as well. Drainage properties determine the fill ratio of each fill point and therefore the mine production ratio as well. Drainage properties of hydraulic fill are significantly influenced by the finest particles in the backfill product. The finest particles lead to excessive abrasion of the pipes used for transport and degrade drainage properties of the backfill body. Using hydraulic backfill, no particles <75 microns should be used. Generally particles <10 microns in the backfill product should not exceed 10%.Using paste fill, particles <20 microns should not exceed 15% for transporting reasons (details can be found in Chapter 6.2).

When using binding agent, the pH value must be monitored as well. [11]

An environmentally critical factor is the influence of the backfill system on the ground water quality. The disposal ordinance can be used as basis for reference values concerning the influence of backfill on ground water.

4.2.2 Technical demands

Technical demands on backfill mainly concern the efficiency and reliability of the backfill system. Especially when using hydraulic backfill, conflicts between efficiency and mechanical properties arise, as transportability increases with decreasing heavy liquid density but mechanical properties can deteriorate. This is even more significant when using binding agents to increase the fill strength.[11]

4.2.3 Organizational and economic demands

Organizational demands related to the establishment, operation and maintenance of the backfill infrastructure, i.e. backfill preparation, transport, introduction and influence on the water system to mining activities. A crucial fact is that in general backfill activities are executed in a reverse flow to the production activities.[11] This can give rise to considerable difficulties in the area of mine logistics. To guarantee the long term objectives of the mining business, backfill activities should be prioritized. This means that mining activities can only be continued in a special area, when backfill works are completed. As a result of this, multiple headings should be conducted simultaneously to assure a continuous production. From this it follows that the capacity of an underground mine using backfill is determined by the backfill infrastructure and facilities.[11]

If quantifying the cost effectiveness, not only costs but also benefits resulting backfill use have to be considered. This might be very difficult, because useful effects of backfill like longer life time of a mine, more effective extraction of the deposit and better reuse of the surface area of former mining activities, come into effect in the future. When assessing the economics of a backfill system, the negative influence of production losses due to difficulties during backfill placement, have to be considered as well. Therefore it might be practical to overdesign the backfill infrastructure. [11]

4.2.4 Geomechanical demands on backfill

As one of the main purposes of backfill is its ability to contribute to the support of underground openings, the geotechnical demands on backfill are particularly significant. Four basic cases in geotechnical demands on backfill have to be distinguished according to Wagner (2009):

- prevention of bed rock swelling
- reduction of extraction loss because of higher stability of pillars
- reduction of rock burst hazard

- increasing safety during backfilling as a result of higher stability of backfill benches

11

Prevention of rock bulking

Through mining often large areas of roof strata are exposed, which can cause caving of the roof or swelling of the bed rock. In shallow depths these fracture phenomena can extend up to the surface and can cause mining damage to the surface. The free height of the opening h_0 determines in tabular mining the height of the rock bulking zone h_c (Jacobi, 1960):

$$h_c = \frac{h_0}{K-1} \quad 11$$

h_0…free height of opening [m]
h_c…height of rock bulking zone [m]
K... bulking parameter of the roof strata (from 1,2 for thin bedded schist formations up to 1,8 for blocky rock masses)

The critical material parameter is here the bulking parameter of the roof strata K. By filling the opening, the free height can be reduced and therefore the height of the bed rock swelling zone can be reduced.[11]

Improvement of the pillar stability

A detailed description of the effect of backfill on pillar stability can be found in chapter 5.2.

Reduction of rock burst hazard

Through mining, stress and energy redistributions occur which can result in stable and unstable rock fracturing processes. The frequency and severity of these increases with depth and can result in a release of a large amount of energy. The Energy Release Rate is a good measuring tool for the extension of the fracturing and for the rock burst hazard. [11]

$ERR = \Gamma \sigma_v D_h$

Γ…factor of stress concentration in the mined-out area (controlled largely by the mining geometry)

σ_v…Primary vertical stress

D_h…convergence in the mined-out area

Especially in very deep room and pillar underground mines, backfill is placed between pillars because of rock burst hazard. In this case the fill's demand is not to stabilize the opening, but to reduce the impact of very sudden and abrupt rock burst hazards. By the backfill application the roof and the pillar are supported in such a manner, that no void is available for collapse.[14]

Backfill has the potential to reduce the convergence volume in the mined-out area. In order to be very effective backfill must fill the openings as completely as possible. Further on backfill should possess a high initial stiffness and a small pore volume. Experience shows that backfill can reduce the rock burst hazard by 50-70%. [11]

4.2.5 Quality demands on backfill

The demands on the quality of backfill concern not only the backfill mix but also the in situ backfill body. The main factors requiring an increased backfill quality are the mining method and the geotechnical conditions.

Using overhand cut and fill mining, the backfill should be of a certain composition to make it a suitable and safe work surface for workers and mining machinery. Of particular importance are that the backfill does not liquefy under the dynamic operating conditions, that it does not deteriorate under the transport loads and that it provides adequate confinement to the rock structures in the fill. Therefore the backfill body must possess a certain strength not to dilute the material. Main influencing parameters to achieve these properties are the density, the cohesion and internal friction. [9]

Using underhand cut and fill mining, the backfill serves as an artificial roof, which must be stable. [9]

In the course of mining operations, the backfill body may be exposed and therefore it should possess sufficient stability. In the case of exposed backfill walls the stability is influenced by the wall geometry, the cohesion and internal friction of the backfill body. The backfill wall might be additionally loaded by blasted rock, geotechnical stresses or dynamic loads resulting from equipment operations .[9]

Through mining operations it might be necessary to penetrate through the backfill body. Therefore the backfill body must have a sufficient stability to support its own weight and to withstand the ground stresses.[9]

From a geotechnical point of view, the backfill should replace the excavated rock mass as good as possible. It should show a quality which is similar to the rock mass. Additionally the backfill body should not easily be compressed and should have an early load-bearing capacity. Backfill without binding agents normally has a lower quality than the rock mass, because of a high pore volume which arises during backfill placement and the absence of a cohesive strength. As backfill should oppose a resistance against the ground displacements as early as possible, a high placement density is required.

In addition to that, when using hydraulic backfill, preferably it should dispense a small amount of water and when using binding agents, the heat development or gas development during hydration must be low. [9]

5 Backfill in underground mining

Mineral exploitation, in which all operations are carried out beneath the earth's surface, is called underground mining. The choice of the underground mining method mainly depends on the geologic conditions. Strong ore and rock conditions might not need artificial support to guarantee a stable and safe underground opening, whereas weak rock conditions require additional support. [18]

In order to avoid underground excavations in weak ground conditions from collapse, a filling material, called backfill, can be introduced into the open voids of a mine. Backfill refers to any waste material that is placed into underground openings of disposal or for engineering functions.[24] The primary purpose of backfill is to support these openings, prevent them from caving, avoid surface subsidence, pressure on the work place and rock bursting. By filling the openings, the structural integrity of the mine is improved, which guarantees a lifelong stability.[4] Beside this application field, backfill has many other functions.

Underground mining methods are employed, when the depth of the deposit or the stripping ratio of overburden to ore are too high to apply surface exploitation. The choice of the underground mining method heavily depends on the geology governing the ground conditions and the resulting necessary support. Hartman and Mutmansky generally distinguish three classes of underground mining methods based on the extent of support:

- Unsupported methods
- Supported methods
- Caving methods

Using an **unsupported mining method**, the rock is self-supporting and no major artificial support is necessary to guarantee the stability in the mine and the safety of the workers. [18] Either the ground does not need any support, or the ground is naturally supported by pillars, which ware left in place to control the stability of the extracted areas. A typical mining method using pillars for major ground control is room and pillar mining, which is often employed in shallow dipping ore bodies. In these cases backfill is less effective, as fill transport is often gravity based and is therefore less efficient. [6]

Supported methods require some kind of **backfill**, which means that they need artificial support to remain open during mining operations. Even after closure of the mine, the placed backfill can prevent significant surface subsidence and major caving.[18]

In **caving methods**, the ore or the rock or both are collapsing in a controlled way. [18]This includes block caving, where ore bodies are undercut to start the caving process. Also sublevel caving is part of caving methods, where the hanging wall caves gradually to fill the openings created by ore extraction. Concerning the surrounding area, surface subsidence is tolerated. [6]

As the aim of this report is to make a state of the art review of backfill technology, for this reason artificially supported mining methods using backfill are discussed in detail.

5.1 Artificially supported mining methods

Generally, artificially supported mining methods are applied, where a safe and/or complete extraction of the deposit is not possible with a different mining method. This concerns:

- massive deposits with weak ore conditions
- highly precious deposits
- deposits where special care of the surface has to be taken
- deposits with water where the integrity of the adjoining rock has to be assured (often talcum and magnesite deposits or evaporite deposits)

[11]

The term cut and fill stoping describes underground mining methods which require support, which can be local or short-term and general or long-term. Supported mining methods are a set of cut and fill stoping methods using backfill for general support and several different local support methods.[4]

5.1.1 Cut and fill stoping

In cut-and-fill stoping, a tabular or irregular shaped deposits are mined in horizontal slices and replaced with backfill in underhand or overhand direction. Backfilling is normally performed after each slice is removed and different **backfill materials** (waste fill, pneumatic fill, hydraulic fill with dilute slurry or high density hydraulic fill) are used. Like in other vertical exploitation openings, the stope is often bounded by pillars for major ground support, which can often be totally recovered because of the use of backfill. [4,18] This mining method is performed in conditions, where the stope boundary rock cannot sustain stable, free-standing spans suitable for open stoping. [1]

The mechanized equipment for conventional extraction (drill, blast, load, haul) makes cut and fill stoping to a moderate productive method with high mining costs due to backfill application and expenditure of time to fill the extracted areas. As a result of backfill introduction, cut-and-fill mining is flexible and adaptable to changing conditions. It can be applied in weak to strong ore and in weak rock conditions. The lower the rock quality designation of the ore and the surrounding rock mass, the more likely is the use of cut and fill stoping as mining method. It is predominantly used for moderate to steep vein deposits (dip>45°), as these methods generally rely on gravity flow of broken rock, and so the dip of the deposit should exceed the angle of repose of the material. Especially in large to irregularly-shaped ore bodies cut and fill stoping is suitable. [4,18]

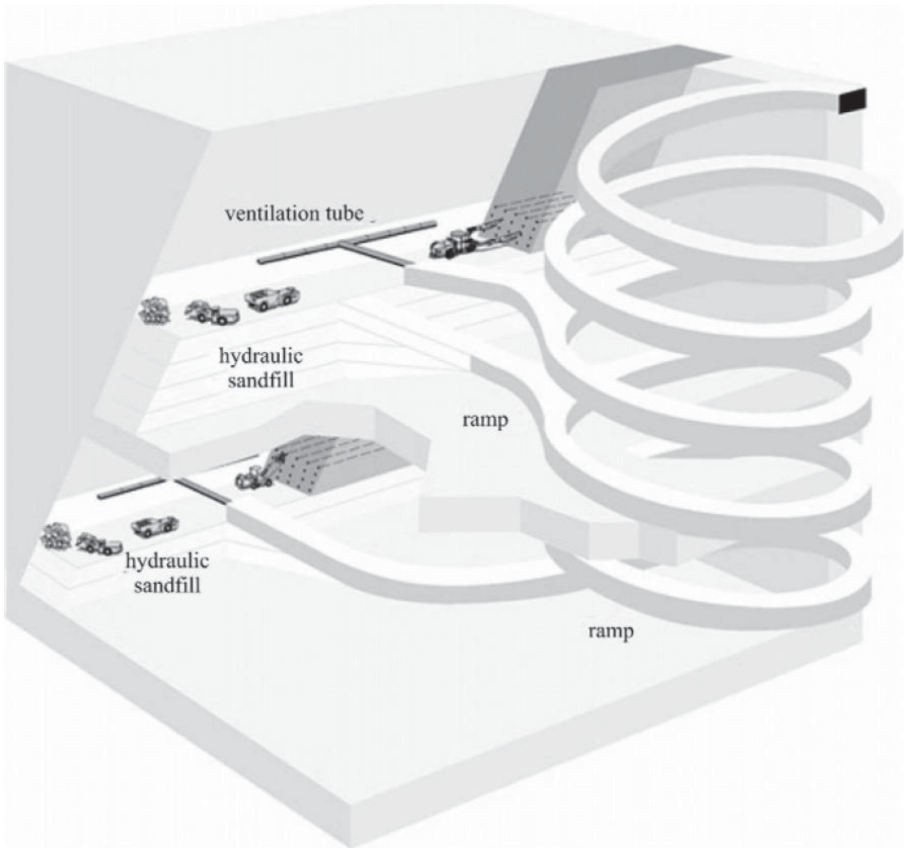

Figure 31: Mechanized Cut and Fill Stoping [1] p.349

Hartman et al. (1992) distinguish four main variations of cut and fill stoping:

- Overhand cut and fill stoping
- Post-pillar stoping
- Undercut and fill stoping
- Drift and fill stoping

Overhand cut and fill stoping

In overhand cut and fill stoping ore is excavated by horizontal cuts (1,8-4,6m in height). As the stope advances in upward direction, the excavated ore falls onto the backfill, which was placed in the previous cut and fill cycle. The roof in overhand cut and fill stoping can be unsupported in competent ore, rock-bolted, with timber stull back and rib support when the roof and ribs are in poor conditions and with square set timber support in wide stopes. Overhand cut and fill stoping can be conducted as breast stoping, where the ore is successively mined in horizontal slices of 1,5 to 4,5m thickness from the back of the stope. If operating overhand cut and fill stoping as drift and fill stoping, normally very wide and moderate to poor ore bodies require that the block is divided into series of parallel drifts with vertical walls. First a drift is stoped and then the stoped area is filled with cemented sand fill sometimes reinforced with steel nets or masts, which provides roof support for the following drift.

Back stoping in overhand cut and fill stoping is similar to breast stoping, with the difference of vertical blast holes compared to horizontal blast holes in breast stoping. [4]

The standard backfill procedure for overhand cut and fill stoping remains hydraulic backfill, whereas either the whole backfill is cemented or just the upper part of one horizontal cut. Over the time pump stowing obtained a certain importance as well. [9]

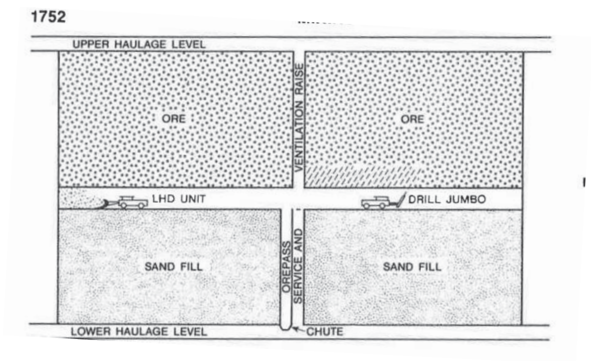

Figure 32: Overhand cut and fill stoping [4] p.1752

Post pillar stoping

The post-pillar mining method is predominantly used for the extraction of regional, irregular three-dimensional deposits. [25] Ore bodies, where post-pillar mining is used, normally have a greater vertical extent than can be mined with regular room and pillar mining. [4] Using this method, the deposit is mined in slices in overhand direction with the implementation of backfill after each slice (Figure 33). Hence Post-pillar mining represents the combination of a traditional room and pillar mining method with the application of backfill.

The slenderness factor of the pillars increases from slice to slice, so the risk of pillar failing due to buckling increases.[25] Backfill is used to provide lateral support to the slender pillars.[4] On the one hand the introduced backfill supports the pillars and increases their load-bearing capacity; on the other hand the last slice is used as working platform for the following slice. The pillars which are left standing in the backfill support the direct roof. Especially in massive, moderately steep to steep deposits post-pillar mining is used. Examples from alpine mining show that pillars with widths of 5m and heights of 100m in backfill remained stable. [25,26]

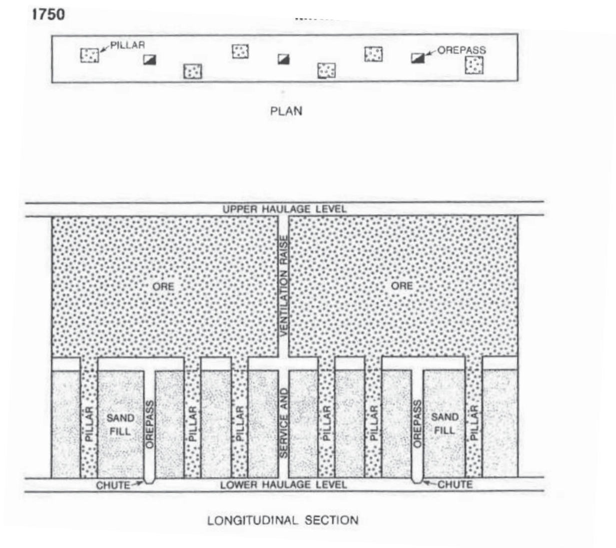

Figure 33: Post pillar stoping [4] p.1750

Undercut and fill stoping

In undercut and fill stoping or underhand cut and fill stoping the mining direction is downward. Horizontal cuts are done like in overhand mining and cemented backfill is placed, which represents the new roof, under which the stoping of the next cut proceeds downwards (Figure 34). The initial floor is created on the top level, where then excavation proceeds by driving a conventional drift round. The opening is supported by square sets or rock bolts and mats. When the cut is completed, a timber or wire mat is placed on the floor of the cut and the void is filled with cemented hydraulic backfill (Figure 32). [4]

In the majority of cases, highly cemented hydraulic backfill is used for filling but rarely pneumatic stowing is used. To reduce the amount of binding agents, limited roof support in form of props or roof bolts has been applied to supplement the lower strength backfill. A second possibility to reduce the amount of binders and hence the costs for backfill, is to place one slice of highly cemented and often reinforced backfill on the floor to act as a competent roof beam for the next mining horizon. On top of this reinforced backfill slice low strength backfill can be placed to fill the remaining void of the stope. This is only possible when using hydraulic backfill or paste fill. A special case is the application of concrete as backfill. Because of the high costs of this procedure, the opening is only partly filled and the remaining void between the roofs can be used for mine ventilation. [9]

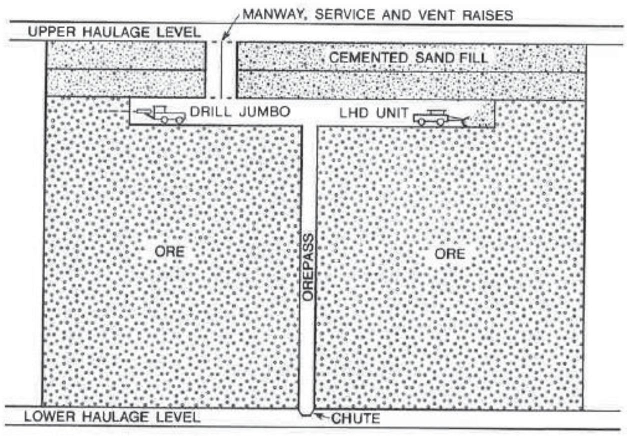

Figure 34: Underhand Cut and Fill mining [4] p.1753

Drift and fill stoping

As in undercut and overhand mining methods, the ore is mined in horizontal cuts, which is performed as a series of drifts (Figure 35 on the left). Once a drift is mined, it is backfilled, which provides roof support (Figure 35 on the right). After sufficient hardening of the mostly cemented backfill, the next drift can be mined directly adjacent to the backfilled drift. [9] Drift and fill stoping is predominantly used in wide flat, seam-like or tabular ore bodies, with moderate or poor ore competency for extraction without loss. [4,9] A sufficient stability of the surrounding rock mass as well as a tight fill of the openings is required for a successful drift and fill operation. Predominantly slinger stowing is used for this purpose. Other possible backfill methods are pneumatic stowing, or pump stowing, if a certain decline is available. Hydraulic backfill is generally not used for drift and fill stoping, as no complete roof fill is possible due to drainage.[9] In addition the regular construction of backfill barricades makes the application of backfill in this situation costly and unattractive.

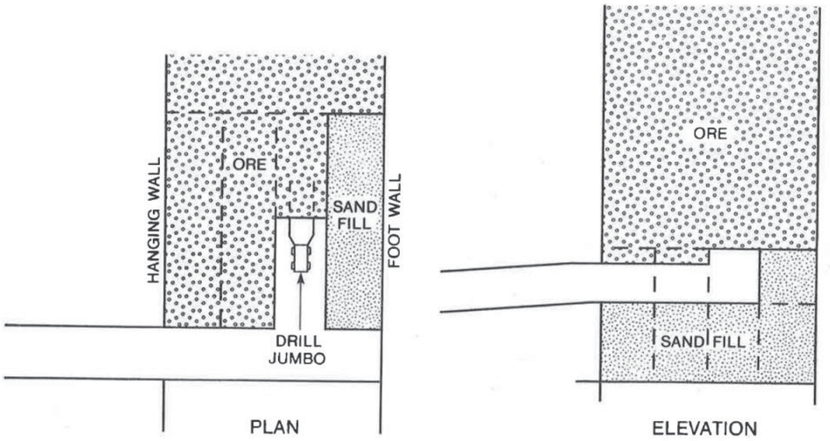

Figure 35: Drift and fill stoping [4] p.1751

5.1.2 Excavation methods

Hartman et al. (1992) classify four different excavation techniques using cut and fill stoping methods:

- Back stoping
- Breasting
- Drifting
- Benching
- Mechanical

Using back stoping, the open space between the backfill surface and the roof of the stope is available to drill vertical or steeply inclined blastholes into the roof. Back stoping can only be applied in Overhand cut and fill stoping and in post pillar stoping with rock-bolted or unsupported roof. Under normal circumstances it will be necessary to support the roof to protect the workforce against roof falls. Only in the case of the uppermost slice will it be possible to leave the support out if back stoping is done on the retreat and remotely operated loading equipment is used.

Employing a vertical working face and horizontal or slightly inclined blastholes, refers to breasting. In this excavation technique backfill is placed to fill the opening of the previous cut but leaving enough space for effective blasting. In undercut and fill stoping, breasting is executed in upward direction, whereas in overhand stoping rounds are breasted downward. Breasting is generally used in weak rock conditions, often with timber support.

In the case of isolated drifts, blasting follows the principles employed in development blasting, i.e. drift rounds are blasted using a burn cut or a different drifting cut.

In undercut and fill stoping, vertical holes can also be drilled from the top downwards, which refers to benching.

With all variations of cut and fill stoping, mechanical excavation techniques can be used.[4]

5.1.3 Geomechanics of cut and fill stoping

The success of cut and fill stoping mainly depends on efficient ground control. Very useful information about the geomechanics of cut and fill stoping can be gained, when analysing the state of stress around a stope. Especially the stress conditions in the crown and the sidewalls during vertical extension are of interest. Regarding the stress conditions in cut and fill stoping, the presence of backfill in the mined and filled zone can be neglected, as support pressure acting at an excavation surface has an insignificant effect on the elastic stress distribution in a rock mass.The geometry of the stope is considered to have a semi-circular crown and as the extraction continues upwards, the state of stress in the adjacent rock mass is directly related to the change in relative dimensions of the opening. From their analysis Brady and Brown (2005) came to the following results: Low states of stress (generally tensile) are generated in the sidewalls of the excavation. As a jointed rock mass will disintegrate under tensile stress, normally narrow ore bodies are mined using backfill. Therefore the function of the placed fill is to prevent disintegration of the stope wall rock. Around the stope crown, crown stress factors exceeding 10 times the vertical stress field are generated . The crown stress factor increases also with increasing H/W ratio. The result from this analysis is an increasing need for crown support and reinforcement when mining progresses upwards. At low stope height the crown requires little or no support. [1]

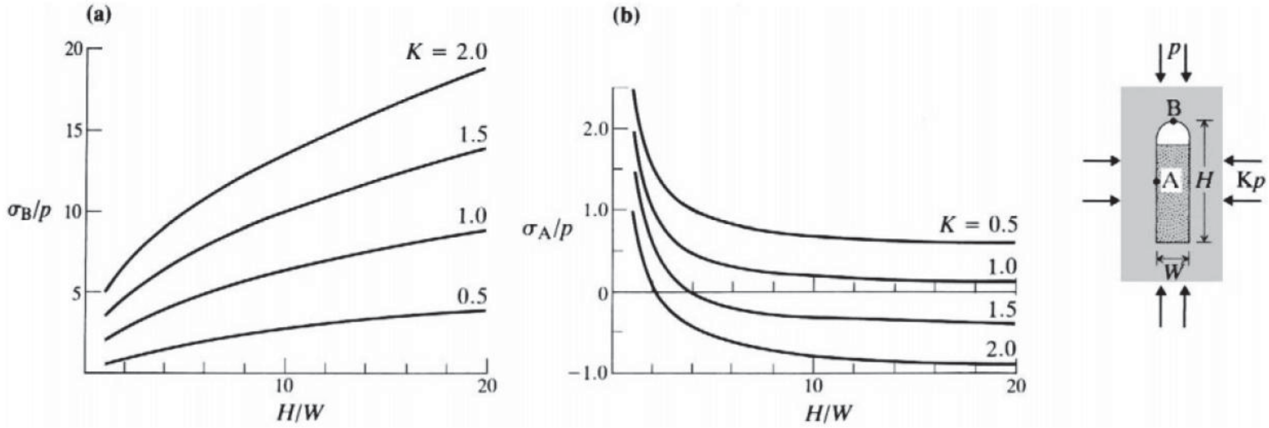

Figure 36: Roof and sidewall stresses around a cut and fill stope [1] p.420

5.1.4 Mining in the vicinity of backfilled stopes

Mining in the vicinity of backfilled areas represents a dangerous operation which requires special attention. Generally the backfilled stope has to be sealed off the new mining area or an integrate backfill body by the addition of binding agents has to be produced to avoid inrush of the backfill material into the new openings. Further on pillars are left between those openings, which might be extracted after finishing the new mining operation, when the rock mass conditions and the backfill body enable this operation. To guarantee safe workings the following points should be considered:

- Additionally to the backfill, pillars should be left for stabilization (in case of poor rock mass conditions the width of the pillars has to be increased).
- The distance of the new openings to the backfilled areas has to be surveyed via drill holes.
- These drillings can be used to control the condition of the backfill body.
- When the fill body is saturated, the stability pillars should not be extracted.

- When using hydraulic fill, a remaining safety pillar of at least 8m has to be maintained.
- Before starting to mine in the vicinity of backfilled stopes, the rock mass conditions above the stability pillar should be investigated. If rock mass conditions are very poor specialist advice is recommended before proceeding with the mining of stability pillar.
- When mining with only one access point, the staff has to be informed and self-rescuers ensuring 90 minutes of oxygen have to be provided in this area.
- The condition of the safety roof and safety pillar has to be monitored by extensometers.

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5.2 Influence of backfill on pillar stability

Backfill and pillars represent two elements for stabilization in underground openings, but are rarely used together because of their different load-deformation behavior. Pillars have a high deformation resistance and high load-bearing capacity, whereas backfill has a low deformation resistance and a low initial strength. Therefore in combined support systems, mostly pillars are loaded. Backfill generally comes into effect, when pillars are overstrained, so the combination of these two systems is rarely reasonable. Exceptional cases, where the combination of pillars and backfill is reasonable are for example:

- irregular deposits of high thickness, where with every slice that is being mined pillars become more slender and buckling to the side becomes a risk factor
- in narrow and tabular deposits at great depths stability pillars are sometimes employed to control rock burst hazard. In these instances structural failure around the usually very wide pillars is due to foundation failure of the hanging wall or footwall strata. To counteract this backfill can be used.

- support of pillars by backfill to increase recovery by minimizing pillar cross-section
- support of under designed pillar systems

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5.2.1 Rock mechanics aspect

From a rock mechanics point of view, backfill supports pillars in three ways (Figure 37):

1. it resists rock wedges sliding from pillar sides; hence backfill works against gradual disintegration of pillars
2. passive backfill pressure increases the strength of very high and slim pillars
3. backfill offers a horizontal pressure on pillars, which works against lateral deformation of pillars and increases their resistance [26]

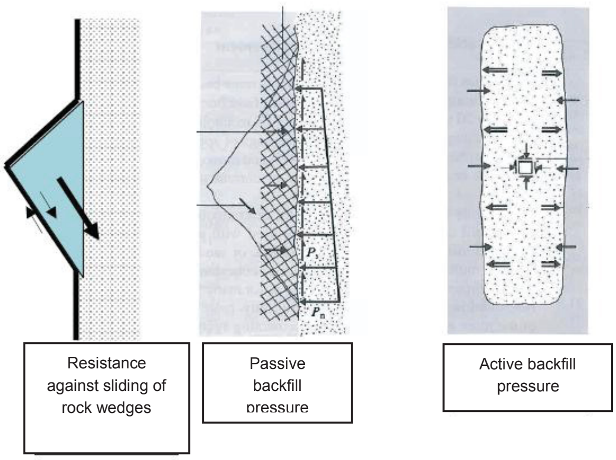


Figure 37: Interaction of backfill and pillars [26]

The passive backfill pressure equals the earth rest pressure for uncemented fill and its horizontal component is dependent on the height, density and angle of friction of the backfill body:

$$\sigma_{pass} = \rho g h_v K_{OH} \qquad 11$$

σ_{pass}… passive backfill pressure [MPa]

ρ… density of backfill body [kg/m³]

h_v…height of backfill [m]

g…gravity [m/s²]

K_{OH}…coefficient of earth pressure at rest (K_{OH}=1-$\sin\varphi$)

For a backfill density of 2000kg/m³ and an angle of friction between 15 and 15°, pressures/meter of 9 and 15 kPa occur. This pressure prevents the sliding of rock wedges from pillars and represents enlacement stresses for the pillar as well, which refers to a triaxial state of stress for the pillar and increases its strength. The triaxial state of stress reduces the driving forces for sliding of rock wedges and therefore increases the safety against sliding of rock wedges.[25]

The horizontal backfill pressure only increases linearly up to a certain backfill height. From 20-30m of backfill height on, no increase in vertical stresses occurs as a result of the silo-effect, which deviates the stresses at the side of the pillars.

The placed fill body has a higher effect on the strength of pillars, when it is subjected to passive deformations, which means that it reacts to deformation by opposing a resistance against this deformation, which is called the **active backfill pressure**:

$$\sigma_{act} = \varepsilon_{lat} E_v$$ [11]

ε_{lat}...deformation of backfill under lateral load

E_v…deformation modulus of backfill [MPa]

The lateral deformation of pillars is induced into the backfill, which then develops a resistance against the deformation. The development of the active backfill pressure depends on the deformation of the pillar and on the strength- and deformation properties of the backfill body. When the backfill body is subject to a deformation of *Δb/2* of the pillar (width of pillar=*b*), the backfill develops reaction stresses as a result of the induced convergences (Figure 38). [25]

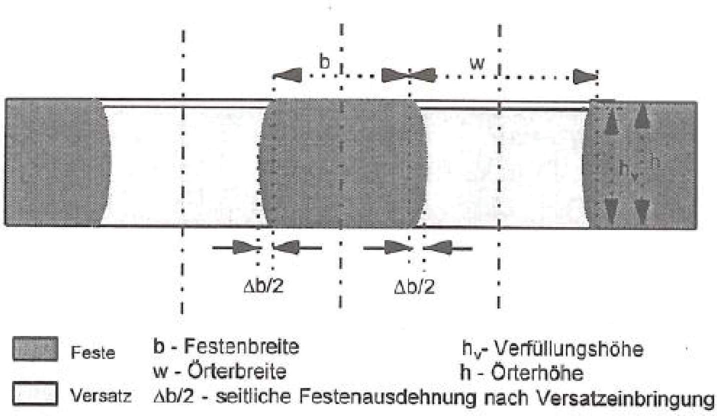


Figure 38: Sectional view of a room and pillar mine using backfill[25] p.51

Figure 39 illustrates how the active backfill pressure, generated in the backfill, influences the pillar strength and its post-peak behavior. In the soft fill little lateral stress is applied but nevertheless 85% of the maximum strength is maintained in the post-peak phase. By the application of a stiff fill, the peak strength of the pillar is increased threefoldly. To achieve these results, the underground openings must be completely filled and backfill has to be placed before any inelastic lateral deformation of the pillars occurs. [1] The optimum behavior of backfill supporting pillars is therefore a high initial stiffness and a fast development of reaction stresses against lateral deformation of the pillars. For these demands a stiff backfill is required.[25]

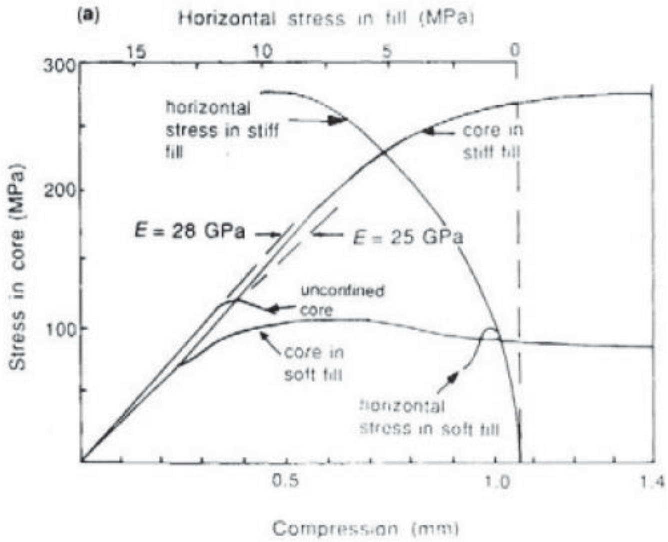


Figure 39: Effect of soft and stiff backfill on pillar properties after Blight (1984)

The overall backfill pressure acting on the pillar is the sum of the active and the passive backfill pressures. By applying backfill, the load-bearing capacity and the fracture behavior of pillars can be improved. Especially the failure by fracturing changes: Slender pillars often fail suddenly with a small residual strength, while stocky pillars fail quite slowly. By backfill application long and slender pillars behave more like short and stocky pillars, which means that the fracturing of the pillars happens slowly and that the residual strength after failure is increased (Figure 40).

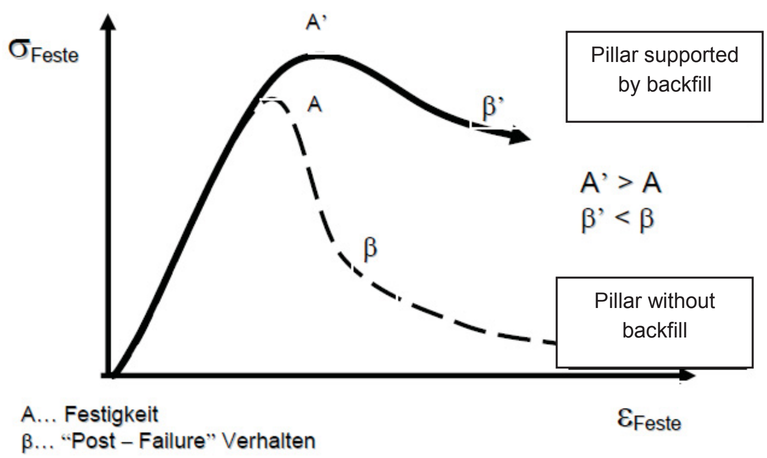


Figure 40: Influence of backfill on pillar deformation[11] p.56

Contrarily to the support mechanisms of backfill described above, a weakening of pillars because of backfill can occur as well. This weakening is due to a relative movement between pillars and backfill, because of settling of backfill as a result of its self-weight (Figure 41). As backfill is much weaker than the pillar, the relative movement is generated between the pillar and the backfill, which leads to shear stress. Because of these shear stresses, a part of the backfill weight is transmitted into the pillar, resulting in tension in the upper part of the pillar and additional compression of the lower part of pillars. This can lead to relaxation cracks in the upper part of the pillars. [26]

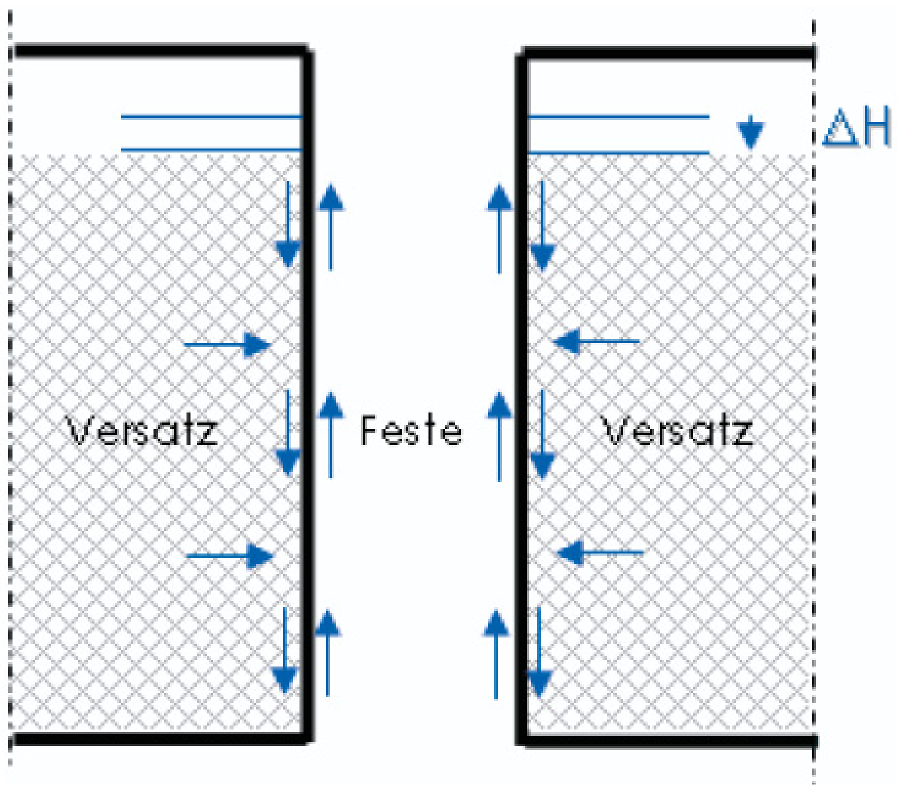


Figure 41: Interaction between pillars and backfill - relative movement [26]

5.3 Case studies

The following case studies represent examples of underground mines using mainly one or a combination of the three discussed fill types:

- Cemented (CRF) or uncemented rock or aggregate fill
- Cemented and uncemented hydraulic fill
- Cement slurry rock fill and modifications of paste fill

The first case study treats Asamara's Cannon Mine in the USA, using cemented and uncemented rock fill. As this is an example of an already closed mine, a more recent example of CRF in northern Manitoba is presented as well.

The Wolfram mine in Mittersill serves as example for all three types of backfill.

In Germany the mine in Unterbreizbach of Werra of the K+S KALI GmbH shows the application of hydraulic fill.

The third example is the George Fisher Mine at Mount Isa, which uses different modifications and types of paste fill.

5.3.1 Asamera's Cannon mine in Wenatchee

In the Cannon mine in Wenatchee, Washington, USA, the second largest underground mine in the USA, run by Asamara, gold and silver were produced until 1994, using overhand cut and fill mining as mining method.[4]

At first primary stopes were mined and backfilled with cemented rock fill (Figure 42 and Figure 43). Afterwards secondary stopes were mined and then filled with dry rock fill (Figure 44), using them as dump points for waste material. Further on several in place pillars for ground stability were left behind. Using this mining method, around 2000 tons of ore per day were produced. [4]

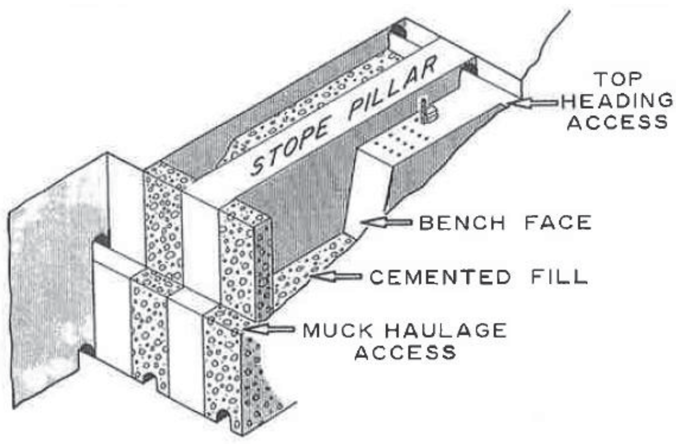


Figure 42: Cut and fill mining method at Cannon mine in isometric drawing[4] p.1757

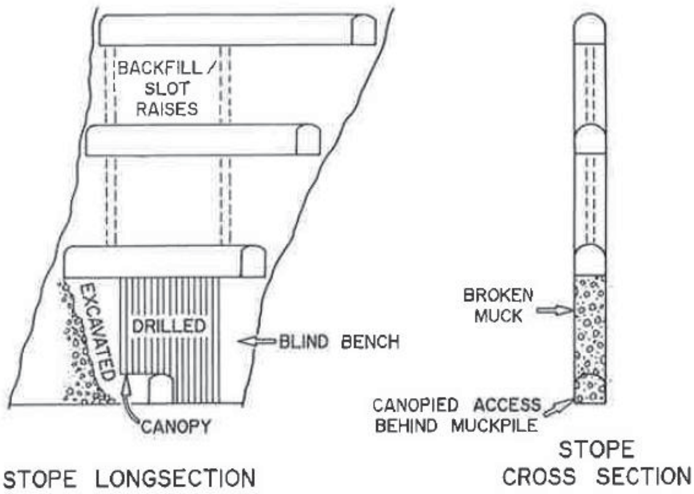


Figure 43: Overhand cut and fill mining at Cannon mine[4] p.1757

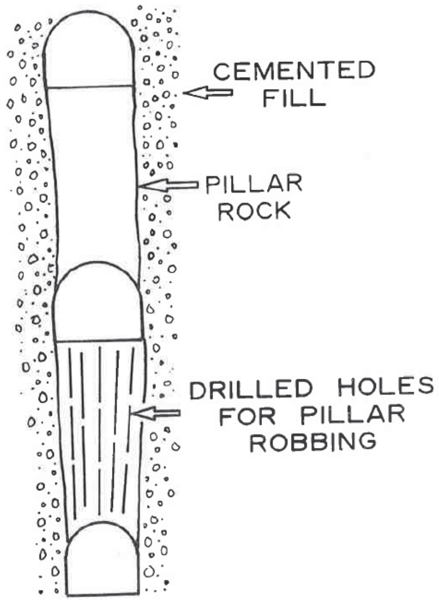


Figure 44: Cross section of secondary stope "pillar" recovery[4] p.1758

For backfill operations, Asamera’s Cannon mine used cemented or uncemented rock fill, taking river sand and gravel from the Columbia River and 5% cement in a very thick mixture. The sand, aggregate and cement mixture did not contain water and was transported to the underground mine by gravity through boreholes to an underground mixing plant, consisting of a pug mill. [4]

Sand and aggregate were dumped through boreholes into underground silos and then fed into the pug mill by vibrating feeders. The cement was stored in surface silos, then transported to the underground plant and fed into the pug mill by a pipeline in a borehole from the surface. [4]

Primary stopes were filled with high-strength cemented fill (up to 8,3 MPa) and secondary stopes were filled with uncemented dry rock fill. High-strength cemented rock fill was introduced into the primary stopes, as during mining of the secondary stopes, full overburden loading was applied onto the already filled primary stopes. The fill product was dumped down a backfill slope with a 38° angle of repose. The low water content of the fill allowed the trucks to drive on the freshly placed fill. When finishing the filling of a stope, both the cemented and the uncemented fill were placed tight against the back at the top of the ore body by a mechanical jamming device in conjunction with a loader. [4]

The cemented fill material consisted of 55% (76mm) aggregate, 40% sand and 5% Portland cement. This resulted in segregation during dumping and that's why the maximum aggregate size was reduced to 51mm, which relieved the problem.

Some of the stability pillars were recovered later and then filled with high-strength cemented rock fill, which allowed complete pillar recovery without significant surface subsidence.[4]

5.3.2 Underground mine in northern Manitoba

The case study mine is located in northern Manitoba near the town of Thompson. The mine uses sublevel stoping with stope dimensions of 18mx12mx30m (*l* x *w* x *h*) and CRF backfill. The backfill product is prepared by mixing rock aggregate with binding agent slurry (Portland cement:fly ash=30:70). The backfill material is graded and it comes from a nearby open pit mine and from development works from the underground mine itself. The waste material mainly consists of biotite-schist, which has a low porosity with an average uniaxial compressive strength of 100 MPa and a Young's modulus of 56 GPa. Trucks are used to transport the waste material into the underground mine, where the material is then dumped into a fill raise. The raise feeds different levels of the mine through finger raises. The binding agents are stored in an underground binder silo. In an underground flash mixer, the binder slurry with a water:binder-ratio of 0,5 is produced and then pumped to the stopes. The stopes are filled from the top using a load-haul-dump mechanism. The mixing of the binder agent slurry and the waste rock is either conducted by dumping a bucket of aggregates in a sump and showering it with binding agent or by showering the waste rock with the slurry directly in the bucket. The mine runs on 2 shifts per day with an average backfill placement rate of 500 tons per shift.[10]

5.3.3 Wolfram Mine in Mittersill

At Mittersill every year 430 000t of scheelite ore are mined from a deposit, with a dip of 55° and which plunges to the WNW. The mining method depends on the thickness of the ore body, the overburden and on the rock mass quality and varies between sublevel caving and sublevel stoping (Figure 45) with hydraulic backfill.

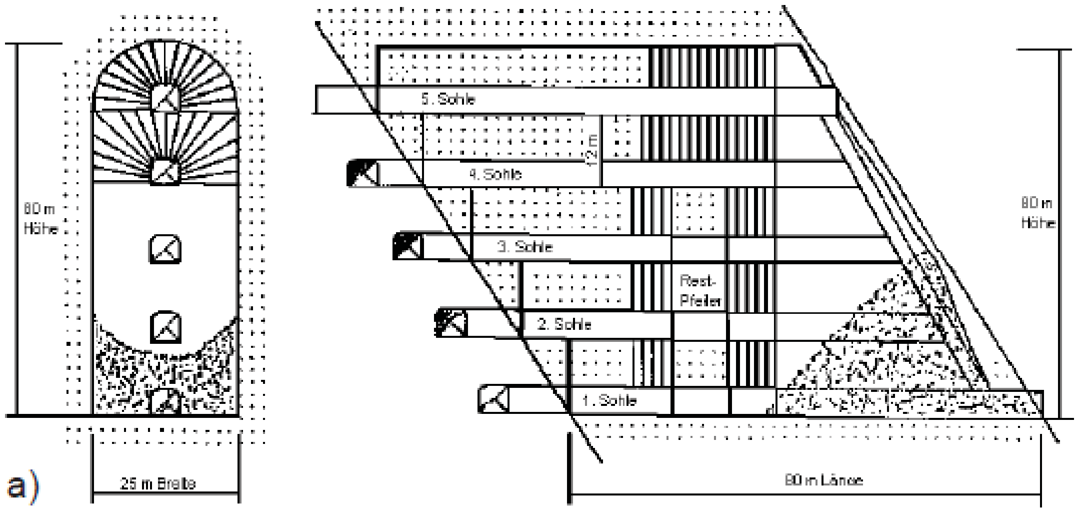


Figure 45: Sublevel stoping at Mittersill[28] p.323

The ore body in Mittersill can be divided into the western and the eastern part. The eastern part of the ore body is mined by surface mining, while the western part is mined in underground mining activities. The part of the deposit exceeding 1175m has already been completely extracted and backfilled with hydraulic fill.

Sublevel caving is used in the lower levels of the mine (<1100m) where the ore body is weak and the surrounding rock mass is stable.

Sublevel stoping with backfill is used either in striking direction or in cross-cut direction depending on the thickness of the ore body.

The main reason for the mine to use backfill was the high amount of waste material which accumulates due to the low mineralization of the ore body. More recently the depth below surface of the deep ore bodies is approaching 1000m resulting in considerable rock pressure problems in the richer ore bodies which are characterized by lower mechanical properties due to be situated in higher mineralized shear zones. Generally Mittersill uses three types of backfill:

- Drop fill
- Hydraulic fill

- Paste fill

The waste material from development works is directly dumped by wheel loaders into the mined openings.

The hydraulic fill consists of fine-grained tailings from the processing and is pumped through a 75 mm pipe from the surface to the underground mine over 3km. No binding agents are used for hydraulic fill, and therefore the underground openings are steined at first and then the hydraulic fill can be placed. When paste fill is necessary due to stability requirements, the tailings from the processing plant are dewatered underground and binding agents are added in a central underground backfill plant. Then the paste fill is pumped into the openings. [28]

5.3.4 Unterbreizbach of K+S KALI GmbH

The mine of Unterbreizbach of K+S Kali GmbH started to place hydraulic fill in the underground openings in 1997 because of the large amount of salt tailings, which naturally occur with potash, with the main reason to reduce the amount of salt load placed in the rivers. This is a typical example for waste disposal as main reason for backfill application. Underground openings with a height of up to 100m are filled with a volume of more than 1 million m³ of backfill product. The backfill infrastructure consists of a surface backfill plant, an underground pipe system and of the liquor recirculation pumping system (Figure 46).

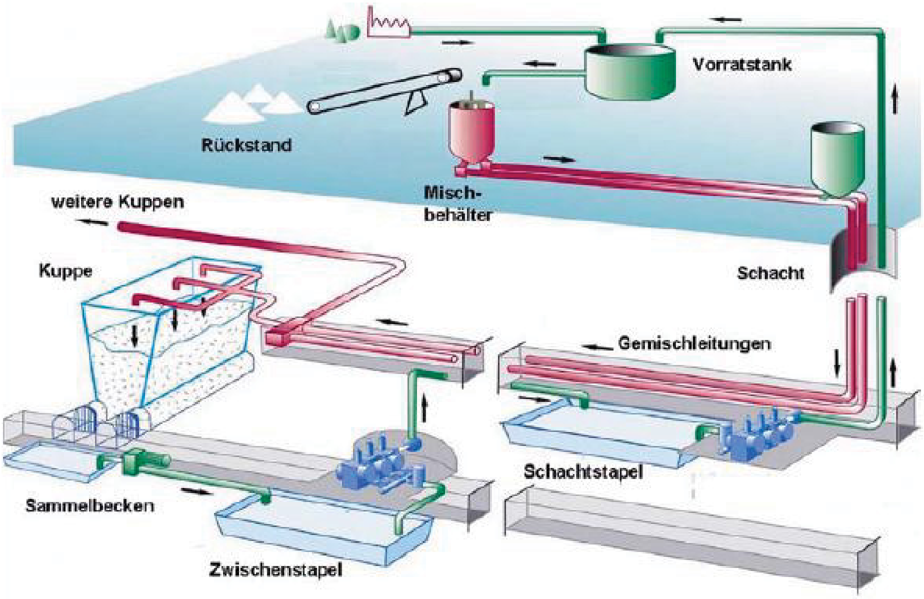


Figure 46:Scheme of backfill infrastructure[29] p.35

In the surface backfill plant, the tailings from mineral processing are mixed with the transporting medium. The transporting medium is an almost saturated $MgCl_2$ solution, which is pumped to the surface after drainage of the backfill body to be reused.[29]

The backfill plant is connected to the processing plant, to guarantee a continuous removal of the tailings from the processing plant without intermediate storage. In the backfill plant the tailings and the transporting medium are mixed and then transported through a shaft downpipe by gravity. The underground openings are connected with the pipeline systems by drill holes from the upper level. The lower levels serve for the drainage of the backfill body, as drainage channels and collecting ponds were created there. From the lower level the transporting medium is pumped to the surface by different pumping stations using piston membrane pumps and centrifugal pumps. The pipelines for recycling the transporting mediums use very resistant cast basalt coatings, as corrosion and crystallization of the salt minerals attack the inner surface of the pipelines.

When starting to place the fill product, a high mixture density has to be used for a long time period to create a filter cake in front of the backfill barricades, which prevent fine grained material to drain through the barricades (Figure 47).

Figure 47: Fill barricades in lower level of the mine[29] p.37

The full drainage of the backfilled openings takes up to 1 year after completing the filling process. As a result of the shrinkage of the backfill body (up to several meters), a refilling process must be conducted to fully fill the openings.

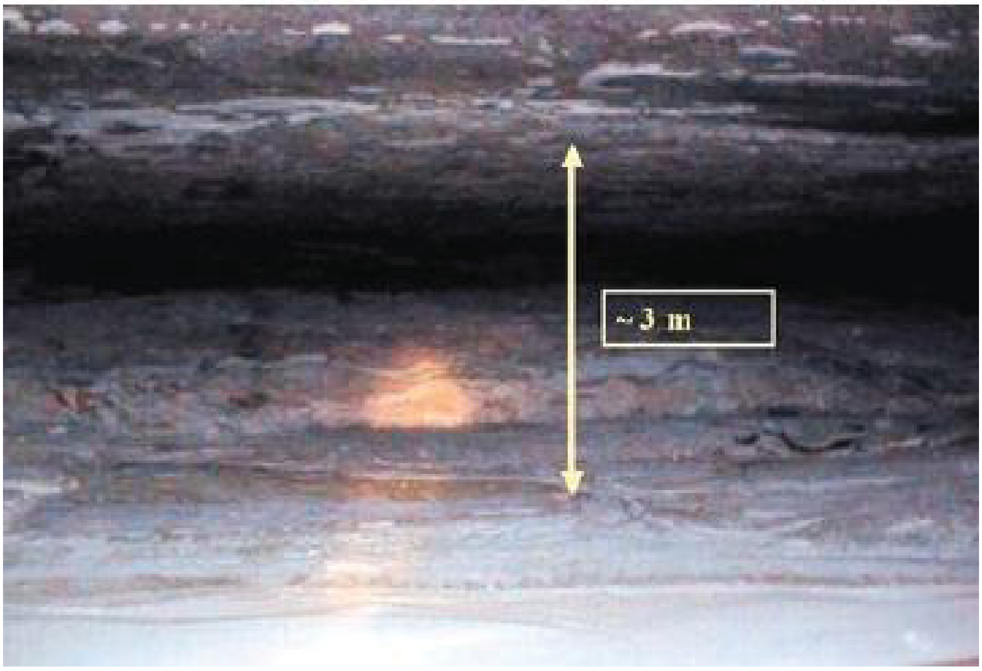


Figure 48: Shrinkage of backfill body[29] p.39

5.3.5 George Fisher Mine at Mount Isa

The George Fisher Mine located 22km north of Mount Isa in Queensland, Australia represents the second largest zinc reserve in the world, run by Xstrata Zinc Group. The zinc reserve consists of eleven stratiform tabular and parallel ore bodies with a near N-S strike and 30 to 90° dip. The mineralization is of sphalerite-pyrite-galena-pyrrhotite type, separated by shales and siltstones with total reserves of 74 million tons. Using Bench stoping and open stoping as mining methods, in 2010 the mine produced 3,5 million tons of ore. The C and D ore body are 15-25m wide and are presently mined using transverse open stoping. The primary stopes possess a height of 30m between sublevels and are 10m deep along the striking direction. The secondary stopes are 30m high with a width of 15m along the striking direction. The main access drives to the ore bodies are parallel to the ore body's strike in the footwall side, accessing the ore bodies by cross-cuts (Figure 50). The primary stopes are filled with paste fill and the secondary stopes are filled with waste rock fill (Figure 49). In Figure 49 the green stopes represent the primary stopes and the grey stopes represent the secondary stopes filled with waste rock fill.

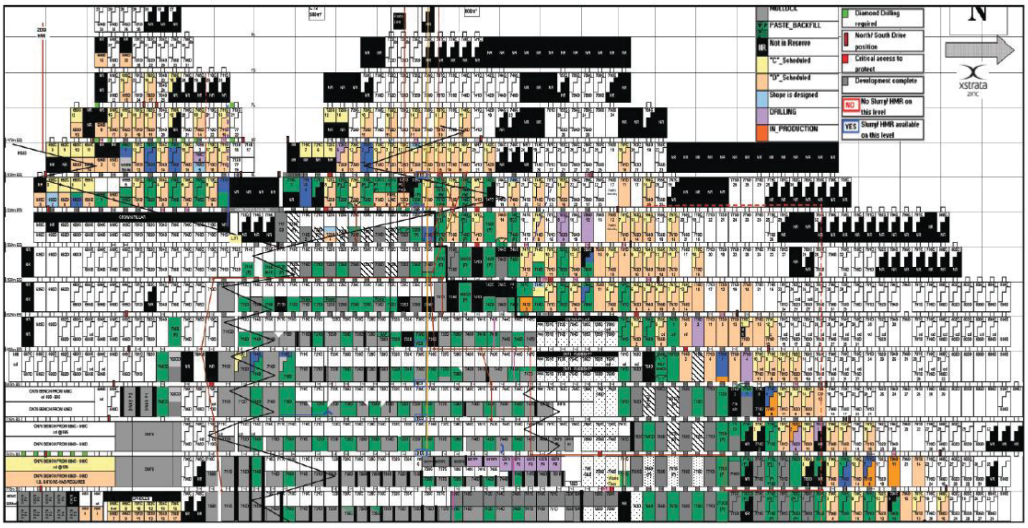


Figure 49: Vertical view parallel to strike direction at George Fisher Mine stoping sequence[30] p.6

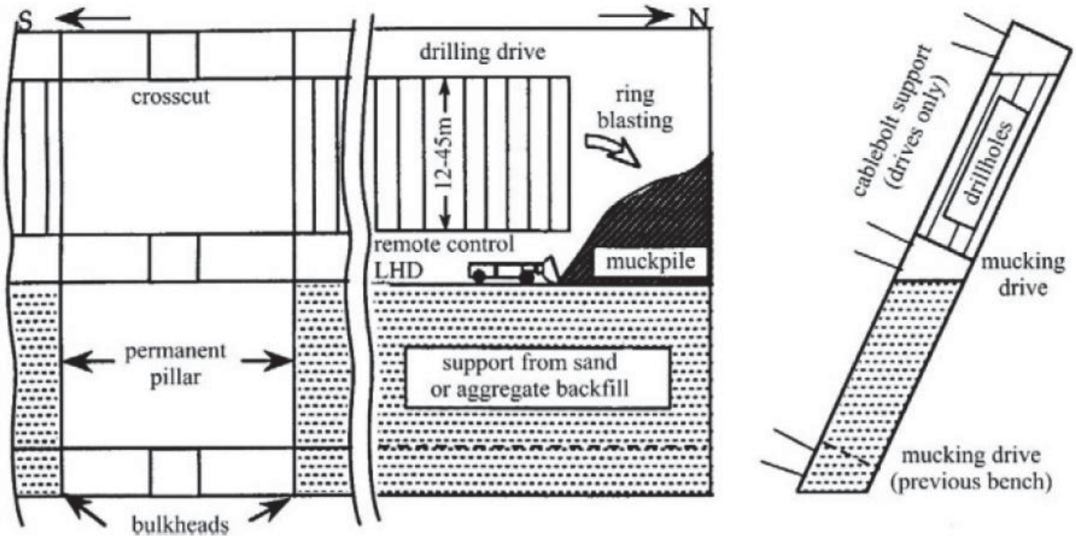


Figure 50: Stoping geometry at Mt Isa, Australia [1] p.362

At George Fisher Mine different mine fill types were developed over time:

- blended tailings paste fill
- rocky paste fill
- total tailings paste fill
- cement slurry rock fill.
- reclaimed tailings paste fill

For total tailings paste fill, thickened zinc-lead tailings would have been transported from Mount Isa through a 22km pipeline to be processed at George Fisher Mine. As the capital expenditure for this pipeline system was higher than for cement slurry rock fill, in 2001 George Fisher Mine decided to use the cement slurry rockfill.[30]

This cement slurry rock fill consisted of graded rock fill with or without fines, 5% Portland cement and water. The water-cement ratio could be found at 0,8. The rock fill used to be loaded onto trucks and sprayed with the cement slurry, which was then dumped into openings by trucks. As one of the main concerns was segregation in the fill product, additional fines were added to the backfill product. The additional fines required a mechanical mixing of the fill product, as fines prevented the rock fill penetration by the cement slurry.

From the end of 1997 on, heavy-medium reject aggregate was a product of the processing plant, which then was used as rock fill for the backfill product. The difference using the heavy-medium reject aggregate can be found in the particle size distribution, which is 150mm maximum for rock fill and 16mm maximum for heavy-medium reject product (Figure 51).[30]

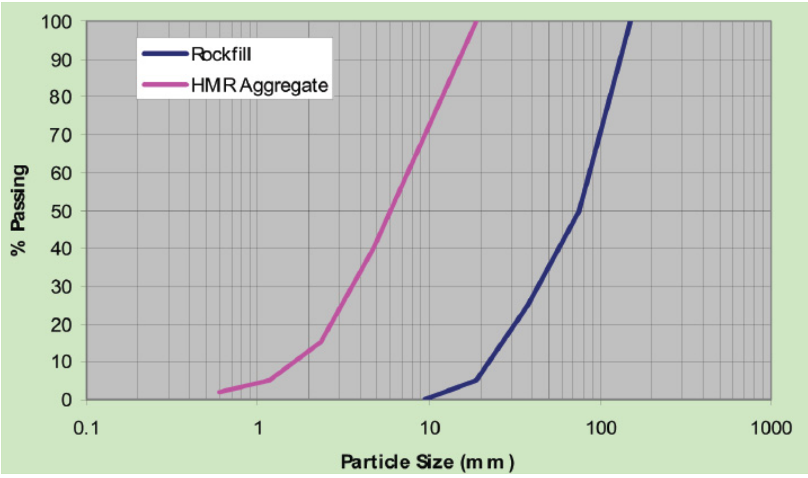


Figure 51: Particle size distributions of rock fill and heavy-medium reject[30] p.7

Through a coarse product, the cement slurry penetration did not represent any concern, whereas when using heavy-medium reject, mechanical mixing was required as well. By mechanical mixing, a 100% coating of the particles with cement slurry could be achieved. Therefore a mixing station for the backfill product was placed 700m below surface to mix heavy-medium reject with cement slurry. This product was then dumped onto underground trucks to place the fill in primary stopes (Figure 52).[30]

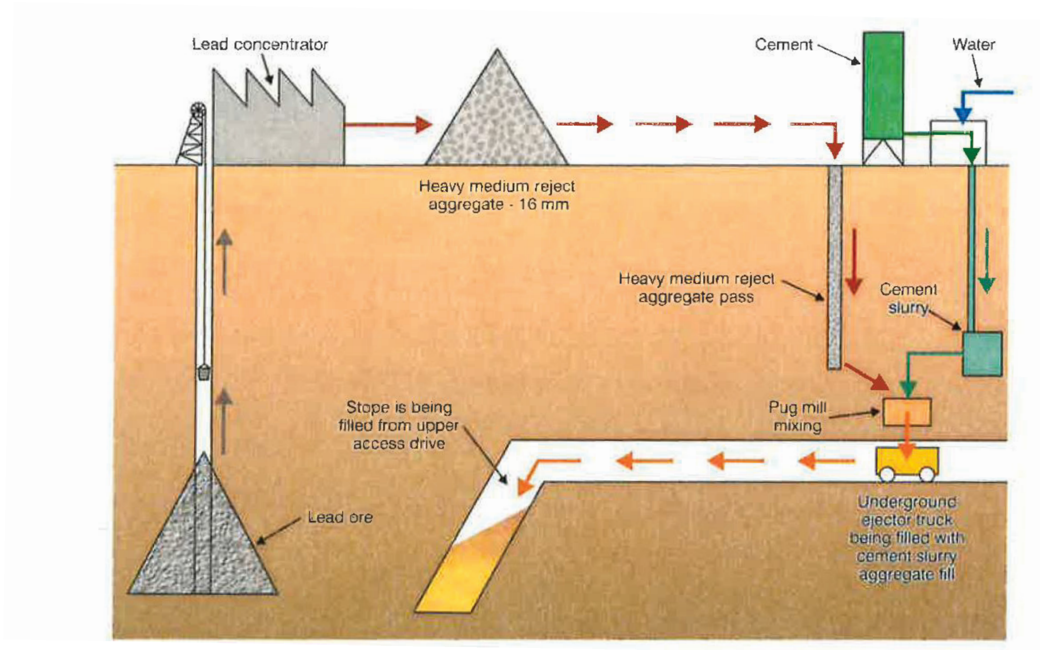


Figure 52: Material flow associated with production of cement slurry rock fill mixing at George Fisher Mine[6] p.114

This system was used during 2000 and 2004 at George Fisher Mine, and the results when exposing the fill mass during extraction of secondary stopes were satisfying.[30] Using this fill at Mt Isa resulted in no reduction of metallurgical recoveries and in constant production rates and accident rates even in remnant conditions. [2] As only one mixing station for this backfill product was created underground, the hauling distances for the fill product increased and this increased the underground traffic development.[30]

After a review of the fill system, the decision to use reclaimed tailings paste fill was taken. The reclaimed tailings paste fill is produced with tailings excavated from a tailings storage facility, mixed with binding agents and water to form a paste. The tailings were transported from the deposits to the George Fisher Mine by land trains, arriving at the mixing facility with moisture contents between 6-10%. Preprocessing of the tailings by a power screen and impact crusher was also necessary due to formation of tailings agglomerates during transport.[30]

6 Influence of backfill properties on backfill performance

The knowledge of backfill properties and their influence on backfill performance is necessary for an efficient and economic backfill operation. Concerning backfill properties, a distinction between rock mass stability and transportability influencing properties and characteristics of minor importance has to be made.

According to Helms (1988), the following backfill properties have to be studied:

- Mineralogical and petrographic composition
- Particle size distribution
- Fines content and maximum particle size
- Type of binding agent
- Type of admixtures
- Water content
- Water-cement ratio
- Porosity and density
- Internal friction and cohesion
- Permeability
- Consistency and viscosity
- Strength
- Weight-volume relationship[31]

It has to be stated that these parameters only concern the backfill material itself and the water and binding agent addition. For simplification, performance influencing parameters concerning backfill production, backfill storage, backfill aging and testing procedures are not discussed in detail.

According to the handbook on mine fill (2005) the relevant backfill properties depend on the backfill type.

In the following the main influencing parameters for backfill performance are discussed. The parameters concerning cemented backfill, like water:cement ratio or properties concerning binding agents and additives, are discussed in chapter 0.

6.1 Chemistry and mineralogy

The chemistry and mineralogy of materials used for backfill mainly influence physical and mechanical properties. In many backfill applications, tailings used as fill material contain quartz, feldspar, mica, clay minerals, sulphide minerals and carbonate minerals. Regarding the different minerals, some might have a negative influence on the strength development, when using binding agents. The presence of clay minerals (Chlorite, Illite and Kaolin) and sulphide minerals (Pyrite, Pyrrhotite) reduce the strength of a fill body for a given cement type and dosage.[31] Clay minerals might gather water, by what the achievement of the desired w/c ratio (for cemented fill) is negatively influenced.[7] In contrast, the presence of carbonate minerals (Calcite, Dolomite) increases the strength of the backfill body. [31]

6.2 Particle size gradation

The particle size distribution mainly influences the achievement of the desired densities and porosities. In Figure 53 particle size distribution curves are presented, which describe the common composition of backfill materials. The U.S. Bureau of Mines developed a particle size distribution (USBM curve) which represents a marginal distribution concerning the finest content (<74 microns) of a backfill material. Most fill materials lie to the right of the USBM curve, and few backfill materials contain more than 25% of <74 micron material. [2]

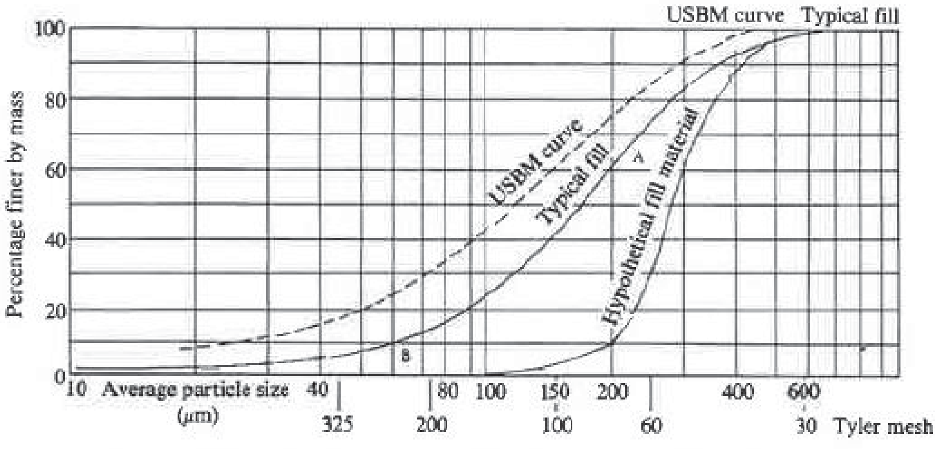


Figure 53: Particle size distribution curves [2]

The particle size also influences the equipment requirements, as coarse material is difficult to pump conventionally and so has to be transported pneumatically. The largest equipment can transport material up to 100mm. [2] The usual pipe diameter for pneumatic stowing is about 225mm and the maximum particle size of the transported material should not exceed half the diameter. Therefore particles coarser than 100mm have to be crushed before being transported pneumatically.[7]

The particle size distribution for quarry or mine blast piles is presented by the following power law equation:

$$P(u) = 100\left(\frac{u}{u_{max}}\right)^{n} \quad ^{6}$$

P(u)… probability of material finer than sieve opening *u*
u…opening size [mm]
u_{max}…maximum particle size [mm]
n…power law exponent

n generally ranges between 0,75 and 2 for quarry or blasting material. For large armor stones the value can go up to 7, so it depends on the maximum particle size. This power law equation generates a family of particle size distributions for a chosen maximum particle size, which is known as a set of Fuller curves. The particle size distribution shows that with an increasing power law exponent, the particle size distribution becomes confined to a narrow range of particle sizes. Therefore the porosity of rock fills with large power law exponents will be high and fills will not develop high shear strengths even when bound by binding agents.

When the power law exponent is 0,5, the optimum particle size distribution for concrete technology is reached, which is considered to work for backfill material as well and therefore results in a dense backfill body:

$$P(u) = 100\left(\frac{u}{u_{max}}\right)^{0,5} \quad 6$$

P(u)… probability of material finer than sieve opening *u*

u…opening size [mm]

u_{max}…maximum particle size [mm]

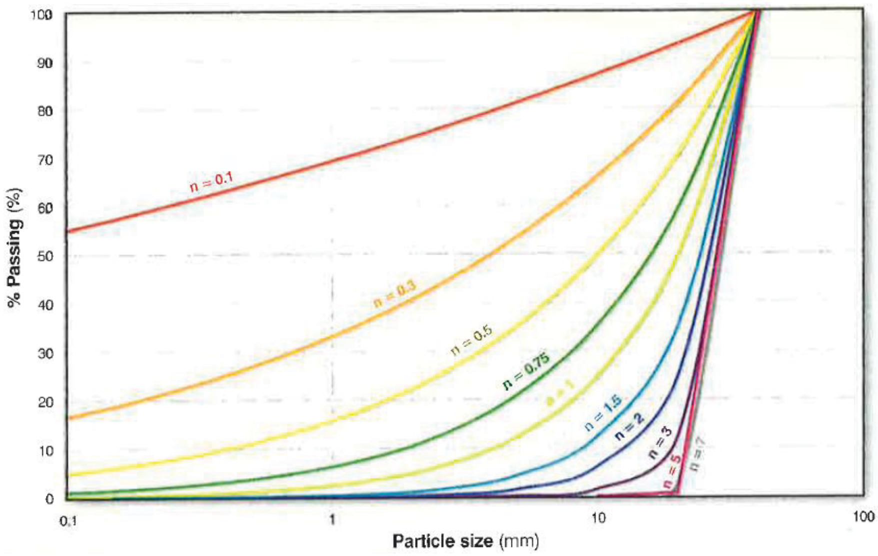


Figure 54: Set of Fuller curves for rock fill with maximum particle size of 40mm[6] p.104

6.2.1 Importance of the coefficient of uniformity

The size gradation of the backfill product mainly influences the final in-situ density. When introducing hydraulic backfill into an underground opening, normally the material is allowed to settle, which leads to segregation and layering. If the particle size distribution is not suitable, the final density will be poor. The coefficient of uniformity gives a very good idea whether a material's particle size distribution is suitable or not. This coefficient C_u is calculated by the ratio of D_{60}/D_{10}, where D_{60} and D_{10} are the diameters of the particles at 60% and 10% pass. Low values of the coefficient of uniformity indicate a uniform particle size which leads to large void spaces between the particles. [2] These backfill materials require a high binder addition to develop backfill bodies of good support properties. Values from 5-7 represent good grading and values between 5 and 10 are typical for **hydraulic fill**.

[2,6] Large values for C_u indicate a wide spread of the particle grading curve and therefore a well graded material. Paste fill has a large C_u (10-20) which forms a well-packed material and can develop high strength with a small amount of binders.

For backfill material often a maximum particle size exists and as the fraction <74 microns is removed, quite a uniform backfill product results. These materials are characterized by an excessive percolation rate, a low density and excessive compression because of high void ratio. The large amount of voids between the particles is a result of the uniform particle sizes. The interstices between the particles remain open as no smaller particles remain in the product to fill these voids. Therefore a careful grading is necessary to improve the performance of the material. [2]

The particle size distribution also influences the pumpability of the fill, when using hydraulic backfill. The larger the grain size, the faster the settling velocity will be. The velocity of the backfill product must exceed a critical velocity at which settling of the solids starts. [6]

6.3 Influence of fines

The particle size has a major influence on the fill properties, especially on the percolation rate and on the used transport mechanism.

The influence of fine particles in the backfill product depends on the backfill type used and the reduction of the fines content lies mainly in problems caused by fine particles. When considering **paste fill**, a certain amount of fines must be part of the backfill product, otherwise the particles will settle and no paste is formed. The fines help to float the coarse grains in the slurry which generates a non-settling state. Therefore in paste fill, at least 15% of fines (<20 microns) should be present. [6]

As particles become smaller, the surface drag forces in slurries and pastes dominate as well, which increases their viscosity. [6]

Fine particles have a higher specific surface and therefore hold more water. This complicates the drainage process and increases the amount of water which has to be added to **hydraulic backfill**. As the fines content in a hydraulic fill increases, it becomes more difficult for water to flow through the fill, which negatively influences the dewatering process. Therefore the amount of fines has to be reduced and most particles <74 microns are removed from the backfill material. [2,6]

When the backfill product contains a high amount of fine material, the percolation rate of the fill will be negatively influenced (see chapter 6.8).

6.4 Water content

Water is added to the backfill material for the following reasons:

- To facilitate transport in pipes
- When using binding agents, for hydration
- Acts as a lubricant and contributes to the workability of the mixture
- Secures necessary space in a paste for development of hydration products

[31]

The water content in a backfill product changes over its life cycle. In the fresh backfill material, the water content is determined by the moisture of the tailings and by the addition of water. During the transport of the backfill product, changes in the water content can occur, depending on the transport type. [9]

When the backfill is placed in the underground opening, drainage of the remaining water might be necessary as the remaining water in the backfill body mainly influences the strength of the backfill. [9]

The water content, *w*, of the backfill product is defined as the ratio between the weight of the water present to the weight of the solids:

$$w = \frac{Weight\ of\ water\ W_w}{Weight\ of\ dry\ solids\ W_s}$$ [6]

It has a major influence on the backfill properties and therefore a good understanding of the effect of water addition to the backfill material is necessary. Depending on the amount of water present and the degree of mixing, a fill material can exist in any one of liquid, plastic, semi-solid and solid states (Modified Atterberg's limits for backfill Figure 55)

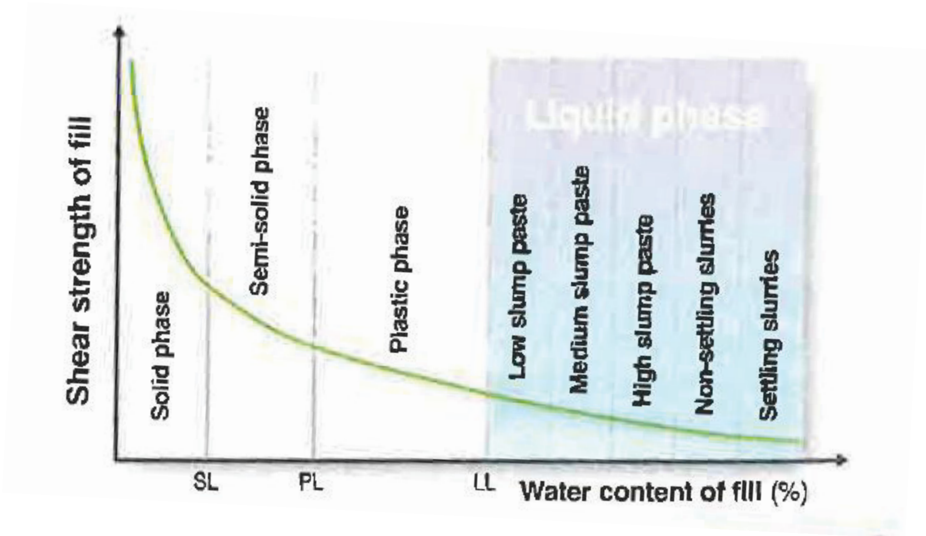


Figure 55: Consistency of tailings water mixture and relative shear strength [6] p. 34

When no or little water is present in a backfill product, the material behaves like a solid. With further addition of water, the material starts to behave like a semi-solid (could be molded with form). With further addition of water the phase of the mixture will change to a liquid phase and therefore its shear strength is decreased to a minimum. [6]

Impurities in the mixing water can cause a strength reduction of the backfill body, depending on the type of tailings and the amount and type of binding agents.[31]

6.5 Density and densification

The density of the backfill material and the densification of the backfill product in situ play a major role concerning backfill performance. Additionally knowledge of the density contributes to an efficient dimensioning of the backfill system.

In general compact fills are better performing concerning backfill properties than loose fills, so backfill should be as dense as possible.[6] The placement method of a fill product is a significant factor concerning achievable backfill body densities.

The bulk density [t/m^3_r] is a parameter to describe the densification achievement of a certain placement method:

$$Bulk\ density = \frac{Density\ of\ the\ material\ [\frac{t}{m^3{}_s}]}{Fill\ parameter\ [\frac{m^3{}_r}{m^3{}_s}]}$$ [7]

By a pneumatic backfill application, higher bulk densities can be achieved, as the impact of blown material increases densification, but segregation also occurs with pneumatic fills, so hydraulic and pneumatic methods nearly achieve similar in-situ densities. [2]

From a certain load on, settlement of the backfill body linearly depends on the bulk density, which is achieved by a placement method. The higher the bulk density, the lower the undesired settlement of the backfill body will be. The addition of water improves the bulk density, as it reduces internal friction between particles. The highest settlement of the backfill body can be found at the beginning of the loading process (Figure 56), but in general all uncemented backfill types start to resist against deformation a certain time after backfill introduction, which is not desired. To make sure that the load-bearing of the backfill body takes place earlier, binding agents can be added to the mixture.[7]

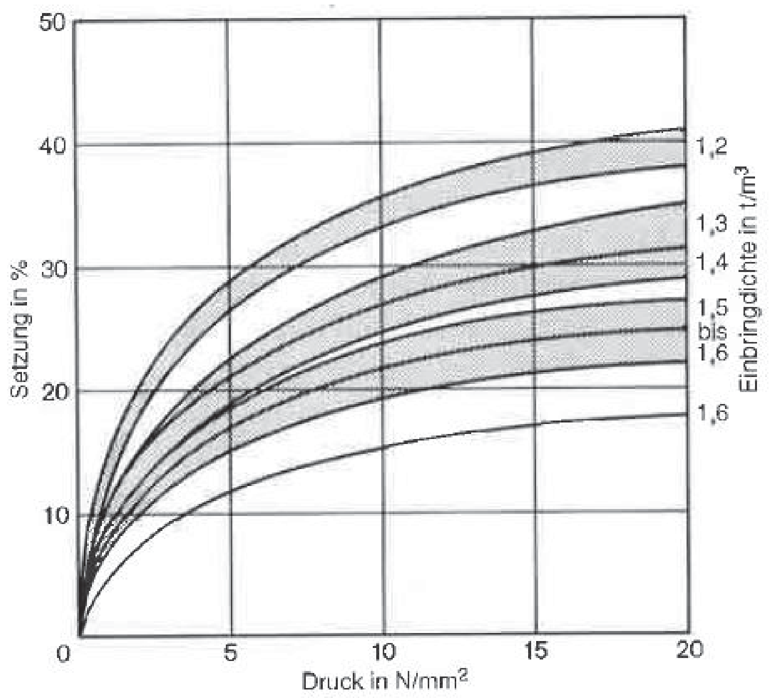


Figure 56: Load-settlement curves of a backfill product at different bulk densities[7] p.558

If the coefficient of uniformity is adapted, the bulk density can be increased and so no physical densification is necessary. Investigations by Nicholson and Busch (1968) showed that

- Decreasing the void ratio by half, decreases the strain at comparable loads by half
- Decreasing the void ratio by half, gives a 100-fold increase in the tangent modulus
- By compaction of loose fill, an eightfold decrease in fill yield resulted

Finally a fill specific gravity of backfill products lies between 1,6 and 2.[6]

The densification process of the backfill body can be achieved by compaction or consolidation. Compaction is a process in which unsaturated fill particles are forced to move closer together by mechanical energy. In compaction only air is removed from the backfill material, so the air-filled pores are reduced. The mechanical energy can come from moving machinery, blasting or by compaction methods. [6]

Consolidation is a process, in which saturated fill particles are forced by gravity related static forces to move closer together. These gravity related forces occur normally naturally by the backfill's own weight. During this process, pore water escapes from the fill. If this pore water cannot escape from the backfill material, no consolidation can occur. Regarding paste fill, consolidation will occur very slowly, as paste fill is thick and possesses a low permeability. [6]

6.6 Backfill strength

Talking about backfill strength, requires an exact definition of what is conceived using the term "strength". In general the generic term strength comprises all properties of a backfill product, which concern the resistance against deformation. Using backfill, different types of strength can be distinguished:

- Uniaxial compressive strength
- Cohesion
- Internal friction
- Splitting (indirect) tensile strength
- Bending tensile strength
- Shear strength

All these properties determine the backfill capacity, its stability and geomechanical behavior.[9]

According to Wilson et Calverd (2011) the strength of a fill material mainly depends on the following factors:

- Tailings mineralogy
- Chemical composition
- Particle size distribution
- Water to binder ratio

12

According to Saw et Villaescusa (2014) the necessary backfill strength is a function of the mining method, the geometry of the ore body and of possible failure modes. Possible failure modes could be sliding, crushing, flexural and caving. Sliding can occur as a result of low frictional resistance between the backfill body and the rock mass wall. Crushing can occur when the applied stresses exceed the uniaxial compressive strength of the backfill body. Flexural failure results out of a low tensile strength of the backfill and caving occurs because of low shearing resistance at the rock wall. [31]

The most frequently used strength parameter concerning backfill is the uniaxial compressive strength. According to Saw et Villaescusa (2014) it is a function of the fill material, binding agent type, amount of binding agents, water, solids percentage, water:cement ratio, curing time and curing temperature.[31]

6.7 Load-deformation behavior

According to Wagner (1996) one of the main geomechanical demands of backfill is its behavior during compression, which is described by the following equation:

$$\sigma_v = \frac{a * \varepsilon}{b - \varepsilon}$$ [32]

σ_v…reaction stress [MPa]
a…material parameter describing the initial behavior (characteristic value *a*=10MPa)
b…material parameter for the pore volume describing the maximum compression of the backfill body (characteristic value *b*=0,3)
ε…axial compression of a non-cohesive backfill

By the addition of binding agents, the initial behavior is controlled as a result of an increased uniaxial compressive strength and indirectly by the initial stiffness. This is of special interest in shallow depths with small convergence rates. In very deep mines, the behavior under significant compression has to be considered, which is controlled by the parameter *b.*[25]

Usual backfill shows a late load-bearing behavior as a result of a high pore volume. During the loading of the backfill body, the grains are redistributed and when the pore volume is reduced, the settlement of the backfill body starts to stabilize and the fill starts to carry the applied load. This represents a late load-bearing behavior, which is not desired for backfill. Cemented backfill possesses a lower pore volume. When applying a load on a cemented fill body, the material starts bearing the load from the beginning and the body only settles due to deterioration of the cement-fines-matrix. [7]

Figure 57 presents a typical stress-strain curve from an oedometric test on cemented and uncemented backfill. The results reveal an asymptotic approximation of a maximum densification, which is identical for cemented and uncemented fills. The difference in both curves can be found at the beginning of the loading, describing the initial stiffnesses of the materials. The initial stiffness of cemented fill is increased by the addition of binding agents, which significantly increase the cohesion among the particles. It should be noted that this test must be conducted under pore water pressure control.

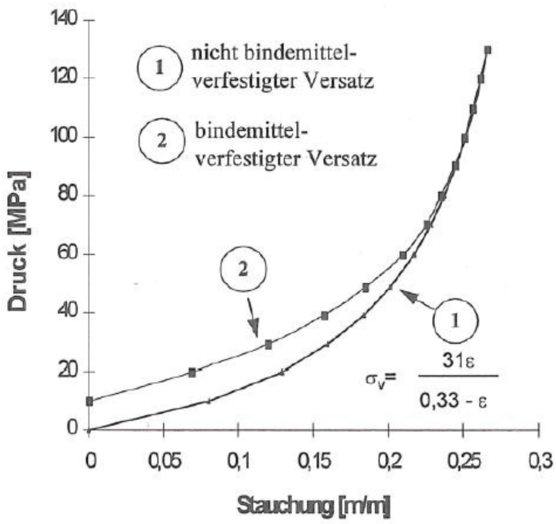


Figure 57: Axial load-deformation behavior of uncemented (1) and cemented (2) fill[25] p.51

Under lateral pressure, the uncemented fill though, needs to be compressed to be able to withstand lateral deformation of rock masses. The cemented fill body possesses an initial strength to withstand lateral deformation resulting from a higher stiffness at the beginning of the load-deformation relation. Yet a low uniaxial compressive strength of the backfill implicates the development of reaction stresses which are able to withstand lateral deformation.[25]

6.8 Permeability and backfill drainage

The backfill body usually contains, depending on the transport method, a significant amount of water, which negatively influences its performance. Therefore the water has to be removed from the backfill body by drainage. [9] For efficient backfill drainage, the backfill body has to be permeable. The drainage of the backfill body is not only influenced by the permeability of the backfill body, but also by the sedimentation behavior of the material. The sedimentation behavior is influenced by the particle size distribution, the particle shape distribution and the particle density. [9]

If a backfill body is subjected to a water head difference *h.* the applied head difference would induce a water flow *Q* through the backfill body, which is described by the equation of Darcy:

$$Q = k\frac{(h_1 - h_2)}{L}At = kiAt$$ [6]

$$q = \frac{Q}{t} = kiA$$ [6]

Q..quantity of water [m³]
q… rate of water flow [m³/s]
k… Darcy's coefficient of permeability [m/s]
h_1-h_2… water head difference
A…cross-section [m²]
L…length [m]
i…hydraulic gradient [-]
t…time [s]
Applied to a stope, where water drainage is conducted (all flow occurs through the draw point), the equation of Darcy is presented as follows:

$$Q = KA\frac{\delta H}{\delta L}$$ [6]

Q…flow rate out of the stope [m³/s]
K…fill mass permeability [m/s]
A…cross sectional area of draw point [m²]

$\frac{\delta H}{\delta L}$...hydraulic gradient in the draw point [m/m]

Darcy's coefficient k describes the permeability of a material. The denser the packing the lower the flow rate and therefore the lower the permeability k will be. [6] A dense packing depends on the particle size distribution as well as on the particle shape distribution, hence on the pore content. Therefore also the permeability of the backfill material depends on these properties. [9] Additionally the value of k depends on the viscosity of the permeating fluid. [6]
As it can be observed from the equation adapted to a drained stope, the flow rate out of the stope linearly depends on the hydraulic gradient. As a result of the area reduction between stope and draw point, this gradient will be smaller within the stope than within the draw point to maintain flow equilibrium.[6]

The percolation rate is a measure of the permeability of a saturated hydraulic fill sample where the driving head is just equal to the flow path length. The higher the slimes content, the lower the percolation rate will be. [6] All particles smaller than 74 microns control the percolation rate; the higher the amount of this fraction is, the lower percolation rates will be. Generally percolation rates of 100mm/hour are acceptable; values above 100mm/hour don't have benefits, as they don't contribute to a faster mining cycle. So a cut at 74 microns is done in an attempt to remove all particles smaller than 74 microns, which is impossible. But the remaining small particles usually do not cause problems. [2]

Hazen's relationship for filter sands allows an estimation of the percolation rate in mm/hr:

$$K_p = c(P10)^2$$ [6]

K_p...percolation rate [mm/hr]
c...constant with typical value of 3600 [1/mmhr]
P10...particle size at 10% passing [mm]

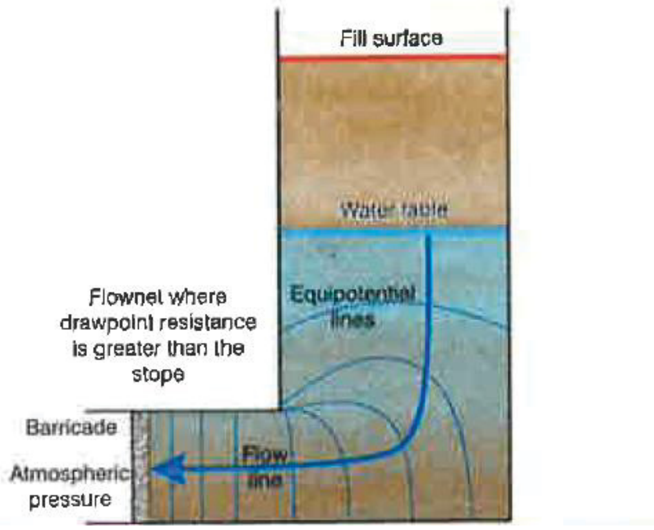


Figure 58: Water head variations along flow lines [6] p.59

6.9 Consistency and viscosity

A tool for the description of the rheology of a paste fill is its yield stress. The yield stress is the stress at the limit of the elastic behavior. It describes the minimum force necessary to initiate the flow in a paste at almost zero shear rate. For design of a paste backfill transportation system, a deep understanding of the yield stress as a function of the solids percentage is essential. [31]

6.10 Weight-volume relationship

The weight-volume relationship of a backfill product is determined by its porosity, void ratio and relative density. To describe this relationship the specific gravity of a material is used.

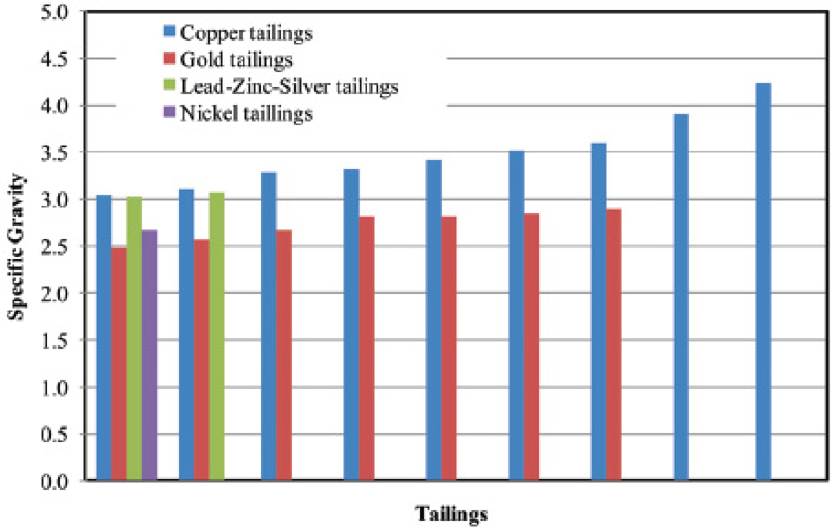


Figure 59: Typical specific gravities of different tailings for backfill[31] p.144

7 Influence of binding agent addition on backfill performance

Backfill materials are from grained nature, but have different particle size and particle size distributions, therefore the pore volume in backfill can be extremely different (mostly between 25 and 60%). Because of the pore volume the backfill has got a strongly non-linear load-deformation behavior. By cementation, properties of backfill like the initial strength and deformation behavior can be improved. [11]

Cementation means using binding agents to bind the material particles together. The strength of the backfill with binders depends on the particle size distribution of the fill material, the type and amount of binders and the density of the backfill heavy liquid. The most commonly used binding agent is cement, which can be used in two ways: as ultra-low cement content (20:1-40:1 sand-cement ratio) or for used in slushing and mucking floors (10:1-5:1). [2] By binding agent addition to the backfill mixture, uniaxial compressive strengths of 1-10 MPa are reached.[7]

7.1 Types of cemented backfill

Different types of cemented backfill are available, which can be distinguished according to the following properties:

- Particle size range
- Structure
- Consistency of the fresh mixture
- Water content of the fresh mixture
- Strength
- Conveying and placement characteristics

According to the particle size range, a distinction between fine grained and coarse grained backfill products can be done. Normally material from fine grained mineral processing like flotation tailings belongs to fine grained backfill, with a maximum particle size of 1mm. Coarse grained backfill contains fine grains as well but in a very low grade. The maximum particle size of coarse backfill products holds several decimeters. [9]

7.2 Components of cemented backfill

Cemented backfill consists like concrete of tailings, binding agents, water and additives.

Tailings represent quantitatively the biggest portion of the mixture and as their quality is not constant their properties have a strong influence on the strength of the backfill product. Tailings can be divided into coarse and fine grained tailings. Fine grained tailings have a maximum particle size of 1mm and coarse grained tailings of several decimeters. Fine tailings are often fine mineral processing tailings, which accumulate generally in heavy liquids. As sedimentation characteristics and permeability of the backfill product are important for its transport and drainage properties, it might be necessary to remove finest tailings, which have to be placed in settling ponds which represents an environmental drawback and can have adverse effects on the stability of tailings dams. [9]

Coarse tailings are normally used for drop fill, slinger stowing and pneumatic stowing, rarely for hydraulic or paste fill. To guarantee convenient transport properties of coarse tailings fill, a certain amount of fine tailings might be added to the backfill product, especially for paste fill. Concerning admixtures, generally substances with special chemical-physical effects are added to the mixture to modify the properties. [9]

Water represents the necessary component for setting of hydraulic binders. The amount of water mainly depends on the placement type, as drop fill, slinger and pneumatic stowing use low water contents and hydraulic fill uses a high amount of water for transport. [9]

Additives are added to the backfill product in order to modify its properties concerning consistency or setting progression. The same additives as in concrete technology are used: setting retarder, setting accelerator or fluxing agents. A setting retarder might be used when the transport distances for cemented backfill are very long. Setting accelerators are used when an early load-bearing strength for stable backfill slopes is required. Because of high accelerator amounts for this purpose, reasonably priced substances like calcium chloride are applied. Fluxing agents are used to improve the transport of highly viscous backfill mixtures. Flocculating agents may be used as well, when sedimentation or drainage properties have to be improved. [9]

7.2.1 Binding agents

Binding agents represent the smallest portion of the mixture but their influence on the total costs of the backfill product is considerable. The most commonly used binding agent is Portland cement but also other substances with pozzolanic or latent hydraulic properties like metallurgical slag, filter ash and sulfides. For backfill products, normally Portland cement of low to medium strength class is used, as a higher strength class would not be effective because of the low binding agent content. Binding agents with a resistance against sulfides and sulfates might be useful, as these chemical compounds occur in considerable amounts in tailings and water and as they negatively influence the long term strength of the backfill body.

During hydration heat is generated, which does not affect the mine climate, as the binding agent amount is extremely low.

Metallurgical slag as alternative for cement has the same physical properties and accounts for hardening in presence of substances like calcium hydroxide.[9] It consists of the same essential components as cement but in different proportions.[6]

Filter ash is formed during combustion of lignite or hard coal and shows pozzolanic properties. The filter ash particles are spheres, which influences the viscosity of the cemented backfill mixture and therefore the consistency of the backfill material and body. The properties of filter ash mainly depend on the type of coal and the combustion procedure and in particular the fineness of filter in the stack.

Sulfides harden in contact with moisture accompanied by an extreme formation of heat, which can lead to self-ignition and release of sulfur dioxide. Therefore their application as binding agent in backfill products is rare. [9]

Gypsum has also been considered as an alternative binding agent to cement. The production of calcined gypsum is cheaper because of its lower energy requirement. For a comparable backfill performance concerning strength, 2,5 to 4 times the amount of cement must be used. Using gypsum, the final strength is reached within 10 days, but the source of gypsum has to be close to the mine to be an economic alternative for cement. [6]

When using ash, slag, filter ash, sulfides or gypsum (which are cheaper than Portland cement) instead of cement, more of these binding agents have to be added and chemical reactions are slower, so higher strength only occurs after a longer period.[11]

When replacing 5-40% of Portland cement with fly ash in fine-grained hydraulic fill, in general it is not recommended to replace more than 21,9% of cement with fly ash, to meet the technical requirements of ordinary Portland cement.[33]

Cement content (wt%)	Curing time (days)	Specimen tested	Cohesion c [MPa]	Friction angle φ [deg]
4	7	22	0,13	30
	28	23	0,15	
8	7	24	0,24	33
	28	24	0,31	
16	7	24	1,02	36
	28	24	1,46	
0	205	11	0,03	32
4	207	12	0,21	37

Table 7: Strength parameters as a function of cement content and curing time [1] p.412

By increasing the binding agent content, the compressive strength of a cemented tailings backfill body increases, with regards to suitable water:cement ratios (Figure 60). Binding agents, water and fines form a matrix in the backfill product, whose strength and amount increases with increasing cement content. This mainly influences the strength of the backfill body, as the strength of the matrix is much lower than the strength of the tailings. [9]

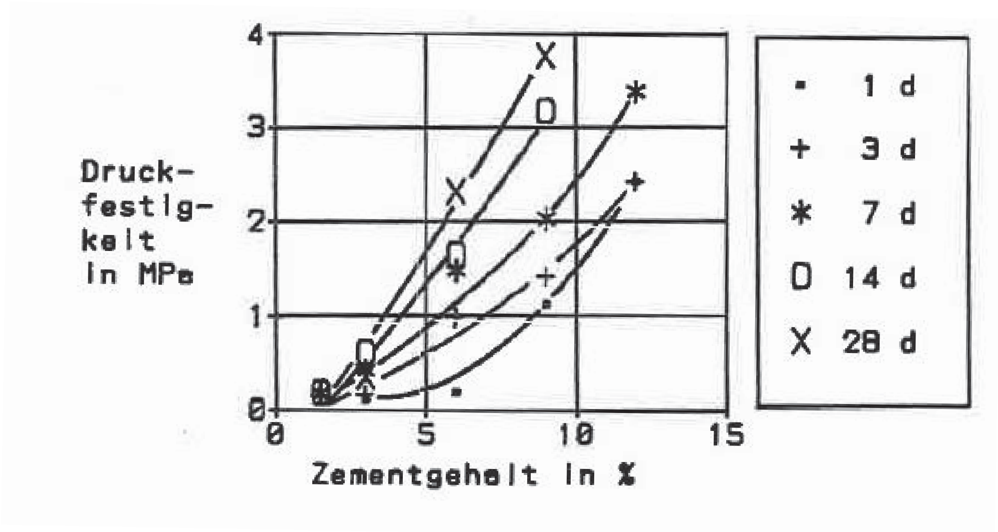


Figure 60: Relation between compressive strength and cement content using tailings backfill [9] p.59

The relationship between binder content and fill strength for cemented tailings backfill however is not linear.[9] Generally about 5% cement addition (total dry weight fill 95%, cement 5%) produce suitable strengths for most fill applications. [6]

Helms (1988) observed that the strength of cemented hydraulic fill increases disproportionately with the cement content (Figure 61), but special types of cement don't show any advantages. An important parameter for cemented hydraulic fill represents the grinding fineness of the cement, which is optimum at a specific surface between 2500 and 3000 cm²/g. By the addition of metallurgic slag the strength of hydraulic fill can be increased, especially at low cement contents.[9]

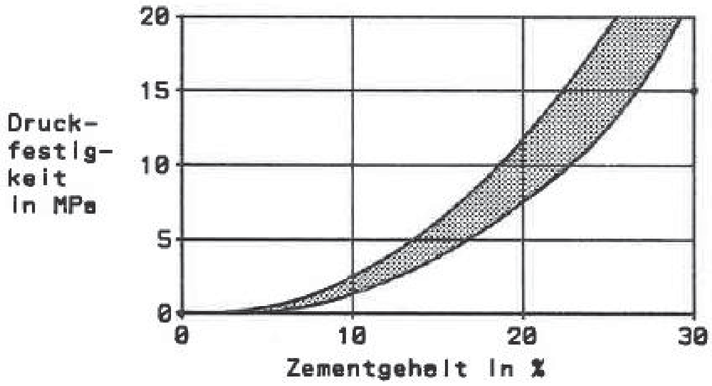


Figure 61: Strength of hydraulic fill as a function of cement content[9] p.84

By the addition of filter ash, the strength of hydraulic fill can be increased as well, even until a rise in strength of about 20% (at least 10% cement content are necessary). At low cement contents, filter ash can cause a decrease in strength. [9]

7.2.2 Water-cement ratio

The water-cement ratio is defined as the relation of the total amount of water in the backfill composite to the binding agent mass:

$$\frac{w}{c} = \frac{m_w}{m_c} = (m_{wk} + m_{wc})/m_c$$

m_w…mass of water [kg]
m_c…mass of binder [kg]
m_{wk}…mass of water contained in tailings [kg]
m_{wc}…mass of additional water [kg]
9

Concerning ordinary concrete the early strength and the final strength increase with decreasing w:c ratio, but the undercut of 0,4 reduces the strength. In practice normally higher values are used as a result of better consistency and preparation. 99

In comparison to concrete, water-cement ratios in backfill products are much higher as a result of the low binding agent content. In contrast to concrete, low water-cement ratios don't lead to the highest strength of the backfill body. The reason for this fact can be found from studies about soil consolidation with cement. For the backfill product a high load-bearing capacity is required which results from a dense packing, a low pore content and a high number of particle contact points. The required binding agent content should be low because of costs. For a high strength of the fill mass, the pore volume should be filled with a slurry out water, fines and binding agents. If low w/c ratios are used, high binding agent contents are required which is not desired in terms of costs and at low w/c ratios, there is not enough water available for creating a slurry to fill the available pore volume.

When studying the relation between water content and compressive strength of a cemented tailings backfill body (after 28days), the curves show clear maxima (Figure 62).

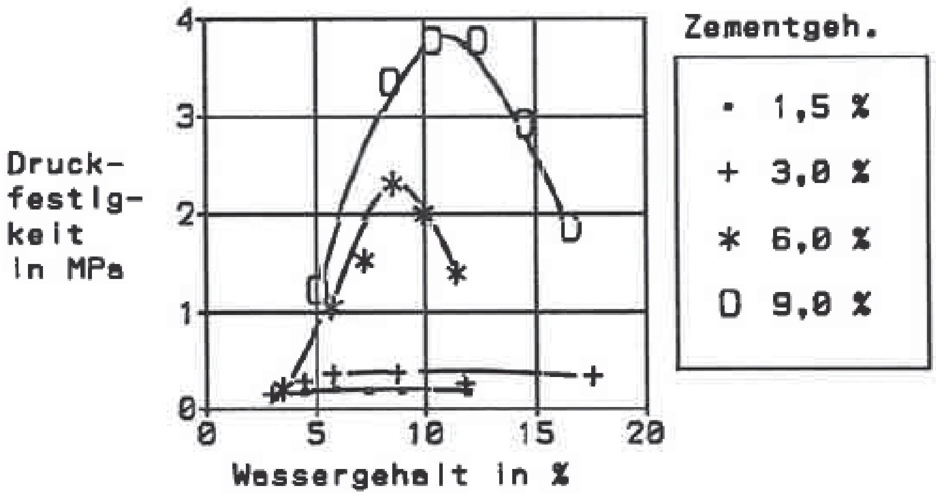


Figure 62: 28-day compressive strength of cemented tailings backfill as a function of the water content [9] p. 58

The backfill product consists of tailings, the binding agent matrix (binding agent, fines, water) and of pores (filled with water or air). Too low water contents (and therefore low water-cement ratios) lead to high pore contents and therefore to low strength of the backfill body. [9]

So the addition of water is extremely important, as a certain amount is required to cover the particle surfaces, especially fines require a high amount of water because of their elevated specific surface area. Additionally, water is required for the hydration reaction with the binding agent. Increasing the binding agent means that more water is required for the hydration reaction. The optimum water-cement ratio for a cemented tailings backfill is therefore decreasing with increasing cement content (Figure 63). [9]

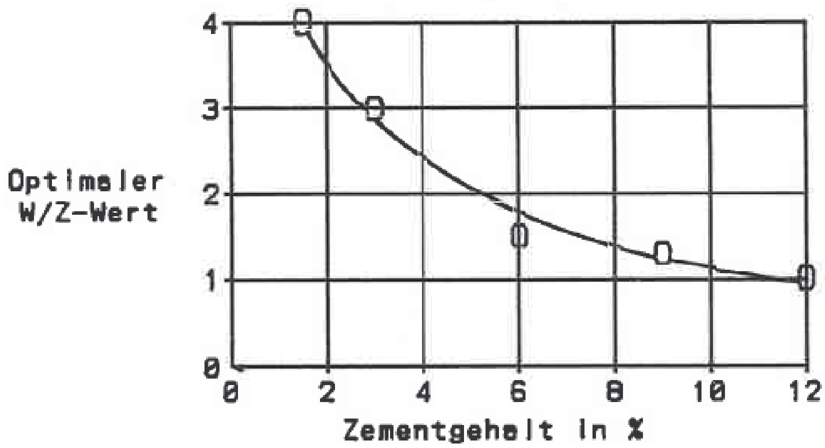


Figure 63: Optimum water-cement ratio as a function of cement content [9] p.61

The theoretically necessary water-cement ratio for cemented tailings backfill can be found to fall between 0,3 and 0,4.If this necessary amount of water is not available, the hydration reaction cannot be complete. If an excess of water is available, pores are filled with water, which reduces the strength of the backfill body.

For cemented hydraulic fill the optimum water-cement ratio decreases with the cement content (Figure 64). A high water content leads to low strengths of the backfill body and the lost cement as a result of drainage additionally decreases the strength of the hydraulic fill body. [9]

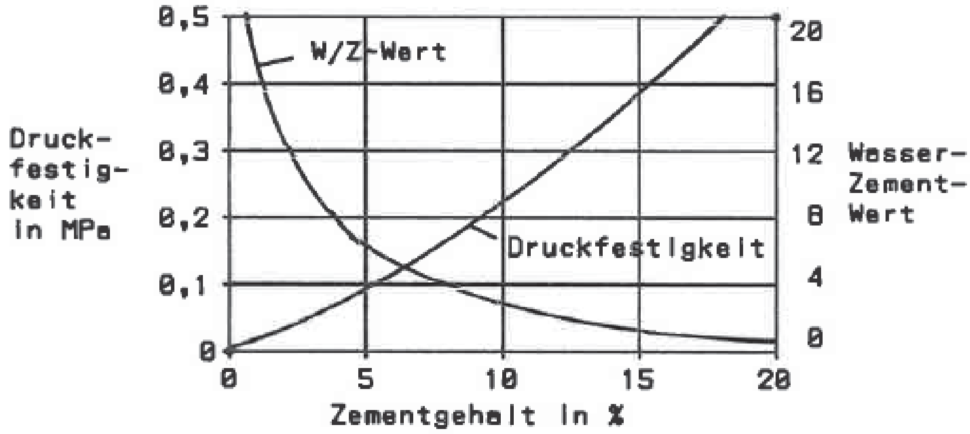


Figure 64: Optimum water-cement ratio as a function of cement content for cemented hydraulic fill[9] p.87

For cemented tailings backfill and cemented hydraulic fill below a certain (optimum) w/c ratio, the strength of the backfill body increases linearly with increasing w/c ratio up to a maximum value. Above this point there is not enough water available for the hydration reaction. After this optimum water-cement ratio the strength decreases because of water-filled pores, which reduce the strength of the backfill body. [9]

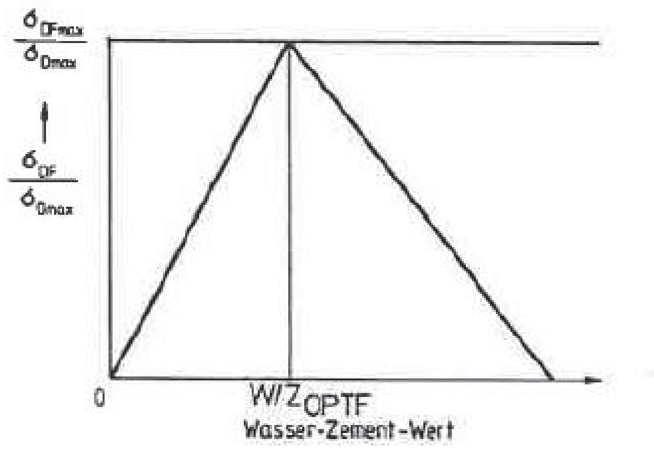


Figure 65: Compressive strength of completely compacted backfill as a function of w/c ratio [9] p.63

Another important point concerning the water content is the necessary water required for consolidation. The highest possible strength can be achieved at low w:c ratio and optimum consolidation. [9]

7.2.3 Density

The density of fresh cemented backfill product mainly depends on the water content and the densification energy applied during placement. An almost linear relation between density of the backfill product and water content can be found (Figure 66). [9]

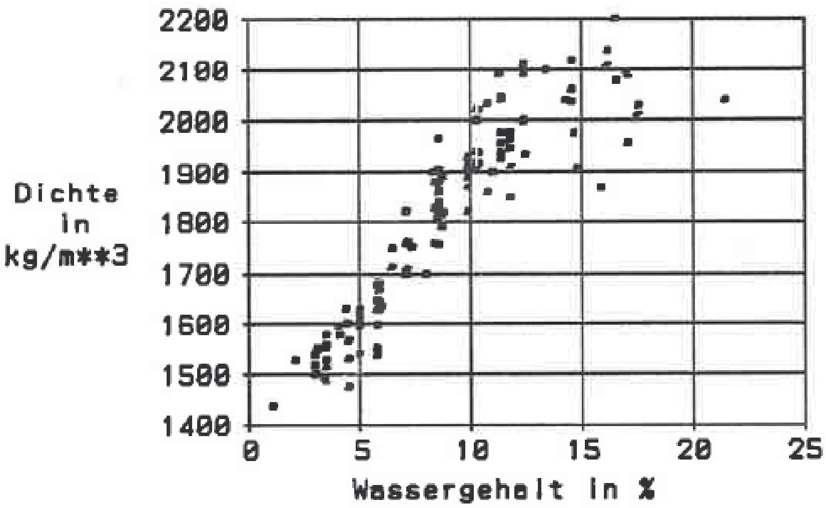


Figure 66: Relation between density of backfill product and water content [9] p.58

The effect of density on strength of cemented fill can be seen in Figure 67. (Patchet, 1983)

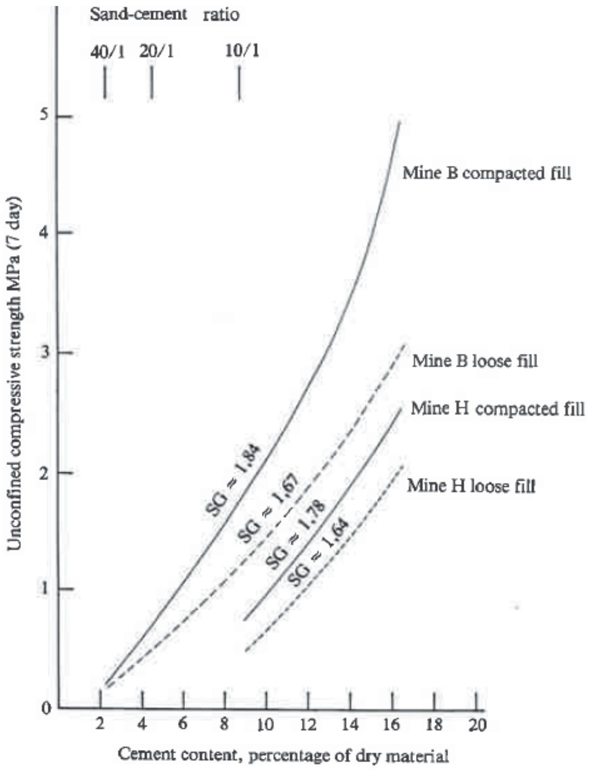


Figure 67: Effect of density on strength of cemented backfill (Patchet 1983 p.250)

7.2.4 Shear strength

Concerning unconfined environments and free-standing backfill walls, shear strength is a critical issue. Uncemented fills cannot form vertical faces and therefore only cemented fills can be exposed. The appropriate shear strength is developed by the addition of binders. By laboratory testing the strength of cemented fill using different binding agents at different curing times can be evaluated. Additionally to binding agents, the interlocking of particles, unsaturated moisture and compaction contribute to shear strength. [6]

Regarding confined mining environments, usually friction, density and porosity are critical issues.

7.2.5 Aging

The strength of the cemented tailings backfill body increases over time (Figure 68), but an additional increase can be found even after several months. [9]

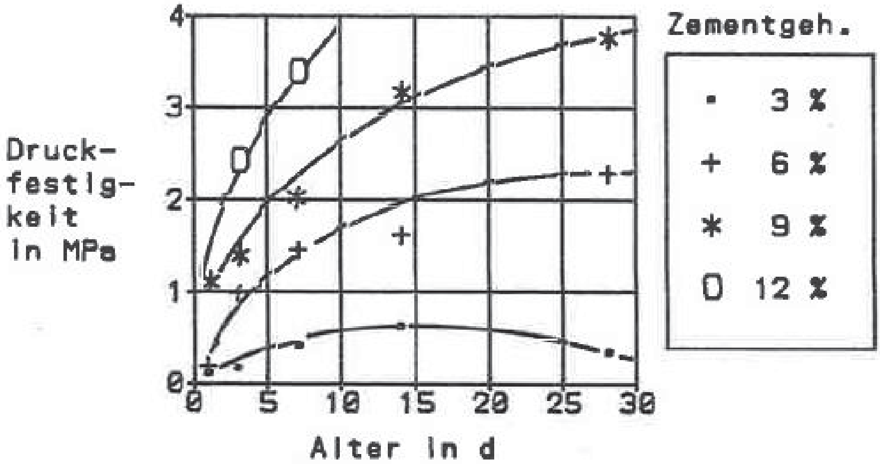


Figure 68: Compressive strength of tailings backfill as a function of its age [9] p. 60

The strength of cemented hydraulic fill increases with its age in a characteristic way. After three days 50% and after 7 days 75% of the 28-day strength can be reached. As for cemented tailings backfill, after 28 days a considerable increase in strength of the backfill body can be found (Figure 69).[9]

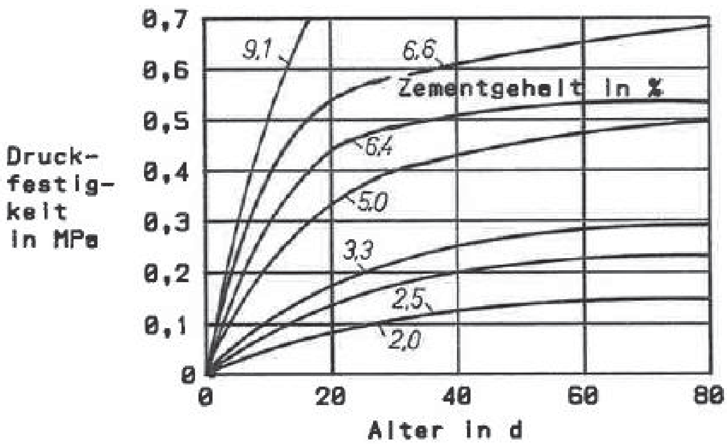


Figure 69: Strength as a function of age for cemented hydraulic fill[9] p.89

8 Measurement technology and laboratory and numerical investigations on backfill

8.1 Measurement technology and testing procedures

The sampling and testing of backfill and its properties is an issue for the technical and economical control of the mining method. As the dimensioning of an underground construction depends particularly on the properties of the used backfill materials, a need for controlled and repeatable testing and examination of these materials exists, which is valid for sampling and preparation of the samples as well. Generally the main raw material for backfill is waste material, so no constant properties and quality of the backfill material can be guaranteed, which represents a certain challenge for the testing of backfill. [5]

The implementation of a backfill system for the production of a backfill body with defined properties requires a detailed investigation of the system rock mass-backfill:

- Determination of the required backfill properties (from a mining and rock mechanics aspect)
- Evaluation of the available raw materials for backfill production (technical, economical and legal aspects)
- Definition of the mine planning geometry in terms of a backfill body with defined properties
- Choice of a backfill system with regard to the raw materials, the fill ratio, transport distance and other influence factors like mine drainage (using a pipeline for transport, factors like abrasiveness, rheological properties of the backfill and altitude difference of the pumping station have to be considered)
- Determination of laboratory testing in order to develop a suitable backfill material for the required conditions

- Implementation of a quality management system for control and documentation of the backfill properties

5

Currently laboratory testing procedures for cemented backfill are geared to existing regulations and procedures for investigation of concrete and rock, as no appropriate regulatory for backfill examination is available. Since the main influence factor for the type of backfill examination is its intended purpose, the testing of backfill cannot completely be carried out with the existing regulatory for concrete and rock. If certain testing methods and parameters from concrete and rock testing were chosen for backfill investigations, the results of these investigations would have a limited significance as the desired properties depend on the backfill purpose.[5]

To optimize the use of backfill and its components, a quality management system has to be implemented. This entails amongst others the investigation of the backfill material, the backfill product and the backfill body. According to Helms (1988) three types of testing procedures exist for this purpose:

- Qualification test (Eignungsprüfung)
- Quality testing (Güteprüfung)
- Retesting (Nachprüfung)

Qualification tests represent an essential component of an efficient backfill system. During these tests, the optimal composition of the backfill product is determined and suitable production-, transport-, and placement facilities are identified.

By quality testing of the mixed backfill components, the quality of the backfill product can be surveyed continuously and adapted if the desired properties are not achieved.

With retesting the designated backfill properties are controlled and influencing parameters from the mine environment on the backfill properties can be discovered.[9]

8.1.1 Testing procedure

The main influencing parameters on the results of backfill tests are either related to the sample itself or the experiment. Sample-related influencing parameters are:

- Sample geometry
- Sample preparation
- Duration and nature of sample storage

The influencing parameters related to the experiment are the following:

- Type of load
- Type of load ratio (increase in load or deformation/time unit)
- Treatment of pore water
- Testing machine stiffness

[5]

The difficulty in sampling of backfill depends if it concerns qualification tests, quality testing or retesting. For qualification and quality testing, the backfill product is easily available and sampling should not represent any difficulty. Concerning retesting of placed backfill, the importance of obtaining undisturbed backfill samples out of the backfill body arises. The best possibility to control the designated backfill properties would be the measurement of backfill properties in situ, with a plate-load test or sound transmission. [9] As in situ tests are very expensive and complicated to conduct, most backfill property tests are conducted in the laboratory on backfill samples. Additionally it is very difficult to relate plate test results to uniaxial compressive strength tests.

In the following, the influencing parameters on the test results are discussed, whereas sample preparation and testing equipment stiffness are not considered, as they don't have a significant influence on the test results on backfill according to Hohl (2009).

Geometry of specimen

The size of the sample is adapted to the type of backfill, especially its grain size composition and the designated testing procedure. The sample size is controlled by the maximum grain. [9] Concerning the sample shape, generally cubes, cylinders and prisms are used for testing backfill. Cubes are frequently used as specimen geometry in backfill technology, as this arises from cement technology. In cement technology predominantly cubes with 100mm edge length are applied. When regarding test results one has to be aware of the influence of the sample geometry on the results. Very often in the industry backfill is provided by subcontractors and the strength of the backfill is determined in a contract. In this contract the type of specimen and the geometry have to be specified as well, as these factors influence the strength of the specimen. In case of samples of different slenderness ratios the effects of this parameter on the test results has to be taken into account. In Figure 70 the relation between the uniaxial compressive strength and different sample geometries is illustrated. A very simplified conclusion of this illustration is that the strength of cylindrical specimen is indirectly proportional to its slenderness ratio. [5]

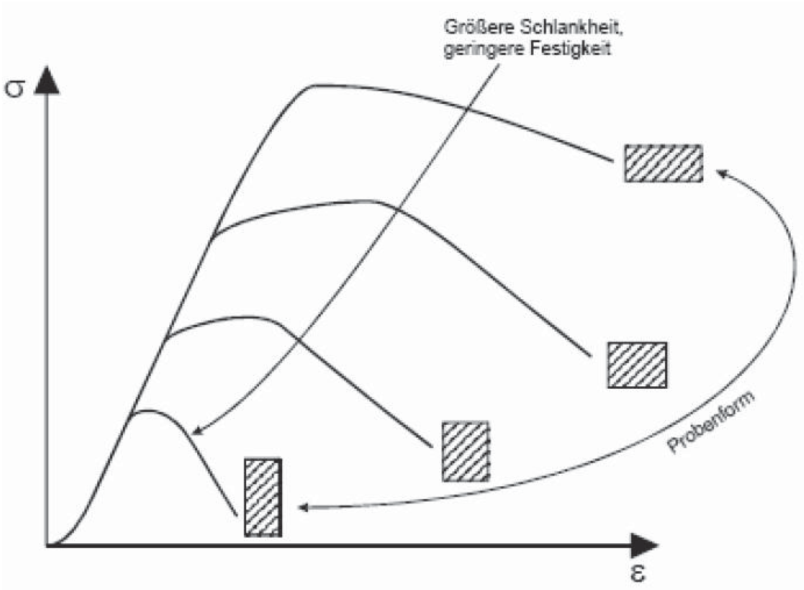


Figure 70: Relation of uniaxial compressive strength and sample geometry[5] p.76

For a detailed and reliable strength result of the backfill, cylindrical specimens with a ratio of height/diameter of 2 are recommended, to reduce the end effects on test results. The diameter of the specimen depends on the size of the largest particle and should be at least three times the largest grain diameter.

Cubic specimens are valid as index tests as well as for a routine quality control, provided a repeatable specimen fabrication can be secured. When comparing laboratory tests on cylindrical and cubical specimens, a conversion factor between cylinder strength and cube strength can be calculated. [5]

Sample fabrication and storage

Backfill samples can be produced by molding or core drilling, but when applying core drilling a possible erosion of the specimen might occur due to water used for flushing. [5]

Regarding sample storage of cemented samples, the amount of binding agents and the duration of storage predominantly influences the strength development of the samples.[5] Further on it has to be mentioned that the samples for testing are stored under controlled temperature and humidity conditions. The problem with these conditions is that the backfill product in situ is not subjected to the same conditions. Therefore the test results of the samples might deviate from the real backfill properties.

Load type

For the investigations on mechanical properties of backfill samples, three load types exist:

- Controlled by deformation (deformation rate independent from stresses is applied)
- Controlled by stress (constant stress augmentation until failure)
- Controlled by strain (constant strain augmentation in axial and lateral direction)

Treatment of pore water

Pore water treatment is only necessary, when simulating a triaxial stress state, like by the triaxial compression test or by the oedometric test. As a result of pore water pressure development during the testing procedure, falsification of the results can occur. When uniaxial compressive strength tests are conducted and drainage of the sample is not possible due to a low permeability, pore water pressure development has to be considered as well. In this case the load ratio must be adapted.[5]

8.2 Measurement of different backfill properties

To investigate backfill properties, testing procedures from various special fields like rock and soil mechanics or concrete technology can be adopted.

In Chapter 6 the most important backfill properties concerning backfill performance are discussed. Since these properties mainly influence the behavior of the backfill body, they have to be determined and controlled by defined laboratory testing procedures. The following properties should be observed:

Property	**Laboratory testing procedure**
Chemistry and mineralogy	Common methods: microscopy, x-ray diffractometry, electron microscopy
Particle size gradation	Common methods: sieving, classification or sedimentation analysis
Fines content	Common methods: sieving, classification or sedimentation analysis
Water content	Drying and weighing
Density	Common methods: gravimetry, volumetry, radiometry
Densification	Oedometric test, compaction test
Permeability and fill drainage	Constant head permeability test and in terms of percolation rate by a standard percolation tube test
Consistency and viscosity (flow properties)	Slump test, Vane shear viscosimeter, pipe loop testing, L-type pipeline resistance test
Uniaxial compressive strength	By uniaxial compression test
Shear parameters	In terms of c' and φ' by a standard triaxial compressive test

Table 8: Observed properties and testing procedures

Table 8 shows numerous backfill properties and testing procedures. It has to be mentioned that in practice, only the properties according to the specific demands on the backfill are tested.

Mineralogy and petrography are determined by common methods like microscopy, or if chemical reactions should be investigated, by x-ray diffractometry or electron microscopy. [9]

Using gravimetry or volumetry or radiometry, the density of a backfill material can be determined.[9]

The water content of a material is determined by drying until constant weight.[9]

The particle size distribution is normally investigated by sieving, classification or sedimentation analysis. [9]

8.2.1 Fill drainage properties

Among the most important properties of backfill material are its permeability because of drainage and its strength and deformation behavior.

The testing procedures on permeability as many other backfill tests arise from soil and rock mechanics. Two types of permeability tests for sands and finer soils exist. For coarser grained soil the constant head permeability test is used, and even fine-grained fills can be tested. In practice the best dimension to assess drainage properties of hydraulic fill is the time after placement, when the fill body is walk able.

Constant head permeability test

During this test a sample of the fill product is subjected to a constant water head difference. The result of this test is the water flow rate, which is measured. A description of the experimental set-up can be seen in Figure 71.

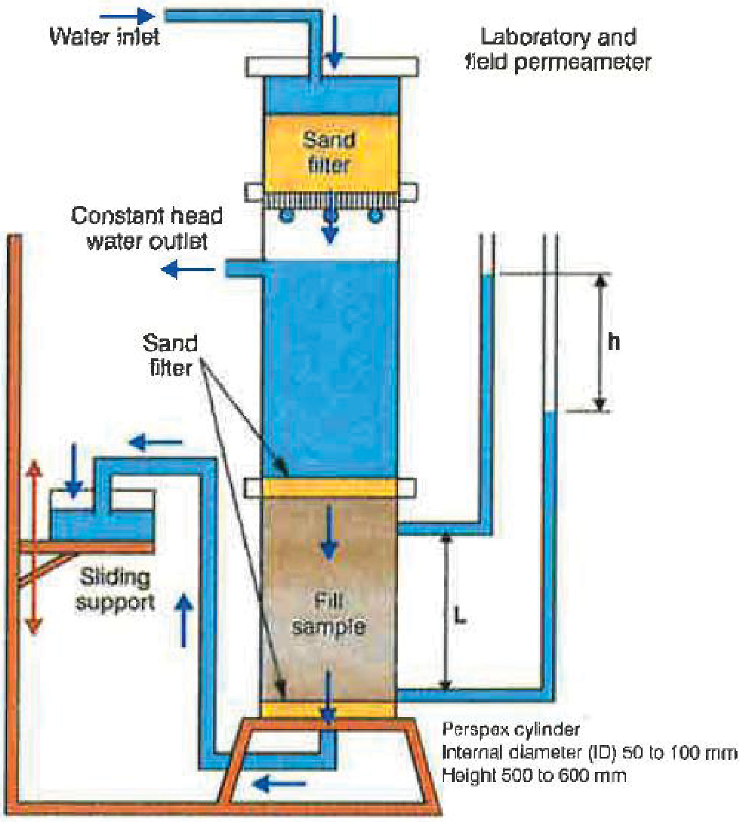


Figure 71: Experimental set-up for constant head permeability test[6] p.37

Percolation tube test

Fill drainage properties can be specified in terms of percolation rate determined in a standard percolation tube test. During a percolation test, one or several holes are dug in the backfill product to a specified depth. Then a perforated pipe is introduced into the hole and the holes are filled with water. Afterwards the holes are filled until a specific level with the backfill product and the time is measured until the water level drops as the water percolates into the surrounding backfill product. [34]

The percolation rate is equivalent to the permeability of the fill (measured under a gravity gradient close to unity) Using the following relation Mitchell (1983) estimates the in situ percolation rate (P) from a percolation tube test and in situ measured void ratios:

$$P_{corr} = P_{measured}\left(\frac{e_{insitu}}{e_{measured}}\right)^2 \quad 1$$

P_{corr}… estimated percolation rate [mm/h]
$P_{measured}$… from percolation tube test measured percolation rate [mm/h]
e_{insitu}… in situ measured void ratio [%] (undisturbed sample from in situ)
$e_{measured}$… void ratio during percolation tube test

For a free-draining fill, a percolation rate of at least 25mm/h is required. At this percolation rate no ponded surface water will be generated.[1]

At the base of a stope the permeability P_d must be higher than in the backfill body, as the water follow is choked through the filled draw point and bulkhead. To guarantee free drainage the following relation must be fulfilled:

$$P_d > P_{corr}\left(\frac{A_s}{A_d}\right)$$ [1]

A_s…cross section of stope [m²]
A_d…cross section of draw point [m²]
1

8.2.2 Densification

Oedometric test

The oedometric test comes from soil mechanics and is based on the application of an axial load with completely hindered lateral deformation (Figure 72). The sample is placed in a steel ring and then loaded in axial direction. The constrained modulus and the working line can be derived from the oedometric test and estimations for densification and deformation can be done.

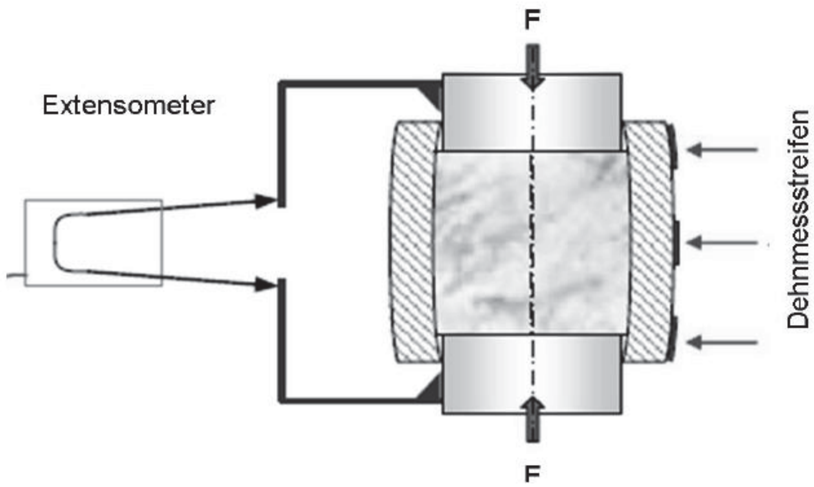


Figure 72: Oedometric test[5] p.78

Compaction test

Another laboratory test for the observation of compaction characteristics of **hydraulic backfill** or **cemented hydraulic backfill** is the compaction test or the Proctor compaction test. During such a test, the backfill material is mixed with the water and compacted into a steel mold with a falling weight hammer, with standardized compaction energy. For the mixture different water contents are used to reach the dry density of the product under different water contents. When plotting the dry density against the water content, an optimum moisture content at the maximum dry density can be observed from the curve. This water content represents the optimum water content for the mixture in terms of compaction properties.

8.2.3 Consistency and viscosity

Consistency and viscosity are significant parameters describing the flow properties of a backfill product. To analyze the flow properties by laboratory-scale tests, slump tests, cone tests, tests on yield shear stress and pipe loop tests are conducted.

Slump test

The slump test originally comes from concrete production and is an empirical test for measuring the consistency and workability of fresh concrete. For backfill applications predominantly tests on paste fill are conducted. The testing equipment comprises a metal mold in the shape of the frustum of a cone or of a cylinder, which is open at both ends and provided with a handle at the side. Its top internal diameter measures 102mm and its bottom internal diameter measures 203mm with a height of 305mm. The cone/cylinder is placed on a hard non-absorbent surface and is filled with the fresh backfill product in three stages. After each stage it is tamped with a rod of standard dimensions. At the end of the third stage, the backfill material is struck off at the top of the mold. Then the mold is removed from the backfill product. After removing it, the backfill product results in a small heap, which is subsided in relation to the cone/cylinder (Figure 73 (2)). This subsidence is called the "slump" and its height is measured.[35]

According to the profile of slump, it is termed as true slump, shear slump or collapse slump (Figure 73 (1)). In a true slump the concrete simply subsides, keeping its shape. In a shear slump the top portion of the concrete shears off and slips sideways. In a collapse slump the concrete collapses completely. Only from a true slump, measurements for further conclusions can be taken. When the backfill product results in a shear or collapse slump, the test has to be repeated. In general a collapse slump shows a too wet product in concrete technology. Very dry mixes; having slump 0–25mm are used in road making, slumps from 10-40mm are used for foundations with light reinforcement, and slumps with a height of 50-90mm for normal reinforced concrete placed with vibration. High workability concrete has a slump > 100mm.[35]

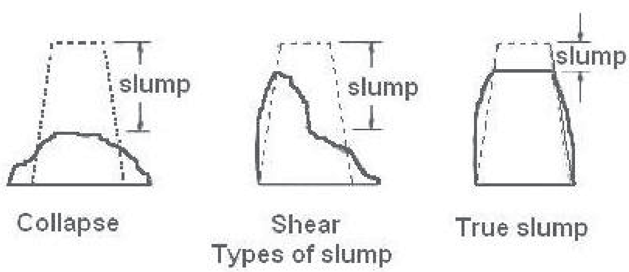


Figure 73: Slump types (1)[36] and cylindrical slump test (2)[6] p.60

No conclusions on the flow characteristics from these slumps of backfill can be taken, as they refer to concrete application.

However, Pashias et al. (1996) have developed a solution to calculate the yield shear stress from a cylindrical slump test. For this cylindrical slump test generally a steel mold with a diameter of 200mm is used. [6]

According to Clayton et al. (2003) the following equation relates the yield shear stress and the slump height:

$$s' = 1 - 2\tau_y'(1 - \ln(2\tau_y'))$$ [37]

s'=*s/H*…dimensionless slump height

$\tau_y' = \tau_y/(\rho g H)$…dimensionless yield shear stress

ρ….density [kg/m³]

g….gravity [m/s²]

H….cylinder height [m]

s…slump height [m]

Using this equation, the relation between the yield stress and the slump can be illustrated in Figure 74.

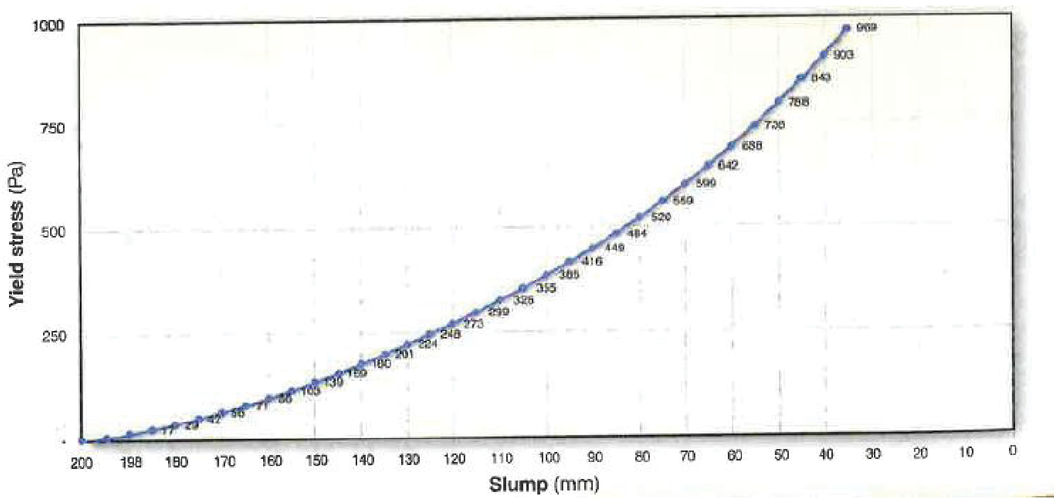


Figure 74: Relation between yield shear stress and slump (for 200mm cylinder, density=2 t/m³)

From the yield shear stress conclusions on the flow properties of paste fill can be taken, as it indicates the flow ability of a paste. In Table 9 an empirical relation between the pulp density and the yield shear stress deriving from a testing program is demonstrated.[6] It shows pulp density limits for paste fill with different cement contents resulting from yield stress functions.

Portland Cement content [%]	Function	Lower limit pulp density φ [$\%_{cw}$]	Upper limit pulp density φ [$\%_{cw}$]
0	τ_y=1,36x$10^6$$\varphi^{25,28}$	66,0	74,0
2	τ_y=4,06x$10^5$$\varphi^{20,48}$	63,0	74,0
4	τ_y=5,47x$10^5$$\varphi^{20,72}$	62,0	73,0
6	τ_y=6,75x$10^5$$\varphi^{21,50}$	63,0	73,0

Table 9: Example of yield shear stress functions for paste fill[6]

Other methods for flow characteristics determination

Various other methods to determine the flow characteristics of a paste fill exist. With the vane shear viscosimeter the yield stress can be measured, and so information about the flow characteristics of paste fill can be gained as well.[6]

Viscosity measurements for example can be conducted by standard capillary rheometers as well. [6]

By pipe loop testing, the pressure distribution in a fill system can be measured. It is carried out by pumping slurries around different diameter pipelines at different slurry densities and flow rates. The pressure drop over known lengths of pipes is recorded and the data can be illustrated as friction losses versus density and flow rate. From this test information about reticulation design of backfill systems can be gained.[6]

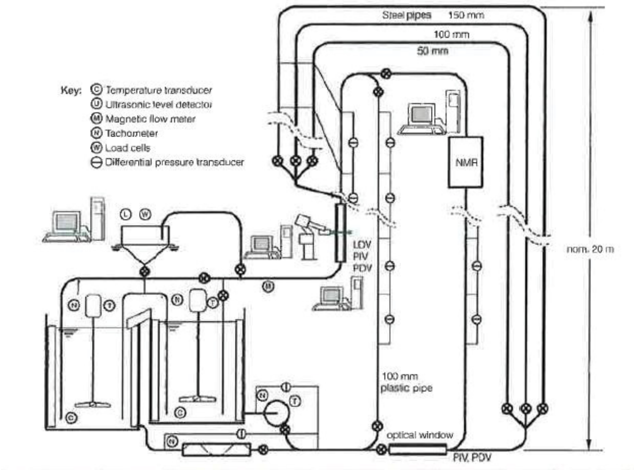


Figure 75: Example of a test loop equipment[6] p.62

As the loop pipeline fill test is expensive to realize and takes a lot of time, the L-type pipeline resistance test was developed to test the resistance in paste fill pipeline transportation. Compared to the loop pipe test, it is less expensive and complex. By the L-type pipeline test, the resistance of paste fill during transportation can be measured. The experimental equipment consists of a conical funnel and a vertical and horizontal pipe. The slurry for the test is prepared by a concrete mixer. In a first step, the fill is mixed in batches. Then the batches are loaded into the funnel. When the funnel contains the necessary quantity of slurry, the plug at the bottom of the funnel is opened, which allows the fill to flow through the pipeline. The velocity of the fill is measured and is used to calculate the resistance. [38]

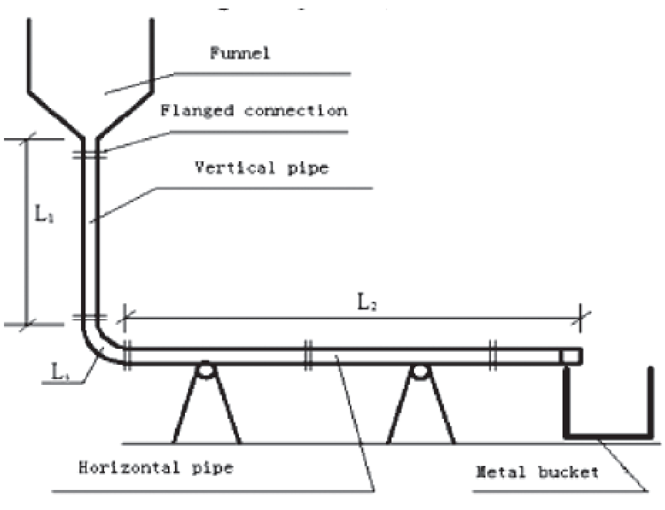


Figure 76: Testing device for L-type resistance test[38] p.176

When the fill overcomes the initial shear stress, the slurry begins to flow. When the velocity increases, the pipeline resistance increases as well. Using the initial shear stress and the viscosity of the slurry, the resistance loss of a unit length can be calculated as follows:

$$i = \frac{16\tau_0}{3D} + \frac{32\eta v}{D^2} \quad 38$$

i…resistance loss of a unit length of pipeline [Pa/m]
τ_0…initial shear strength [Pa]
η…viscosity [Pas]
v…flow rate of the slurry [m/s]
D…Diameter of pipeline [m]

8.2.4 Fill strength

Fill strength is generally determined by standard soil mechanics procedures like triaxial tests. The strength can be represented by the parameters *c'* and *φ'* or by a low stress bond strength (C_b) which can be determined by unconfined compression test results. At low cement content (<5% by dry weight) and high confining stress, the stress-strain behavior is ductile. At higher cement content and low confining pressure, brittle behavior dominates. Mitchell (1983) recommends to conduct preliminary backfill design on the basis of uniaxial tests but for further analysis triaxial compression tests are required to obtain *c'* and *φ'.* [1]

Uniaxial compression test

The uniaxial compression test is by far the most used laboratory test for investigation of mechanical properties of backfill, as it is not very expensive to conduct. The basic principle is to load the specimen in axial direction without confining stresses until rupture (**Fehler! Verweisquelle konnte nicht gefunden werden.**).

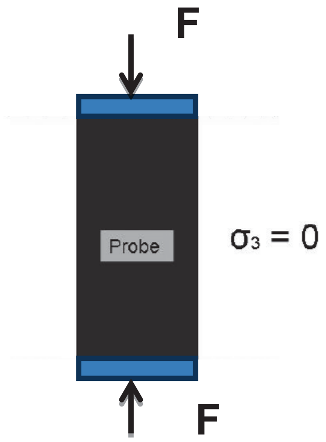


Figure 77: Uniaxial compression test[5] p.77

For the investigation of deformation properties, like Young's modulus and the modulus of deformation, cycles of charges and discharges can be conducted during the test. From the axial and lateral deformation of the specimen, Poisson's ratio can be calculated. The uniaxial compression test is the most applied test for mechanical properties, but gives limited information for real backfill application, as in general a backfill body is subject to a triaxial stress state. Frequently load-deformation properties have more influence on backfill performance.[5]

Although in reality segregation occurs, laboratory control test results are comparable to the average strength. Mitchell (1983) proposes a relation for the uniaxial compressive strength which is increasing primarily with the cement content:

$$\sigma_c = A + BC^2 \log t \ \ kPa \quad ^1$$

A, *B* … constants
C… cement content [wt%]
t… curing time in days
1

Triaxial compression test

From the triaxial compression test, shear parameters of the backfill body can be obtained. The shear strength of a fill is a function of void ratio, confining stress, loading rate, degree of cementation or age of fill, degree of saturation, size, shape and grading of the particles.[6]

During a triaxial compression test the specimen is loaded gradually in axial direction under lateral confining pressure (Figure 78). The whole experimental set-up is therefore installed in a pressure vessel, which is filled with a fluid. As a result of pressure change of the fluid, different confining pressures can be applied easily.

From experiments with different stress states, cohesion and angle of friction for the Mohr-Coulomb failure criterion can be determined (Figure 79). Costs for this test are quite high due to an elaborate specimen and test preparation.[5]

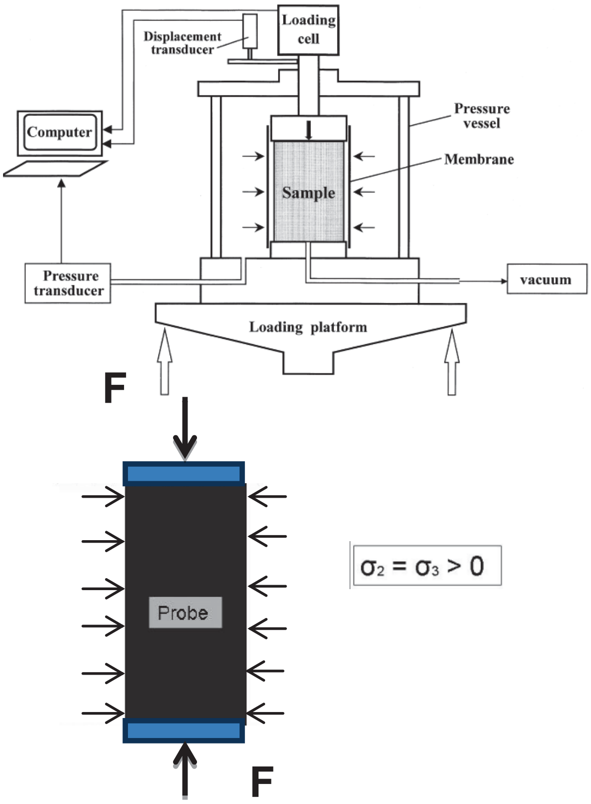
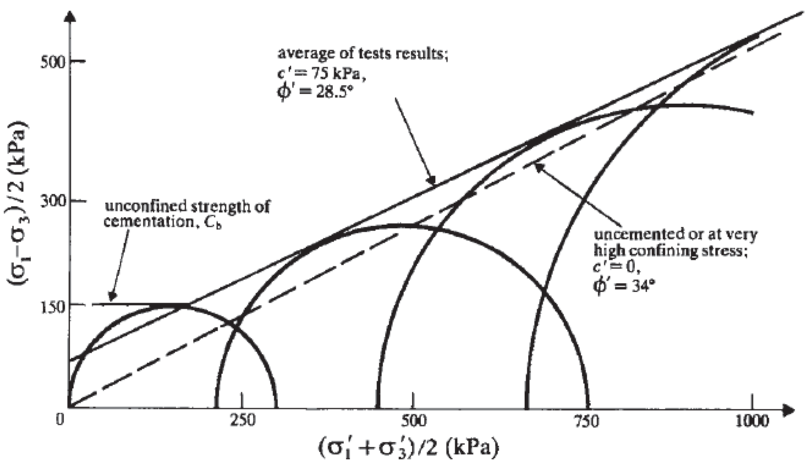


Figure 78: Triaxial compression test (1)[40] p.133 and general principle (2)



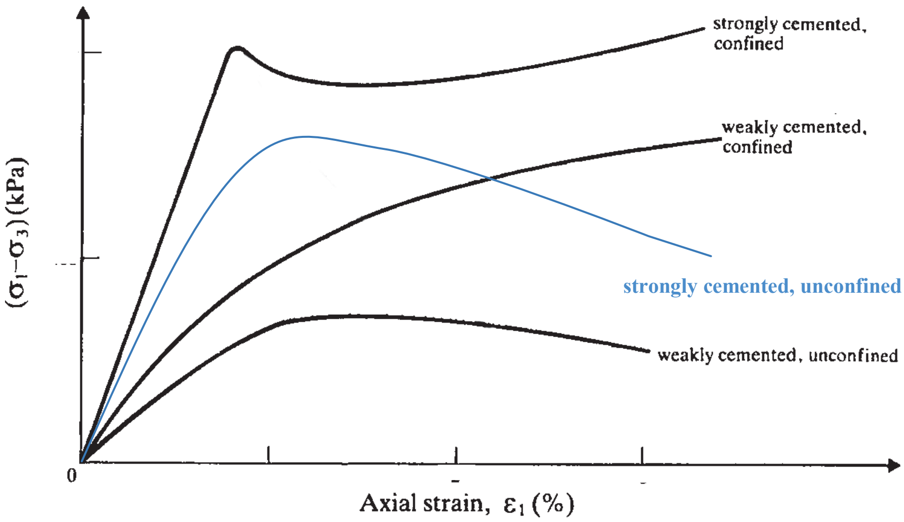


Figure 79: Typical triaxial test results on cemented hydraulic backfill [1] p.414 (modified[14])

In Figure 79 qualitative curves for the triaxial load behavior of different cemented hydraulic fill materials are presented. This diagram originates from Brady and Brown (2004) who presented a quantitative assessment of different hydraulic fill masses. A modification concerning the curve of strongly cemented, unconfined hydraulic fill was conducted as Brady and Brown (2004) presented a behavior for this fill mass which equals the behavior of confined fill, until the peak of the curve. According to Wagner (2014) a difference in the strength behavior of strongly cemented confined and unconfined fill must occur, as the confinement strongly influences the strength of the fill mass. Therefore this curve was modified and the modified curve describes lower peak strength and a lower residual strength of the backfill.[14] Subsequently the results should be considered in qualitative sense.

Regarding a dense hydraulic fill when increasing the axial compression, the shear resistance decreases due to an overcome in the interlocking between the particles. By a furthermore increase of the axial compression, a steady-state shearing with constant void ratio is obtained. At this point, the dense fill has a constant volume, which is larger than the initial volume. The void ratio at this stage is called the critical void ratio. The initial void ratio has a significant effect on the stress-strain behavior during a triaxial compression state, which shows the importance of a dense fill. The denser the fill, the more important the interlocking of the particles will be and therefore the greater the friction angle and the shear resistance. [6]

Conclusion

The relevance of laboratory tests for in situ behavior is restricted as they are specification tests and not performance tests. By all the testing methods the parameters, having an influence on backfill performance are identified but their effect on the performance of backfill is not investigated. Additionally the reproduction of in situ behavior by laboratory testing is limited. For example triaxial compression tests are conducted under artificial conditions using hydraulic pressure to simulate the enlacement stresses. This causes a non-uniform rupture, which does not represent in situ conditions. Further on testing and storage conditions are normed and do not reflect reality. However, laboratory tests are useful to investigate the properties of the backfill according to its individual demands and give a good idea of which behavior can be expected in situ. It is very important to analyze the results from laboratory testing very carefully and to interpret them based on in situ conditions.

8.3 Laboratory and numerical investigations on backfill

8.3.1 Investigations on uniaxial compressive strength development by addition of aggregates to paste fill

Paste aggregate fill represents a combination of paste fill with aggregates (graded waste material). The aggregates are added to the paste in the mixer and once placed, this variation of paste fill does not separate into aggregate and paste fractions. Wilson et Calverd (2011) conducted investigations on paste fill with different additions of aggregates to investigate the influence of aggregates on the uniaxial compressive strength development of paste fill. Three mixtures of paste fill with aggregates were tested:

1. Metal ore tailings (72%<20 microns) with waste rock (k_{max}=16mm)
2. Precious metal ore tailings (62%<20 microns) with crushed development waste rock (k_{max}=16mm)

3. Precious metal ore tailings (93%<20 microns) with waste rock (k_{max}=12mm)

Metal ore tailings (72%<20 microns) with waste rock (k_{max}=16mm)

The addition of 50% by volume of aggregates resulted in a 320% increase in strength at comparable binder dosing.

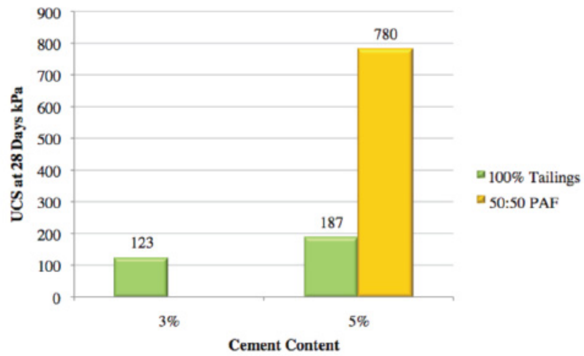


Figure 80: UCS results of Metal ore tailings with waste rock[12]

Precious metal ore tailings (62%<20 microns) with crushed development waste rock (kmax=16mm)

Due to the addition of 50% by weight of crushed development rock showed a clear improvement in the strength development of the mixture.

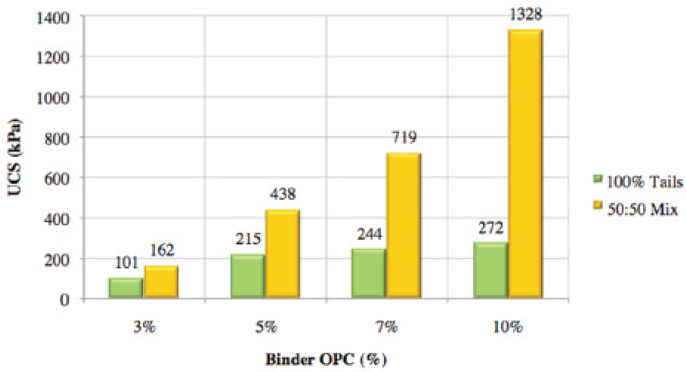


Figure 81: UCS results of precious metal ore tailings with crushed development waste rock[12]

Precious metal ore tailings (93%<20 microns) with waste rock (kmax=12mm)

Again in this case, 2 different amounts of aggregate addition (50%, 70%) resulted in a significant increase of the UCS of the mixture.

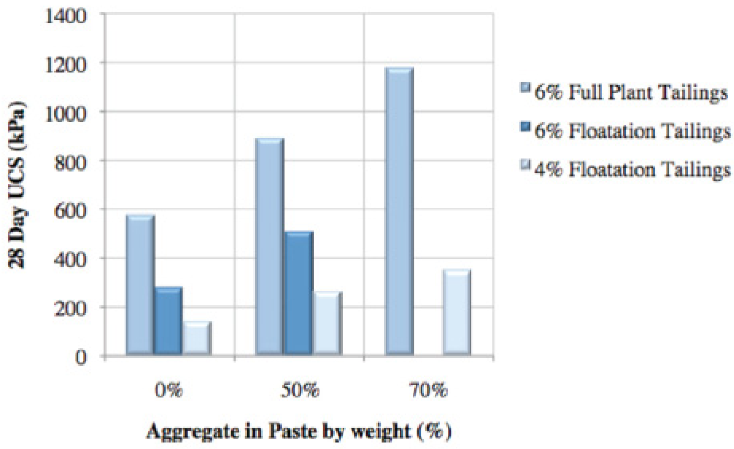


Figure 82: Influence of aggregate addition on 28-day uniaxial compressive strength[12]

Conclusion

The overall conclusion of the work carried out by Wilson et Calverd (2011) showed that by the addition of aggregates to the paste fill significant improvements in the fill strength can occur. The investigations on the first two products show, that without aggregate addition, the paste fill strength would not be sufficient for mining requirements, whereas in the third case, binding agents can be reduced by aggregate addition. An important factor pointed out by the authors is the importance of the ratio of tailings to aggregates, which is fundamental for the strength development. When adding aggregates to a paste fill product, a critical issue is also presented by the stability of the material while flowing. According to Newman et al. (2003) using a continuous aggregate grading curve and sufficient fines to fill the voids between the particles, results in sufficient stability in the fill product while flowing. The optimum fines content for good flow properties can be found at >15% particles smaller than 20 microns.

8.3.2 Investigation of composite cemented sand fill/rock fill at Mt Isa

Brady and Brown (2005) report about investigations by Gonano (1975,1977) on the in situ determination of a well-cemented zone in a composite cemented sand fill/rock fill mass at the Mount Isa Mine in Australia. These investigations showed that rock fill inclusion in the hydraulic fill medium caused a significant increase in the cohesion of the fill mass. [1]

Fill type	c' [MPa]	φ' [deg]	E [MPa]
8% cemented sand fill (CSF)	0,22	35	285
Composite of 8% CSF and rock fill	0,60	35,4	280

Table 10: In-situ properties of composite backfill [1] p.415

Further investigations were made on the effect of cement content on the 28-day uniaxial strength of cemented rock fill. Swan (1985) found the following relation between the compressive strength and the volumetric cement content C_v:

$$\sigma_c = \alpha_r C_{Cv}^{2,36}$$ [1]

α_r… Characteristic of rock type

C_{Cv}…Volumetric cement content [%]

Typical results for σ_c could be found in the range between 1-11 MPa and deformation modulus in the range 300-1000 MPa. In general in situ strength is considerably lower than laboratory strength, due to effects of segregation, porosity and cement distribution. Barrett and Cowling (1980) proposed a relation of in situ strength to laboratory strength of ½.[1]

8.3.3 Investigation of Helms on different backfill types

Helms (1988) investigated the properties and the development of properties of different cemented backfill types as a function of water content, binding agent content and the resulting w:c ratio. He investigated the following types of backfill:

- Cemented tailings fill as drop fill (CT – drop fill)
- Cemented tailings fill as slinger fill (CT – slinger fill)
- Cemented Hydraulic fill – fine grained (CH)
- Cemented paste fill using pumping (CP)

Cemented tailings fill as drop fill

The drop fill product consisted of sandstone tailings, blast furnace cement and water. The maximum particle size was limited to 32mm and the coefficient of uniformity valued 2,8. The most important influencing parameters are the cement content, the water content and the water-cement ratio.

During compressive strength tests, generally the matrix was destroyed. In specimen, which showed higher strengths, also larger tailings particles were affected. [9]

Cemented tailings fill as slinger fill

The tailings for slinger fill mainly consisted of sediments with a high percentage of limestone and argillaceous schist, whereas the maximum particle size was limited to 63mm and the coefficient of uniformity valued 16. As binding agent Portland cement and blast furnace cement were used. For slinger fill the same parameters as for drop fill were investigated: cement content, water content and water-cement ratio.

Investigations on the uniaxial compressive strength using cement contents between 1,5-12% showed a variation between 0,5-20 MPa. The compressive strength of backfill products using Portland cement was considerably higher than the strength when using blast furnace cement (Figure 83).

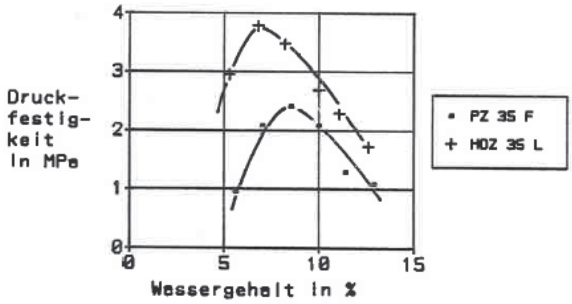


Figure 83: Strength of backfill with Portland cement and blast furnace cement [9] p.76

Concerning slinger fill the water content and accordingly the water-cement ratio have an important influence on the strength of the fill body. For each cement content in the mixture, an optimum water content or water-cement ratio exists, where a maximum strength can be achieved.

With increasing cement content the strength maxima can be found at lower w/c ratio, but to higher absolute water content.

Cemented Hydraulic fill –fine grained

For the investigations of hydraulic fill Helms (1988) used sand as raw material with a maximum particle size of 2,5mm. As binding agent blast furnace cement was used.

For this fill type the influence of water content and w:c ratio on uniaxial compressive strength and the splitting tensile strength were investigated. The uniaxial compressive strength was found between 0,3 and 8 MPa.

Cemented paste fill using pumping

The raw material of the paste fill consisted of flotation tailings and coarse grained tailings from a heavy liquid separation with a maximum size of 20mm. The mixture has a water content of 11-26% and as binding agent2-9% blast furnace cement was used. The influence of water content, cement content and water/cement ratio on the performance of the paste fill was investigated.

At defined cement content in the backfill material, a high strength can be achieved by keeping the water content and therefore the w/c ratio low. For delivery reasons a certain w/c however must not be undercut.[9]

Comparison of Helms's results

Fill type	σ_{ucs}=f(c)	σ_{ucs}=f(w)	σ_{ucs}=f(age)	σ_{ucs}=f(w/c)
CT – drop fill	linear increase	σ_{ucs} max. at 8%	4-12fold increase in 28 days	-
CT – slinger fill	linear increase	optimum water content increases with cement content	strength increase even after 28 days	with increasing c, σ_{ucs} max at lower w/c
CH	-	σ_{ucs} max for all cement contents at 14%	-	influence of w/c increased with c
CP	linear increase	decrease	-	decrease

Table 11: Overview over Helm's results

For all fill types a linear increase in the uniaxial compressive strength with increasing cement content could be observed, except for cemented hydraulic fill, where no results are available. For cemented hydraulic fill and cemented tailings drop fill, a maximum strength could be observed at 14% or 8% water content. For slinger fill, an optimum water content could be observed as well, but it strongly depends on the cement content of the mixture. For paste fill the strength decreases with the water content. Drop fill shows a 4-12 fold increase in strength between the first and the 28th day, slinger fill reports to a continuing strength increase after 28 days. Regarding the w/c ratio, slinger fill shows that with increasing cement content, the maximum strength could be found at lower w/c. For hydraulic fill the influence of w/c increased with the cement content, and paste fill showed a decrease of strength with increasing w/c ratio.

8.3.4 Model for backfill rock mass interfaces

Manaras et al. (2011) developed a model to describe the contribution of the rock wall roughness to backfill behavior using an experimental program. Shear strength testing was conducted with 450 specimens. Five different surface roughness profiles were tested, defined based on the joint roughness coefficient scale. Three mixtures with different cement contents were tested (2,5; 5 and 7,5% cement content per dry mass) and specimens were tested after 14, 28 and 56 days. Five levels of normal loads (25-2000kN) were tested as well.[41]

The shear tests were conducted under constant normal load and some specimens were tested under multistage loading, to provide the intact normal strength and the residual shear strength.[41]

Results showed that the shear strength of the paste fill-rock interface strongly depends on the surface roughness, but the cement content was proved to be the most important influencing parameter. Further on the paste fill cure time is of importance.[41]

The equation of Barton (1973,1976) was used to predict the interface strength. The results of the predicted interface shear strengths were very low to the measured ones (Figure 4).

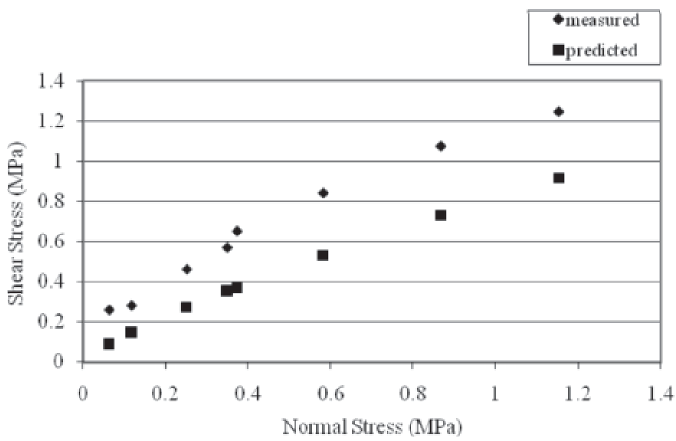


Figure 84: Relation between shear stress and normal stress predicted by Barton's equation[41] (modified diagram order in comparison to the paper, mistake in the diagram order from the paper)

Further on Barton's equation does not consider the additional strength which comes from the paste fill binding strength. Manaras et al. (2011) revised Barton's equation to involve the increased peak shear strength due to paste fill adhesion associated with its cohesional strength to predict the interface shear strength as follows:

$$\tau_p = C + \sigma_n \tan(\varphi_b + JRC * \log_{10}(\frac{JCS}{\sigma_n}))$$

JRC…Joint roughness coefficient (estimated by back analyzing or visual comparison, 0-20)

JCS…Joint compressive strength (unconfined compressive strength of the rock at the joint surface)

φ_b…friction angle

σ_n…applied normal stress

C…constant to involve increased peak shear strength

The prediction of the revised formula is strongly improved (Figure 85).

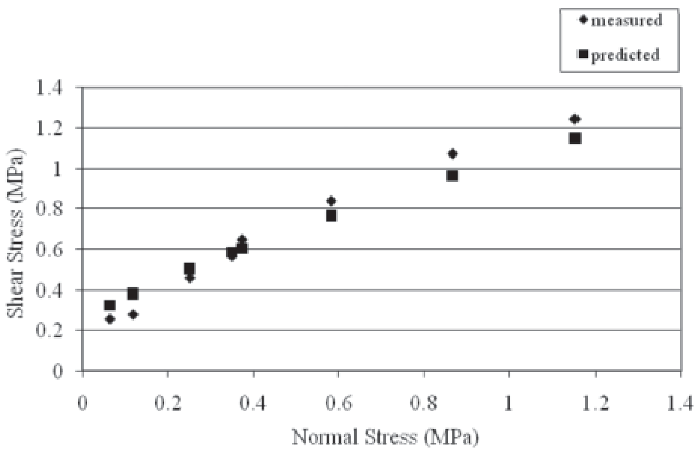


Figure 85: Prediction of shear stress - normal stress relation by the revised equation[41] (modified diagram order in comparison to the paper, mistake in the diagram order from the paper)

The revised formula is a combination of Barton and Mohr Coulomb’s equation, in which cohesion, friction angle and JRC parameters are combined in a logarithmic relationship.

Over all it was concluded that the fill-rock mass interface strength is developed as a function of frictional effects and cohesion of the interface. Friction and cohesion effects increase, when the cement content, wall roughness and mixture curing time are increased.[41]

8.3.5 Model for design of stable free fill walls

One of the most common applications of cemented backfill is for pillar recovery, where the fill mass is exposed during the operation. To describe the behavior of a free-standing fill body, models considering the fill mass as a free standing wall with a two-dimensional slope were developed, but according to Mitchell these models don't consider support forces mobilized at the surface of the backfill. Mitchell used a different model for static analysis of fill stability. In this model, where the backfill and local rock support are in contact, shear resistance mobilizes some of the self-weight of the block, which can slide on the inclined base plane (Figure 86).

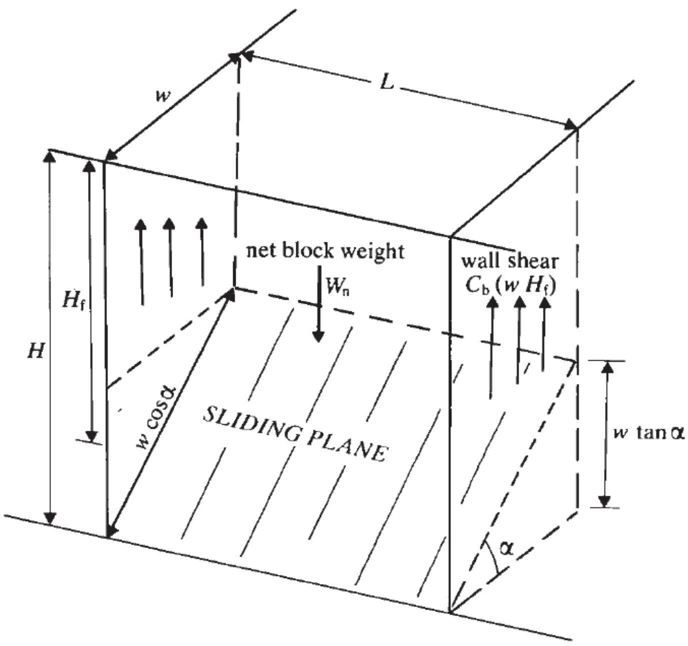


Figure 86: Confined block model for backfill stability analysis by Mitchell [1] p. 418

To study this failure mechanism, he implemented a factor of safety against plane failure. Additionally a surcharge can be applied on the backfill surface, to investigate also the influence of an additional load on the plane failure mechanism. The net weight W_N of the backfill body is so described as:

$$W_N = wH_f(\gamma L - 2c')$$ [1]

$$H_f = H - \frac{1}{2} w \tan\alpha$$ [1]

α…inclination of critical failure surface ($\alpha = 45° + \varphi'/2$)

The factor of safety against plane failure conforms to:

$$F = \frac{\tan\varphi'}{\tan\alpha} + 2c'L/[H_f(\gamma L - 2c')\sin 2\alpha]$$ [1]

The smallest value for the friction angle φ' for hydraulic fill is assumed as 30°. At this minimum value, the cohesive strength requirement on the fill mass is described as:

$$c' = \gamma L H_f \sin 2\alpha\, /2(L + H_f \sin 2\alpha)$$ [1]

9 Final remarks and conclusion

The goal of this work was describe the role of backfill as essential part of mining activities. In doing so in the first chapter basic information about backfill was discussed. At first different materials are described which compose the backfill product: Tailings, natural sands and aggregates, water and binding agents. Furthermore a classification of backfill into backfill types is approached. For this reason different literature was considered and several backfill classification approaches were discussed. It was noted that backfill classification can be done according to the material used, binding agent addition and transportation type. It was stated that the best way to classify backfill types is according to the transportation system.

These are described in the next part of the work. The most important backfill transportation types are transportation by gravity, pumping, pneumatic stowing and slinger stowing.

Backfill is placed in underground excavations for different purposes, but the primary purpose remains stabilization of underground openings and to guarantee regional stability. In doing so backfill acts in three different ways as support: by active backfill pressure by passive fill strength and it works against local disintegration of the rock mass as well.

In the next chapter duties and demands of backfill are discussed. Backfill has many different duties like ensuring long-term stability, creating a working platform, underground waste disposal and for mine ventilation purpose as well. Demands on backfill can be divided into five groups: health, safety and environmental demands, technical demands, organizational demands, geomechanical demands and quality demands on backfill. It has to be mentioned that every backfill application derives from different purposes and that every mine is an individual case, requiring different specifications of backfill.

In the third chapter the application of backfill in underground mining methods is discussed. The general mining method employing backfill placement is called cut and fill stoping. In cut-and-fill stoping, a tabular or irregular shaped deposits are mined in horizontal slices and replaced with backfill in underhand or overhand direction. Backfilling is normally performed after each slice is removed and different backfill materials are used. Further on backfill can be used in combination with pillars to provide additional support for the underground openings.

In Chapter 4 all possible parameters, having an influence on backfill performance are discussed. These parameters derive from the used backfill material and its chemical composition, particle size gradation, permeability, consistency and load-deformation behavior. These are all possible parameters having an influence, but again, the individual purpose of backfill application defines, if these parameters have an influence on the required backfill properties.

An important point in backfill technology is the addition of binding agents to the mixture. By cementation, properties of backfill like the initial strength and deformation behavior can be improved.

In the final part of the work measurement technology of the discussed backfill properties is presented and further on laboratory investigations and experimental results from the literature are discussed.

The conducted work gives an overview over backfill technology and several important fields which require further investigations could be discovered.

Critical points which were not discussed in detail during this work and which require further treatment are the discussion of backfill concerning extraction ratio versus backfill activities. The application of backfill in highly efficient mining activities like in longwall mining could be critical. In modern longwall mining daily face advance of 20m can be achieved, whereas backfill placement advances around 2m to 3m a day. In this case backfill activities severely restrict and limits the efficiency of this potential high performance mining method. As this subject was not treated in this work it would require special attention to develop a satisfying solution concerning backfill application in very efficient mining operations.

Another important point is the treatment of backfill placement and mining activities as concurrent activities. In general two ways for the mining and filling sequence exist: either the stope is completely mined and then backfilled or the stope is mined and filled simultaneously. Both solutions have advantages and disadvantages which need to be discussed to figure out the best solution for mining sequences. Further on the treatment of blocked backfill transportation systems is not regulated and no guideline could be found. As plugged pipelines represent a significant safety hazard, this subject should be further investigated.

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