CONTROLLED LOADING RESPONSE CEMENTED BACKFILL SUPPORT FOR DEEP TABULAR STOPES

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A thesis submitted to the Faculty of Engineering, University of the Witwatersrand, Johannesburg, in fulfilment of the requirements for the degree of Doctor of Philosophy.

Johannesburg, 1993

DECLARATION

I declare that this thesis is my own, unaided work. It is being submitted for the Degree of Doctor of Philosophy in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in any other University.

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ABSTRACT

Hydraulically placed cemented backfill is increasingly being used as a means of stope support in South African hard-rock mines. The addition of binder provides backfill with a primary compressive strength, the property for which the material has traditionally been utilised. Binder-induced cohesion, however, is not the only factor determining the quality of cemented backfill and the material's utility can be enhanced significantly by applying all of its attributes to the task of stope support. The main purpose of this thesis is, therefore, to examine those factors which influence the performance of cemented backfill, and to provide relevant and useful information for the design of improved cemented backfill mine support for tabular mining excavations.

In an extensive laboratory investigation, a number of relevant factors were tested for their influence on the loading behaviour of cemented backfill. Twenty two cemented backfills, grouped according to aggregate type, water content, binder content and binder type, were analysed to determine the influence of composition on their material properties. This study was augmented by several test series on the effects of curing conditions on cemented backfill quality. In a second major investigation, the effects of geometrical parameters, including sample volume and sample width to height ratios were analysed. A further study dealt with the effect of spatially separating sample backfill ribs under normal compression, as well as at high closure rates.

It is concluded, that by co-ordinating the composition design of cemented backfills with the spatial configuration of backfill support elements, it is possible to modify all phases of the cemented backfill loading response. This implies the control of the binder-induced compressive strength at low strains, the large-scale yielding behaviour of backfill support, as well as the stiffness of the backfill body at high stresses. In the light of stope support requirements, particularly under rockburst conditions, the capacity of backfill support to yield and absorb rapid stope closure and then to decelerate the hangingwall by the rapid strain-hardening of the, now, large width to height ratio backfill mass, has the potential of substantially increasing mine safety in tabular stopes.

ACKNOWLEDGEMENTS

The work presented in this thesis was carried out at the Chamber of Mines of South Africa, Research Organisation and formed part of their research programme.

I would like to acknowledge the contributions of the following persons and express my sincere gratitude to them:

Professor S. Budavari, for the supervision of this thesis, the generous sharing of his vast experience and his kind guidance and encouragement.

Professor H.R. Phillips, for his generous support and encouragement, and his supervision of this thesis in its early stages.

The Chamber of Mines of South Africa, Research Organisation, for providing the facilities and funds for the research. Many colleagues and friends in the organisation contributed directly or indirectly to this work. I would like to mention the following: Dr J.M. Stewart and Dr M. Nami, for their permission to utilise the data gathered in various research projects, and for their friendly encouragement.

Mr A.A. Bradley, for his selfless and invaluable help and advice in many of the investigations. Mr J.G. Sachse, for the preparation and testing of countless samples with unending patience and enthusiasm. Thanks also to all the staff of the laboratory, particularly Mr R.W. Quail. Mr S.G.E. Ashworth, for his help with the empirical strength equation and Ms J.M. Churcher for the analysis of the curing time effect on sample strength.

Mr C.P. Kramers, for sharing his extensive experience in backfill engineering, and teaching me much of what I know about backfill technology. Mr H.J. Ilgner, for his friendly help and his contribution on the hydraulic transport behaviour of backfill. Dr R.G. Gürtunca and his staff, particularly Dr P.N. Taggart and Mr P.H. Swart for their advice and help with unconventional backfill testing.

The outstanding researcher and friend, who set me on the path of this work, and to whom I am very grateful indeed: Dr Ian H. Clark.

To my dear wife Ulla, who has prepared many of the drawings in this thesis, but much more importantly, has never failed in her support for me, I would like to dedicate this work.

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α	shear angle
α	fraction of stoping width filled with backfill
a	backfill characteristic stress parameter
A, B	power function parameters
β	backfill areal fill fraction
b	backfill ultimate strain parameter
c	stope closure
C_u	coefficient of uniformity
d	maximum tolerable stope closure
d_{50}	mean particle diameter
D_{max}	maximum working deformation, yieldability
E	Young's modulus
F_y	support yield load
ϕ	friction angle
g	gravitational constant
h	height of fluid column
h	thickness of the damaged hangingwall beam
k	coefficient of permeability
k_s	initial stiffness of support system
M_f	backfill figure of merit
M_R	mass of rock to be supported
n	porosity
ρ	specific gravity
$\underline{R}$	rockburst support resistance
R_0	initial support resistance
σ	specimen compressive strength
v	maximum controllable ground velocity
$\overline{v}$	average ground velocity
w, h	specimen width and height
X	curing time, days, in curing time model
Y	primary compressive strength, in curing time model

NOMENCLATURE

C/W	cement/water ratio
COMRO	Chamber of Mines (of South Africa) Research Organisation
CSF	condensed silica fume
CT, CCT	cyclone classified tailings
CW, CMW	comminuted waste, or crushed, milled waste
ER	energy release
ERR	energy release rate
ESS	excess shear stress
FA	fly ash
FT, FPT	full plant tailings
GGBS	ground granulated blastfurnace slag
HAC	high alumina cement
NCS	non-cementitious solids
OPC	ordinary portland cement
PBFC	portland blastfurnace cement
PFA	pulverised fuel ash
RD	relative density
RHC	rapid hardening portland cement
W	water
W/S	water/solids ratio
W/H	width/height ratio

1 INTRODUCTION

Deep South African hard-rock mines are characterised by narrow, laterally extensive, inclined orebodies (reefs), in many cases extending to depths below three kilometres. Stresses around mining excavations at such depth are extremely high, with stopes and service excavations surrounded by extensive envelopes of fractured rock. The mining of such deep deposits would be impossible without the provision of competent stope support.

The primary purpose of support in stopes is to provide a safe environment for people working in them; secondly, support must ensure that the excavation can serve its intended purpose (Budavari, 1984). Technically, the support must, in the first instance, prevent the build-up of high stresses and the associated severe fracturing at the stope face (regional support) and secondly, prevent fractured rock from collapsing into the workings (local support). Support must, furthermore, protect the excavations, as far as possible, from the effects of dynamic loading, such as rockbursts and seismic tremors.

A wide variety of support elements and systems have been developed to provide the necessary local and regional support, some of which have become indispensable in deep level mining: hydraulic props and backfills are among the most important. Hydraulically placed bulk backfill is particularly well suited to deep level mine support, as it can combine many of the properties required of both local and regional support.

This thesis deals with hydraulically placed backfill stope support, with particular emphasis on cemented backfills. It will be shown, that cemented backfill can be a fully engineered support medium, that its performance and behaviour can be designed in detail, and that it can be as sophisticated a support medium, as any other support type.

1.1 Backfill Mine Support

For decades, many traditional support types and systems have been employed successfully in protecting deep stopes in South African hard-rock mines. As mining developed beyond three kilometres in depth, however, the number of rockfalls and rockbursts increased dramatically as conventional mine support proved increasingly inadequate to cope with the immense rock stresses. These problems eventually led to fundamental changes in mining strategy and mine layouts and resulted in the development of a new generation of deep level stope support, including rapid-yielding hydraulic props and competent backfill.

Backfill has periodically been used as stope support in Witwatersrand mines since the turn of the century, but only from about 1970 onwards, has it been introduced on a large scale, in the realisation that this material offered unique properties for the support of deep level mining excavations. Since then, backfills have essentially developed along two separate paths: uncemented backfills for their regional support characteristics, and cemented backfills, mainly used for their support qualities at low strains and for their self-standing ability in high stoping widths. Both types of bulk backfill are invariably utilised for their material properties, and are placed in bodies as large as practically possible.

The loading behaviour of backfill is, however, not exclusively determined by its material composition, but also by the shape of the backfill body. Up to now, the concern in this respect has been to exceed the 11:1 width to height ratio of backfill bodies in order to provide effectively infinite confinement and thereby guarantee the stiffest possible compression response. It will be shown later in this thesis that by modifying this shape, the backfill compression response can be significantly influenced, which, in conjunction with material composition, allows accurate control of the overall deformation response of backfill support systems.

There is a tendency in South African deep level mines to use either uncemented or cemented backfill exclusively, according to perceived overall support needs and company policy. Through such a categorical commitment, a company might inadvertently deprive itself of the benefits of a whole range of possible backfill support properties offered by adding or omitting binders, as the case may be. If backfill mine support is to be utilised to its full potential, a holistic approach will have to be followed in future backfill support design which is purely merit based and in which the distinction between cemented and uncemented backfills becomes entirely site-specific. In addition to the backfill properties dictated by the material composition, such a design approach will then also need to consider the effect of backfill support element shape, a domain largely overlooked today.

1.2 Review of Cemented Backfill Research

Publications on cemented backfill began to appear regularly from the end of the 1960's onwards and, by the nature of the subject, were mainly published in countries with large mining industries. Each region, naturally, developed in the direction, most relevant for the prevailing mining methods practiced there. Australian and Canadian papers are mainly concentrated on filling very large vertical stopes in base-metal mines, South African papers tend to deal with filling deep, tabular stopes in hard-rock mines, while German papers concentrate almost exclusively on filling coal mines. The great majority of publications are concerned with site-specific investigations, only a few individuals or institutes seem to be studying general principles of cemented backfill properties. This is not surprising, considering the inter-disciplinary, very specialised nature of the work and the world-wide recession in mining.

The first major work published on cemented backfill, and still often quoted, is the thesis of E.G. Thomas (1969). In this work, Thomas laid the foundation to many techniques for the investigation of cemented backfills, and touched on most issues, still considered important today. Thomas worked on Australian materials, mainly Mount Isa residues, utilising portland cement and cement/fly ash binders with relatively small fly ash contents. Thomas saw the connection between cemented backfill strength, sample porosity, and moisture content. He reported the effect of tailings classification on the strength of cemented backfill, but did not quantify the effect. He pointed out the importance of fill permeability and how this was effected by ultrafines in the backfill.

In 1975, Gonano published a paper on the size effect behaviour of cemented backfills. He concluded, that the effect of sample size on the strength of cemented backfill is not very pronounced. (His work will be discussed in more detail in Section 4.2.1 in the thesis .) In 1976, Avalle wrote a dissertation on cemented South African gold mine backfill slurries. Avalle used classified tailings and full plant tailings, as well as portland cement with pozzolans. He realised the importance of drainage on the liquefaction potential of these backfills. In his evaluation of the compressive strengths of backfills, however, he did not consider the porosity of the materials, a serious shortcoming in his work, which generally lacked a systematic approach to the backfill compositions tested.

From 1977 on, G.E. Blight began to publish in the field, systematically applying soil science and civil engineering principles to the design of backfills (Blight, et al, 1977, Blight, 1979, Blight and Clarke, 1983, Blight, 1988). Blight is an authority on the liquefaction of backfills and has long advocated the mechanical reinforcement of cemented backfills.

During the 197[illegible] number of papers appeared, dealing with massive mine backfills and specifically with the replacement of portland cement with pozzolans, such as fly ash, blastfurnace slags or reverberatory furnace slags. Most of these publications were site-specific, not concerned with general principles (Corson, 1970, Weaver and Luca, 1970, McGuire, 1978, Sallert, 1978, Schwartz, 1978 , Mitchell and Smith, 1978).

In 1978, Thomas dealt with backfill permeability, he recommended a minimum percolation rate of 100 mm/h for acceptance of backfill and stated, that the addition of portland cement eliminated the liquefaction problem. He also stated that permeability is not of great concern with the addition of binders and the associated hydration reactions. In this respect, it must be remembered that portland cement actually uses very little water in its hydration reaction, 28 % of the mass of cement, far less than what is usually present in hydraulic backfills. In the same year, 1978, Thomas and Cowling reported on the use of Mount Isa reverberatory slag as a pozzolan and note a decrease of backfill porosity with an increase in binder content.

1981 saw the first of a number of publications by W. Helms and W. Knissel on German cemented backfills (Helms, 1982, 1984, 1988 a,b, Helms and Knissel, 1981, 1982, 1985 , Knissel, 1987). Their work is concerned with the filling of German coal mines for the prevention of surface subsidence. The fill material is invariably coal washery refuse of a wide particle size spread and the cemented backfill is placed at very low water contents by pneumatic or mechanical stowing. Helms's thesis and habilitation (1982 and 1988, respectively) are excellent summaries of cemented backfill properties, they deal with all relevant strength factors, including the effects of backfill on supported rock strata, but are, naturally, tailored to German coal refuse backfills.

In 1985, Swan published a much quoted paper on cemented backfill design. He introduced a cemented backfill binder number, a function of mean particle distance - related, of course, to porosity. This binder number produces a correlation between backfill strength and cement content, but the errors are too large for practical use. In this publication, Swan did not utilise the well known cement/water ratio, even though that would have explained many findings satisfactorily. A single quantity was introduced, describing an entire particle size distribution. His paper offers a good, unconventional approach to cemented backfill design.

From 1986 on, a large amount of work has begun to emanate from the Chamber of Mines of South Africa Research Organisation (COMRO). Much of the work was published for the use of the South African mining industry but, unless confidential, has always been freely available to international researchers. Publications at conferences included: Clark, 1988 a,b, Lamos and Clark, 1989, Hinde, et al, 1990, Hinde and Kramers, 1989, Kramers, et al, 1989. Most of the work was concerned with deep level mine backfill for South African hard-rock mines. The work presented in these publications and in this thesis formed part of that research effort and the publications are referred to repeatedly. Many internal reports by Bradley and Lamos dealt with basic issues, including strain rate effects on strength (Bradley, 1986), binder content determinations in backfill (Lamos and Scalla, 1986), specimen shape effects (Bradley, 1987), curing of backfill under stress (Lamos and Bradley, 1987), curing temperature effects (Lamos and Bradley, 1987).

In 1989, Aref, et al published a paper on the liquefaction potential of cemented paste backfills. The paper gave a good general summary of the subject, but did not add substantially to the knowledge at the time. In the 1980's site-specific work on backfill binders continued, but were mainly of local interest: Atkinson, et al, 1989, Chan, et al, 1989, Douglas and Malhotra, 1989, Hopkins and Beaudry, 1989, Viles, et al, 1989.

In conclusion, the study of the relevant literature shows that:

* internationally, at least two thirds of the publications deal with site-specific or material-specific backfill problems, the remaining third being dedicated to general, basic backfill behaviour;

* the unique support challenges faced in deep South African mines have necessitated and stimulated a large amount of research work, both general and site-specific, and have made South Africa a leader in the field of mine backfill research;

* a comprehensive study on South African cemented backfills is lacking up to now;

* there is no detailed account of cementation reactions, as they apply to cemented backfill slurries;

* there is very little information on the 'post-failure' region of cemented tailings materials, and

* the design concept of a fully controlled loading response for cemented backfill mine support has not yet been described.

1.3 Purpose and Scope of this Study

A survey of the published literature shows that until now a comprehensive study on South African cemented backfills, dealing with, both, material and structural aspects of cemented backfill support design, has not been undertaken. After appropriate motivation, therefore, the South African mining industry, through the Chamber of Mines, agreed that there was a need to undertake a detailed study of cemented backfills with the objective of providing relevant information for the design of improved, cost-effective cemented backfill mine support; that was the main purpose of this study. The objectives were to be achieved by acquiring a basic understanding of the materials and parameters controlling the support behaviour of cemented backfill, including:

i. The properties of the most important aggregate and binder materials used in South African backfills.

ii. The effects of aggregate and binder characteristics on the primary strength of cemented backfills.

iii. The effects of material composition, curing environment, sample geometry and testing methods on the primary strength of cemented backfills.

iv. The effects of tailings classification, ultra-fines content and associated water on the primary strength of cemented backfills.

v. The precise functioning of binders in cemented backfills, including the effects of cement crystal growth in backfills.

vi. The relative influences of the various strength controlling factors.

vii. The behaviour of cemented backfill under rapid compression.

The investigation should further provide:

viii. A holistic approach to the design of cemented backfill stope and gully support, taking into account, both, material and structural design factors in order to be able to fully control the loading response of cemented backfill stope support.

ix. Practical design data for cemented backfill stope and gully support.

x. Empirical equations for the estimation of cemented backfill strength, based on material composition and curing time.

xi. Clarification of ambiguous backfill terminology.

1.4 Structure of the Thesis

Aside from the peripheral sections, the introductory and concluding chapters, the core of this thesis consists of four main chapters:

Chapter 2: Backfill Stope Support Requirements deals with the problems of supporting deep tabular stopes in South African hard-rock mines. It presents an overview of mining conditions at great depth, with special emphasis on local and regional stope support requirements under, both, static and dynamic loading conditions. In the chapter, various support elements and systems are compared with the specific support capabilities of cemented and uncemented backfills. The information in this chapter outlines the requirements which need to be considered in the design of competent backfill stope support.

Chapter 3: Backfill Constituent Materials describes in detail all common South African backfill materials, including aggregates, cements and cement extenders (pozzolans). Their physical, chemical and mineralogical properties are evaluated in the light of backfill requirements. The chapter provides an overview of cement and pozzolan chemistry and their hydration reactions. It also deals with the engineering properties of backfill constituents and backfill slurries, including backfill preparation and the hydraulic transportation of slurries. The information contained in this chapter forms the basis of the work presented on the load-bearing behaviour of cemented backfills in Chapter 4.

Chapter 4: Load-bearing Characteristics of Cemented Backfills is the largest chapter of the thesis and is concerned with the effects of compositional, environmental and structural factors on the strength of cemented backfills. It contains most of the experimental results and gives a comprehensive summary of the behaviour of cured cemented backfills under load. The factors investigated include, among others: aggregate and binder types, porosity, water content, curing conditions, loading rate and sample geometry. This chapter provides most of the background for the following chapter on backfill support design.

Chapter 5: Design of Controlled Loading Response Cemented Backfill integrates the results of Chapters 3 and 4 to describe how all aspects of the load/displacement curve of cemented backfill can be controlled separately. It is shown that by combining material design and the configuration of cemented backfill support elements, the behaviour of a backfill support system under load can be predetermined.

2 BACKFILL STOPE SUPPORT REQUIREMENTS

The working of deep, tabular gold and platinum deposits in South African hard-rock mines presents some of the most formidable mining challenges on earth today and the provision of safe working conditions is vital for their continued viability. This chapter outlines the stope support problem in deep tabular mining excavations, provides information on the requirements placed on competent support, and compares various support systems which have developed in response to specific mining conditions. Backfill will be shown to be an important support medium with characteristics offered by no other support type; characteristics which, in fact, make it indispensable for the support of mine excavations at great depths.

In comparing different support mediums, it must be borne in mind, however, that the use of backfill for regional support involves a fundamentally different concept to traditional support duties: whereas conventional stope support is essentially used to keep the fractured hangingwall intact, prevent falls of ground and absorb seismic rock movement, backfill regional support is capable of restraining elastic closure, thereby substantially reducing stress build-up at the mining face. This characteristic is common to, both, cemented and uncemented backfills and is undoubtedly their most important support quality.

2.1 Mining Conditions

Deep South African gold and platinum mines in the Witwatersrand system and the Bushveld Igneous Complex are characterised by narrow, laterally extensive, inclined orebodies. In some lease areas of Witwatersrand gold mines, these orebodies, or reefs, extend below 4000 m in depth and if mined, present considerable challenges in the provision of competent mine support.

Stresses around tabular excavations at such depth are extremely high, with a large proportion of these stresses concentrating at the stope faces. Unless some of these stresses can be routed through stiff support at the centre of the excavation, stress concentrations at the stope face will reach destructive magnitudes. At great mining depths, even well supported stopes are surrounded by an extensive envelope of fractured rock, as is illustrated in Figure 2.1. It is the purpose of stope support, in the first instance, to prevent the build-up of such high stresses and the associated severe fracturing at the stope face (regional support) and secondly, to prevent fractured rock to collapse into the workings (local support). Furthermore, support must, as far as possible, protect the excavations from the effects of dynamic loading, such as rockbursts and seismic tremors.

2.2 Local Support Problem

The purpose of local support is to prevent rockfalls in stopes and gullies, caused by the deterioration and collapse of fractured rock strata immediately overlying or adjoining the mining excavation. The fracturing around deep stopes is mainly due to intensive stress fields surrounding the excavations and is exacerbated by structural factors such as geological discontinuities, faults and dykes, rock type, and by blasting shocks (Brummer, 1987, Ryder, 1988 (1)). In addition to these semi-static mining problems, local stope and gully support is required to ameliorate the effects of dynamic loads imposed by rockbursts and seismic tremors.

2.2.1 Stope Closure

If mining progresses in a stope at a constant stoping width, the height of the excavation at a given point will decrease, as mining advances away from it. This is due to a combination of elastic convergence of the hanging and the footwalls, and to inelastic closure, which, at depth, is the larger component of the two. The closure profile of a stope is a graph describing the extent of closure in relation to the distance from the

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1) The excellent book *An industry guide to methods of ameliorating the hazards of rockfalls and rockbursts, 1988 Edition*, edited by John A. Ryder and published at the Chamber of Mines of South Africa in 1988, has been used extensively in the preparation of this chapter.

face (Figure 2.2). If mining advances at a reasonably uniform rate, the distance is sometimes expressed in terms of time. Figure 2.3 shows stope closure as a time-dependent phenomenon: whereas elastic closure is effectively instantaneous, inelastic closure takes place over an extended period of time. Closure rates range from as little as 1-2 mm per metre of face advance in shallow mining, to over 50 mm/m in deep stopes.

2.2.2 Static Loading Conditions

The exposed rock around mining excavations is subject to elastic and inelastic deformation. At shallow mining depths, the unsupported hangingwall is basically under tensile stress as a result of gravitational forces; at small spans, the hangingwall may be intact and perfectly capable of supporting itself. If the safe span is exceeded, however, the tensile stresses in the hangingwall will exceed the tensile strength of the rock and the hangingwall beam (1) will fail.

In deep-level mining operations, the high resultant stresses around excavations cause increasingly severe fracturing of the surrounding rock, and stope closure becomes an acute problem. In deep stopes, this closure is much greater than the expected elastic convergence and even with very stiff support, this inelastic closure can only be reduced by about 50 % (Ryder, 1988).

Figure 2.1 illustrates the stress-induced fracture pattern around a hypothetical deep stope. The severity and extent of the fracturing increases with the higher stress levels at greater depth. Several types of fractures are distinguished (Ryder, 1988):

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1) The term *hangingwall beam* has come into common usage, describing, in a structural sense, the rock strata directly above a mining excavation; it is understood that *plate* may be a more appropriate term.

* *Extension fractures*, forming a short distance ahead of the face and parallel to it, mainly as a result of high compressive stresses and the dilation of rock; *flat extension fractures* form where stresses curve around stope corners or at small spans, such as at ledges, in lags and leads.

* *Shear (or longwall-parallel) fractures* form well ahead of the face, as a result of the shear stress fields projecting ahead into the hanging and the footwall from the edges of the face; in unstable conditions these shear planes can facilitate slip displacements of large blocks of rock.

* *Blasting-induced fractures* (not shown in the illustration) which tend to aggravate hangingwall problems in critical areas.

The principal role of temporary and permanent local support is to keep the hangingwall intact. This is done by reducing the unsupported span to appropriate, safe limits and by preventing the falling of keyblocks. The task of supporting the hangingwall at great depth is aided somewhat by the dilation of the severely fractured rockmass ahead of the face, which results in compressive stresses being set up in the hangingwall at right angles to the face. These stresses help to clamp the steeply dipping rock slabs and can make a highly fractured hangingwall essentially self-supporting.

2.2.3 Static Load Bearing Requirements

The function of support elements within a support system is to prevent falls of ground, particularly of keyblocks from the hangingwall, and to attenuate the damaging effects of seismic tremors. The level of support required for the task is assessed according to the extent of fracturing around the excavation and the presence of geological weaknesses. These factors determine the mass of potentially unstable rock which needs to be supported. Under static conditions, the required support resistance, R, is essentially determined by the deadweight of unstable keyblocks:

$$R \geq \rho g h \quad . \qquad \text{(Eqn 2.1)}$$

It has been observed that most falls of ground in intermediate to deep mines involved hangingwall rocks of up to about one metre thickness, a deadweight equivalent of about 25 kN/m^2. Since support not only holds up loose rocks, but also serves to frictionally stabilise the fractured rock strata overlying the excavations, a support resistance of about twice the carried deadweight is considered appropriate. In practice, a minimum support resistance of 50 kN/m^2, applied as soon as possible, is regarded as necessary under static, or slowly changing stress conditions. Local conditions can, however, vary considerably and significantly higher values could be indicated.

2.2.4 Dynamic Loading Conditions

In deep South African mines, dynamic conditions are defined as being caused by seismic events. Seismic events are sudden relative displacements of unstable, stressed rock masses that set free potential energy. Seismic energy is transmitted through rock by compressional and shear stress waves, travelling at different velocities. In undamaged quartzite, compressional waves travel at about 5,4 km/s, whereas shear waves travel at about 3,7 km/s. Particle velocities in rocks around underground excavations, however, rarely exceed a few metres per second, the actual value being dependent on the magnitude of the seismic event and the distance from the source (Wagner, 1984, Roberts and Brummer, 1988).

In the South African mining context, rockbursts are defined as seismic events which cause damage to mining excavations. Various types of rockbursts are distinguished, either based on the severity and extent of damage (Wagner, 1984), or by their origin, as 'shear' or 'burst' type events (Ryder, 1988). *Shear events* account for most of the large seismic events (M > 2), causing severe damage to excavations. They tend to occur on large, steeply dipping geological planes of weakness, either natural faults or dykes, or mining-induced fracture planes, up to 50 m above or below the reef plane. *Burst events* are caused by the crushing of volumes of overstressed rock, either at the mining face, on pillar sidewalls, or removed from the excavations at the foundations of pillars. If these events occur near mining excavation, rock might dilate into workings.

When a seismic shock wave travels through rock, particles are accelerated forwards and backwards, reaching maximum velocities dependent on the frequency and amplitude of the wave. As a wave emerges at an excavation, the surface rock mass is accelerated into the cavity, and unless it is part of a solid rock mass with adequate tensile strength, or firmly supported, it will be thrown into the excavation at approximately the peak ground velocity. It has been observed that very little damage results to mining excavations at peak ground velocities below 1 m/s; nearly all rockbursts causing severe damage indicate peak ground velocities above 2,5 m/s (Wagner, 1984).

2.2.5 Dynamic Load Bearing Requirements

Dynamic loading conditions are much more complex than static ones, as in addition to gravity, the variables of ground velocity, seismic event magnitude and distance need to be considered. As the seismic wave arrives at an excavation surface, loose rock is accelerated into the cavity and the purpose of support now becomes to absorb the energy of the rockmass and restrain the resulting stope closure to acceptable limits.

The amount of work required from the support to decelerate the rockmass is proportional to the mass of rock to be supported, M_R, and the square of the ground velocity, $\bar{v}^2$ (Wagner, 1984). For the estimation of rockburst support requirements, a further important parameter is the ratio of the support yield load, F_y, to the rock mass to be supported, M_R. Figure 2.4 shows the relationship between the stope closure, c, the average ground velocity, $\bar{v}$, and the ratio F_y/M_R, the support density per unit rock mass, expressed in terms of gravitational acceleration [g]. If, at a ground velocity of 3 m/s, the ratio F_y/M_R falls below 3 g, the stope closure rapidly rises beyond the tolerable 0,3 m. This ratio therefore indicates the yield force required to carry a given mass of fractured rock in the hangingwall under dynamic conditions.

Figure 2.5 shows the support resistance required to restrain a given thickness of hangingwall strata to a maximum stope closure of 0,3 m at ground velocities from 1 to 3 metres per second. From the graph it is evident that if a stope, supported for static conditions at 50 kN/m^2, is shaken by a seismic wave with ground velocities of 3 m/s, less than one metre of fractured hangingwall could adequately be constrained. To support about 3 m of hangingwall under these conditions requires at least 200 kN/m^2 support resistance.

Figure 2.6 shows the distance from the source of a seismic event up to which stope closure of over 0,3 m is expected, as a function of the event magnitude and the available support resistance; three metres of fractured hangingwall is assumed. It can be seen that the areal extent of damage increases sharply below a support resistance of 200 kN/m^2. Conversely, there is little benefit to be gained by increasing the support resistance to over 400 kN/m^2.

For a quick estimation of the required average rockburst support resistance, R, assuming a fracture zone of 3 m and a ground velocity of 3 m/s, the following equation can be used:

$$\underline{R} \geq \left(1 + \frac{1}{2}\frac{v^2}{gd} \right) \rho g h \quad , \qquad \text{(Eqn 2.2)}$$

where v is the maximum controllable ground velocity (at current hydraulic prop technology: 3 m/s),

h is the thickness of the damaged hangingwall beam which needs to be supported, and

d is the maximum tolerable stope closure, in narrow stopes considered to be 0,3 m (Wagner, 1984, Ryder, 1988).

Accepting the above values for v and d, support systems in rockburst conditions should carry up to about three times the static deadweight load. In practice, support resistances of 200 kN/m^2 are presently being suggested for rockburst conditions.

2.3 Regional Support Problem

The high virgin and induced stresses, and the exceptionally large lateral extent of the narrow mining excavations in deep South African mines, have necessitated the provision of regional support. Stiff, voluminous support is required to provide, as soon as possible, passageways for the transmission of field stresses, away from the working faces, and to constrain the movement (convergence and ride) of large rock masses which would otherwise exacerbate the build-up of high stresses around the stope faces and associated service excavations. At great depth, these stresses can far exceed the shear strength of the rock and would cause excessive seismicity and catastrophic failures of the excavations.

2.3.1 Regional Support Requirements

Regional support specifications are closely linked to the concepts of energy release and the energy release rate. It is known that virgin rock stresses increase by 27 MPa for every 1000 m of mining depth, but near the edges of mining excavations these stresses are considerably higher. A substantial amount of strain energy is, therefore, present in the rock around the mining excavation, most of which (> 97 %, Ryder, 1988) is released in the form of fracturing, crushing, and slow relative movement of blocks of rock in the fracture zone ahead of the mining face. The amount of energy released through fracturing and by mining is an indication of the severity of mining conditions in a stope and it was found that high average levels significantly increase the probability of the occurrence of rockbursts.

In mining practice, a spatial rate of energy release is utilised, which relates the released energy in a specific mining section to the volumetric convergence associated with mining advance in that section. That *energy release rate, ERR*, expressed in

MJ/m^3 (1), is governed by mining depth, the shape and layout of the excavation, and the amount of volumetric convergence into the excavation. The average energy release rate is not a good measure for the prediction of falls of ground, but is a very useful indicator of prevailing mining conditions and can be used for the specification of regional support requirements. Past experience in deep mining suggests acceptable values of ERR between 20 MJ/m^2 and 40 MJ/m^2, depending on geological conditions and rock mass characteristics (Budavari, 1983, Piper, 1991).

2.3.2 Stabilising Pillars

Traditionally, stabilising pillars were left unmined to provide regional support; at moderate depths, their positions can to some extent be selected to coincide with low value ground (as in scattered mining). With increasing depth and stress levels, however, they can no longer be positioned according to ore grade, as mining layouts are dictated more and more by rock engineering requirements, with ever larger stabilising pillars locking up valuable ore reserves.

The reintroduction of backfill into deep South African mines since the late 1960's, now also as a regional support medium, has made it possible to substitute at least part of the support pillars with residue materials (Piper, 1991). In principle, it should be possible to replace stabilising and shaft pillars entirely with backfill. At even greater mining depth, this may become vital, as intact rock pillars become too stiff, threatening foundation failure in the hanging and footwalls through excessive shear stresses around the pillar edges. At that stage, backfills with accurately controlled compression behaviour, containing coarse aggregates and possibly binders, may become indispensable.

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1) The nearly constant stoping width of tabular stopes in South African hard rock mines makes it convenient to express the ERR in terms of MJ/m^2.

2.4 Support Elements and Systems

Since no single support type is able to meet all of the specialised requirements, a variety of support elements and systems have been developed for any practical mining situation. Provided that the selected support type is available and cost-effective, the choice of any one combination depends on the capacity of that system to carry the imposed loads and to absorb, as far as possible, the energy of seismic tremors. The design of effective support is largely determined by the characteristics of the rock, the deformation of the excavation to be supported, i e the stope closure profile, and the force generated within the support in reaction to its deformation (Ryder, 1988).

Each support element and system exhibits characteristic behaviour in response to loading, and the choice of support will be based on the relative merits of these characteristics. In comparing the performance of individual support elements, the deformation of the element under load is related to the reaction force generated within the support. If several elements of a support system are being considered simultaneously, the support force is expressed as *support resistance*, force per unit area [kN/m^2]; this is a useful measure of the total load bearing capacity of the installed support system.

2.4.1 Support Characteristics

Table 2.1 lists stope support characteristics and ranks them from poor to excellent. Table 2.2 compares different stope support systems according to the rating in Table 2.1 (both tables are adapted from Ryder (1988)).

The comparisons of support types in Figures 2.7 and 2.8, as well as in Table 2.2 are based on the average generated reaction force per unit area. In such comparisons, backfill, especially cemented backfill, appear as very attractive alternatives. The physical bulk of backfill support and its hydraulic placement method, however, effectively prevents this type of support to be placed closer than about 5 m from the working face. This must be taken into account when comparing the merits of backfill as a face area support. As a result of this limitation, and the fact that these first five

Table 2.1 Stope support system characteristics

	Rating				
Parameter	**1 Poor**	**2 Fair**	**3 Good**	**4 Very Good**	**5 Excellent**
Early stiffness [$kN/m^2/mm$]	< 0,75	0,75-1,5	1,5-2,5	2,5-5	> 5
Yieldability [mm]	< 25	25-75	75-250	250-500	> 500
Rockburst support resistance [kN/m^2]	< 75	75-150	150-250	250-500	> 500
Areal coverage [%]	< 3	3-10	10-30	30-50	> 50
Rockburst cost effectiveness [(1987)c/kN]	> 20	10-20	5-10	2-5	< 2

metres of the stope are particularly vulnerable to falls of ground and rockburst damage, it is unavoidable that backfill support be complemented by other means of support, preferably hydraulic props.

Figure 2.7 compares typical system characteristics of standard hydraulic props, composite packs, uncemented backfill, and cemented backfill at 1 m stoping width [1]. Figure 2.8 shows the same relationship between hydraulic props and the two types of backfill on a larger scale. Two important bench-mark values are used for comparison (Ryder, 1988):

k_s the initial stiffness of the system, i e the average slope of the characteristic curve up to 50 kN, expressed in [$kN/m^2/mm$], and

1) By its nature, backfill is evaluated in terms of generated reaction force per unit area, i e as a support system, and it is therefore appropriate to compare it with other support systems, rather than individual elements.

Table 2.2 Stope support system comparison. Ranking from 1 (poor) to 5 (excellent), according to Table 2.1.

	Early stiffness	Yield-ability	Rockburst resistance	Areal coverage	Cost per kN
Mat pack	1	5	3	3	2-3
Endgrain pack	1	4	4	3	3
Sandwich pack	2	4	4	3	3
Skeleton pack	1	5	2	3	2
Composite pack	1	4	2	3	2
Elongate pack	2	5	2	3	2
Grout pack	2	3	2	2-3	
Mine pole	4	2-3	1	1	(2)
Elongate/pipestick	4	3	2	1	2-3
Mech. yielding prop	5	2-3	1	1	(2)
Hydraulic prop	5	4	4	1	5
Barrier chock	5	4	5	1	4
Uncemented backfill	2-3	5	5	5	4
Cemented backfill	4-5	5	5	5	3-4

Support Densities:	
Packs, 1,1 x 1,1 x 1,4 m:	9 m^2 spacing
Poles, elongates, barrier chocks:	3 m^2 spacing
Mechanical and hydraulic props:	1,5 m^2 spacing
Backfill:	80 % coverage

D_{max} the maximum working deformation in [mm], also referred to as yieldability; for rockburst conditions in a stope typically 300 mm.

A further distinguishing attribute of stope support systems is immediately apparent: whereas ordinary composite packs exert no support resistance without being deformed (they are 'passive'), hydraulic props are prestressed and show an initial support resistance, R_0, commensurate with their hydraulic setting pressure, typically 200 kN per unit; such prestressed support is called 'active'. Backfills provide passive support and the initial stiffness of uncemented fills is very low; in contrast, cemented backfills, even though passive, are comparatively very stiff.

Under rockburst conditions, the capacity of a support system to absorb energy is a very important one. In Figure 2.7, this energy is represented by the area under the characteristic curve, up to the deformation limit (300 mm). A more practical indicator of this capacity is the rockburst support resistance, $\underline{R}$, in [kN/m^2], taken as the average support resistance up to the deformation limit.

A further important support property is areal coverage, that is the percentage of hangingwall area actually supported. If that proportion is small, there are the dangers of rockfalls occurring between the support units, and of the support units 'punching' into the hanging and/or footwall if shear stresses in the rock above or below the units exceed the shear strength of the rock. In this comparison, backfill scores very high, as it is normally placed in adjoining ribs over a contiguous area; total area coverage, however, rarely exceeds 80 %.

2.4.2 Regional Support with Backfill

The purpose of regional support is to prevent, as far as possible, the movement of large rock masses over the mined-out area giving rise to the build up of stress at the periphery of the excavation. In practice, rock or tailings backfills are the only artificial support mediums capable of restraining elastic closure. If backfill is to be used to provide the required very stiff support, its quality becomes of primary importance. For regional support duties the quality of backfill is best described by the figure of merit, M_f, a measure of its stiffness, as defined further below; backfill stiffness is directly correlated to the material's porosity, n.

Backfill, like soil, is a strain hardening material, i e as it undergos volumetric strain its porosity reduces and it becomes stiffer. That relationship, or 'confined compression curve' is a reproducible characteristic of any given backfill. Figure 2.9 shows the compression curves of two backfill materials of identical tailings composition. Curve (1) is the trace of the cemented tailings backfill, whereas curve (2) is that of the uncemented tailings.

The curves in Figure 2.9 show the two measures in use to describe a backfill's regional support quality: its ultimate strain value, b, along with the related parameter, a (1), and M_f, a figure of merit, defined as the mean strain undergone by a fill in reaching its design load (Ryder, 1988) (2). The graphs in Figure 2.9 also demonstrate the potential effect of cement addition on the regional support capacity of a backfill. Both backfills were placed at similar porosities, but as a result of the growth of (virtually) incompressible cement crystals inside the pores of backfill (1), its effective porosity decreased with time, making it significantly stiffer than its uncemented counterpart.

Figure 2.10 shows the regional backfill support requirements, in terms of the backfill figure of merit, for a specific mining situation. The parameter α denotes the fraction of the stoping width which is filled. Incomplete filling to the hangingwall or premature stope closure result in low values and penalties in potential ERR reductions. The parameter β denotes the areal fill fraction; above 20 % fill, the ERR value is not very sensitive to this parameter but modern practice aims at a minimum of 80 % areal coverage, mainly for local support benefits.

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1) The parameters a and b are used as input parameters for mining simulation programs, such as MINSIM-D.

2) In an unpublished Chamber of Mines research report of 1978 (Ryder and Wagner, 1978), a different definition was proposed for a figure of merit: $M = ERR_{(no\ fill)}/ERR_{(with\ fill)}$ - a ratio of the energy release rates in otherwise identical, filled and unfilled stopes, providing a direct measure of the support merit of a particular backfill.

The graph shows that high quality backfills can definitely provide high quality regional support. It can also be seen, however, that at a mining depth of 3000 m, standard quality fill at a M_f of 0,5 becomes marginal at α fractions below 1 and that it can certainly not reduce the ERR to an average of 30 MJ/m^2.

2.4.3 Support Evaluation

In the evaluation and design of support systems the type of graph shown in Figure 2.11 has come into use. The support resistance of a support system is related to theoretical closure profiles at one or more closure rates. In the support resistance graph, the design target (here: the 200 kN/m^2 rockburst design recommendation) intersects the characteristic curve of the proposed system (here: backfill). That intersection point is projected into the closure profile graph, indicating in this case, that backfill by itself is an inadequate face area support, since even at the highest closure rate, the required 200 kN/m^2 support resistance can only be achieved about eight metres behind the face. At lower closure rates, this distance becomes even greater. This again demonstrates that backfill support, even when placed only 5 metres from the face, must be complemented by effective face area support, preferably hydraulic props.

2.5 Summary and Conclusions

Narrow, laterally extensive mining excavations at great depth present acute problems with regard to the provision of satisfactory stope support. The concentration of stresses around the excavations result in a build-up of strain energy in the rock surrounding the excavation and in the release of a portion of the total gravitational energy primarily through the formation of a fracture zone around the excavation. High average levels of energy release are indicative of severe mining conditions and increase the probability of the occurrence of rockbursts.

Three types of stope support are differentiated in deep, high stress mining environments: local face area and gully support under static loading conditions, local face area and gully support under dynamic loading, i e rockburst conditions, and regional support. A number of support elements and systems are available for these duties, some temporary, such as hydraulic props and sticks, and some permanent by nature, such as timber packs, composite packs and backfill.

The purpose of local support in face areas and gullies is to prevent the fall of rocks from the fractured hangingwall and to provide frictional stability to the fractured hangingwall beam by supporting keyblocks and preventing the separation of rock strata along geological parting planes above the stope. At great mining depth, the task of local support is assisted to some degree by compressive stresses set up in the hangingwall by the dilation of fractured rock into the excavation, and the resultant clamping of the steeply inclined face-parallel rock slabs. Since local support not only holds up loose rocks, but also serves to frictionally stabilise the fractured rock strata overlying the excavations, a support resistance of about twice the carried deadweight is considered appropriate: under static, or slowly changing stress conditions, therefore, a minimum support resistance of 50 kN/m^2, applied as soon as possible, is typically recommended. Specific mining conditions may, however, demand significantly higher support resistance.

In deep South African mines, dynamic loading conditions are defined as those caused by seismic events; rockbursts are loosely defined as seismic events resulting in damage to mining excavations. The purpose of rockburst support is to absorb, without loss of integrity, the energy of the rockmass violently accelerated into the workings and to contain stope closure to tolerable levels. The maximum acceptable stope closure under rockburst conditions is considered to be 0,3 m in narrow stopes.

Seismic energy is transmitted through rock by compressional and shear waves and when such waves terminate at mining excavations, rock at the surface of the excavation is accelerated into the opening at a velocity dependent on the frequency and amplitude of the wave. Seismic events resulting in ground velocities of under 1 m/s do not normally cause serious damage, almost all cases of severe damage to mining excavations are caused at peak ground velocities of over 2,5 m/s. For a 3 m thick hangingwall beam to be decelerated within 0,3 m, at a ground velocity of 3 m/s, a support resistance of at least 200 kN/m^2 has to be provided. Little benefit is, however, gained by increasing the support resistance to over 400 kN/m^2.

Regional support specifications are closely linked to the energy release rate, ERR, a measure of the amount of excess gravitational energy released through rock fracturing around excavations, and seismic events, including rockbursts. Normally, it is aimed to reduce the ERR to a value below 30 MJ/m^2, by restraining volumetric convergence through suitable stoping layouts, the provision of regional stabilising pillars, and with high quality backfills. Backfills with a low figure of merit, i e low porosity, placed early in the mining cycle to avoid excessive stope closure, can provide very effective regional support.

F · the review of stope support requirements it is concluded that:

* a minimum system support resistance of 50 kN/m^2 is recommended for static loading conditions;

* a system support resistance of between 200 and 400 kN/m^2 is recommended for dynamic loading conditions; little benefit is gained by exceeding 400 kN/m^2;

* support with a pre-loading capability is indicated for face area support, which should also be able to yield at up to 3 m/s;

* the stiffest possible support system is required for regional support duties;

* backfill is the only cost-effective support medium capable of restraining elastic closure;

* uncemented backfill is able to provide the 200 kN/m^2 required under dynamic loading conditions at a stope closure of about 150 mm; at closure rates of 50 mm/m, with the fill placed 5 m behind the face, this occurs about 3 m inside the backfill, i e 8 m behind the face;

* the addition of a binder can stiffen the backfill to the extent where the 200 kN/m^2 can be provided at a closure of only a few millimetres;

* standard quality backfills restrict the total closure to a maximum of about 25 %, very stiff tailings/coarse aggregate backfills can transmit full field stresses at closures as low as 15 %, and

* in practical mining situations, backfills cannot be pre-stressed, nor can they provide face area support.

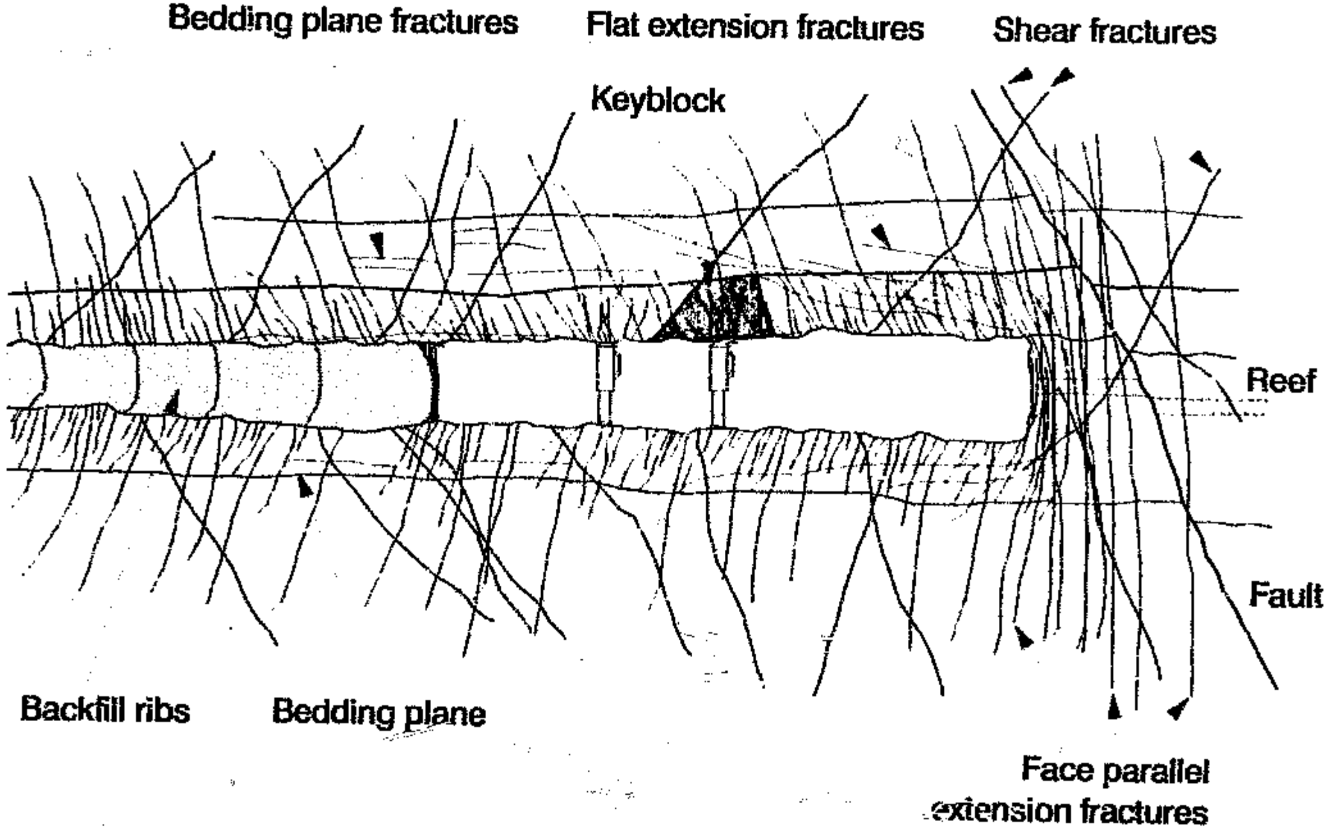

Figure 2.1 Schematic cross-section through deep narrow stope, showing various types of fractures and geological features.

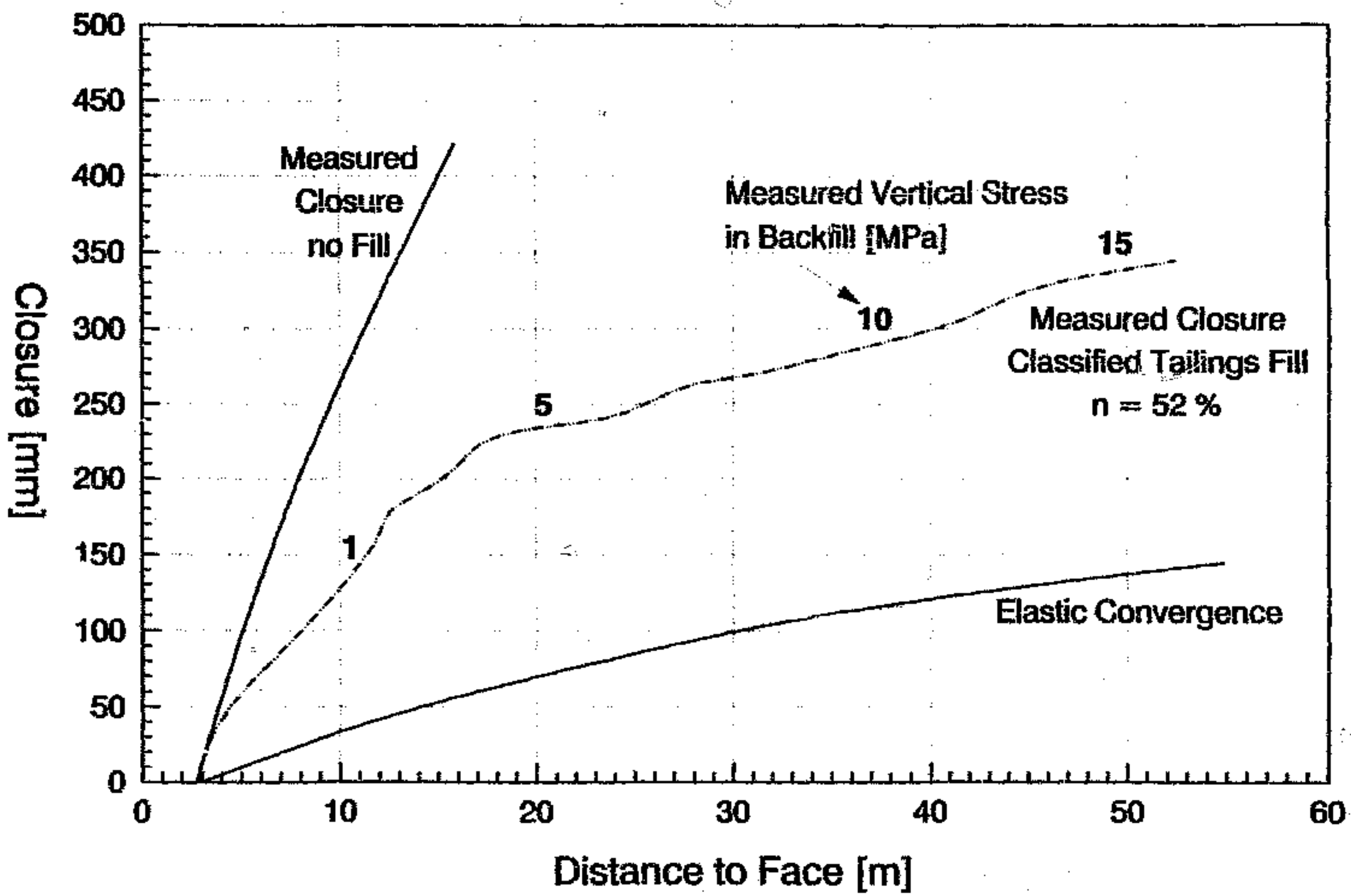

Figure 2.2 Stope closure profiles (after Ryder, 1988).

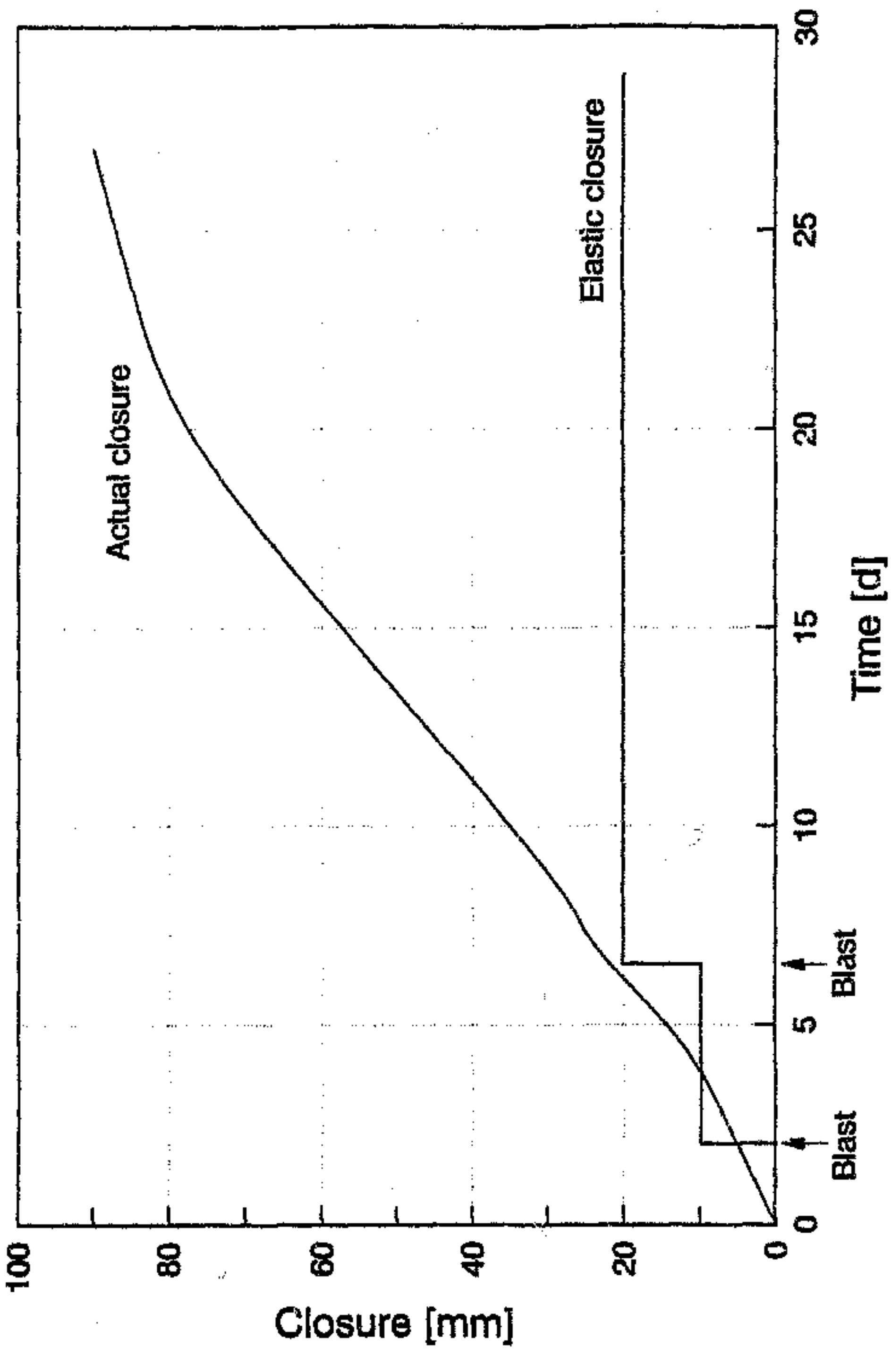

Figure 2.3 Time dependence of stope closure profiles (after Ryder, 1988).

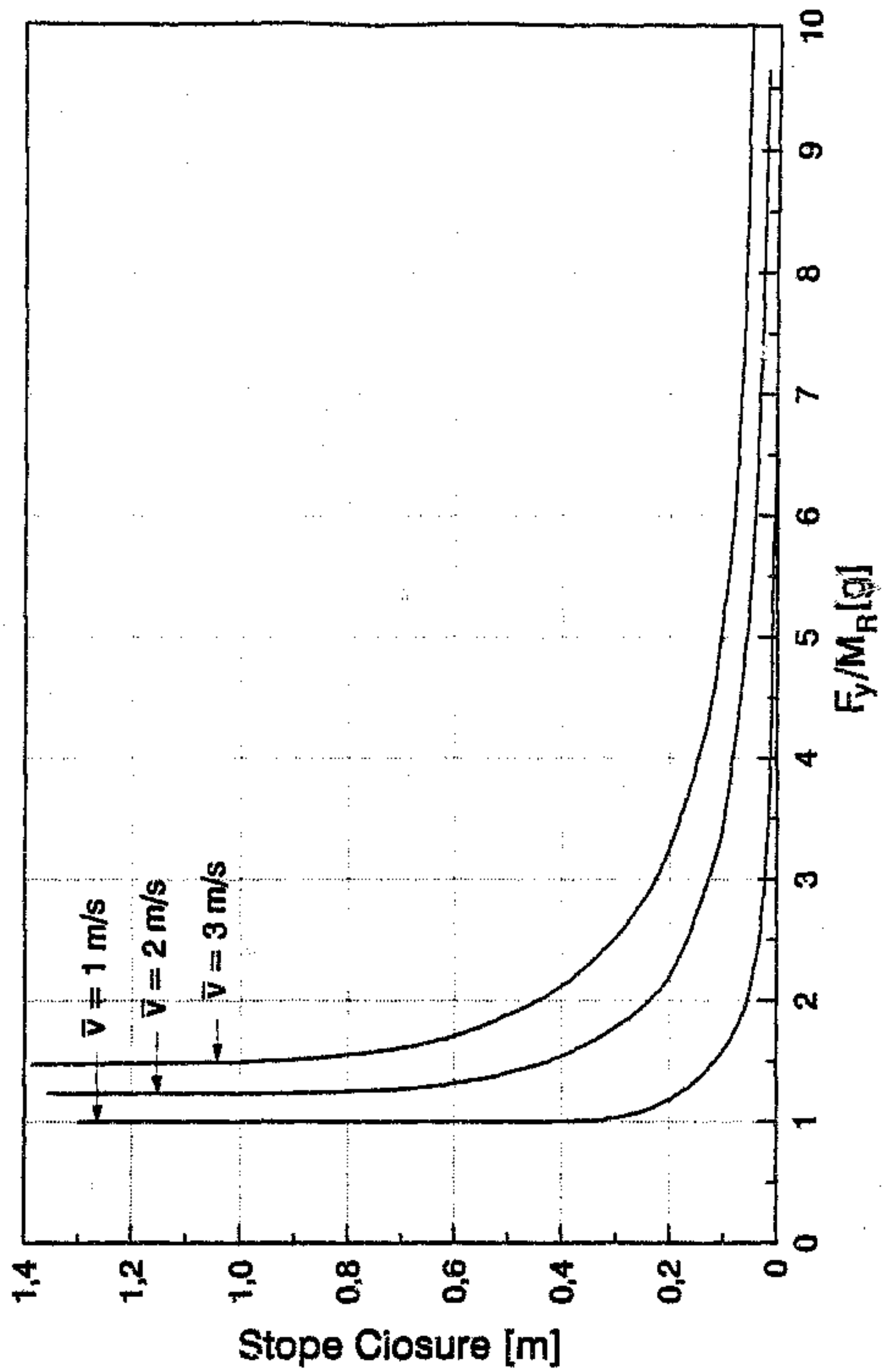

Figure 2.4 Relationship between ground velocity, support density per unit rock mass and stope closure (after Wagner, 1984).

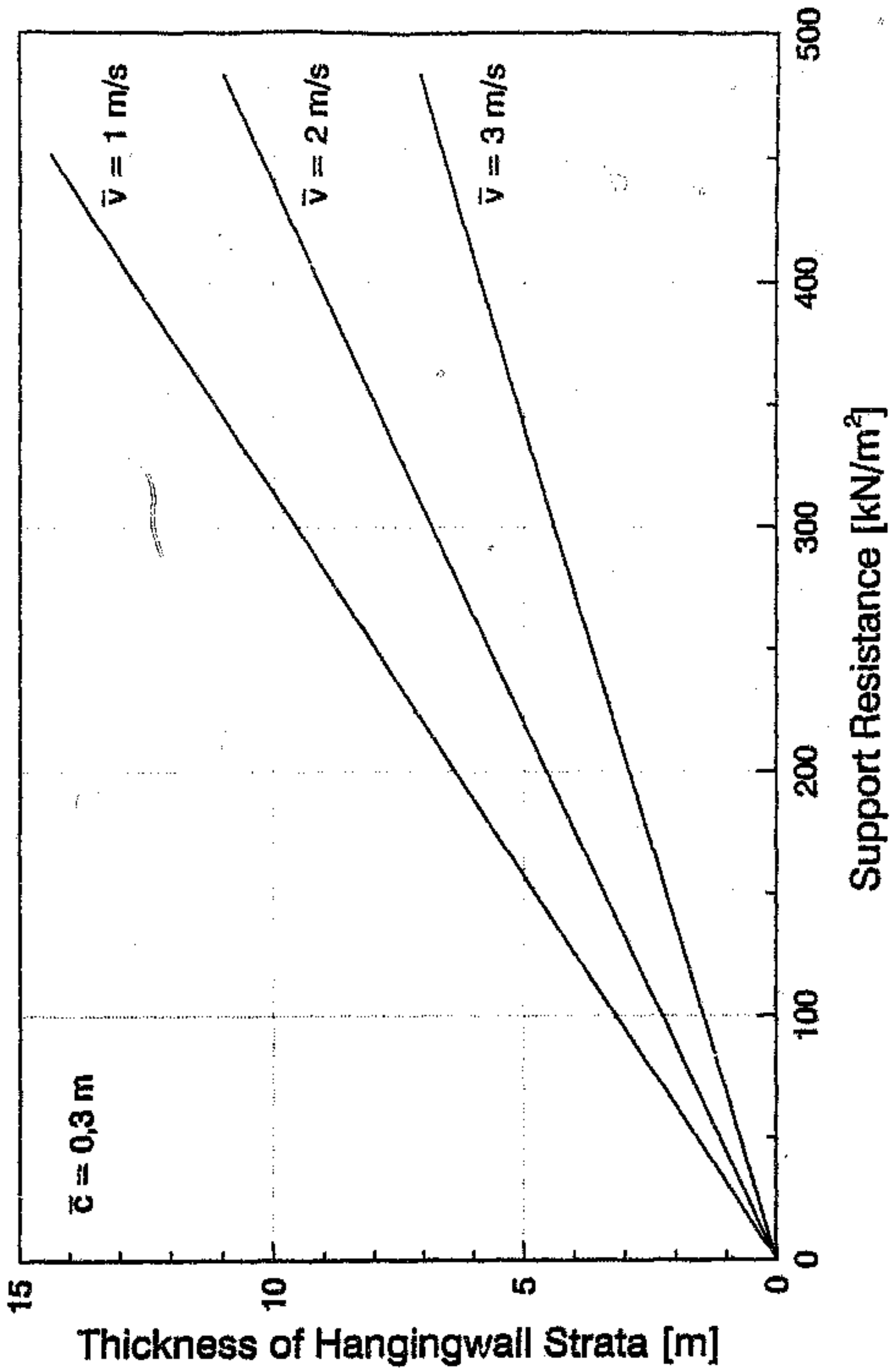

Figure 2.5 Thickness of hangingwall supported at various peak ground velocities [m] (after Wagner, 1984).

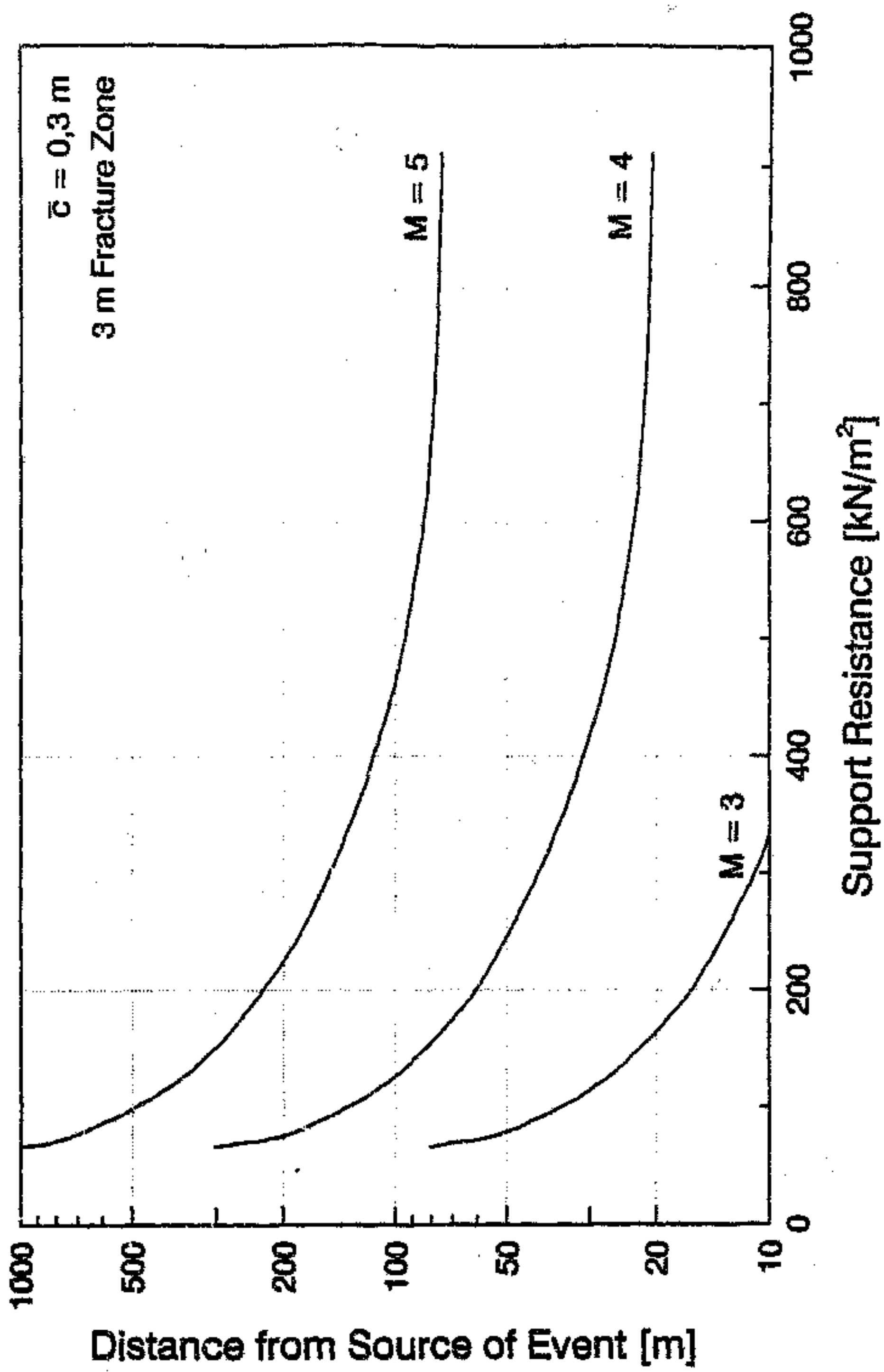

Figure 2.6 Magnitude and distance of seismic event causing 0,3 m stope closure (after Wagner, 1984).

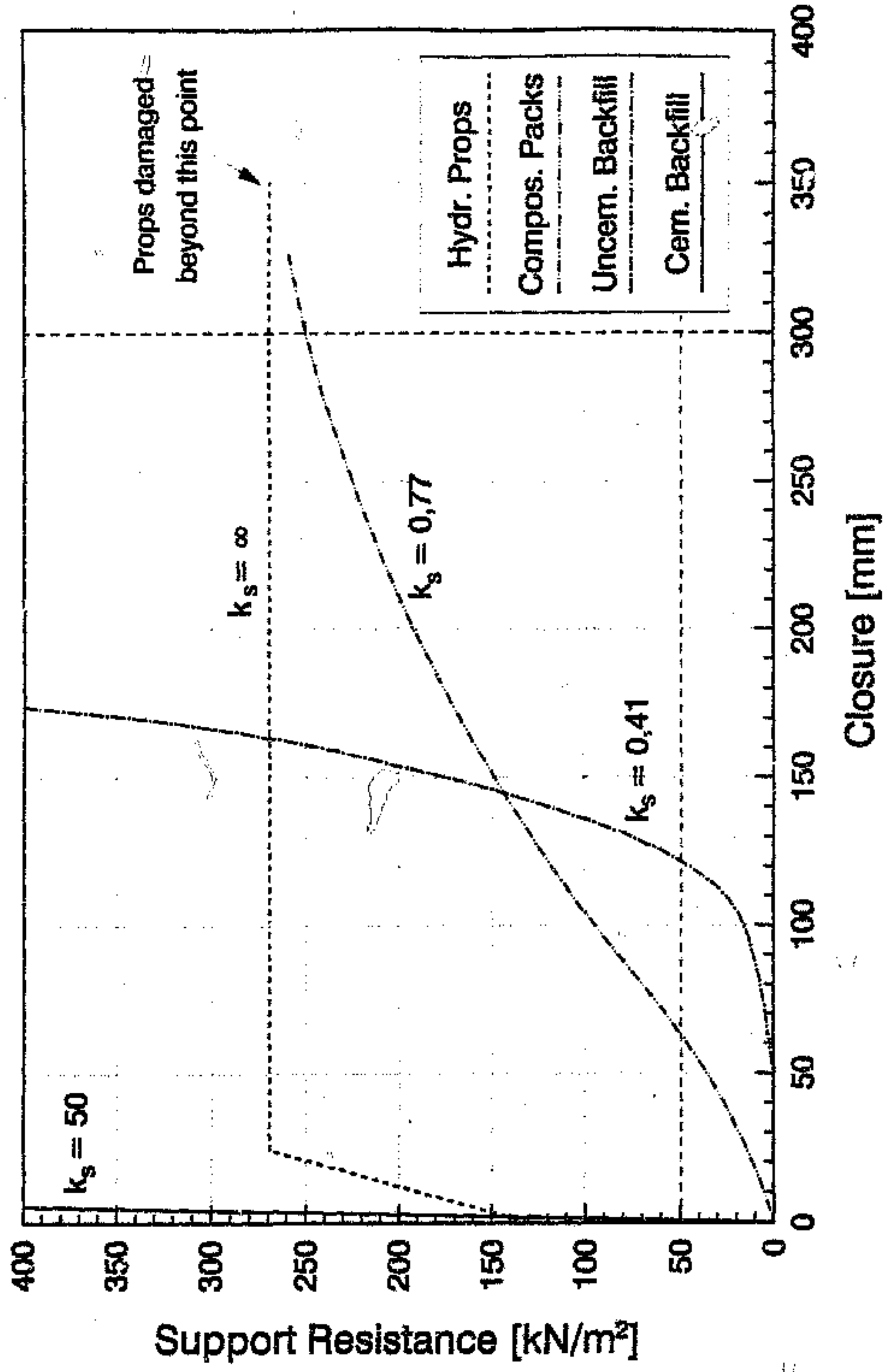

Figure 2.7 System support resistance, low stress regime [kN/m^2] (after Ryder, 1988).

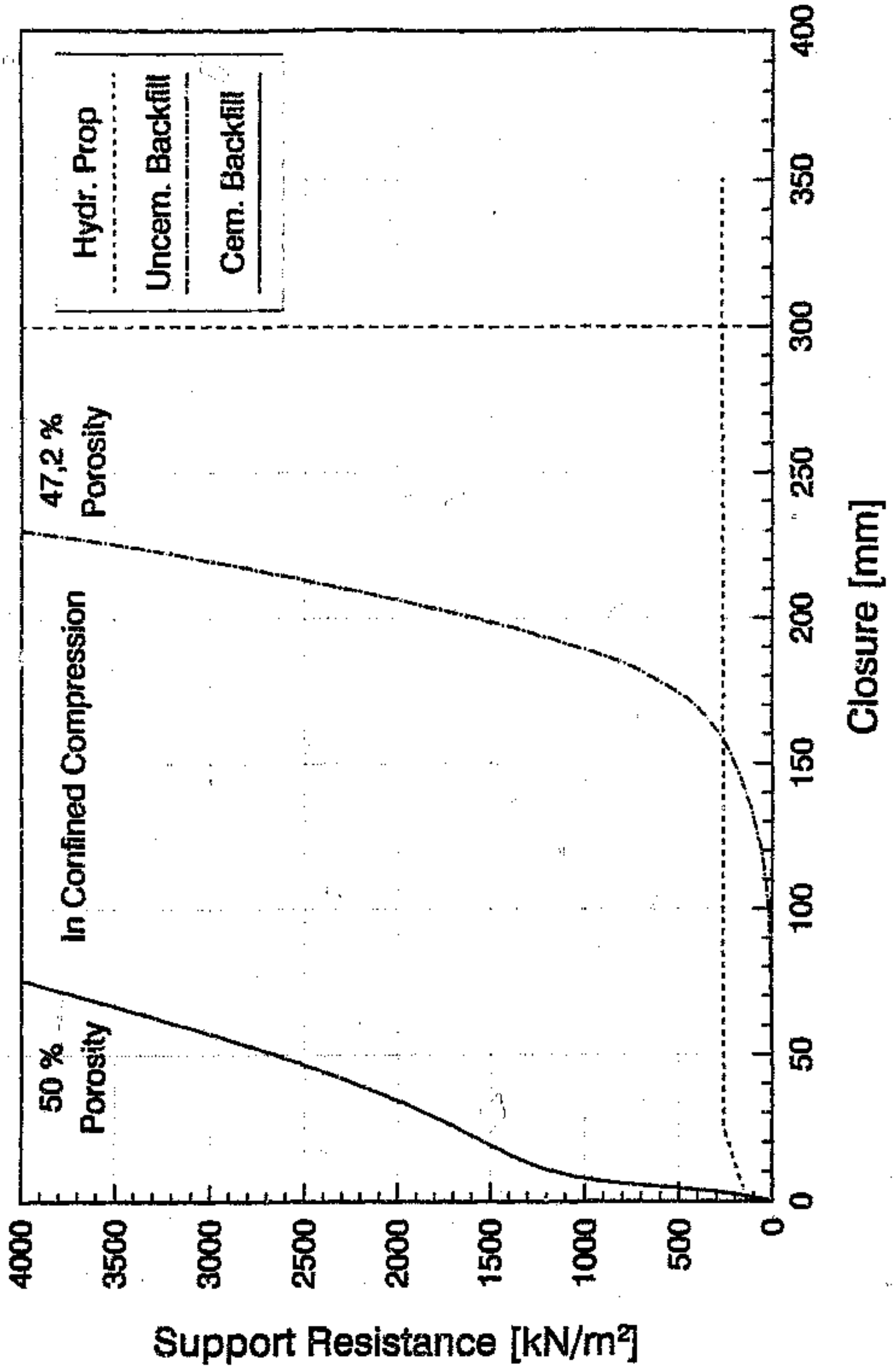

Figure 2.8 System support resistance, high stress regime [kN/m^2] (after Ryder, 1988).

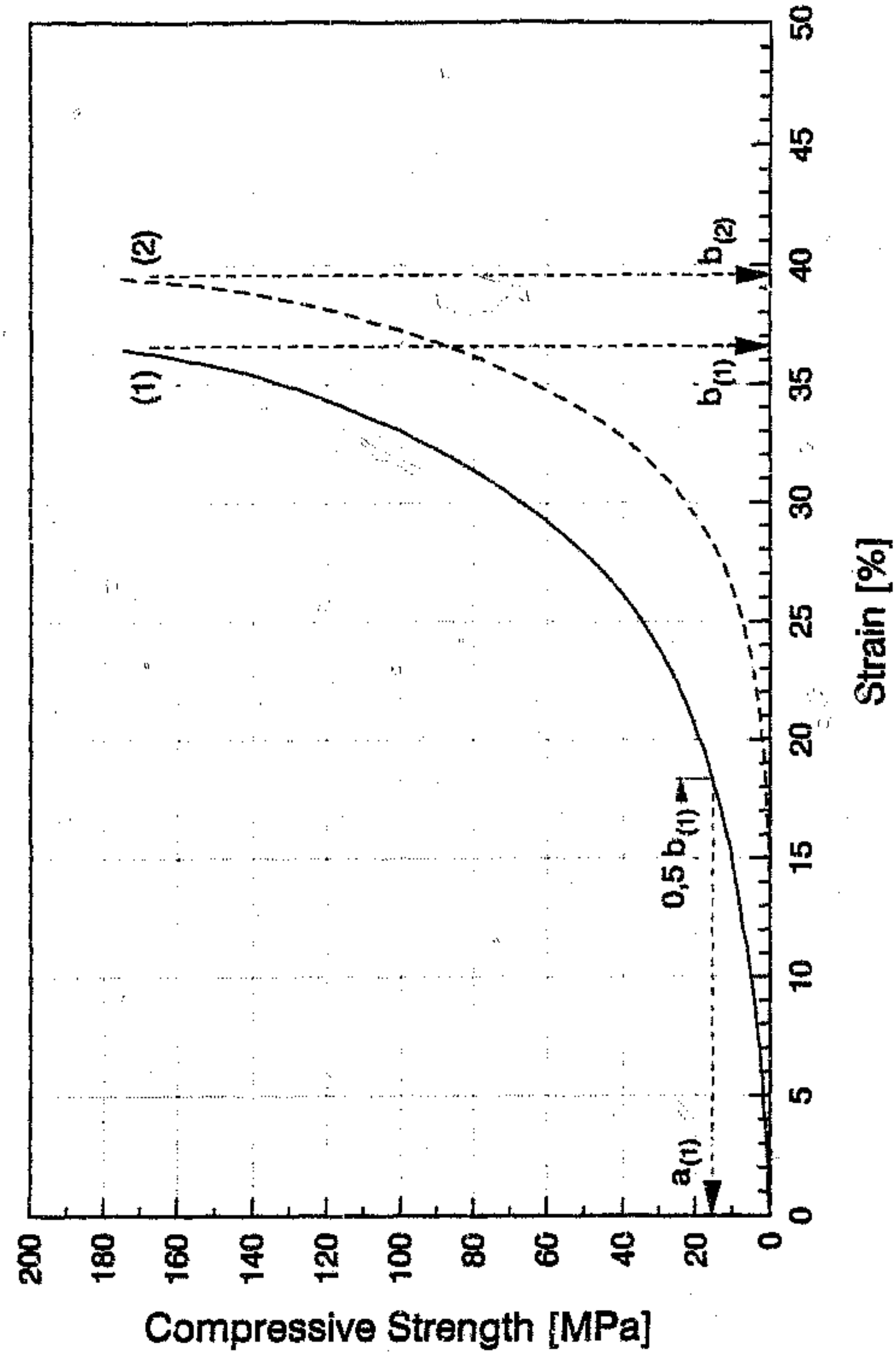

Figure 2.9 Confined compression behaviour of cemented backfill compared to its uncemented equivalent.

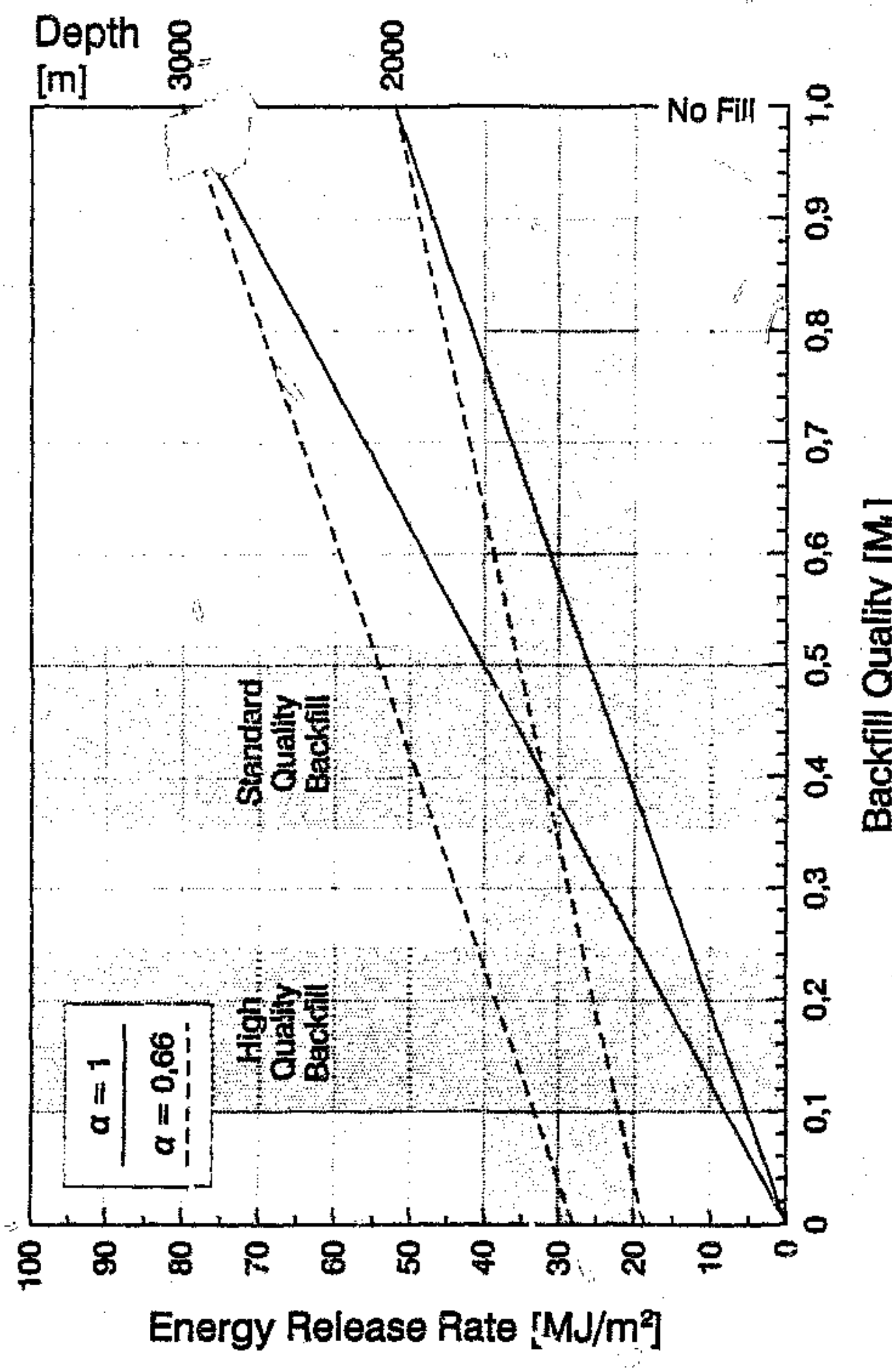

Fig...e 2.10 Effect of backfill quality on the reduction of the energy release rate in regional support applications (after Ryder, 1988).

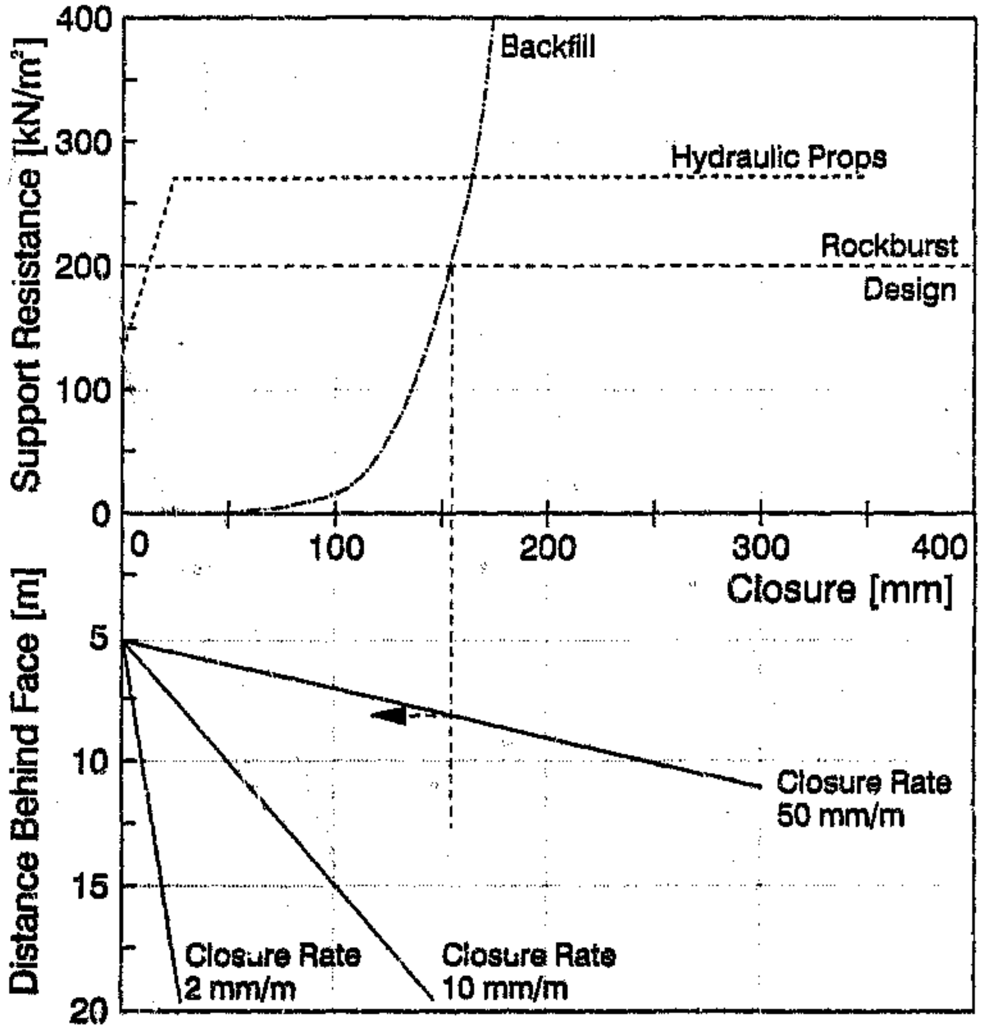

Figure 2.11 Stope support selection graphs (after Ryder, 1988).

3 BACKFILL CONSTITUENT MATERIALS

In the previous chapter, the objectives and requirements of stope support were introduced and it is evident that no single type of support is capable of performing all the required support functions in a mining environment as complex as that encountered in deep South African hard-rock mines. To provide the best possible support, appropriate for specific areas of stoping or service excavations, a variety of support types have evolved, each designed to have specific mechanical characteristics, in terms of both structure and material.

As would be expected of efficient designs, most support types have been given equivalent consideration regarding their structural and material properties. Until now, hydraulically placed bulk backfill stope support has been different in this regard: placed as a fluid, its structure is for the most part determined by the shape of the containment and of the mining excavation itself. Consequently, backfill support design has primarily been concerned with its material properties. Since one of the aims of this thesis is to promote a holistic approach to backfill support design, both, material and structural aspects of backfill design will be considered in detail. Chapter 3 and the first part of Chapter 4 of this thesis are concerned with the material properties of mine backfills.

The properties of cemented and uncemented backfill support materials are to a large degree governed by the properties of their constituent materials. The study of backfill constituents is, therefore, a very important part of the design of effective backfill support. Most of the work presented in this chapter was undertaken as an indispensable part of the experimental process and the results were essential to the understanding and control of final backfill properties.

For reasons of economy, mine backfills consist for the most part of site-specific waste materials and the *a priori* specification of bulk constituents is only practical, insofar as these can be met by local materials. The purpose of this chapter is, therefore, not so much to set bench-marks and material specifications, as it is to describe, in detail, the specific materials used in the experiments and thereby to highlight the crucial properties which determined the properties of the produced backfills. Unlike bulk aggregates, cementitious backfill constituents are commonly imported to the mines and their selection and specification are important design decisions. These materials will be discussed separately, along with their chemical properties which provide cemented backfill support materials with their specific qualities.

Once constituted, some of the components of cemented backfills enter chemical reactions which continuously transform the properties of the raw materials and the bulk of the fill. Provided that sufficient water is available in the fill, these changes can continue for several years, often long after the backfill has outlived its original purpose. It follows, that any description of backfill materials will have to be time-specific; an appropriate cut-off is provided by the final placement of the fill. Up to that moment, the current and eventual properties of the backfill material can be influenced by design, from then onwards, the placed backfill must be regarded as the support medium, even if its properties continue to change with time. This chapter, therefore, in addition to the physical and chemical properties of cemented backfill constituent materials, also deals with backfill slurries prior to placement underground.

Cemented backfill support, by its nature, involves the placement of large masses of material and for it to be cost-effective, waste materials are utilised as far as possible. This is equally applicable for the bulk of the fill, i e the aggregate, and the binder proportion. In the following sections, therefore, all materials discussed in detail, with the single exception of portland cement, are industrial waste or by-products, mainly from the mining industry itself.

3.1 Definitions of Terms

With the gradual introduction of backfill in South African mines over the last decades, a number of terms have been used, and continue to be used, describing backfill constituent materials. Most words were adopted from other disciplines, including soil science, metallurgy, sedimentology and concrete technology. Backfill technologists too, have adopted some common terminology, but many conflicting expressions persist. To prevent any misunderstanding, the following terms are being defined for this thesis:

Aggregate is the bulk material in a backfill, usually chemically inert, frequently a metallurgical or mining waste product. Backfill aggregates may also consist of other comminuted materials, such as natural sands or quarried rocks. The term aggregate does not imply any grain size specification.

Fines or fine aggregate - backfill bulk material of grain size smaller than 4,75 mm, synonymous with the usage of the term fine aggregate or sand in concrete technology.

Ultra-Fines or ultra-fine aggregate - the backfill bulk material fraction smaller than 0,010 mm (1).

Rock or stone, or coarse aggregate - backfill bulk material with at least 50 % of the mass having a grain size larger than 4,75 mm.

Binder is the cementitious material added to backfill to provide it with real cohesion. A backfill binder often consists of more than one component, typically an active cement and a pozzolan. A binder is the combined mass of all materials added to the backfill to provide cohesion.

Cement is the active component in a backfill binder - it has cementitious properties of its own. Ordinary portland cement is a typical, often used cement.

Pozzolan is a binder component with no cementitious properties of its own, but which becomes cementitious by utilising excess reaction products from the cement

1) After Thomas and Holtham (1989).

hydration; pozzolans can be seen as cement extenders. In South Africa, the most commonly used pozzolans are fly ash and ground granulated blastfurnace slag (trade name: slagment).

The main components of backfills are therefore: aggregate, water and, if applicable, binder. Tailings would be classified as *fine (backfill) aggregate*, and the material at or below the chosen bench-mark particle size fraction (e g 0,010 mm), as *ultra-fine (backfill) aggregate*.

Over the last two decades, the bench-mark size, separating ultra-fines from fines has progressively drifted from 0,075 mm, over 0,045 mm, 0,038 mm, to the present 0,010 mm. Beside the gradual fining of mill products, this shift indicates an increased awareness of specific backfilling requirements, rather than extraction metallurgical considerations where the previous control sizes have originated. The 0,010 mm bench-mark now lies in the region of high specific surface areas (Figure 3.1) and has therefore become an appropriate indicator of many important, water related backfill properties.

3.2 Backfill Aggregates

Most backfill aggregates consist of tailings products after the metallurgical extraction of the ore minerals, or of comminuted mining development waste. They are normally grouped according to their metallurgical origin or preparation method, e g full plant tailings, cyclone-classified tailings, or rod-mill product. In the following subsections, the various aggregate types are described generally; later on, their detailed material properties are presented and compared.

3.2.1 Full Plant Tailings

Full plant tailings are single- or multi-stage mill products which have undergone extraction-metallurgical processes: these might include gravity concentration, leach treatment, filtration, flotation, and thickening. Most of these processes modify the product particle size distribution to some extent, usually removing portions of certain size fractions. Normally, those changes are insignificant to the overall size distribution, but they can have a marked influence on the amount of ultra-fine particles remaining in the tailings. Figure 3.2 shows the size distributions of several South African gold and platinum full plant tailings backfills, illustrating the wide range of product fineness.

Even though uncemented full plant tailings were successfully backfilled underground (Clark, 1988 c), they have a number of disadvantages which render them unattractive as hydraulically placed bulk backfills; some of these are mentioned below. Beside forming the source material for classified tailings backfills, their use today is largely restricted to cemented backfills.

3.2.2 Classified Tailings

One widely adopted method for improving the hydraulic transportation, placement and drainage behaviour of metallurgical plant tailings is to reduce their ultra-fines content. Figure 3.1 demonstrates that the largest surface areas are contributed by particles of the small size fractions, below about 0,010 mm (1). As most of the detrimental properties of full plant tailings, such as water retention and high pumping pressure losses can be attributed to the presence of very fine particles, it is of advantage to remove these fines from the material by classification.

Full plant tailings materials can be classified in a number of ways, including centrifuging and hydraulic settling; today the most common procedure is, however, to use hydrocyclones. The operation and performance of hydrocyclones for backfill classification, as well as the influence of different classification parameters on the behaviour of backfill is summarised by Hinde and Kramers (1989).

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1) The surface areas in Figure 3.1 are calculated for a sub-angular particle shape. It is likely, however, that the fine size fractions contain more phyllo-silicates than the coarse particles and their specific surface area may therefore be underestimated in the graph. This is a common difficulty in the measurement and interpretation of specific surface areas.

The classification of tailings, however, can also have significant disadvantages:

* Reduced fill tonnage. The low recovery of an inefficiently operated cyclone classification plant might lead to a shortage of fill material.

* Disposal of overflow product. The ultra-fine overflow product of the cyclone classification has to be disposed of. The extremely slow settling rate and the high water retention of the material may pose special problems with the design and operation of tailings dams (McPhail and Wagner, 1987).

In spite of these drawbacks, the r[illegible]h improved transportation, placement and drainage properties have made cl[illegible]ied tailings the most extensively used backfill materials in deep South African mines.

3.2.3 Comminuted Waste Rock

Comminuted waste rock can be a [illegible] or multi-stage crusher or rod-mill product. Typically, it is characterised by a v[illegible]aded size-distribution (Figure 3.6), allowing placement at very low porosities. As this material does not normally undergo metallurgical processing, its size-distribution can be optimised for the task of backfilling. If the size-grading is suitable, comminuted waste backfill can be pumped as a hydraulic slurry, otherwise it must be moved mechanically.

Waste for backfilling can be produced and comminuted underground; this reduces tramming costs and relieves the shaft of unproductive waste hoisting [1]. In modern deep mines, shaft time is very tightly controlled and any reduction in waste hoisting can improve mine productivity.

By far the largest proportion of waste rock is development waste. With more efficient mining methods, however, and development within ore bodies, the production of waste is minimised, as far as possible; its availability is declining steadily (Hinde, et al, 1990). With increasing mining depth and the resulting more stringent

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1) A complete backfill crushing and milling plant was in underground operation at the Western Deep Levels Gold Mine (South Africa) in the late 1980's.

regional support requirements, stiffer backfills will, nevertheless, be required in future deep mines and resue mining may have to be re-introduced to provide an adequate supply of waste backfill material. Underground ore-sorting may also then prove to be a feasible source of waste rock (Hinde, et al, 1990).

There are some practical problems associated with the use of waste rock as backfill:

* Waste grades. Sometimes waste rock shows significant ore grades and there is a reluctance to backfill potential mineral resources.

* Loss of ore. In extensive underground mining operations, which are difficult to oversee and control, the accidental or unknowing loss of ore into the backfilled waste is of great concern to many operators. The same concern holds true for underground ore-sorting, which by its nature operates with cut-off grades.

The philosophy in the past has been, rather to get all the waste to surface, and, if viable, hold it as a low-grade ore reserve. In very deep mines, the transportation of waste becomes so expensive, however, that its projected future value on surface might not balance its transport costs.

3.2.4 Mixtures of Tailings with Comminuted Waste Rock

A very effective way to reduce the porosity of backfills - a property strongly related to the ultimate compressive stiffness of a backfill - is to replace some of the tailings with solid rock, normally comminuted waste rock. The fines portion of such fills can consist of either full plant tailings or classified tailings; in terms of material stiffness there is little difference between the two types of backfill. Classified tailings are usually chosen, however, for their better hydraulic transportation and drainage properties.

Tailings/waste-rock mixtures can be produced on surface and hydraulically transported underground, but there are often strong economic incentives to produce and add the coarse aggregate portion to the tailings underground. This involves the provision of a crusher plant underground, where run-of-mine waste is comminuted to the required sizes. The materials are crushed and/or milled to a specified top size (e g $\leq$ 25 mm) and then mechanically mixed into the tailings slurry which is gravitated to the plant from surface (1). The coarse aggregate contains only little water and this provides an opportunity to increase the placement density of the backfill; binders or binder additives might also be mixed in at that point. The final transportation to the stopes usually has to be done by positive displacement pumps at high pressures.

The problems associated with mixtures of tailings and comminuted waste rock, arising from the shortage of run-of-mine waste, are very similar to those described in Section *3.2.3*. Tailings/waste-rock mixtures are less problematic in this respect, however, as the backfill properties are not very sensitive to the relative proportions of the backfill components, i e a lack of coarse aggregate can be compensated with fine aggregate (tailings) at only a slight penalty in backfill quality.

3.2.5 Mineralogy of Backfill Aggregates

The deep South African hard-rock mines, considered in this study, are located within two major geological systems: the Witwatersrand Supergroup (gold mines) and the Bushveld Igneous Complex (platinum mines). The ores and host rocks associated with these systems produce two different types of backfill mineralogy, as exemplified by two typical XRD analyses listed in Table 3.1 below.

The gold tailings are of fluvial sedimentary origin and consist mainly of quartz, whereas the platinum tailings originate from magmatic differentiation and consist mainly of mafic minerals, such as pyroxenes and amphiboles, as well as feldspars. Both tailings types are largely inert and yield stable backfills, provided that the amounts of phyllo-silicates, such as mica, talc and chlorite do not significantly exceed 10 %.

Phyllo-silicates, and especially clay minerals, such as kaolinite, montmorillonite and smectite, have high specific surface areas and thus high water demands. This can lead to long-term water retention and unfavourable cement/water ratios in cemented

1) At the time of writing of this thesis, an underground backfill waste crushing plant is being commissioned at the South Deep Project (South Africa).

Table 3.1 X-ray diffraction semi-quantitative mineralogical analyses of typical South African gold and platinum mine tailings, [%].

Mineral	Au Tails	Pt Tails
Quartz	80	2
Chlorite	8	-
Pyrophyllite	5	-
Hypersthene	-	69
Plagioclase	-	12
Talc	-	8
Mica	7	7
Amphibole	-	2

backfill materials. The layered structures of these minerals, furthermore, present weak points in the fill mass, where shear slip originates readily. Most test work for the present study was conducted with quartzitic tailings from Witwatersrand mines.

3.2.6 Particle Size Distributions

The particle size distribution is one of the most important characteristics of any backfill material. It has a direct and primary influence on backfill properties such as placement porosity and hence ultimate backfill stiffness, water drainage or retention, and hydraulic transportability.

Tailings backfill materials, consisting mainly of a single mineral species, such as is quartzite in most Witwatersrand ores, and which are derived from comminutive metallurgical processes, tend to show unimodal size distributions, i e distributions with statistically only one maximum size class. If substantial amounts of a softer, more easily crushed mineral are associated with the ore, for instance chlorite (a common phyllo-silicate), bimodal distributions can result, in principle similar to the one shown in Figure 3.3 (c). The blending of different materials, as in tailings/coarse aggregate mixes can also produce multimodal distributions. Figure 3.3 shows the major types of size distributions of backfill aggregate materials.

Most tailings backfills considered in this study are of the unimodal type and can for the most be described by two parameters, derived from soil science:

* the mean particle diameter, d_{50},

describing a size distribution in terms of fine or coarse grained (1) and

* the gradation index, d_{80}/d_{20},

describing the gradation of a distribution in terms of uniform or well graded (2).

The distinction between fine and coarse is self-explanatory; uniform distributions are characterised by the predominance of one single size class, whereas in well graded distributions particle sizes are spread over a wide range of size classes. Table 3.5 later in the section will provide details of these indices for the backfill aggregates.

In metallurgical process plants, product particle size distributions are normally described in terms of a mass fraction (percentage), smaller than a designated sieve size. This provides a simple control measure in milling circuits and extraction processes. Since the provision of backfill aggregate materials was traditionally the responsibility of plant metallurgists, the particle size bench-marks were naturally

1) The mean particle diameter d_{50} of a granular material is defined as the sieve size in [mm], which would permit 50 % of the mass of that material to pass. Other indices, such as d_{80}, are defined in an analogous way; in this case, as the sieve size which would permit 80 % of the mass of the material to pass.

2) Various ratios are being used, e g d_{60}/d_{10}, depending on specific circumstances.

adopted for the description of backfill fineness. In South African gold and platinum mine backfill practice the bench-mark size has over the last decades progressively shifted down from 0,075 mm over 0,045 mm, 0,038 mm, to the present 0,010 mm.

The comparison of backfill aggregates in terms of their ≤ 0,010 mm fraction is, in part, a reflection of increasingly finer mill products, but also of an increased awareness of the needs of backfill technologists. This 0,010 mm bench-mark size has at last come to lie in the region of high specific surface areas (Figure 3.1) and can therefore appropriately gauge many important, water related backfill properties.

Thomas and Holtham (1989), considering world-wide metallurgical practice, categorises ore grinds between 'very coarse' and 'extremely fine', with the ≤ 0,010 mm fractions ranging from under 10 % , to over 50 % by mass, respectively. In this classification, most important South African ores are comminuted to between medium and fine mill products, with typical Witwatersrand gold plant tailings ranging from 20 % to 40 % ≤ 0,010 mm. Platinum ores are being ground to very similar finenesses (20 % to 40 % ≤ 0,010 mm), but with high talc contents can occasionally show ≤ 0,010 mm fractions of over 60 % (McPhail and Wagner, 1987).

Table 3.2 and Figure 3.4 show the particle size distributions of some South African full plant tailings. The ultra-fine particle fractions with their high specific surface areas crucially effect the materials' physical, chemical and mechanical properties. They have a negative influence on hydraulic transportation, placement properties and permeability, and if associated with phyllo-silicates, are detrimental to cemented backfill strengths. If the ultra-fine fractions consist of tecto-silicate particles (e g quartz), small enough to be chemically reactive in highly alkaline environments, they can become part of the cementitious binder and thus enhance cemented backfill strength.

Table 3.3 and Figure 3.5 show particle size distributions of cyclone-classified tailings products. It can be seen, that compared to full plant tailings, the size fractions below 0,010 mm have been reduced significantly, thus removing the detrimental fines portions. For comparison, the line marked *centr* in Figure 3.5 shows the size distribution of a backfill classified with a centrifuge; its product has a slightly more even size distribution.

Table 3.2 Particle size distributions of some metallurgical full plant tailings (FPT) used in this study. Cumulative [%] passing.

	Mass Passing [%]			
Size Class [mm]	FPT 1	FPT 2 *	FPT 3	FPT 4
0,005				25,0
0,010	15	17,4	21,8	39,0
0,015	22,1	24,3	29,3	48,0
0,020	28,7	31,1	34,9	54,7
0,025	36,4	36,7	39,1	62,5
0,038	42,3	43,7	49,2	72,0
0,053	51,9	53,3	60,6	80,0
0,075	61,9	63,5	65,9	87,5
0,106	81,1	84,4	79,2	92,3
0,150	94,7	95,6	79,9	97,0
0,212	99,2	99,2	100,0	100,0
0,300	99,8	99,7		
0,425	99,9	99,9		
0,600	100,0	99,9		
0,850		100,0		

* Material used in most of the original experiments for this study.

The particle size distributions of some comminuted waste materials are shown in Table 3.4 and Figure 3.6. If the materials are to be transported hydraulically, the largest aggregate size is usually kept to less than one fifth of the dimension of the transportation pipes, to prevent particle bridging and resulting pipe blockages. Tailings/coarse aggregate backfills are produced specifically for their potentially low placement porosities and the resulting high compressive strengths; their fines content allows hydraulic transportation with positive displacement pumps. Uncrushed run-of-mine rock normally has a top size of 300 mm, larger rocks are removed to prevent waste-pass and bunker blockages.

Table 3.3 Particle size distributions of some cyclone classified tailings (CCT) used in this study. Cumulative [%] passing.

	Mass Passing [%]			
Size Class [mm]	CCT 1	CCT 2 *	CCT 3	CCT 4
0,010	3,2	4,0	6,8	3,6
0,015	4,2	4,8	8,0	5,1
0,020	4,8	7,4	9,7	9,0
0,025	5,2	10,9	12,2	11,9
0,038	7,5	14,0	14,2	16,9
0,053	10,6	17,4	19,5	26,5
0,075	19,8	24,2	33,3	37,0
0,106	31,0	42,5	57,0	51,2
0,150	58,4	73,1	89,6	68,5
0,212	76,1	93,7	99,1	82,3
0,300	97,1	99,5	99,9	92,1
0,425	99,9	99,9	100,0	96,4
0,600	100,0	100,0		98,4
0,850				100,0

* Material used in most of the original experiments for this study.

Table 3.4 Particle size distributions of comminuted waste products and a tailings/coarse aggregate 30/70 blend. Cumulative [%] passing.

	Mass Passing [%]			
Size Class [mm]	Cone	U/G Rod *	Op Circ	30/70
0,038	1,6	16,6	3,7	16,2
0,053	2,0	18,7	4,1	20,0
0,075	2,4	21,4	5,0	26.1
0,106	3,0	25,3	6,0	32,5
0,150	3,7	29,8	7,4	39,5
0,212	4,4	34,7	8,8	41,4
0,300	5,3	41,7	10,4	42,8
0,425	6,1	48,0	12,4	44,0
0,600	6,7	55,3	14,0	45,0
0,850	7,6	62,5	16,0	46,1
1,180	8,9	71,5	18,6	47,8
1,700	10,2	79,1	21,8	49,5
2,360	13,1	88,8	25,3	52,5
3,350	16,9	94,9	28,3	56,5
4,750	24,6	98,9	35,9	62,6
6,700	36,9	99,8	46,7	72,5
9,500	63,6	100,0	64,1	91,5
13,200	99,5		86,5	100,0
19,000	100,0		94,2	
26,500			96,5	
37,500			98,6	
53,000			100,0	

* Material used in most of the original experiments of this study.

The following abbreviations are used in the table:

Cone	Cone crusher product,
U/G Rod	Rod-mill product produced underground,
Op Circ	Open circuit jaw and cone crusher run and
30/70	A 30:70 blend of classified tailings and crushed waste.

Table 3.5 lists the properties of the backfill materials with which most work in this study was undertaken. The respective size distribution curves are shown in Figure 3.7.

Table 3.5 Particle size distribution properties of backfill materials used in this study.

Backfill Aggregate	**d_{50} [mm]**	**C_u (d_{60}/d_{10})**	**d_{80}/d_{20}**	**≤ 0,010 mm**
Full Plant Tailings	0,050	12,2	8,3	17,1
Cyclone-classified Tailings	0,120	6,8	2,8	4,0
Crushed and Milled Waste Rock	0,470	110,7	26,1	10,0

Clark (1988 c) states that for well graded size distributions, the d_{80}/d_{20} mass ratio is a better indicator of uniformity, than is the conventional coefficient of uniformity, C_u, defined as the d_{60}/d_{10} ratio of masses.

The addition of binders to backfill aggregates changes the combined size distribution only slightly from that of the aggregate. The reasons are that normally less than about 10 % of binder is added and secondly, the binder size distribution is not very much finer than the aggregate's. Figure 3.8 shows the component and combined particle size distributions of a cemented full plant tailings backfill. The addition of 10 % of the binder raised the ≤ 0,038 mm size fractions by about 4 %.

3.2.7 Porosities and Placement Properties

The load-bearing properties of both uncemented and cemented backfills are to a large extent dependent on the porosities of the placed materials. The placed porosity is, therefore, one of the most important determinants of backfill aggregate quality. A material with a uniform size distribution, such as a well classified cyclone product, has a high porosity and a low stiffness at low strains. The low porosity provided by a well graded particle size distribution, on the other hand, such as in comminuted waste, leads to high compressive strength. The stiffness of these materials increases with increasing strain - they are strain-hardening.

The phenomenon is explained by the micro-structure of a consolidated particulate material: at high porosity all stress applied to the bulk of the material is conducted through relatively few contact points between the aggregate particles. As compression commences, some slippage occurs at these points until the densest possible packing is achieved; then the compressive strength of the mineral grains is exceeded at the most highly stressed points, fracturing grains and thereby producing the fines necessary to fill the voids between larger grains - i e producing more contact points to share the load and thus a stiffer material with reduced porosity.

The placement properties test relates the water content of a backfill material to its vibrated minimum porosity at that water content (Clark, 1988 a). As the porosity of a fill is directly related to its compressive strength, the placement properties test can be seen as a measure of the quality of backfill aggregates, applicable to both uncemented and cemented backfills. Figure 3.9 shows the placement properties curves of the three typical backfill materials used in the first suite of experiments. It can be seen that classified tailings and full plant tailings have similar minimum porosities, with the classified material marginally higher at about 41 %, compared to 40 % of the FPT. The rod mill product is substantially lower at about 22 %, which is reflected in its high compressive stiffness. As part of another test series, blends of tailings and coarse aggregate, approximately at the 30/70 ratio, have yielded minimum porosities of about 18 %, the lowest achieved in our laboratory with binary mixtures of backfill aggregates. The addition of cement up to about 10 % by dry mass to a fine backfill aggregate changes the minimum porosities by only about 1 to 2 percentage points.

It is important to remember that the minimum vibrated porosities determined in the placement properties test merely serve as a basis to compare different materials, but cannot economically be achieved on placement underground. Experience shows that good actual placement porosities are about 1,25 times higher than the minimum values measured in the laboratory.

3.2.8 Drainage and Permeabilities

The permeability of a backfill material is principally determined by its content of ultra-fine particles with large specific surface areas. Figure 3.10 shows the coefficients of permeability for a wide range of backfills, in terms of their $\leq$ 0,010 mm particle size fraction. The relationship is a logarithmic one and suggests an average minimum permeability of about 8×10^{-4} cm/s for a perfectly classified Witwatersrand tailings material. Most classified tailings lie around 3×10^{-4} cm/s (approximately 26 cm per day).

The permeability of a backfill material determines the rate of drainage from that fill, and thus the time after placement, when the fill has developed sufficient shear strength to sustain static and dynamic compressive loads. The addition of cement to a backfill reduces the initial permeability only slightly and these fills drain quite readily. With the onset of the hydration reaction, however, the permeability gradually drops to very low values, effectively stopping drainage [1] .

3.2.9 Backfill Preparation

Mill tailings for backfilling purposes are generally drawn from the pipeline or launder leading from the metallurgical plant to the tailings dams as dilute slurries with relative

1) This low permeability creates practical problems in the triaxial testing of cured backfills, as the pore water pressure dissipates extremely slowly, often necessitating consolidation periods of several days.

densities of approximately 1,35. Depending on the type of backfill used, this slurry needs to be processed further; its density, however, has to be increased for all types of fill to reduce the quantities of water in the material.

If full tailings are to be used as backfill aggregate, with or without cement addition, they are normally de-watered to slurry relative densities of between 1,65 to 1,90. This would typically include an initial de-watering stage in a thickener, using flocculants, and subsequent filtering on a vacuum belt-filter or in a disk-filter. These are expensive de-watering operations and are mainly used when high backfill tonnage is essential (1).

The preparation of classified tailings is a cheaper procedure, involving the use of hydrocyclones, dimensioned to provide a specific solids cut-size and recovery. A single-stage cyclone typically produces an underflow relative density of between 1,65 and 1,70; uncemented classified tailings in the gold-mining industry have traditionally been placed in this density range. Relative densities of up to 1,85 can be achieved with hydrocyclones and are being advocated for gravitational hydraulic transportation of uncemented fills (Kramers, et al, 1989). Decantation from settling tanks is also being used as a cost-effective process of backfill slurry classification and densification.

Comminuted waste backfill is produced from (- 300 mm) run-of-mine waste by crushing and milling, the latter only if the provision of fines is necessary for hydraulic transportation. Various plant layouts are possible, a typical arrangement would include a primary jaw-crusher, a cone-crusher or secondary jaw-crusher and a rod-mill. If only coarse aggregate is to be produced for tailings/coarse aggregate mixtures, a primary and secondary crusher are used. Other, less conventional options include single-stage wet cone-crushers and autogenous mills; both can produce pumpable products in a single operation. To produce tailings/coarse aggregate backfill mixes, the dry coarse aggregate fraction is blended into the tailings slurry, using proprietary spiral or paddle mixers. The dense backfill is then pumped into suitable bags or paddocks, using positive displacement pumps.

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1) An intermediary option between filtering and full cyclone-classification is offered by de-watering (or 'de-sliming') cyclones, with underflow recoveries of about 85 %.

If a binder is to be used, this can be added to the backfill on surface or underground, dry, or as a slurry. It is thoroughly blended into the backfill with suitable mixing impellers or shearers, or with static in-line mixers, which form part of the pipeline, and contain an arrangement of shear-blades, or simply a length of loose chain to disseminate the binder in the backfill slurry. The provision of dry binder underground can be extremely costly, if pneumatic cement distribution systems are employed. These require special, purpose-designed shaft columns, dedicated compressors, operating with dry, filtered air, and air-agitated cement silos. The transportation of cement to underground mixing stations using bags or bulk cars are much cheaper alternatives.

3.3 Binders

In principle, mine backfills can be cemented with many different adhesives and binders, both organic and inorganic. Considering, however, that relatively expensive organic adhesives are not well suited for the use with hydraulic backfill slurries, and that very high tonnages are normally involved, only cheap inorganic binders are used for bulk cementing. These are, therefore, the binders dealt with in this study.

Binders and adhesives essentially represent un-bonded chemical energy - they are chemically unstable. If they are to be effective, they must react chemically, and the solid, stable reaction products must adhere to the substrate to be bonded. The binders normally considered in backfilling are based on hydraulic calcareous cements, consisting of mixed metal oxides, which react with water to form crystalline or crypto-crystalline hydration products: these adhere to the backfill aggregate particles, bonding them into a more or less rigid, porous structure. Most cements used in backfilling technology are based on the portland cement clinker.

Organic adhesives, by contrast, usually attain strength by the polymerisation of reactive, monomeric compounds. Excess water, as is always present in backfills, is usually detrimental to the curing of adhesives, or prevents it altogether. In addition, polymerisation reactions often liberate toxic volatile compounds which, if released in large quantities, would seriously threaten the underground environment. For these and other reasons, therefore, and on account of their high cost, the use of organic binders in mining is restricted to specialised applications such as the fastening of rock-anchors in bore-holes; in bulk backfill cementation they have no practical application today.

The chemistry of cementitious calcareous binders can be understood as comprising two converse aspects:

* The burning of the raw-materials to produce the cement clinker, and
* The hydration of the finely ground cement to bond the aggregate.

Essentially, in the first phase, the volatiles, i e water and carbon dioxide, are expelled from the raw-materials by heat. In the second phase, when required, those components are re-introduced to react with the cement, forming stable compounds, often similar to the raw-materials. The heat required to activate the raw-materials can be natural, e g volcanic - the source of natural pozzolans, or more commonly artificial, in fired kilns.

3.3.1 Chemical Composition of Binder Components

Table 3.6 lists the chemical compositions of a range of cements and pozzolans (cement extenders).

Table 3.6 X-ray fluorescence analyses of commonly used cements and pozzolans. As [%] oxides.

Oxide	OPC	PBFC	HAC	OPC/FA 1:2	FA	GGBS	CSF
CaO	63.80	48.80	36.70	28.20	10.40	31.40	1.04
SiO_2	22.70	30.00	3.80	37.77	45.30	35.20	88.70
Al_2O_3	4.70	9.60	39.80	19.70	27.20	17.20	1.70
MgO	2.50	5.60	0.40	2.83	3.00	11.30	1.50
Fe_2O_3	2.66	1.8[illegible]	15.30	3.98	4.64	0.66	1.93
MnO	0.14	0.75	0.01		0.03	0.83	0.17
Cr_2O_3	0.01	0.01	0.01			0.01	0.01
V_2O_5	0.01	0.01	0.04			0.01	0.01
TiO_2	0.37	0.46	1.71		1.57	0.64	0.04
P_2O_5	0.12	0.08	0.06		1.54	0.01	0.11
Na_2O	0.10	0.10	0.20		0.50	0.40	0.10
K_2O	0.15	0.80	0.08		1.10	1.08	1.50
L.O.I.	0.73	0.05	1.04		1.62	-1.26	3.48
Cl^-	0.04	0.01	0.01		0.02	0.01	0.02
$SO_3^=$	2.16	1.60	0.04		0.70	1.39	0.20

The following abbreviations are used in the table:

OPC	ordinary portland cement,
PBFC	portland blastfurnace cement,
HAC	high alumina cement,
FA	fly ash,
GGBS	ground granulated blastfurnace slag and
CSF	condensed silica fume.

The compositional tolerances of commercial cements are invariably adhered to in their manufacture, usually even exceeded with potentially detrimental elements such as magnesium and the alkalis. Similarly, the fineness of grind (Blaine number) normally exceeds standard minimum values.

The chemical analysis of a binder, expressed as oxide percentages, is not sufficient for the evaluation of a binder's quality. Several other factors, including the cement mineralogical composition, the proportion of vitreous compounds and the average particle size must also be considered. The oxide analysis is, however, useful in the determination of the amount of binder used in a cemented backfill, provided standard samples of all raw materials are available for calibration.

In the search for cheaper binders for mine backfilling, it has variously been suggested to produce a sub-standard portland cement, since, for instance, sulphate resistance is of little practical value in backfills. The manufacture of cement is such a large, established industry, however, that this would not be viable: the largest components of the cost of cement are energy input and transportation; savings on raw materials would only be marginal. In addition, any change in clinker composition would necessitate higher calcining temperatures, more than negating any savings in raw material costs.

3.3.2 Hydration Reactions

At a cursory level, the chemistry of hydraulic cement reactions appears straight-forward; detailed reaction-kinetics and -phases are, however, by no means fully understood (Bensted, 1983). On account of the complex stoichiometry of the phases involved, it has become customary in the cement industry to describe the various compounds with abbreviations of the included oxides (Table 3.7).

Table 3.7 Nomenclature used in the cement industry for oxidic components.

CaO	=	C
SiO_2	=	S
Al_2O_3	=	A
Fe_2O_3	=	F
FeO	=	f
MgO	=	M
K_2O	=	K
H_2O	=	H
SO_3^{2-}	=	$\bar{S}$

Calcium hydroxide, $Ca(OH)_2$ is thus designated as CH. A bar over a letter, e g $\bar{S}$, denotes the chemical element symbol, here: sulphur.

Since practically all active cements used in cemented backfills are based on the portland cement clinker, the hydration reactions of those compounds are the most relevant (Bensted, 1983, Krüger, 1990), they are given below.

$$2C_3S + 6H \rightarrow C_3S_2H_3 + 3CH$$

(C_3S + water → calcium silicate hydrate + calcium hydroxide)

$$2C_2S + 4H \rightarrow C_3S_2H_3 + CH$$

(C_3S + water → calcium silicate hydrate + calcium hydroxide)

$$C_3A + CH + 12H \rightarrow C_4AH_{13}$$

(C_3A + lime + water → tetracalcium aluminate hydrate)

$$C_3A + 3C\bar{S}H_2 + 26H \rightarrow C_3A(C\bar{S})_3H_{32}$$

(C_3A + gypsum + water → ettringite)

$$C_3A + C\bar{S}H_2 + 10H \rightarrow C_3A(C\bar{S})H_{12}$$

(C_3A + gypsum + water → calcium monosulfo-aluminate hydrate)

$$C_4AF + 4CH + 22H \rightarrow C_4AH_{13} + C_4FH_{13}$$

(calcium aluminoferrite + lime + water → iron (III) substituted compounds and aluminium analogues)

The most important compounds formed are CSH (calcium silicate hydrate), the main 'cementing' crystals, and CH, the excess calcium hydroxide, which can activate pozzolans added to the cement (Section 3.3.4). Most of the reaction components and products form very complex and often non-stoichiometric mixtures, the precise compositions of which are still being researched.

Even though cementation is a chemical process and the reactions are largely determined by the material's composition, two important physical factors critically influence the performance of cements; they control the rate of reaction of the cement with water:

* The fineness of grind of the cement, i e its specific surface area; this provides the necessary interface for the reactions to occur at a satisfactory rate; small particles also allow the complete reaction of the material.

* The proportion of minerals in the vitreous or crypto-crystalline state, which is primarily determined by the cooling rate of the clinker: the water-soluble free lime present in cement forms a highly alkaline solution which is able to attack the glassy components chemically, if these minerals had crystalised, the much stronger ionic bonds in the crystals would slow down dissolution and virtually prevent cementation.

The fineness of grind of cement and pozzolans is expressed in terms of the material's specific surface area, i e in [cm^2/g]. In South Africa, this quantity is usually determined by the modified Blaine air permeability method (SABS 471: 1971). Table 3.8 lists the ranges of specific surface areas of South African cements and pozzolans. Figure 3.11 shows the particle size distributions of several South African cements and pozzolans.

Table 3.8 Specific surface areas of South African cements and pozzolans. [cm^2/g].

Material	Surface Area
OPC	2800-3200 *
PBFC	3100-3800
RHC	4000-4500
FA	≃ 4500
Slagment	3500-3800

* Average particle size: 0,020 - 0,02? mm

3.3.3 Hydraulic Cements

There are two groups of basic calc...eous binders (Krüger, 1990), both of which have been used in backfills:

* *Non-hydraulic* lime-based cements, and
* *Hydraulic* cements, defined as those cements which can cure under water.

In the latter materials lime is combined with reactive silica to form calcium silicate hydrates on hydration. This section is concerned with the two most common hydraulic cements used in mine backfilling.

Ordinary portland cement

The chemical composition of ordinary portland cement is given in Section 3.3.1, the more important mineral composition is listed in Table 3.9 below. To distinguish the act... occurring minerals C_3S and C_2S in the cement from the pure synthetic compounds, the industrial equivalents are also named *Alite* and *Belite*, respectively.

Table 3.9 Mineral composition range of South African ordinary portland cements [%] (Addis, 1987, Fulton, 1977).

Mineral	[%]
C_3S	35-55
C_2S	20-40
C_3A	5-12
C_4AF	5-10
$C\bar{S}$	4-7
C	0,5-2,5
M	0,3-4,0

The strength development and relative contributions of the above cement minerals to the total strength of a concrete or cemented backfill differs drastically. By far the largest contribution is made by C_3S; it has all the essential qualities of portland cement, contributes early strength and high ultimate strength. C_2S develops strength much slower, but eventually, after years, reaches similar ultimate strengths as does C_3S. C_3A develops negligible compressive strength but generates large amounts of heat during the first few days of its hydration. C_4AF develops little strength, it is, however, valuable as a flux in the manufacture of cement. Gypsum, $CaSO_4 \cdot 2H_2O$ ($C\bar{S}$) is added to the portland cement clinker to control the rate of setting of the cement paste.

Portland blastfurnace cement

Portland blastfurnace cement, PBFC, is an inter-ground mixture of equal amounts of ordinary portland cement and ground granulated blastfurnace slag. Since the slag components are physically harder than the cement minerals, inter-grinding results in a material finer than OPC (Table 3.8) with a corresponding increase in reactivity.

Portland blastfurnace cement has very similar curing properties to OPC, but is said to be more tolerant of high water contents - an advantage in hydraulically placed backfills. This similarity in properties is predictably also reflected in the cement's price, which is only marginally lower than that of ordinary portland cement. The hydration reactions of the slag component in PBFC are identical to those of slags in blends with portland cement; the binder is chemically no different to such blends, it is only ground finer.

3.3.4 Pozzolans

Pozzolans are a group of siliceous inorganic compounds, which need the addition of soluble calcium compounds (usually calcium hydroxide) to develop cementitious properties of their own (1). They do possess potential chemical energy, but their composition is incomplete for the formation of strong hydration products. Since pozzolans are cheap and need only small amounts of cement to activate them, they are very important components of backfill binders (Krüger, 1990).

In most cemented backfill systems, pozzolans comprise a significant proportion of the binders used; indeed, in many materials the cost of the pozzolan fraction exceeds that of the active cement. Low-cost pozzolans are, therefore, essential to the design of more cost-effective cemented backfills, irrespective of the required fill strength or setting time. Listed below are some common artificial pozzolans:

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1) The name is derived from the town of Pozzuoli near Naples in Italy. Civil engineers in ancient Rome used the volcanic ash, *pulvis puteolanus*, excavated near the town, with lime to produce a durable concrete.

* Fly ash (FA)
* Pulverised fuel ash (PFA)
* Ground granulated blastfurnace slag (GGBS)
* Condensed silica fume (CSF)
* Granulated base-metal smelter slags

All of these materials are by-products from burning coal or from pyro-metallurgical processes. Since high energy input is essential for the formation of pozzolans, the list of potential materials is limited. Natural pozzolans, such as volcanic ashes and tuffs are not normally used in binders, as their occurrence is generally limited to regions of active volcanism.

The preservation of the vitreous phases of mineral components is a further important requirement for the formation of active pozzolans: not only is it essential to have a high temperature environment, in which the right mineral components melt and fuse, but this molten slag must be cooled ..pidly, to preserve the compounds in their glassy, reactive state.

In coal fired power-stations, burning fine milled coal, the ash escapes from the boilers with the flue gases and cools rapidly in the heat exchangers and the elec ro-static precipitators, thereby preserving most of the slag in the glassy state (normally 75 % to 80 %). Metallurgical slags are commonly poured onto dumps and left to cool slowly - the slag minerals crystalise and lose their reactivity. For metallurgical slags to remain reactive, the melt has to be granulated i e chilled in water or with compressed air, and subsequently ground fine to increase the surface available for fast reaction.

When very finely divided, and thus chemically reactive, pozzolanic silica enters the hydration reactions, compou..ds similar to those produced in portland cements result:

$$3\,Ca(OH)_2 + 2\,SiO_2 \rightarrow 3\,CaO{\cdot}SiO_2{\cdot}3\,H_2O$$

(building lime + pozzolan → calcium silicate hydrate)

In the above reaction, the role of building lime is usually taken by the excess calcium hydroxide liberated from portland cement in its hydration reaction. Because of the complete composition of portland cement and the fast formation of CSH, the relatively slow pozzolanic reaction can be tolerated. Pure lime does not provide that early strength and the pozzolanic reactions proceed at their slow pace, providing equivalent compressive strengths several weeks later.

Fly ash, pulverised fuel ash

The chemical composition of fly ash is given in Section 3.3.1. Mineralogically, fly ash consists mainly of aluminosilicate glass (75 % to 80 %) and mullite (A_3S_2). Fineness is the single most important determinant of the quality of fly ash, the finer the ash, the higher is its pozzolanic activity (1). As a standard of quality, a maximum of 20 % by mass larger than 0,045 mm has been proposed; the major producer in South Africa supplies at about 10 % $\leq$ 0,045 mm.

In South Africa, sufficient amounts of high quality fly ash are produced to obviate the need for grinding bottom ash (Krüger, 1990). In power stations, fly ash is normally extracted in a series of electrostatic precipitators, with the finest product accumulating in the last field. The proportion of this, however, amounts to only a few per cent of the total mass of fly ash produced. The combined ash from all fields is, therefore, air classified to produce a sufficient amount of high quality ash.

Pulverised fuel ash, PFA, is the ground residue of coal-fired power stations. It consists of fly ash, collected in the filters of the stations, blended with bottom ash from the boilers. Since bottom ash is fused into coarse lumps, the material has to be ground to increase the materials specific surface area and thereby its reactivity.

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1) Small slag droplets cool very rapidly to ash and preserve most of the vitreous phase. Additionally, they have very high specific surface areas.

Ground granulated blastfurnace slag

Ground granulated blastfurnace slag (GGBS) is a material produced by quenching (granulating) blastfurnace slag with air or water to preserve its vitreous phase and thereby its pozzolanic activity. It consists mainly of calcium-magnesium-aluminium-silicate glass. Its pozzolanic reactions differ from FA in that it does not react directly with calcium hydroxide, but that the lime acts like a catalyst; the main reaction product is CSH, as with OPC.

GGBS is a much more reactive, but also more expensive, pozzolan than fly ash. It can be blended with portland cement at the binder mixing stage, or inter-ground with OPC by the manufacturers (portland blastfurnace cement, see Section 3.3.3). GGBS by itself often shows weakly cementitious properties, but this cannot be relied .. and it is normally used in conjunction with OPC.

3.3.5 Lime

Lime is a non-hydraulic cement, which under normal circumstances cannot cure under water, as it needs atmospheric carbon dioxide to harden. This is a clear disadvantage in backfills which are normally placed at very high water contents and drain slowly. There have been several attempts to use lime as an activator for pozzolans, but very slow curing rates, and a price nearly identical to OPC, have effectively ruled out its use in cemented backfills. The reactions governing the activation and the hardening of lime are given below.

Limestone is burned at 900 °C:

$$CaCO_3 \rightarrow CaO + CO_2$$

(Limestone → Quicklime)

Quicklime (or unslaked lime) is slaked to building lime, which is weakly cementing:

$$CaO + H_2O \rightarrow Ca(OH)_2$$

(Quicklime → Building lime)

In time, building lime is carbonised with atmospheric carbon dioxide and reverts to the strongly cementing agent calcium carbonate:

$$Ca(OH)_2 + CO_2 \rightarrow CaCO_3 + H_2O$$

(Building lime → Limestone)

When impure limestone, containing some clay, is burned at 900 °C, hydraulic lime is formed, which produces pozzolanic hydration reactions.

3.3.6 Other Binders and Additives

Recently, the use of fast curing binder systems has become popular in the South African gold and platinum mining industries. These mainly 'silicated backfills' are expensive alternatives, with an accelerator based on sodium or potassium silicate solution. Sodium (or potassium) silicate reacts instantaneously with excess calcium hydroxide, produced in the portland cement hydration reaction, to form calcium silicate, a hard, stable cementing compound. Calcium silicate tends to form cellular structures within the backfill, physically trapping liquid water.

Where needed, such binders can be of great help, providing hangingwall support within hours and stopping water run-off, a continuing annoyance to underground mining personnel. The entrapment of water in the backfill has, however, serious disadvantages:

* The backfill retains a high porosity with the associated soft compression response.

* The cellular silicate structure within the backfill strongly reduces the permeability of the material and if saturated, the dissipation of pore water pressure can be inhibited.

* When weakly cemented and placed at low porosities the backfill could fail by liquefaction (Blight, et al, 1977).

* Upon the formation of shear zones within a failing backfill body, the entrapped water is liberated, lubricating the shear plane, thus facilitating slip failure.

It has been shown in practice that in narrow stopes (< 1,5 m), equivalent local support benefits can be provided with backfills using conventional binder blends of portland cement with pozzolans. In wide stopes (> 1,5 m), however, with comparatively low closure, silicated backfill with its self standing ability and low shrinkage can provide fast and effective local support.

3.4 Backfill Slurries

This section is mainly concerned with the engineering properties of prepared cemented backfill slurries or pastes. Once a backfill composition has been decided upon, it is a matter of metallurgical engineering practice to prepare the constituents, blend them in the right proportions, and transport the slurry in pipelines to its destination underground.

In order to produce an optimal backfill material for any given specification, a number of compromises have to be made, affecting all aspects of the backfill system. This applies as much to the handling of hydraulic slurries, as it does elsewhere in the process of backfill design. All properties of hydraulic backfills are determined to a large extent by the amount of water in the fill at any one stage in the system and it is really the addition and the management of water which is the crucial issue in the successful production of high quality backfill.

3.4.1 Water/Solids Ratio, Relative Slurry Density

The water/solids ratio is a measure for the amount of water in a backfill and as such it is one of the most crucial parameters in backfilling technology. As the term implies, it is the ratio of the total mass of water in a backfill to the total mass of solids components, both aggregate and binder, in the material. In practice, the ratio is often expressed in terms of the slurry relative density (or slurry RD), taking the solids relative density into account (1).

The practical range of slurry relative densities for quartzitic tailings is largely determined by hydraulic transportation requirements. Classified tailings slurries have traditionally been gravitated underground at relative densities of about 1,65 , but extensive work by Kramers, et al (1989) suggests safe operation at up to RD 1,75. Cemented full plant tailings have been gravitated at up to RD 1,95. Where sufficient gravitational head is not available, centrifugal or positive displacement pumps have to be employed to move the backfill over large horizontal distances. Comminuted waste backfills have been successfully pumped and placed at relative densities of up to 2,15.

3.4.2 Cement/Water Ratio

The term *cement/water ratio* originated in concrete technology and refers to the ratio of the total binder content in a concrete or cemented backfill to the total water content in the material. The ratio of the two constituents has the most profound influence on the strength of the cured product. Basically, the higher the ratio, the stronger the cured concrete or cemented backfill. This is provided the minimum amount of water necessary for the hydration reaction is present and evenly distributed in the paste, for portland cement theoretically 28 %, i e a cement/water ratio of 3,57 (Addis, 1987). In practice, this is very difficult, as the mixture would be almost dry and could not be handled by conventional means, let alone, be mixed homogeneously.

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1) Other related measures are: moisture content and volume- or mass concentration of solids in the slurry.

The cement/water ratio is mainly of chemical significance and its influence on the strength of cemented backfill materials will be addressed in detail in Chapter 4. Its contribution to the physical properties of slurries is only indirect, in that the amount of binder in a slurry has an important influence on its pumping pressure losses (next Section).

3.4.3 Hydraulic Transport Behaviour

Most mine backfills are transported and placed as hydraulic slurries. There are several reasons for this:

* most backfill components are processed with water and its removal is expensive,
* slurry pumping provides a continuous means of transporting materials,
* hydraulic transport technology is well established, and
* hydraulic slurries can flow under gravity all the way into stopes.

The flow resistance of a backfill slurry in a pipe is determined by a number of variables, including: the particle size distribution of the backfill, slurry density, flow velocity as well as pipe diameter and geometry. The particle size distribution influences hydraulic transportation in two sometimes conflicting ways:

* the finer a backfill material is, the higher is its internal friction and with that, its pressure loss; on the other hand,
* the slow-settling fines and ultra-fines effectively increase the relative density of a backfill's fluid component, enabling it to suspend larger backfill grains.

These opposing effects suggest that for most backfills, a compromise will have to be found in terms of the optimal particle size distribution for any given transportation task. The rheology of backfill slurries and pastes (1) is not yet well understood and pressure losses of slurries in pipelines cannot be predicted reliably from laboratory measurements, let alone from the many published theories. The detailed design of reticulation systems still has to be based on practical, preferably full-scale, measurements (Kramers, et al, 1989).

In general, pressure losses increase with slurry density, increase with flow velocity, decrease with pipe diameter, and increase with fines and ultra-fines contents. As an indication of the relative differences of pipeline pressure losses, all other factors being constant, it has been reported that typical Witwatersrand full plant tailings show about ten times higher pressure losses than do classified tailings, with well designed crushed waste products being even lower than classified tailings. Kramers, et al (1989), however, point out that these comparisons do not reflect realistic conditions, since tailings/coarse aggregate slurries could not be pumped at densities as low as those used for classified tailings.

More realistic indications of relative slurry pressure losses are provided by Kramers, et al (1992), in comparing the slurry relative densities, at which the flow regime of each respective slurry type changes from settling to non-settling. Figure 3.12 shows slurry pressure loss graphs of four different aggregate types in the transition region. All backfills were of quartzitic type (i e Witwatersrand materials), contained no cementitious additives, and were tested in a 100 mm diameter pipeline at a constant flow velocity. The inflexion point in each curve indicates the optimal transportation slurry relative density. In this presentation it can now be seen that comminuted waste backfill may theoretically well show lower pressure losses than classified tailings at some intermediate relative slurry density (e g RD 1,85), but it is also clear that the two materials occupy quite different fields in the graph. Even though the classified tailings shows a pressure loss of only about 1,5 kPa/m, as com-

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1) Rheologically, pastes are mixtures of solids and liquids, which exhibit *Bingham-plastic* and *pseudo-plastic* flow behaviour. In the placement properties graph, slurries and pastes can loosely be defined as two different phases of backfill/water mixtures - mixtures at water/solids ratios near the minimum porosity region are described as pastes, those of higher water content as slurries (Clark, 1988 a).

pared to the 5 kPa/m of the comminuted waste, the latter is tested at a relative slurry density of 2,0 , transporting a much higher load of solids than the CCT slurry at RD 1,78.

It is a commonly held believe in the backfill community, that the addition of cement to a backfill slurry will automatically reduce pressure losses of the material, owing to a lubrication effect of the cement, similar to that of chemical pumping aids. It is known that some cementitious binder components improve the pumpability of concretes and it was therefore assumed that the same applied to backfill slurries. In a series of full-scale measurements, quite the opposite was found: the addition of a cementitious binder increased the backfill slurry viscosity, and with that, its pressure losses (Ilgner, 1991). Figure 3.13 shows normalised pressure loss curves of a suite of platinum tailings backfills, containing varying amounts of an OPC/GGBS binder blend. The slurry relative densities of the backfills were kept at a constant RD of 2,00 and the measurements were carried out in a 100 mm diameter pipeline. It can be seen, that the addition of increasing amounts of binder progressively increased the pressure losses up to a slurry velocity of about 3 m/s. This is explained by the increased internal friction caused by the addition of ultra-fine binder components. At a slurry velocity of 3 m/s, the flow regime changes from laminar to turbulent and the additional ultra-fines no longer makes any difference to the relative pressure losses. This velocity is, however, beyond the 2 to 2,5 m/s preferred to minimise pipe wear.

In horizontal hydraulic transport, full plant tailings and comminuted waste products at high relative slurry densities generally have to be moved with positive displacement pumps; classified tailings at similar densities have lower pressure losses and can often be moved with cheaper centrifugal pumps. If fill slurries can be gravitated down shaft columns or bore-holes, the resulting hydraulic head can transport slurries over long distances. Figure 3.14 relates the relative horizontal distances, through which backfill slurries of different densities and specific pressure losses can be transported, given a particular hydraulic head.

3.4.4 Slurry Densification

The final tailings slurries from metallurgical plants are normally pumped away for disposal at relative densities between 1,20 and 1,35. These very dilute slurries have to be densified, both for tailings disposal and for backfilling, as the large amounts of water run-off upon placement would create serious problems. If used as backfills underground, the placement porosities and cemented strengths of materials dumped at such low densities would, moreover, be very poor.

A number of options are available for densification and, depending on the process, the tailings size distribution can remain unaltered, or some degree of classification can occur. This can have important implications for the materials' properties. Most commonly, slurries are treated with flocculants and are allowed to settle in large thickeners. In this way, slurry relative densities of about 1,75 can be achieved routinely. In normal operation, the decant water is clear, and filtered through a sand filter, re-enters the metallurgical plant as process water.

Another method used to de-water tailings slurries is filtration. A wide variety of filtering equipment is available, including vacuum belt-filters, drum-filters, and pressurised disk-filters. With this equipment, slurries can be de-watered to a nearly dry consistency (typically to 14 % residual moisture). The clarity of the filtrate is dependent on the filter cloth used. Flocculants and thickeners are often used in conjunction with filters to improve their efficiency.

De-watering hydrocyclones are yet another method to densify slurries. This method results in a particle size classification of the product. Underflow densities of over 1,80 can be achieved. This is a cost-effective method of de-watering, particularly since most operators prefer some degree of product classification. The problematic disposal of the ultra-fine overflow product has to be considered, however.

One of the simplest methods is the densification of slurry by means of settlement and decantation in tall settling tanks. In principle, this method is similar to settling in thickeners, but avoids the large capital outlay and space requirement of thickeners. If properly designed, this batch process can achieve an adequate rate of densification, coupled with controlled classification by the decantation of super-natant fines. This method, in conjunction with limited hydrocyclone classification, provides a very efficient means of de-watering backfill slurries.

3.4.5 Placement Properties and Permeability

On placement underground, full plant tailings drain very slowly, often remaining hydraulic for several days, even weeks (1), presenting the dangerous possibility of liquefaction under seismic loading and mud-rushes, should the containing barricades or geo-fabric bags fail. Classified tailings drain rapidly and usually attain sufficient shear strengths (> 5 kPa, (Copeland, 1990)) after only a few hours. Comminuted waste products, and mixtures thereof, drain similarly fast, with rapid shear-strength development.

The geo-fabric bags normally used in the containment of backfills underground are an important factor in their drainage behaviour. A wide variety of fabrics with different permeabilities are being used in the industry; a review of their influence on fill placement is given by Copeland (1990). The use of flocculants in backfills underground has at times been shown to be ineffective, dosing is often irregular and excessive. In those cases, solids loss from the backfill is barely improved, and the backfill porosity is usually increased.

3.4.6 Post-placement Shrinkage

The shrinkage of backfill after placement is one of the practical problems of backfill support. Volumetric shrinkage of backfill is due to the drainage of water, as well as the loss of fines and ultra-fines from the backfill with the drainage water after the termination of placement. The reduction of the placed backfill volume results in a gap between the backfill and the hangingwall; it is most pronounced in near horizontal stopes.

The problem is particularly acute in local support applications, where intimate contact with the hangingwall is required to prevent the deterioration of the fractured hangingwall. The most effective means to minimise shrinkage include the careful matching of the backfill material properties and the retaining geo-fabric, the placement of backfill at the highest practical slurry relative density and the addition of binders.

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1) In an experimental stope in a platinum mine, the author has seen very fine grained full plant tailings which had been placed two years earlier. The material was still saturated and water seeped out upon slight vibration of the geotextile bag.

3.5 Summary and Conclusions

In Chapter 3, a detailed catalogue has been provided of the characteristics of the material components which make up hydraulically placed backfills in deep South African hard-rock mines. The results of the study on the backfill materials properties and backfill slurries formed an integral part of the work on the strengths of cemented backfills, as it will be presented in the next chapter. But they also provide a useful catalogue of the properties of South African backfill materials and highlight some of the concerns regarding detrimental material properties.

The most commonly used backfill aggregates are: full plant tailings, containing on average, about 20 % ultra-fines; classified tailings, with about 5 % ultra-fines; and crushed waste, characterised by a well graded size distribution and a large top grain size. The various size distributions are the major determinants of these aggregates' physical characteristics: they determine their final placement porosities, their drainage rates and water retention, as well as their hydraulic transportation characteristics. The discussion also touched on the main preparation processes for backfill aggregates, including hydrocyclone classification and waste rock comminution.

Backfill binders typically consist of an active cement, such as ordinary portland cement, and a cement extender (or pozzolan), such as fly ash or ground, granulated blastfurnace slag. The purpose of the binder is to provide backfill with cohesion and this is possible by the cement hydration reactions, which produce two most important compounds: calcium silicate hydrate (or CSH), the main cementation compound, and calcium hydroxide, which makes possible the additional hydration of otherwise inactive pozzolans, such as fly ash.

The section on backfill slurries included some detail on the hydraulic transportation characteristics of various backfill types. It was shown that pipeline pressure losses of classified tailings slurries are considerably lower than those of full plant tailings. This fact, combined with the vastly improved drainage rate and shear strength development of classified tailings, compared to full plant tailings, explains why nearly all mines practice backfill classification.

A number of factors determining backfill quality were discussed in the chapter, but only a few can be classified as primary, causal agents, crucial in the design of hydraulically placed backfill support; two of them stand out as particularly critical.

Ultra-fines Content This is probably the single most important material characteristic of backfill aggregates, affecting almost every backfill property. Mineral particles smaller than about 0,010 mm have high specific surface areas and, therefore, a high water demand which results in an unfavourable cement/water ratio and poor cemented backfill strength. The large surface areas of ultra-fine particles, furthermore, require large amounts of cementitious additives to bond them effectively. The high water retention of ultra-fines retards shear strength development in placed backfills, a problem exacerbated by the particles blocking drainage paths, resulting in very low permeabilities and, if not compacted, high porosities. All these factors combine to present the potential of backfill liquefaction under cyclic loading such as seismic tremors or blast vibration. Finally, ultra-fine particles significantly increase pipeline pressure losses of backfill slurries and after placement intensify the practical problem of fines losses through geo-textiles into mine workings. It is indicated that ultra-fines contents of more than 5 % by mass are detrimental to the quality of all types of backfills.

Phyllo-silicates Among the common rock-forming minerals in the investigated backfill aggregates, phyllo-silicates are the only detrimental mineral components. This mineral class includes micas, talc and clay minerals, all of which share a crystallographic structure, characterised by weakly bonded laminae. Most of the phyllo-silicates are soft minerals and are, therefore, readily comminuted in the milling process and during backfill slurry agitation. Phyllo-silicates cause two kinds of problems, those related to their high intrinsic surface areas and those caused by their low shear strength. The soft mineral laminae of clays and talc are easily separated mechanically; therefore, they tend to concentrate in the ultra-fine particle size fractions with all the implications of high surface areas described in the previous paragraph. Some of the clay minerals, notably montmorillonite and smectite, show the additional characteristic of expanding with the absorption of water ('heaving clays'). This weakens them further and enlarges their specific surface areas many-fold. Within the shear zones of stressed backfill bodies, phyllo-silicates present weak points where slip is readily initiated; this is exacerbated by the re-orientation of these mineral grains parallel to the shear plane. Depending on the mineral species, phyllo-silicate contents of over 10 % can be detrimental in backfills.

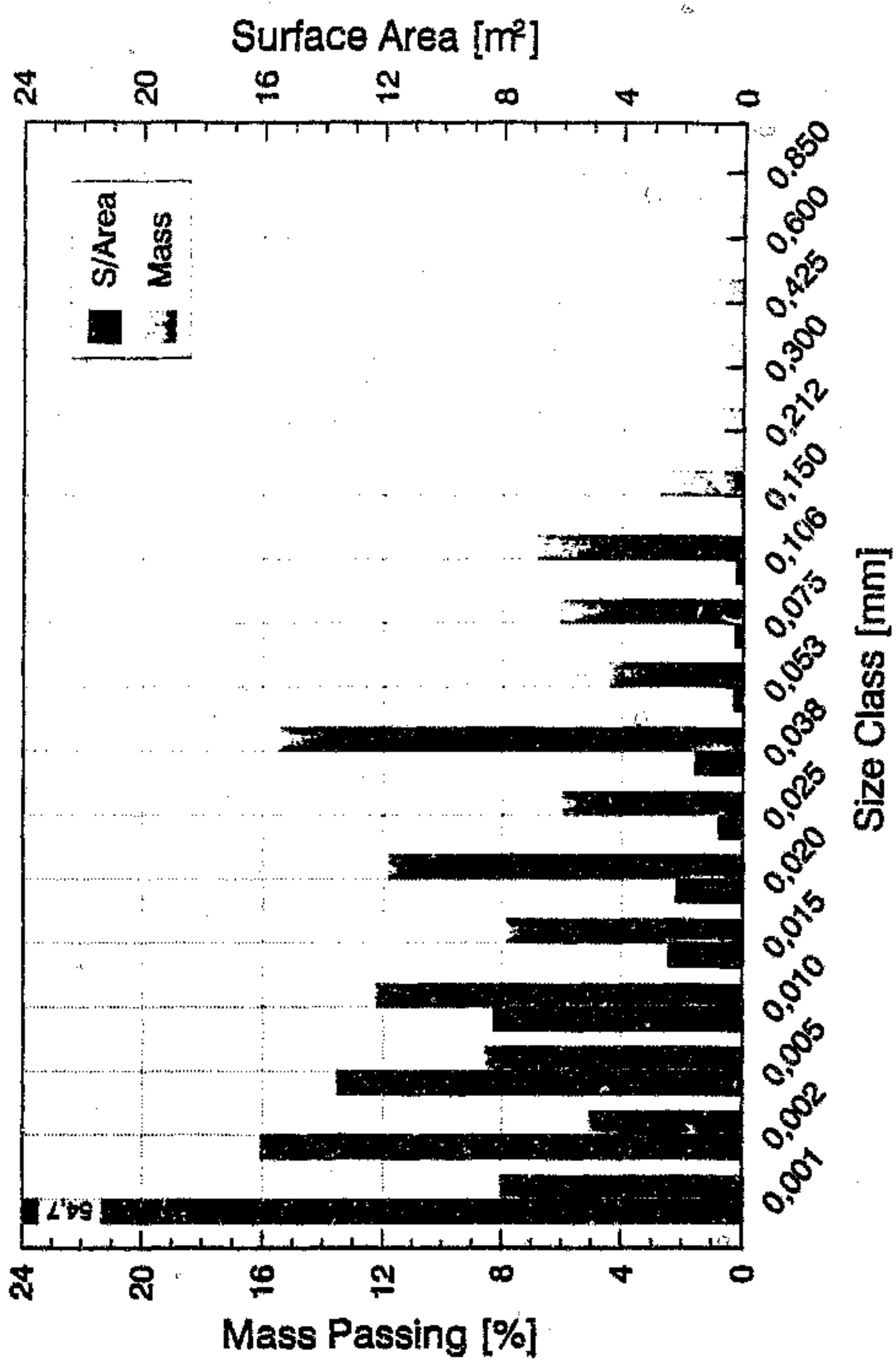

Figure 3.1 Mass-size distribution [%] and total surface areas [m^2] of size classes of a full plant tailings sample, assuming sub-angular particle shape.

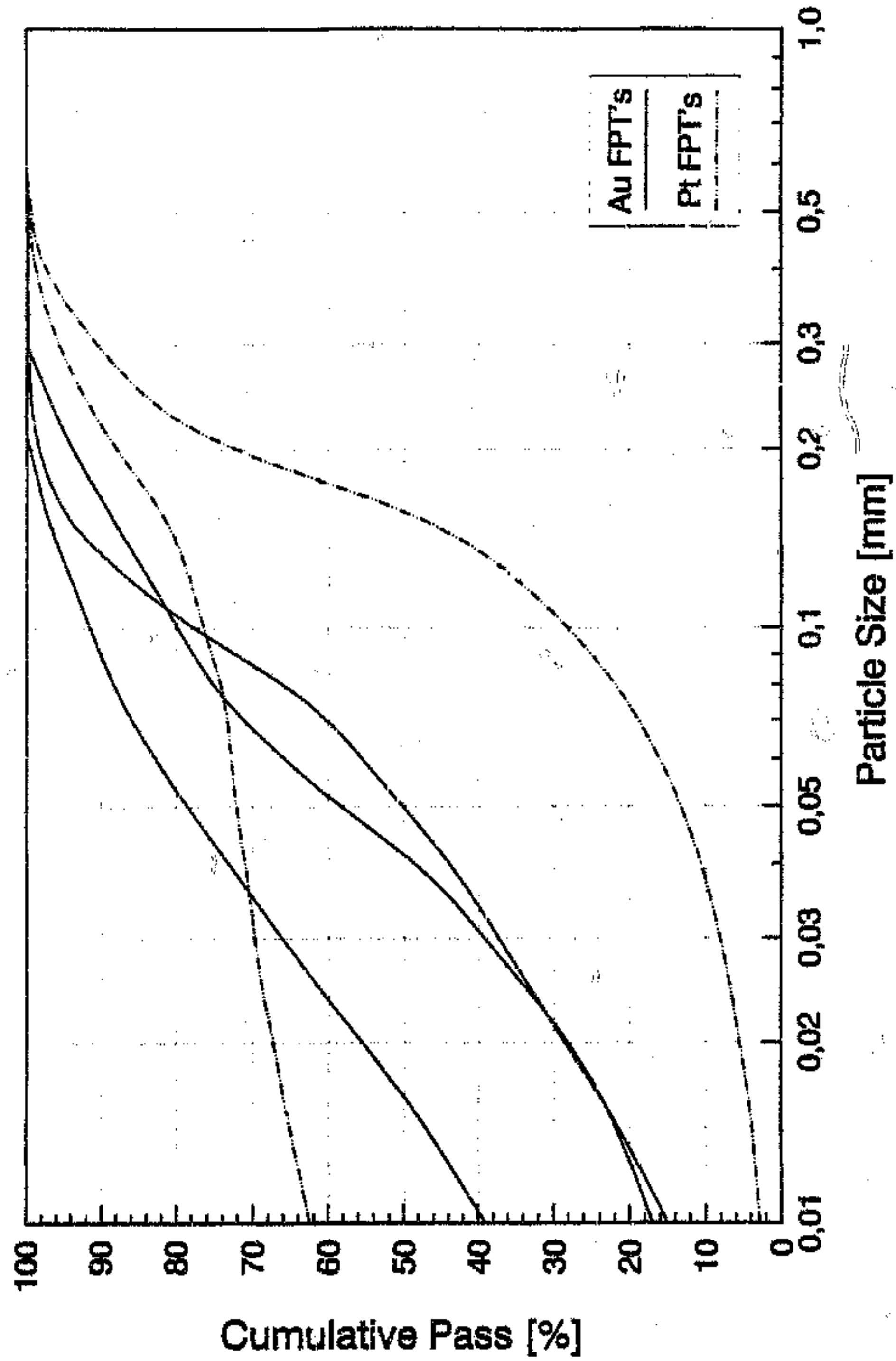

Figure 3.2 Particle size distribution ranges of gold and platinum full plant tailings (FPT) [Cumulative Pass %].

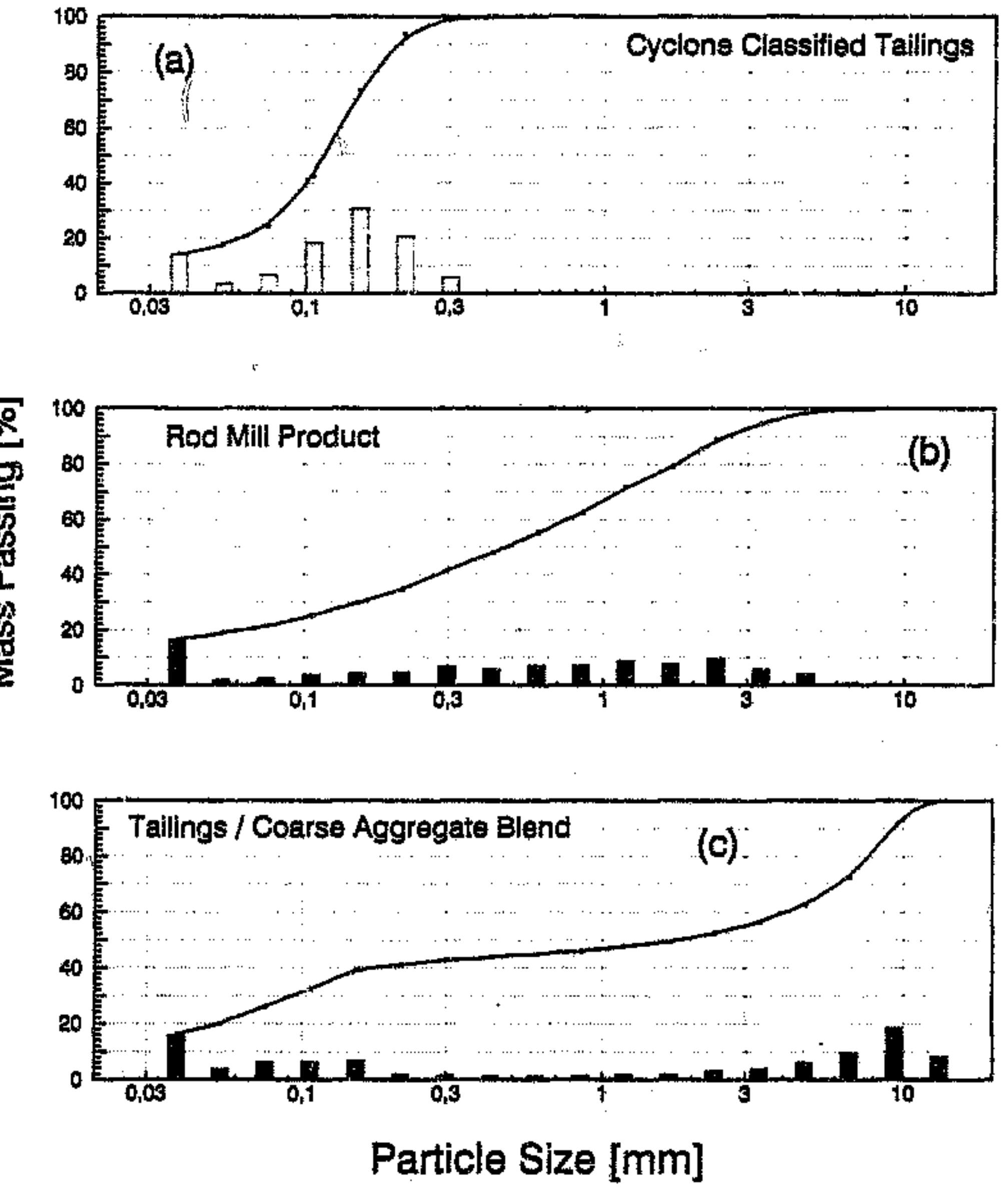

Figure 3.3 Basic particle size distribution types.

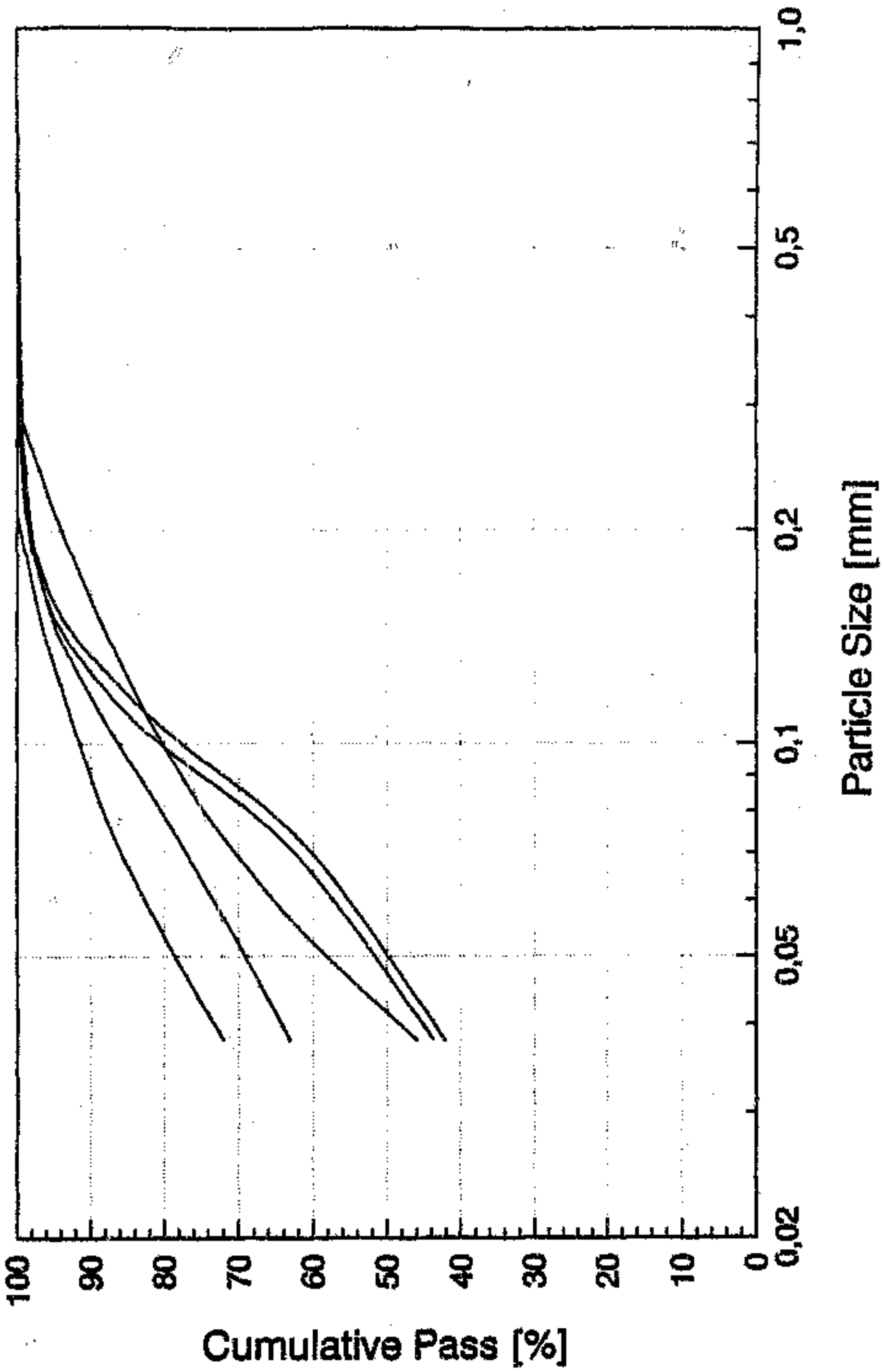

Figure 3.4 Particle size distributions of a range of full plant tailings from the gold mining industry [Cumulative Pass %].

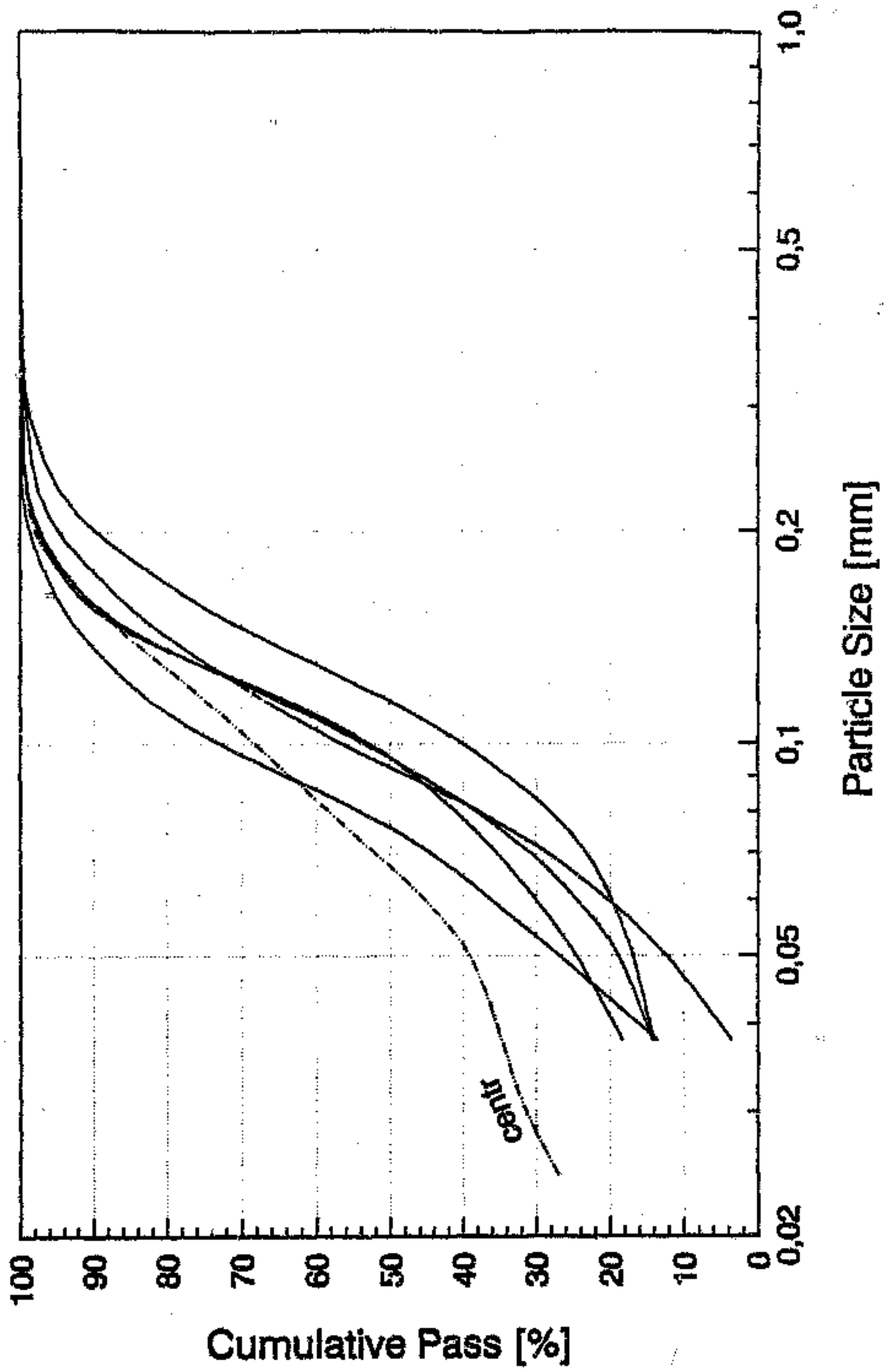

Figure 3.5 Particle size distributions of a range of classified tailings from the gold mining industry [Cumulative Pass %].

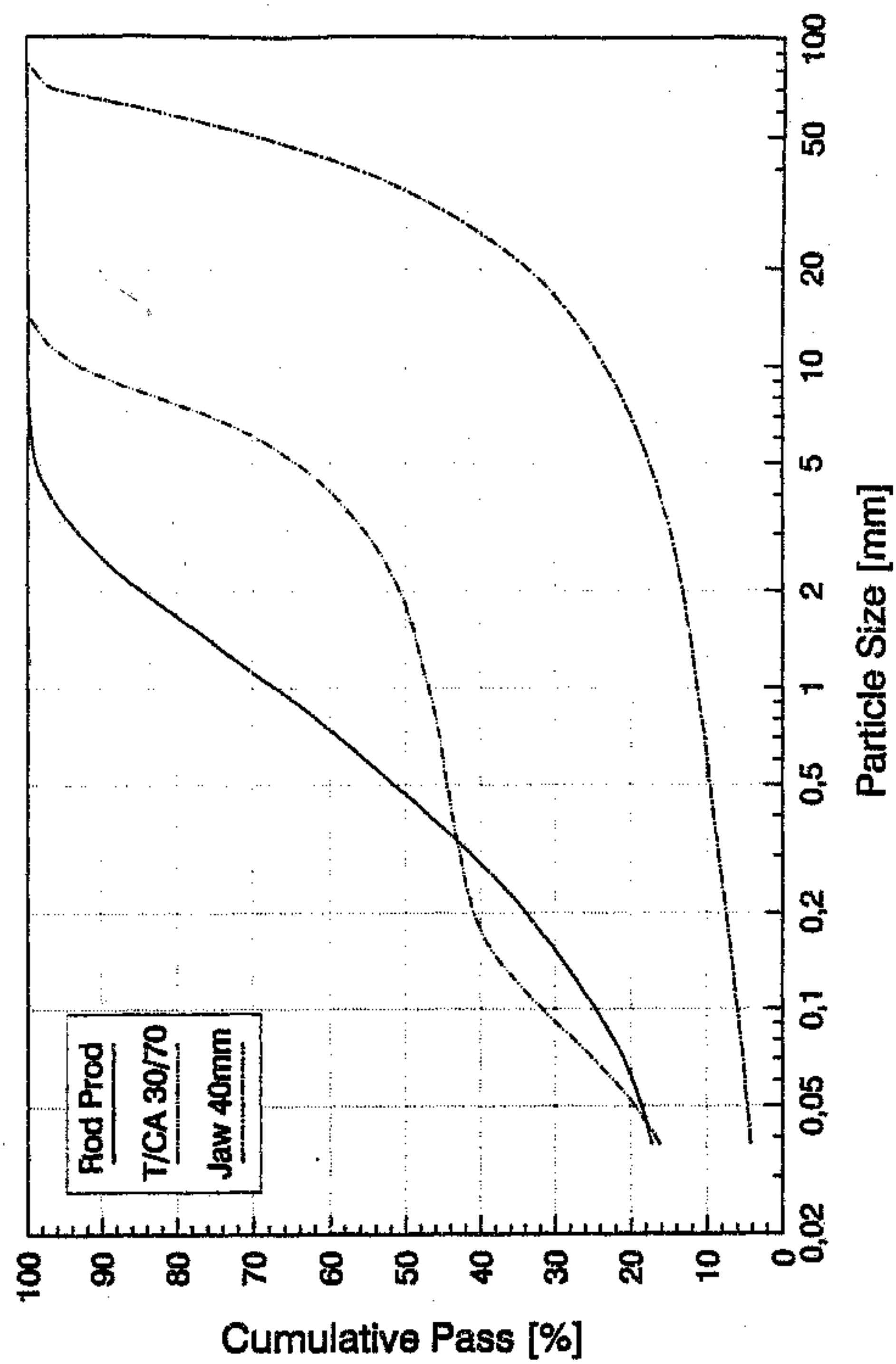

Figure 3.6 Particle size distributions of comminuted wastes (Rod Prod = rod mill product, Jaw 40 mm = 40 mm jaw crusher product) and a 30/70 tailings/coarse aggregate blend (T/CA 30/70) [Cumulative Pass %].

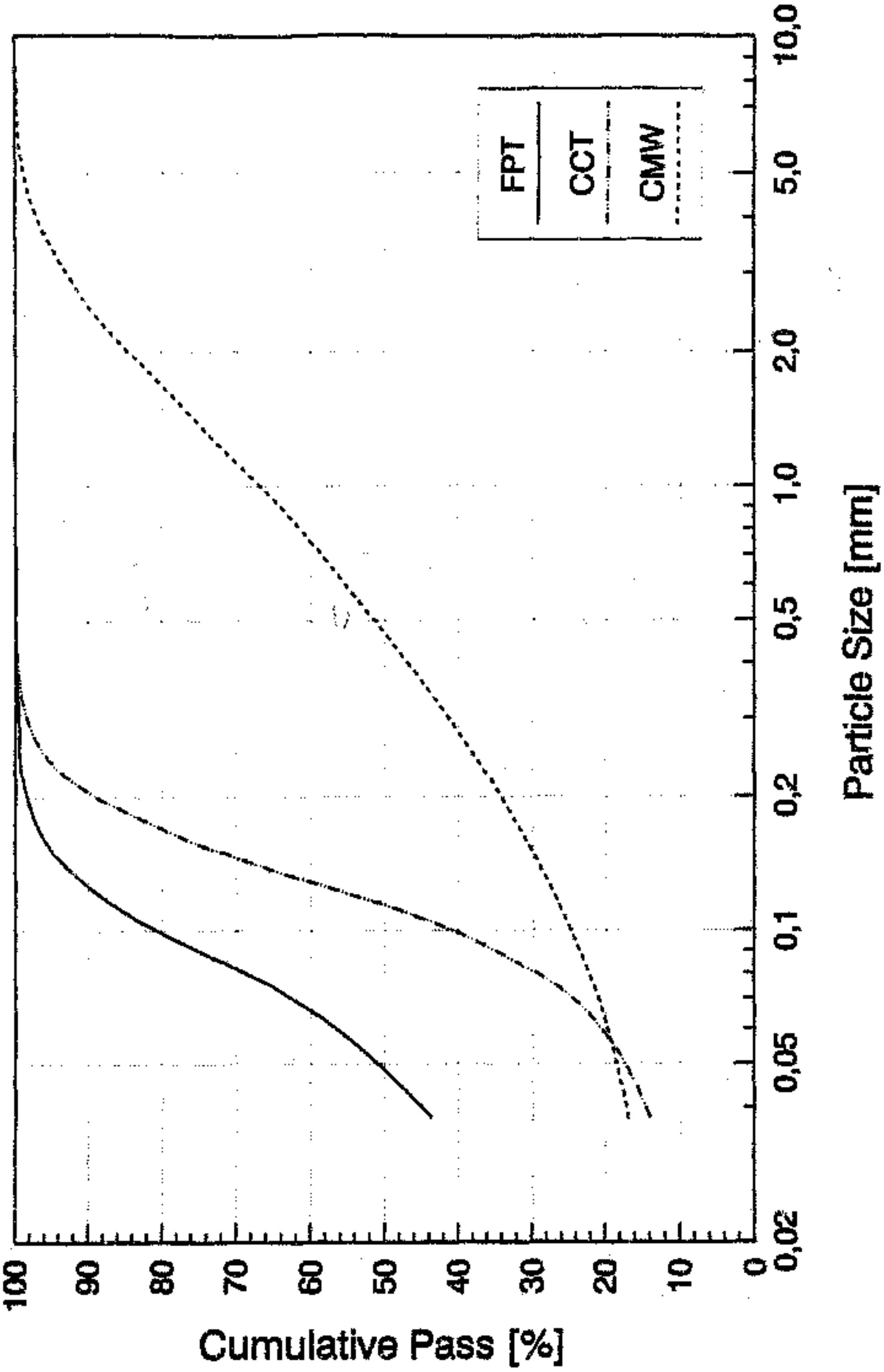

Figure 3.7 Particle size distributions of full plant tailings (FPT), cyclone classified tailings (CCT) and crushed, milled waste (CMW), as used in the initial investigation [Cumulative Pass %].

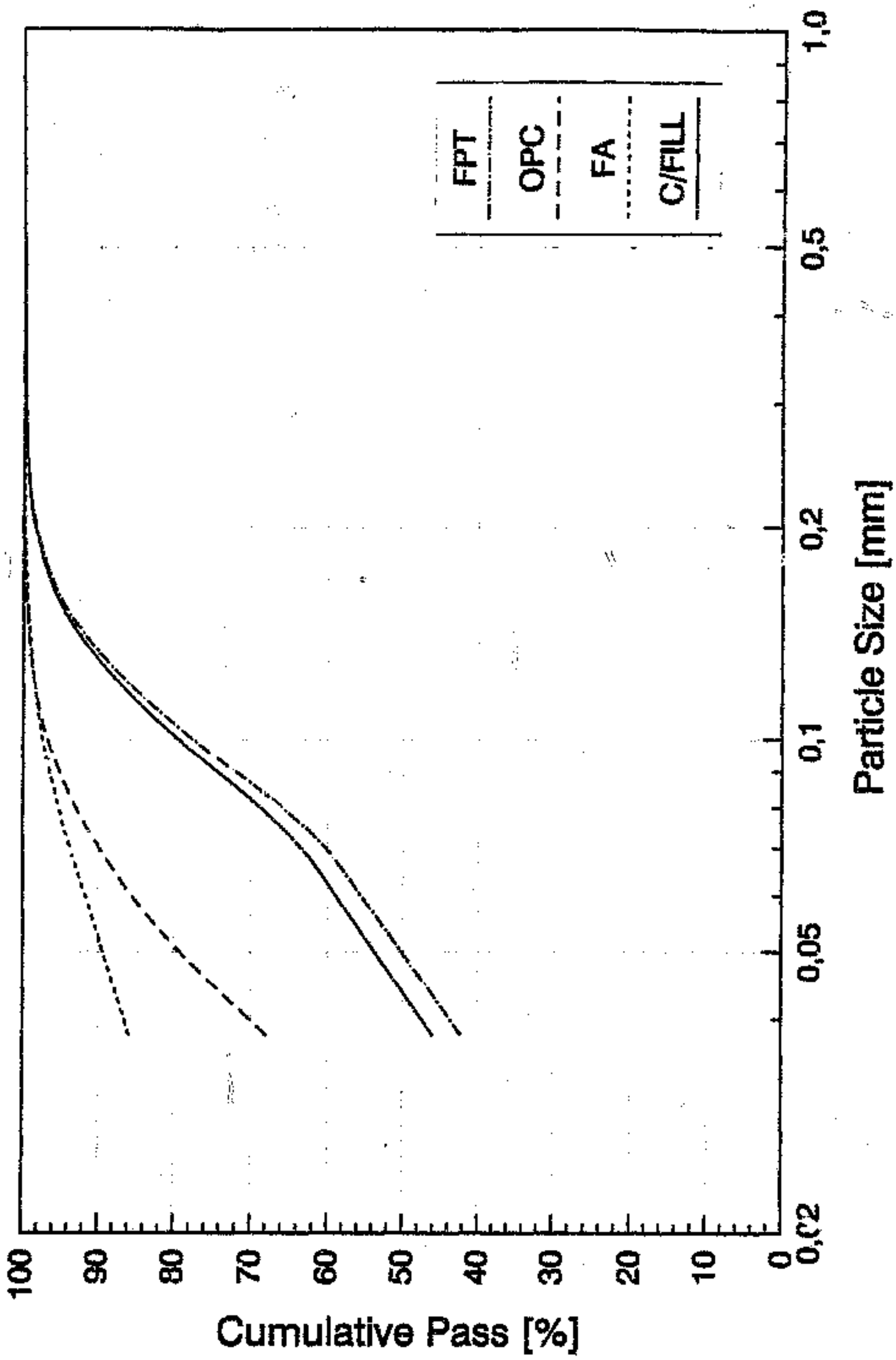

Figure 3.8 Particle size distributions of a cemented backfill and its component materials: full plant tailings (FPT), ordinary portland cement (OPC) and fly ash (FA) [Cumulative Pass %].

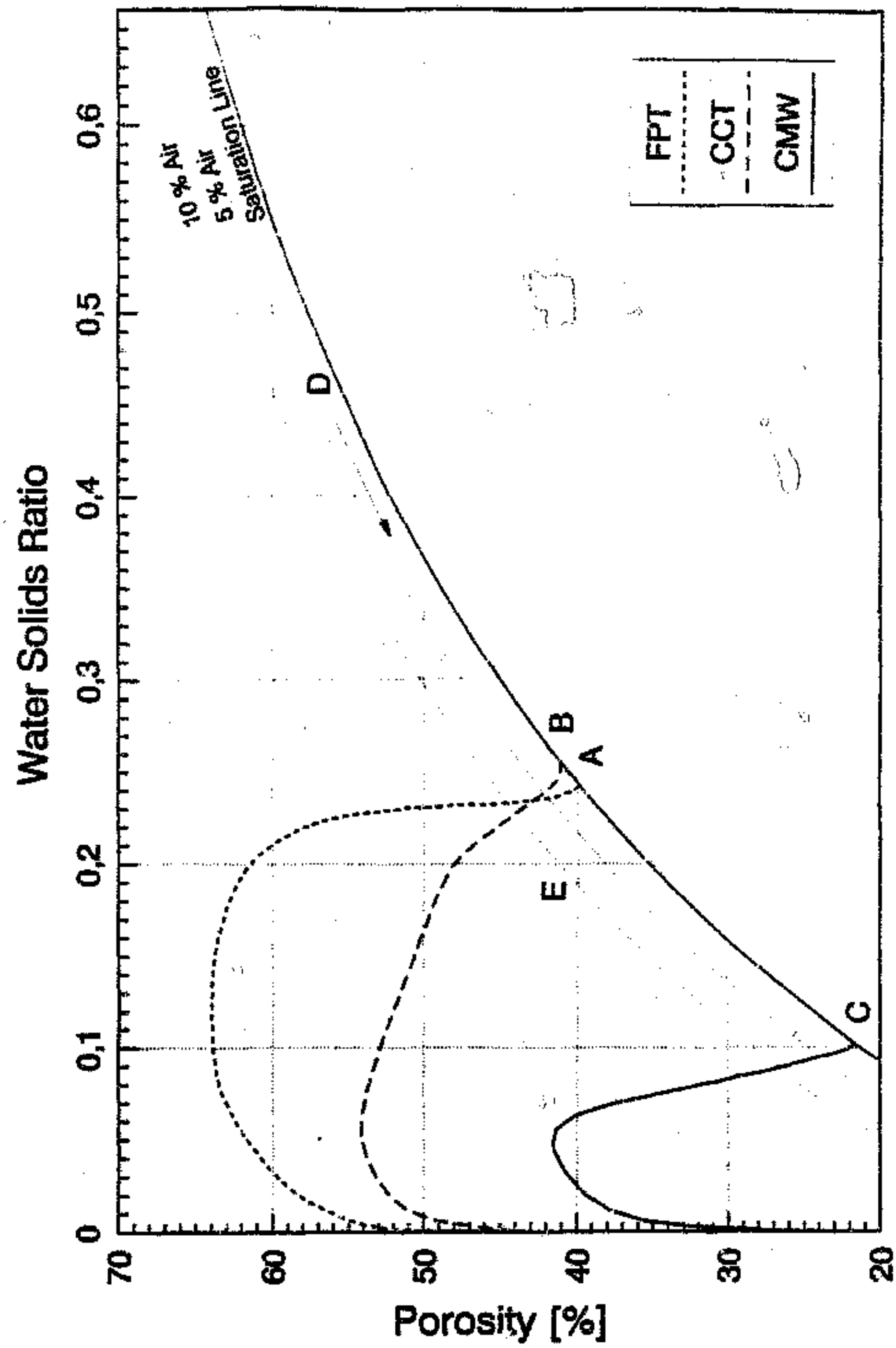

Figure 3.9 Placement property curves of full plant tailings (FPT), cyclone classified tailings (CCT) and crushed, milled waste (CMW).

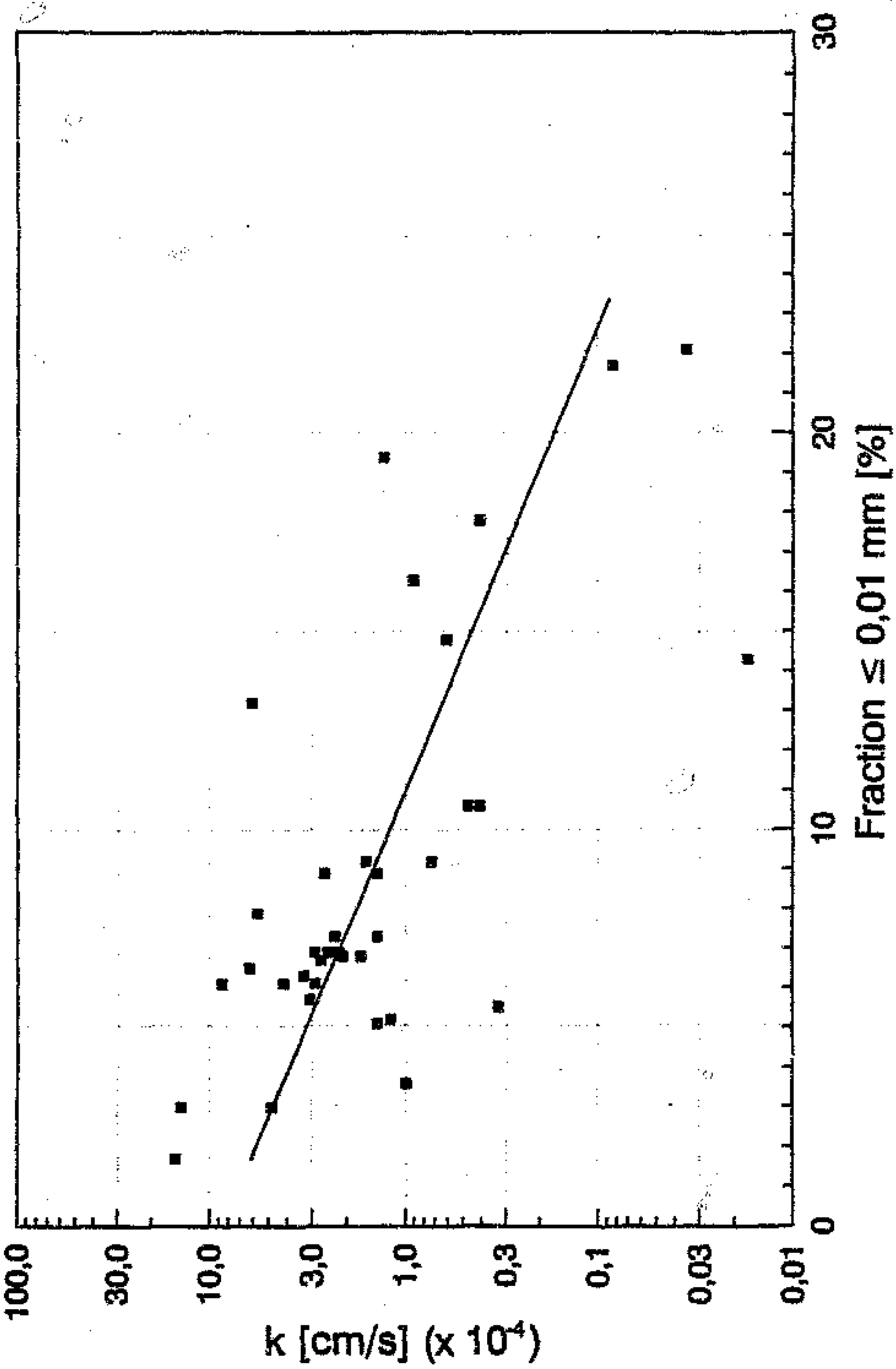

Figure 3.10 Decrease of permeability with increasing ultra-fines content.

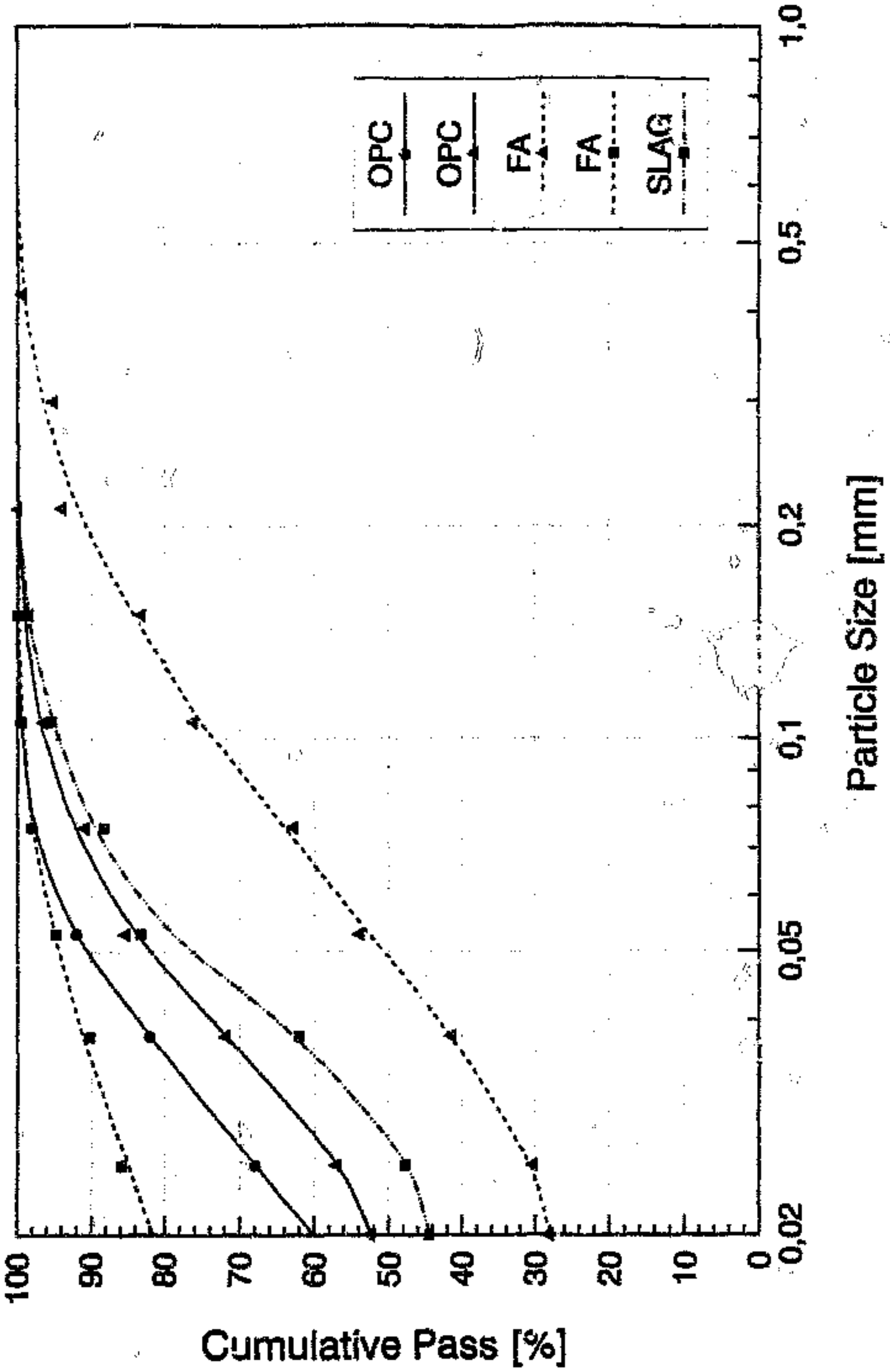

Figure 3.11 Various particle size distributions of ordinary portland cement (OPC), fly ash (FA) and slagment (Slag), as used in the investigations [Cumulative Pass %].

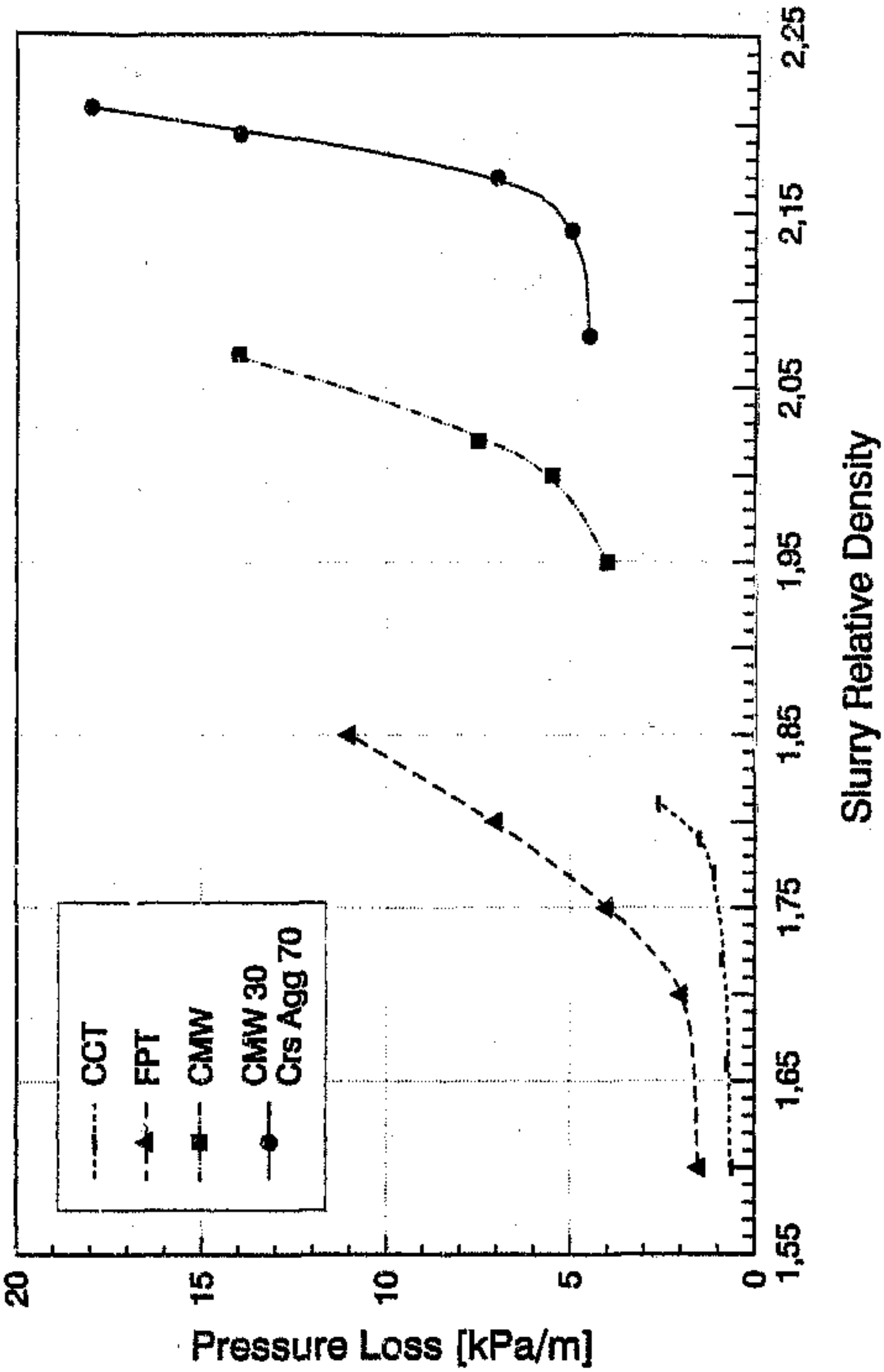

Figure 3.12 Critical transition backfill slurry densities, solids RD: 2,72, uncemented, 100 mm diameter pipeline, full plant tailings (FPT), cyclone classified tailings (CCT) and crushed, milled waste (CMW).

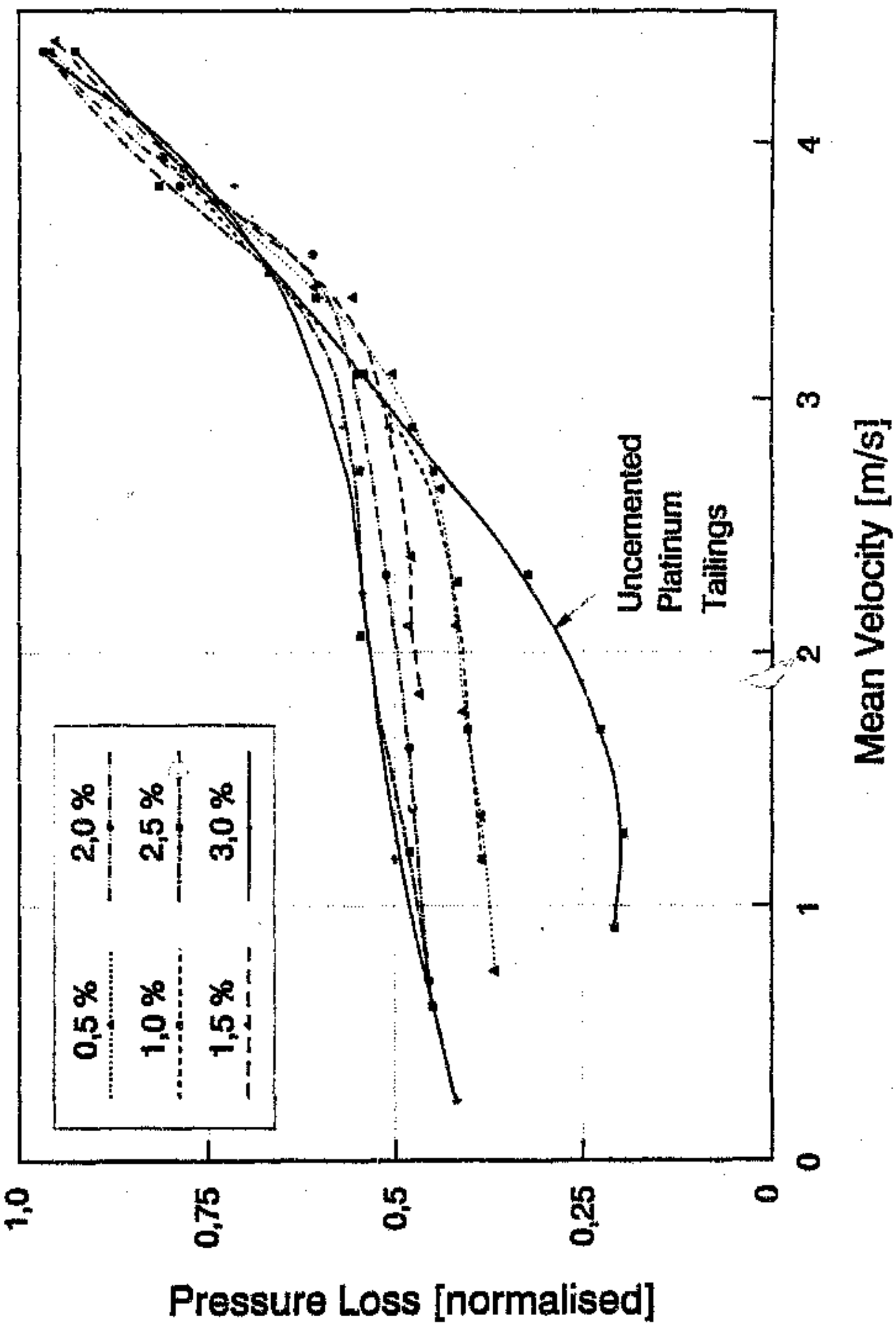

Figure 3.13 Effect of binder addition on backfill slurry pumping pressure losses, slurry RD 2,00, 100 mm diameter pipeline.

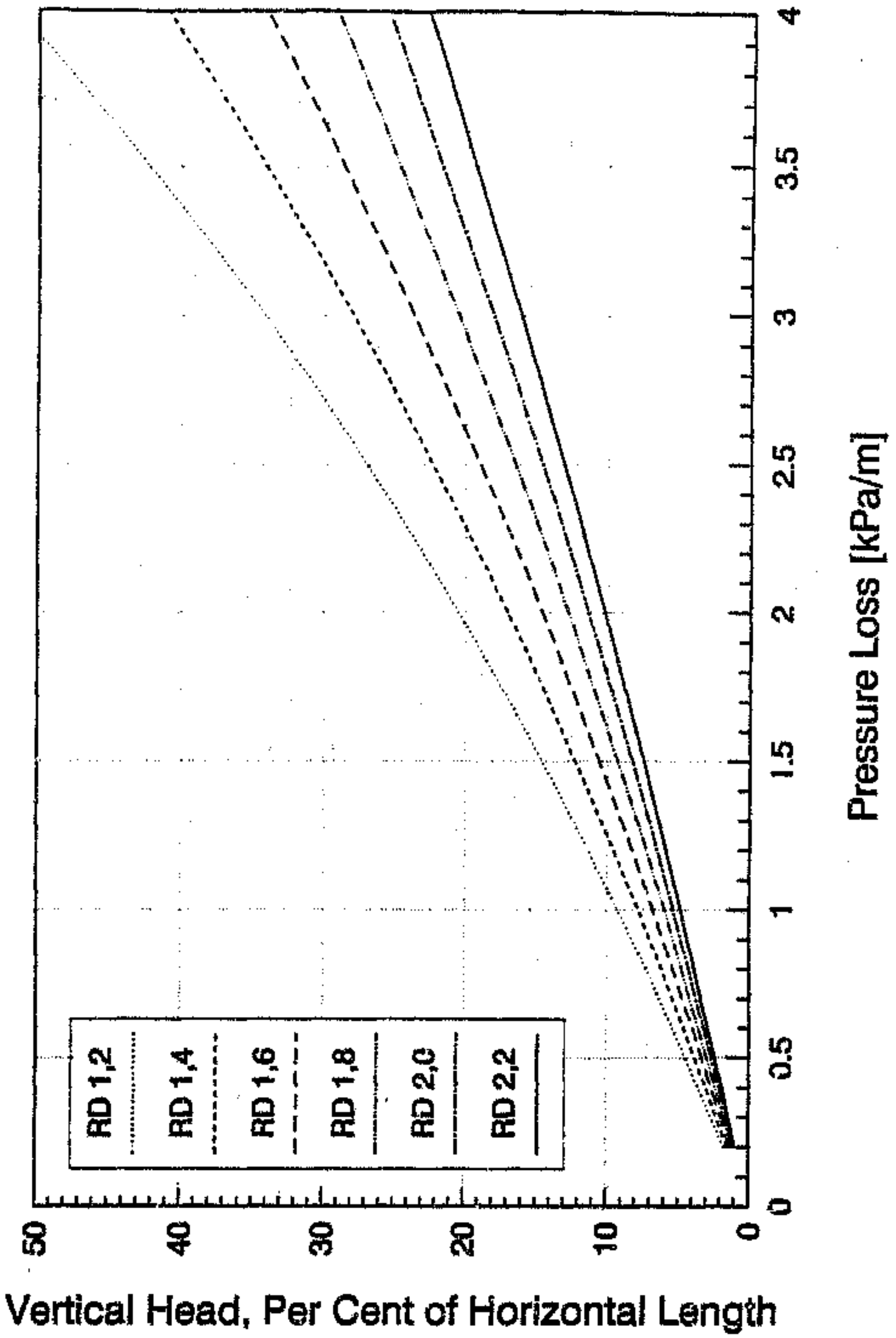

Figure 3.14 Hydraulic gravitation of backfill slurries, vertical head required for horizontal distance.

4 LOAD-BEARING CHARACTERISTICS OF CEMENTED BACKFILLS

In the previous chapter the constituent materials of cemented backfills were described, along with a number of important material and engineering properties of backfill slurries. This chapter deals with the properties of cured cemented backfill materials and how compositional, environmental and geometrical variables relate to their primary compressive strength after placement underground. The original, and largest part of the investigation was carried out with Witwatersrand gold mine tailings and formed the basis of the guidelines for the design of cemented backfills. Subsequently, the database was considerably enlarged with the incorporation of additional tests, including several suites of platinum tailings.

It will be seen, that the typical loading response of backfill with cementitious additives consists of three phases which arise sequentially with increasing deformation of the backfill sample:

* The binder-induced cohesive strength, which will be called the *primary compressive strength*. This is the traditional domain of cemented backfill research and development.

* The *post-failure collapse* after the break-down of the cohesive structure in the backfill.

* The *cohesionless compression region*, or strain-hardening compression, when the backfill compresses as a cohesionless soil. This is the traditional domain of uncemented backfill loading characteristics, mainly concerned with the stiffness of backfill materials at high stresses.

Practical backfill designs are invariably restricted in terms of available backfill constituent materials and it is, therefore, of importance to be able to utilise local materials to their full potential. This usually involves some modification of material properties and a number of compromises to accommodate a wide range of technical and economic requirements. In order to assess the relative influences of the numerous factors which influence backfill support performance, it is necessary to investigate these factors one by one and to provide a comprehensive listing of relevant compositional and environmental design elements, that is the purpose of this chapter.

4.1 Cemented Backfill Strength

After preparation on surface, the cemented backfill is hydraulically transported and placed underground into geotextile bags or paddocks which allow excess water to drain from the fill after placement. In its hydraulic state, the cemented slurry behaves much like its uncemented equivalent, i e it consolidates by de-watering and gradually develops shear strength. Some time after placement, however, the anticipated hydration reactions of the binders set in and the backfill begins to harden (1). From then on the material properties of the cemented backfill change fundamentally from those of the fresh blend of its constituents. Once the curing process has set in, the porosity, permeability and cohesion of the backfill transform gradually towards their material-specific final values.

In principle, cement mortars and cemented tailings backfills are very similar: the mechanisms of bonding are identical: the hydration reactions, as described in Section 3.3.2 apply equally to these materials and differences in material strength are due largely to varying binder concentrations and macro porosities, resulting mainly from different cement/water ratios.

In Section 3.3.2 it was stated that the reactions were of a very complex nature and by no means fully understood. In an analogous way, the same is true for the physical structure of cement paste: even though the broad mineralogical reactions are known, details of the stoichiometry and morphology of the numerous solid phases and the hydrate gels are still under investigation. One of the major causes of this great complexity is the non-stoichiometric nature of nearly all the constituents of the cements and aggregates, being derived from impure natural rocks and minerals.

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1) The addition of sodium or potassium silicate additive to cemented backfill accelerates the gelling process considerably, thereby usually preventing the drainage of water from the backfill. The gelling is, however, not due to normal cement hydration.

For a basic understanding of the setting behaviour and strength of cemented backfill, however, it is not really necessary to consider detailed phase reactions, it is quite sufficient to view the cementation process holistically. Powers (1958) described the structure of hardened cement paste as a gel, made up of hydrate particles of colloidal size [1], probably of platy morphology, in which are embedded larger crystals of calcium hydroxide and remnants of unhydrated cement. Because of the very small constituent particles, the gel has a vast internal surface area, estimated at 2 200 000 cm^2/g (average portland cement: approximately 3000 cm^2/g), and a corresponding minimum inherent (micro-)porosity of about 28 %. Powers established that every 1 cm^3 of portland cement hydrates to 2,2 cm^3 of gel, 1,2 cm^3 of which form outside of the original volume of cement. Figure 4.1 shows an electron micrograph of cured cemented backfill, as well as a schematic view of calcium silicate hydrate crystals, as they cement backfill particles and form a rigid framework within the backfill.

The fact that the volume of the binder increases with the hydration reaction has an important influence on the load-bearing properties of cemented backfills: the resulting slight reduction in the effective overall porosity of the fill significantly improves the stiffness of the materials at high stresses, i e > 35 MPa, where the backfill performs regional support duties.

4.1.1 Strength Definitions

For the comparison of different backfill compositions and configurations, the technically most appropriate measure of strength hrs to be selected. This measure should allow comparisons with results of other investigators in the field, and should, preferably, be uncomplicated.

In general terms, the strength of a material is described by all properties of that material which oppose its mechanical deformation and destruction. An important distinction has to be made between the actual *strength* of a material, related to the force required to completely disrupt molecular bonds within the material - fail it, and the

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1) By definition: at least one dimension of a colloidal particle is smaller than 10^{-4} mm.

strengths of these bonds themselves, which determine the effort required to deform the material - its *stiffness*. Both quantities are expressed in terms of applied stress or pressure.

There are a number of ways to measure and to express various aspects of the strength of cemented backfills, among them:

* Compressive strength
* Tensile strength
* Bending strength
* Shear strength
* Friction angle
* Cohesion

None of these measures can be applied universally in mining and every branch of the industry uses the measures most appropriate to their tasks.

The *unconfined uniaxial compressive strength* is the most widely used measure for the comparison of cemented backfill materials. It is measured on cylindrical or cube samples, usually in an hydraulic press, instrumented to record the applied force and displacement.

The *triaxial compressive strength, or shear strength* is measured in a test cell which allows a controlled confinement pressure to be maintained around the sample during axial compression. The main parameters determined are the friction angle and the cohesion of a material under loading conditions closely resembling the actual stress environment underground. These values are essential for the soil-mechanical assessment of backfills, and are important input parameters for numerical stress prediction models.

The *tensile strength* is applied in the numerical modelling of mining layouts where the direct undercutting of cemented backfill is considered. This quantity can be measured indirectly by the Brazilian testing method, or is deduced from triaxial testing results.

The work presented in this thesis is mainly concerned with comparative material strengths of various cemented backfills, and the parameters which control it. The unconfined, uniaxial compressive strength was, therefore, chosen as the most appropriate measure of strength. Where necessary, other parameters were, of course, measured and will be presented.

4.1.2 General Loading Response

When a backfill is subjected to stress, it will, like any other material, respond by deforming in proportion to the applied load. Two modes of deformation need to be considered: *elastic* and *plastic*, the first recoverable upon the removal of the stress, the second irrecoverable. In granular materials both deformation modes are observed simultaneously to some degree, the relative proportions dependent, among others, on the mineralogy of the material, the applied load, the packing density, and the stress history of the material.

The addition of cement to a tailings backfill produces a material which provides relatively high strength at low strain values. From Figure 4.2 it is apparent that this characteristic is markedly different from the stress-strain behaviour of a cohesionless backfill, which remains inactive at strain values at which its cemented counterpart already carries a useful load. The curves shown schematically are typical of numerous laboratory results and apply in principle to, both, unconfined and confined specimens.

Considering the cemented backfill compression curve in Figure 4.2, the ramp (1) preceding the initial elastic incline is a result of minute imperfections and non-parallelity of the loaded specimen surface. The elastic response of the loading curve (2) is caused by the rigid structure of the cement crystals which bind the mineral grains into a coherent mass that deforms elastically, as a rigid body, under load. Provided the yield strength (3) of this material is not exceeded, this deformation is fully recoverable. Once the material is loaded beyond the yield strength, however, cement crystal bonds are broken and the backfill suffers irrecoverable deformation. The maximum primary compressive strength (4) of the cemented backfill indicates the highest load the backfill can carry on account of the cohesive strength of the cement crystals within the backfill. Beyond that peak, in the post-failure region (5), the cement structure breaks down and the backfill collapses to some extent. Thereafter, the compression behaviour of the backfill approaches that of cohesionless fills, albeit with a somewhat reduced porosity due to the residual fragments of the cement crystals (6). Point (8) indicates the highest strain level obtained, equivalent to the b-value of the particular backfill material (Ryder, 1988).

The cohesionless backfill undergos mainly plastic deformation for the initial portion of the curve. As stress builds up and the porosity of the material decreases, the deformation mode becomes increasingly stiffer and the elastic component begins to dominate, a fact displayed by the modulus of the unloading loop of the material, as indicated in (7).

Figure 4.3 shows two actual load-deformation curves of a typical cohesionless tailings backfill (a) and a cemented tailings backfill (b). To illustrate the two backfill types in one graph, both, stress and strain are plotted on logarithmic scales. It can be seen in the graph, that for the first 10 % of strain, no significant stress is measured in the uncemented tailings backfill, and that at 20 % strain the level of stress has reached only about 1 MPa. Such a backfill will provide little, if any, local hangingwall support in the stope. The cemented tailings backfill shows a markedly different response: at just over one per cent strain, this fill sustains a stress of 1 MPa, sufficient to carry a substantial block of hangingwall; this backfill is capable of providing useful local support in a stope.

Cemented comminuted waste materials are very much stiffer from the outset than any of the backfills shown here and if the water content is kept to a minimum, such fills can approach the strength of mass concretes. A material of such stiffness can provide significant regional support and the replacement of regional support pillars in deep gold mines using high strength cemented backfill or concrete has been proposed (Smith, 1992).

4.1.3 Experiment Design

The properties of cemented backfill are simultaneously affected by all compositional and environmental variables to a greater or lesser degree. To determine the relative influence of each separate variable, therefore, the traditional scientific approach was adopted, whereby all parameters in a series of tests are kept constant, except the one under investigation. The parameter ranges were kept within practical operational limits and the matrix of measurement results, therefore, constitutes a useful data-base for the understanding and design of backfill support materials (1).

In the initial test matrix, four compositional variables were, *a priori*, considered as most influential and the measurements were, therefore, conducted in four groups:

* Aggregate type (i e aggregate particle size distribution),
* Water content (water/solids ratio),
* Binder content and
* Binder type.

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1) This creation of a matrix data-base is an advantage over experiment designs which involve the change of several variables at a time, using statistical techniques to analyse the results (Zhang, et al, 1991).

The non-compositional variable of curing time was accounted for by testing specimens of all compositions after the customary predetermined curing times. The other major non-compositional variable, curing temperature, was kept constant at 30 °C. All these variables will be discussed in more detail further on.

The first test series consisted of 12 different compositions in a matrix of four groups. From early on in the investigation, a number of exttensions were added to the matrix, with a view to supplement data in emerging areas of interest. For that reason, the main matrix, the basis for most of the graphs and the empirical equation presented in the next chapter, grew to 22 different compositions, as listed in Table 4.1.

Over and above the direct matrix tests, a number of peripheral investigations were undertaken, looking at aspects such as:

* The influence of curing temperature on the strength of cemented backfill specimens,
* The influence of ultra-fines contents on the strength of cemented backfill specimens,
* The effectiveness of a wide variety of cementitious binders, not included in the matrix,
* The influence of sample cross-section on the strength of cemented backfill specimens,
* The influence of the testing strain rate on the strength of cemented backfill specimens.

Results of these complementary investigations are reported where appropriate in this chapter.

In accordance with accepted practice in the field, cylindrical samples of a width/height ratio of 0,5 were used (Thomas, 1969, Helms, 1988 a). At this sample geometry, the influence of the press platens on the sample strength is minimal and the material exhibits a well defined peak at the maximum primary compressive stress. The samples were 84 mm high and had a diameter of 42 mm, sufficient for the maximum expected particle sizes in the materials. All tests were done in duplicate or triplicate. Details of all experimental procedures are given in Appendix A.

Aggregate type

To effectively cover the range of backfills used in deep South African mines, three backfill aggregate types were investigated in the study:

* A belt-filter de-watered unclassified gold and uranium plant residue,
* A hydrocyclone-classified gold and uranium plant residue and
* A crushed and rod-milled development waste sample.

All were quartzitic materials, originating from Witwatersrand gold mines. Figure 3.7 shows the particle size distributions of the three materials. At the time of the investigation, these materials were considered typical of the most important backfill types then under consideration, even though uncemented classified tailings were, and still are, by far the most widely utilised. In retrospect, the choice of any one 'classified' tailings material for the comparison seems inadequate. Later studies have shown that classified backfills at different levels of classification react in a much more complex manner than was suggested by one particular material (Section 4.1.5).

Water content

The water content of the samples was expressed in terms of the water : total solids ratio, adopted from soil mechanics practice (Vickers, 1983). The water/solids ratios of the various backfills ranged from 0,18 to 0,44; for cemented quartzitic tailings, this is equivalent to slurry relative densities of about 2,16 to 1,78 , or water contents of about 15 % to 31 %.

The water demand of a backfill, i e the amount of water needed to wet the surfaces of all its particles, changes with different particle size distributions on account of their specific surface areas. It follows, that different backfill types can show widely different consistencies with identical water/solids ratios. Conversely, two backfills with two different water/solids ratios can display similar 'workability' or pumping characteristics. An important consequence of this phenomenon is that the water/cement ratio, and therefore the strength of a backfill, is strongly influenced by its particle size distribution.

Binder content

In the investigation, binder content referred to the total binder content, which in addition to the active component (usually portland cement) may have included one of several pozzolans. The binder content was calculated as a percentage of the dry total backfill mass (total solids). The binder content for the first test matrix ranged from a minimum of 3 % up to 20 %; in some later tests up to 25 %.

Binder type

Four different binders were compared, consisting essentially of ordinary portland cement, fly ash, and ground granulated blastfurnace slag:

* Ordinary portland cement (SABS 471-1971),
* Portland blastfurnace cement (SABS 626-1971),
* A blend of ordinary portland cement and fly ash in the proportions 1:1 and
* A blend of ordinary portland cement and fly ash in the proportions 1:2.

The use of other active cements, such as rapid-hardening portland cement, sulphate-resisting portland cement or high-alumina cement was declined since these are manufactured for specific purposes and are more expensive than OPC. Lime was tested as a binder, but was found almost completely ineffective within the curing periods of the investigation. In South Africa, moreover, the use of lime, by itself, or with pozzolans, is unattractive because of its high cost, approaching that of portland cement.

Curing time

Within dilute cemented backfill slurries, cement crystals increase in numbers relatively slowly and the strength of the fill material is strongly time dependent. The strength of the curing backfill material was, therefore, tested at set time intervals. For this investigation the backfills were usually tested after 3, 7, 14, 28, and 90 days curing. For exceptionally strong materials, the strengths were also measured after one day's curing. In some later experiments, using sodium silicate additive, the curing periods at the time of testing were reduced to a few hours.

Curing conditions

The cement hydration reaction is a chemical reaction and is, therefore, both, time- and temperature dependent. To emulate, as far as possible, underground curing conditions, all samples were cured at 30 °C; this was considered a reasonable average underground temperature. In the investigation of the effect of curing temperature on the strength of cemented backfill, this temperature was, of course, exceeded.

To prevent loss of reaction water from the samples during the curing period, all samples were cured in double poly-ethylene bags. There was some slight loss of water during curing, measured as a slight mass loss of the samples immediately prior to testing. The total mass loss, however, amounted only to about 0,6 % after 28 days and 1,2 % after 90 days curing, and may have included particles of solids.

During the planning phase of the investigation, the option of curing the backfill samples under water was discussed. This is commonplace for concrete specimens and was practised on cemented backfill samples by some investigators. The option was rejected, as in practice backfills do not normally cure under water, neither is this desirable because of the danger of liquefaction of inundated backfill bodies. The insides of cemented backfill bodies do, however, remain damp for long periods of time, indicating an adequate supply of reaction water.

The saturation levels of all samples at the time of testing were computed as part of the determination of sample porosities. All values plotted on the saturation line on a placement properties graph (such as Figure 3.9 , an indication of the full saturation of the cemented backfill specimens at the time of testing.

4.1.4 Cemented Backfill Compositions Tested

For the main part of the investigation, a total of sixteen different cemented backfill compositions were prepared and tested in the laboratory to determine their relative strengths and, if possible, identify a general mode of composition-related behaviour. In Table 4.1, all tested compositions are listed (some of them appear in more than one group, when this is useful for comparison; these compositions are identified in the table).

A cemented backfill, consisting of full plant tailings and containing 10 % OPC (in dry mass), with a water/(total-)solids ratio of 0,39 was selected as a relative standard, against which all compositions were compared. As a result of this experimental design, most tests were done with full plant tailings and/or ordinary portland cement. All samples were cured in sealed poly-ethylene bags at 30 °C. The cement content of a sample was defined as the total binder content in the solids, in mass per cent. In the table, this figure is listed alongside the individual cement contents as a percentage of the total sample mass, i e in slurry.

Table 4.1 Compositions of backfills containing cementitious additives, grouped by principal testing parameters.

Group Sample	Composition									
	Aggr Type	Total Binder [%] in Solids	W/S Ratio	Fill [%] in Slurry	OPC [%] in Slurry	FA [%] in Slurry	PB FC [%] in Slurry	Water [%] in Slurry	Slurry RD	C/W Ratio
Aggregate Type										
FT [1]	FPT	10	0,39	64,8	7,2	-	-	28,0	1,85	0,26
CT [2]	CCT	10	0,32	68,1	7,6	-	-	24,3	1,94	0,31
CW [2]	CMW	10	0,18	76,4	8,5	-	-	15,1	2,19	0,56
Water/ Solids Ratio										
0,28	FPT	10	0,28	70,3	7,8	-	-	21,9	2,00	0,36
0,33	FPT	10	0,33	67,5	7,5	-	-	25,0	1,92	0,30
0,39 [1]	FPT	10	0,39	64,8	7,2	-	-	28,0	1,85	0,26
0,41 [4]	FPT	4,5	0,41	67,7	3,2	-	-	29,1	1,82	0,11
0,44	FPT	10	0,44	62,3	6,9	-	-	30,7	1,80	0,22
CT 28 [3]	CCT	10	0,27	70,7	7,9	-	-	21,5	2,01	0,37
CW 24 [4]	CMW	3	0,24	78,2	2,4	-	-	19,4	2,04	0,12
Binder Content										
3 %	FPT	3	0,39	69,9	2,2	-	-	28,0	1,83	0,08
6 %	FPT	6	0,39	67,7	4,3	-	-	28,0	1,84	0,15
10 % [1]	FPT	10	0,39	64,8	7,2	-	-	28,0	1,85	0,26
15 % [4]	FPT	15	0,30	65,4	11,6	-	-	23,1	1,97	0,50
20 %	FPT	20	0,39	57,6	14,4	-	-	28,0	1,85	0,51
CW 3 [4]	CMW	3	0,18	82,2	2,6	-	-	15,3	2,16	0,17
CW 20 [2]	CW	20	0,18	67,9	17,0	-	-	15,1	2,19	1,13

Table 4.1 cont.

Group Sample	Aggr Type	Total Binder [%] in Solids	W/S Ratio	Fill	OPC	FA	PB FC	Water	Slurry RD	C/W Ratio
				Composition [%] in Slurry						
Binder Type										
OPC [1]	FPT	10	0,39	64,8	7,2	-	-	28,0	1,85	0,26
OP 1:2	FPT	10	0,39	64,8	2,4	4,8	-	28,0	1,83	0,26
OP 4,5 [4]	FPT	4,5	0,39	68,8	1,1	2,2	-	28,0	1,83	0,12
OP 25 [4]	FPT	25	0,39	54,0	6,0	12,0	-	28,0	1,80	0,64
OP 1:1	FPT	10	0,39	64,8	3,6	3,6	-	28,0	1,83	0,26
PBFC	FPT	10	0,39	64,8	-	-	7,2	28,0	1,85	0,26
CT OP [2]	CCT	10	0,32	68,1	2,5	5,1	-	24,3	1,91	0,31
CW OP [2]	CMW	10	0,18	76,4	2,8	5,7	-	15,1	2,15	0.56

1 Identical compositions

2 Water/solids ratio equivalent to full tailing's 0,39

3 Water/solids ratio equivalent to full tailing's 0,28

4 Second test series

The following abbreviations are used in the table :

OPC	ordinary portland cement,
FA	fly ash,
PBFC	portland blastfurnace cement,
W/S	water/solids ratio,
C/W	cement/water ratio,
FT, FPT	full plant tailings,
CT, CCT	cyclone classified tailings,
CW, CMW	comminuted waste and
RD	relative density.

The water/solids ratio was specified for the full plant tailing's size distribution and, both, the classified tailings and the comminuted waste W/S ratios were adjusted to the same relative values. The reason for that was that the three backfill types used have different size distributions (Figure 3.7) and, therefore, different specific surface areas. Consequently, the finer materials have a higher water demand to fully wet their particle surfaces. The water/solids ratio of 0,39 for full tailings is, therefore, equivalent to 0,32 for classified tailings and 0,18 for comminuted waste.

4.1.5 Unconfined Uniaxial Compression

The binder-induced primary compressive strength, as measured in unconfined, uniaxial compression (point 4 in Figure 4.2), was selected to compare the cemented backfill materials in this investigation. It is widely regarded as the single most useful measure for comparing the relative strengths of cemented backfills (Thomas, 1969, Thomas, et al. 1979, Helms, 1988 a); this parameter, at the selected specimen width/height ratio of 0,5 , produces a clearly defined peak value (1) .

In Table 4.2, the maximum compressive strengths of the compositions are listed under their respective curing times. Each sample was tested in duplicate or triplicate, but in the table only the highest reading is given. This is justified, considering that there are numerous factors which can weaken a test specimen, but none which increase it's strength above it's potential maximum. The highest reading should, therefore, be accepted as the one, closest to the actual maximum strength. In practical fill design, however, an average value, or even a low reading would be used, to maintain a realistic safety margin. Figure 4.4 shows a typical range of sample strengths of a single composition.

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1) For cylindrical test specimens, a width to height ratio of 0,5 is commonly chosen to eliminate the frictional restrainment effects of the press platens.

Table 4.2 Maximum primary compressive strengths of the tested cemented backfill compositions, highest readings [MPa].

Group Sample		Curing Time [d] 1	3	7	14	28	90
Aggregate Type							
	FT [1]	-	0,56	0,65	0,81	1,16	1,92
	CT	-	0,51	0,67	0,73	0,99	1,34
	CW	-	2,06	2,99	3,22	3,99	5,24
Water/Solids Ratio							
	0,28	-	1,11	1,42	1,81	2,70	4,03
	0,33	-	0,92	1,12	1,26	1,72	2,83
	0,39 [1]	-	0,56	0,65	0,81	1,16	1,92
	0,41 [4]	-	-	-	-	0,37	-
	0,44	-	0,40	0,47	0,55	0,72	1,19
	CT 28	-	0,73	0,97	1,15	1,45	-
	CW 24 [4]	-	-	-	-	0,23	-
Binder Content							
	3 %	-	0,12	0,18	0,28	0,36	0,48
	6 %	-	0,31	0,36	0,46	0,64	1,08
	10 % [1]	-	0,56	0,65	0,81	1,16	1,92
	15 % [4]	-	-	-	-	3,83	-
	20 %	-	1,39	1,70	2,14	3,26	4,55
	CW 3 [4]	-	-	-	-	0,23	-
	CW 20	3,64	8,45	9,99	13,17	15,17	-

Table 4.2 cont.

Group Sample		Curing Time [d] 1	3	7	14	28	90
Binder Type							
	OPC [1]	-	0,56	0,65	0,81	1,16	1,92
	OP 1:2	-	0,22	0,33	0,58	0,70	1,24
	OP 4,5 [4]	-	-	-	-	0,26	-
	OP 25 [4]	-	-	-	-	2,08	-
	OP 1:1	-	0,26	.40	0,55	0,94	1,68
	PBFC	-	0,38	,51	1,15	2,01	3,68
	CT OP	-	0,21	0,34	0,55	0,88	-
	CW OP	-	0,49	0,92	1,78	3,23	-

[1] Identical samples

[4] Second test series

All samples cured at 30 °C, tested at full saturation

For abbreviations see Table 4.1.

Curing time

As the growth of cement crystals proceeds at a certain rate, dependent on temperature and the supply of reaction products, the curing time is, of course, critical in the estimation of fill strength, as can be seen in Table 4.2 and in the graphs of the following sections. It is, therefore, customary to compare materials at predetermined curing times, usually 28 days. To determine the rate of curing of the cemented backfill samples, specimens were also tested after 1, 3, 7, 14 and 90 days curing (1). For practical purposes, the fill can be regarded as fully cured after about 90 days.

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1) Most compositions were too weak to test after one day's curing and this test was, therefore, restricted to the strongest materials in the series.

Aggregate type

Figure 4.5 shows the primary compressive strengths of the three backfill types investigated, i e backfills with full plant tailings, cyclone classified tailings and comminuted waste aggregates. The binder was ordinary portland cement, at a content of 10 % in dry mass. The water/solids ratio for the full plant tailings samples was set at 0,39 , the ratios for the other materials were adjusted according to their water demand to 0,32 for the classified tailings and 0,18 for the comminuted waste (see also Section *4.1.4*). These water/solids ratios produced equivalent 'workabilities' of the mixes and ensured, as far as possible, equivalent cement/water ratios in the cement pastes.

It is immediately apparent that the comminuted waste material is about four times stronger than the two tailings materials and that of the two tailings backfills, the full plant tailings blend is marginally stronger than the classified material. In general, the findings indicate, that the inherent porosity of the backfill type has a most significant influence on the strengths of the whole range of cemented materials. The relative strengths, as shown in Figure 4.5 approximately follow minimum porosities of the uncemented backfill materials (placement properties, Section *3.2.7*, Figure 3.9): the lower the inherent porosity, the higher the cemented backfill strength. This is simply explained by the fact that a given number of cement crystals can fill a reduced pore space more effectively, and bond more grains simultaneously.

For tailings materials, however, an additional factor plays a very important role: the amount of ultra-fines ($\leq$ 0,010 mm) present in the fine backfill aggregate. An understanding of the influence of ultra-fines on the primary compressive strength of cemented backfills is necessary for the optimisation of cemented backfill particle size distributions.

In cemented tailings backfills the amount of ultra-fines ($\leq$ 0,010 mm) present in the aggregate plays a very complex role. To demonstrate this, a suite of backfill tailings samples were classified to simulate a range of by-pass fractions, representing the efficiency of hydrocyclone-classification. Figure 4.6 shows the particle size distributions of the tailings, classified in the laboratory, with their respective cyclone by-pass fractions (Hinde, 1989). Figure 4.7 shows the measured specific surface areas of these classified tailings, as they relate to the cyclone by-pass fractions.

The difference between the full plant tailings and the perfectly classified tailings material corresponds to a sixfold reduction of the specific surface area, with the various classification efficiency levels proportionally arranged in between.

Figure 4.8 shows the 28 day primary compressive strengths of the different cyclone products at constant binder additions of 10 % (in dry mass) and a constant water/solids ratio of 0,39. A complex pattern emerges, with the perfectly classified material measuring about 1,6 MPa. As some ultra-fines are introduced, the strength rises to a maximum of about 1,75 MPa, only to fall away rapidly with the addition of more ultra-fines to about 0,8 MPa at its minimum, corresponding to a cyclone by-pass fraction of about 20 % (1). From there on, additional ultra-fines improve the strength of the backfill again to reach about 1,2 MPa with the full plant tailings. Any further additions of ultra-fines are expected to worsen the situation drastically, for reasons laid out below.

Various factors contribute to the complexity of the function. First, it should be considered that in the present compositions 10 % of the mass of solids consists of fine grained portland cement with an average particle size of between 0,020 mm and 0,025 mm. Because of its higher solids density, the cement fraction only contributes 8,7 % of the volume of the backfill solids. The second consideration concerns the water demand of the aggregate; this will increase directly with the surface area and thus increase towards the right hand side of the graph. Third, the average particle size of the aggregate will decrease with increasing ultra-fines content. Finally, the porosity of the backfill improves somewhat with increasing amounts of ultra-fines, as was pointed out previously.

Since each sample was prepared with identical amounts of water and portland cement, the difference in strength must be related to the backfill structure and changing water demand. This is a considered interpretation of the results:

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1) It is unfortunate that the primary strength of this particular cemented backfill composition drops to its lowest at a cyclone by-pass of about 20 %, in practice often the region of poorly run cyclones.

* On the far left side of the curve, the aggregate is almost completely devoid of ultra-fines and its water demand is therefore minimal. Almost all the added water is thus available for the cement hydration reaction - producing the weakest possible gel under the circumstances in a relatively large pore space.

* The peak of the graph is produced at an ultra-fines content of about 3 %, when all the added cement, the fine grained aggregate and the ultra-fines produce just the optimum pore space filler at the most favourable cement/(free) water ratio.

* Beyond this optimum, the limited number of cement grains gradually become separated spatially, still in contact with most of the free water though, resulting in a poor cement/water ratio, but reducing the spatial density of the crystal network. The lowest strength is reached at the worst possible combination of these factors.

* The gradual rise of primary strength towards the full plant tailings side is presumed to be caused mainly by the gradually reducing porosity and the reduction of the average particle size of the aggregate - ultimately to about the same size as the cement. This equalisation of particle sizes brings about the almost complete isolation of single cement grains - they are surrounded by an increasing number of aggregate grains (in full plant tailings, on average, about 11 grains). The implication is, that an increasing amount of water is being tied up in wetting grain surfaces remote from cement grains, with the paradoxical result of a favourable cement/water ratio where cement is actually available in the structure. This, together with the marginally improved porosity can account for the rise of primary strength on the right-hand side of the curve. A further, minor contributor to this rise may be the increased availability of sub-micron quartz grains with the increased amount of ultra-fines. Such very small grains can be chemically active and may enter the hydration reaction to some extent as pozzolans.

Regardless of the validity of the given interpretation, it is clear that this very complex behaviour of the different classification products cannot be explained by singular factors. Particularly, the effective cement/water ratio needs to be applied with discretion, considering the possibility of free water and water bound to aggregate surfaces unavailable to isolated cement grains.

Porosity

In the previous section it was pointed out that the porosity of a cemented backfill is a major determinant of the compressive strength of that material. Figure 4.9 shows the relationship between porosity and the primary compressive strengths of the tested cemented backfills. Despite the wide scatter of points, caused by the wide variety of backfill aggregate and binder types, a general decline of strength with increasing porosity is clearly noticeable. The general trend of the relationship remains unchanged in time, except for the increase in strength.

In Figure 4.10, the same data is differentiated according to backfill aggregate and binder type and clear groups emerge, each showing the same basic trend to varying degrees. It is interesting to note that the full plant tailings group with pozzolanic binders (mainly fly ash) shows such a steep curve, confirming findings by cement researchers that fly ash has a modifying effect on the size of pores formed in cemented materials (Krüger, 1990). The steep curve of the comminuted waste group is, however, due to the low inherent porosity of the aggregate material, leaving only little latitude for the cement paste to strongly influence porosities.

The inverse relationship between the primary compressive strengths of cemented backfills and their porosities is to a large extent caused by excessive water in the backfill and the resultant formation of macro-pores. Figure 4.11 shows the clear, linear relationship between the water content of a cemented backfill and its porosity (here after 28 days curing, different backfills). The porosity - water relationship, however, points to second major reason for the weakening of cemented backfills with increasing porosity: the increasingly unfavourable cement/water ratio in the materials. At this stage it is unclear, which is the dominant effect.

The decrease of cemented backfill strength with increasing porosity is demonstrated very clearly in backfills, where porosity is introduced deliberately by mixing stable foam into the backfill slurry. Figure 4.12 shows the decline of cemented backfill primary compressive strength with a reduction of backfill density. The purpose of adding foam to cemented backfill in this test series was in an attempt to produce lightweight support blocks from backfill materials.

Water content

Figure 4.13 shows the primary compressive strengths of cemented bac[illegible], prepared to four different water/solids ratios, ranging from 0,28 to 0,44 [1]. The aggregate was full plant tailings and the binder w[illegible]s ordinary portland cement, at a content of 10 % in dry mass. The range of water/solids ratios chosen for the investigation represents the practical limits for this particular material combination: at 0,28 W/S, the material was too stiff for placement, it no longer flowed and had to be vibrated into the moulds; at 0,44 W/S, the backfill was on the limit of instant segregation, any more water would only have decanted.

The graph clearly demonstrates the advantage of reducing the water contents of cemented backfills to a minimum (i e operate backfill systems at high slurry relative densities), as the strength increases more than threefold over the test range. This substantial increase in strength can be ascribed to a decrease of the placed (macro-) porosity of the fill, and to the more favourable cement[illegible]ater [illegible]o in the binder paste, substantially reducing the micro-porosity in the cement gel.

The influence of the water/solids ratio on classified tailings is shown in Figure 4.14. The same trend as with full plant tailings is apparent: a decrease of the water content brings about a substantial increase of the cemented backfill strength, even though the overall strength levels are somewhat lower than those of their full plant tailings counterparts for reasons explained in the previous section.

In Figure 4.15 the detrimental influence of high water contents on all types of cemented backfills is clearly demonstrated: within the test range the strength declines, on average, by about half with an increase of less than 10 % of water in the fill. All the samples represented in this and the following graph contained 10 % binder of various types. It can be seen, however, that on the whole, the type of cement has little i[illegible]ence on the trend of the graph.

If the same data is differentiated with regard to the aggregate type used (Figure 4.16), it can be seen that the comminuted waste backfill is much more sensitive to variations in the water content, than are the tailings backfills. [illegible]e reason for

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1) For the corresponding water contents see Table 4.1.

this is the low water demand of this type of backfill on account of their relatively low specific surface areas. Any additional water in the backfill is, therefore, immediately translated into a detrimentally low cement/water ratio.

Binder content

Figure 4.17 shows the primary compressive strengths of cemented backfills, prepared to four different binder contents, ranging from 3 % to 20 % in dry mass. The tailings material was full plant tailings and the binder was ordinary portland cement; the water/solids ratio was set at 0.39. The addition of 3 % binder to a backfill was found in practice to be about the lower practical limit, particularly with blends of portland cement and pozzolans. Below 3 %, proper mixing of the binder into the backfill becomes a problem and large pockets of uncemented fill were found in practice. 20 % binder is an unusually high addition of binder and would only be considered in exceptional circumstances, such as for instance the preparation of regional support pillars with inferior aggregate.

The graph shows that the full tailings samples increase in strength approximately in proportion to their cement content. This is due to a proportionally increased number of cement crystals forming in the pores of the backfill material, resulting in increased adhesion and reduced porosity. Figure 4.18 shows the effect of an increase in binder content on comminuted waste backfill, a material of much lower inherent porosity. The addition of 20 % OPC produced the strongest backfill material of the test series at an primary compressive strength of over 15 MPa.

Cement/water ratio

The results of the previous two sections highlighted the important influence of, both, the water and the binder contents on the strength of cemented backfill. Whereas the backfill aggregate type represents a crucial variable in a general sense, it rarely changes, once a backfill system has been implemented and thus can often be regarded as a constant. The cement and water contents, on the other hand, are strategic variables in a very direct sense, particularly if they are considered in tandem, as the cement/water ratio.

Results indicate that for full plant tailings aggregate, the relative influences of the water and the binder contents on the strength of cemented backfill are approximately equivalent. Figure 4.19 shows that the slopes of the 28 day curves of both variables are similar, but reversed: a 10 % increase in the binder content raises backfill strength about as much as a 10 % decrease in the water content does. The compared backfills consisted of full plant tailings and contained 10 % (in dry mass) of cementitious binder; the water/solids ratio was constant at 0,39. The high scatter of values is due to the use of different binder types. The relative influences of cement and water contents on primary compressive strength are different for comminuted waste backfills: because of their low water demand, they are more sensitive to excess water.

Both, the water and the binder contents are such important variables for the cement hydration reaction, that their ratio has long been an important indicator of the optimal use of cement in cemented materials (1). Figure 4.20 shows the (non-linear) relationship between the primary compressive strengths of cemented backfills and their cement/water ratio. It can be seen that the relationship remains unchanged with progressive curing of the materials, except for the gradual increase in strength. The cement/water ratio is a convenient indicator of the effective utilisation of binders in cemented backfills, but the scatter of values is, unfortunately, too large for the ratio's utilisation in the design of backfills, as will be explained in the next chapter.

Binder type

Figure 4.21 shows the primary compressive strengths of cemented backfills, prepared with four different binders: ordinary portland cement (OPC), portland blastfurnace

1) Fulton (1977) recommends the use of the ratio of cement mass to water mass, rather than water/cement, since the relationship between the ratio and the compressive strength becomes a direct one and, at least in concretes, a linear one.

cement (PBFC), and two blends of OPC and fly ash (FA) in the proportions of 1:1 and 1:2. The tailings material was full plant tailings, the binder addition was 10 % in dry mass and the water/solids ratio was set at 0,39.

Of those cements tested in this series, the portland cement backfill shows the fastest initial curing rate: after three days the backfill has reached 0,75 MPa, substantially more than the other backfills. All binders containing pozzolans set out at a slower rate and retained their relative strengths throughout the curing period. After about eight days, the backfill prepared with portland blastfurnace cement exceeded the portland cement backfill in strength and remained substantially stronger from then on. This good performance is probably due to the fine grind of it's (50 %) OPC fraction, which increases it's reactivity (1).

The dilution of portland cement with fly ash reduces its effectiveness approximately in proportion to the mix ratio, i e the 1:1 blend is stronger than the 1:2 blend. In a later experiment (2), it was, however, established that a OPC/FA 1:2 blend was more cost-effective than a 2:1 blend in terms of cost per achieved unit primary compressive strength (Cost/MPa/100 kg backfill). This confirms previous findings (Kruger, 1990) that blends consisting of up to two thirds fly ash and one third ordinary portland cement can be optimal in terms of the utilisation of all available free lime from the cement hydration reaction.

Different fly ashes have different pozzolanic activities, however, and the strengths of cemented backfills can vary widely with the type of ash used. Figure 4.22 compares the (typical) primary compressive strengths of five equivalent backfill compositions (full plant tailings, 10 % OPC/FA 1:2 binder, W/S 0,39), each prepared with a different fly ash sample. The curve marked 'commercial' represents a cemented backfill sample (composition OP 1:2 in Table 4.1) containing commercially available

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1) Portland blastfurnace cement is an inter-ground blend of portland cement and ground granulated blastfurnace slag. The slag is physically harder than the cement clinker and, therefore, tends to comminute the OPC fraction during inter-grinding. Investigators in the cement industry also maintain that PBFC has a higher tolerance of excess water, than has OPC.

2) Unpublished confidential Chamber of Mines consultancy report No. 61/92, Lamos, et al, December 1992.

classified power station fly ash, as widely used in the construction industry and also in the present investigation (SABS 0157); this composition reached a 28 day strength of 0,70 MPa. The other backfills in this test suite were prepared with steam plant ash from the coal-chemical industry and show substantially higher strengths - up to 0,92 MPa. The strengths of these samples are, furthermore, differentiated according to the particle size distribution of the ashes, the finer ashes from precipitator field 3 showing the highest strength; conversely, relatively coarse ash from precipitator No 1 shows the lowest values of the group. A proportional production blend of ashes of the various fields is still substantially stronger than the classified commercial fly ash.

During 1992, unclassified power station ash has become commercially available in South Africa, which performs somewhat worse than the classified material, but at about one fifth of the cost of the classified ash. Careful evaluation of available pozzolan sources, particularly in relation to geographical location, can clearly benefit the cost-effective application of binders in cemented backfills.

Figure 4.23 compares the relative strengths of the investigated aggregate types, using OPC and OPC/FA 1:2 binder. The previously described trends are apparent: in all cases, the OPC compositions are superior to the OPC/FA blend after 28 days curing, but both, the classified tailings as well as the comminuted waste backfills show the rapidly closing gap in strengths, as the OPC/FA binders catch up with portland cement.

Curing temperature

Figure 4.24 shows the effect of curing temperature on the strengths of cemented backfills (Lamos and Bradley, 1987 b). The backfill material for this test series was composed of full plant tailings and 10 % (in dry mass) of an OPC/FA 1:2 binder, at a water/solids ratio of 0,28. The samples were cured and tested as 100 mm cubes; this, combined with their low water/solids ratio accounts for their relatively high strengths.

As expected, elevated curing temperatures accelerate the gain in strength (i e the hydration reaction) of cemented backfills; this is provided that full saturation is maintained and the temperature does not exceed a level, at which cement crystal growth is impaired, most likely by the excessive evaporation of reaction water: it can be seen that with this backfill, the 28 day strength dropped rapidly above a curing temperature of 60 °C. It can be concluded that for this backfill composition 50 °C is the maximum safe curing temperature. If environmental conditions were such, that this temperature was exceeded, the material may not reach it's full strength and it's composition would have to be reconsidered.

Yield strength

The yield strength of a cemented backfill is defined as the highest compressive strength within the initial elastic region of the compression curve (point 3 in Figure 4.2). The yield strengths of the cemented backfill materials amount, on average, to about 72 % of their maximum compressive strengths (Figure 4.25); this result has important implications in the design of cemented fills, as it defines the limit of elastic loadi... ı curing cemented backfill can be loaded with no detrimental result, provided that the yield stress of the material is not exceeded at any time (Section 4.1.8, loading during curing).

Strain

The strains at the primary compressive strengths lie mostly between 0,5 and 2 per cent and there is a slight tendency for them to decrease with higher material strength (Figure 4.26). This tendency is a result of the increasing plasticity of cemented backfills of low strength. The lowest and highest recorded values, out of 165 measurements, were 0,55 and 3,83 per cent, respectively. The trend holds with curing time and the resultant increase in compressive strength. The large scatter of data points is due to the wide range of backfill strengths included and the associated wide range of sample plasticity.

Young's modulus

Cured cemented backfills exhibit two separate domains of elasticity, shown schematically in Figure 4.2: the binder-induced primary elastic region (point 2) and the elastic component of the cohesionless compression curve after the collapse of the binder structure (points 7).

The Young's modulus (or modulus of elasticity, E) considered here is that of the binder-induced primary compression peak. In a compression curve as complex as that of cemented backfill, in which the binder structure collapses during compression, the slope of the tangent to the curve changes at almost any point. It is, therefore, practical to determine the elastic modulus of the material in its intact elastic region, i e on the linear rise of the binder-induced peak (point 2 in Figure 4.2). The initial elastic modulus of a cemented backfill is a measure of its ability to carry a load without permanent deformation.

Figure 4.27 shows the relationship between the primary compressive strengths of cemented backfills and their initial Young's moduli. It can be seen that, overall, the tested backfills were somewhat less stiff for any given compressive strength, than the materials investigated by Swan (1985) and Helms (1988 a); this is not unexpected, as the backfill materials investigated by Swan and Helms are very different from the present suite.

A closer analysis of the data reveals that the cemented backfills are relatively less stiff when fully cured, than before. This can be ascribed to the material properties of the cement crystals themselves, which, with time, grow in numbers, but do not become any stiffer, i e the cohesion of the samples increases more than the elastic modulus. There is no clear trend in the relationship, regarding the type of backfill; comminuted waste backfills are no stiffer than tailings fills. Again this is ascribed to the properties of the cement crystals: since they are the cause of the primary compressive strength, it is *their* performance which determines the Young's modulus.

In conclusion it can be said that the initial elastic modulus of a cemented backfill is principally determined by the stiffness of the calcium silicate hydrate crystals (CSH) which form the rigid framework within the backfill. The backfill aggregate itself is at that stage not part of the elastic behaviour. The only other modifier of the stiffness of the cement gel is excessive micro-porosity, caused by an oversupply of reaction water. To achieve high elastic moduli, therefore, the highest feasible cement/water ratio (the minimum amount of water) must be used.

4.1.6 Confined Uniaxial Compression

The compression characteristic of cemented backfills change with the confinement condition of backfill bodies; the volumetric shapes of such bodies are merely instrumental in effecting specific confinement conditions (Section 4.2). As the unconfined, uniaxial compression of a 0,5 width/height ratio cylindrical specimen represents one end of the characteristics spectrum, confined compression represents the other extreme. All other forms, including triaxial compression (Section 4.1.7) are intermediate to the above extremes.

Confined compression was measured on cylindrical specimens of width/height ratio of approximately 2, 120 mm in diameter and about 60 mm high, depending on the porosity of the constant mass sample. Confinement was provided by a floating steel jacket and the sample was compressed by a pair of drained pistons (Appendix A). Since the massive steel jacket provided virtually infinite confinement, the slightly varying width/height ratio had no influence on the results.

The strength of cemented backfill in confined compression is of specific importance in the provision of regional support in deep mining and as a result of the fact that backfill ribs are normally placed adjacent to each other, forming large, effectively infinite, confined bodies. Because of the direct bearing on regional support requirements, the graphs illustrating confined compression behaviour of cemented backfills are included in Chapter 2.

Figure 2.9 compares the confined compression curves of equivalent uncemented and cemented backfills. The graph shows that the backfills were compressed to approximately 175 MPa, far above the primary binder-induced compressive strength of about 1 MPa. At that scale, the effect of binder addition to the material cannot be seen in the initial rise of strength within about 1 % strain, but very clearly by the significantly reduced ultimate strain (b-value). Figure 2.8 shows that the binder-induced primary peak is there, albeit only barely visible, embedded within the general compression curve.

For the regional support effort, that primary peak is insignificant in comparison to the potential benefits of the binder-induced reduction of porosity. Figure 4.28 demonstrates that by replacing 10 % of the tailings mass in a typical classified tailings backfill with portland cement, this backfill's porosity after full hydration of the cement is effectively reduced from 48 % to somewhere between 42,5 % and 45,3 %, depending on the compressibility of micro-pores in the gel. The reduction of the b-value shown in Figure 2.9 is, therefore, not unexpected and under ideal conditions can amount to more than 25 %.

4.1.7 Triaxial Compression

To estimate the behaviour of backfill in a large mass under compression, the confinement of an actual sample from the backfill body, or of a representative sample from that body is reproduced, as far as possible, in a triaxial test cell. A large amount of work has been done on the triaxial compression behaviour of uncemented and cemented tailings backfills and rockfills (1), and more research in this field will undoubtedly be forthcoming. In this study, however, the purpose of measuring the basic triaxial parameters of cohesion and friction angles was to determine general relationships between those parameters and the cement/water ratios and curing times of a wide range of different backfills.

Undrained triaxial tests were undertaken on 28 day old samples of the first test series (Table 4.1), using a *Hoek* triaxial test cell; the results are listed in Table 4.3 and plotted in Figures 4.29 and 30.

Figure 4.29 shows the relationship between cemented backfill cohesion and the cement/water ratio. Generally, the cohesion increases in direct proportion to the cement/water ratio. A more detailed examination shows that some sample groups are more sensitive to the cement/water ratio than others: the water/solids ratio group

1) For instance: Blight, (1977, 1988), Clark, (1988 c), Helms, (1988 a).

Table 4.3 Friction angles [deg] and cohesions [MPa] of the tested cemented backfill compositions. Triaxial tests using a Hoek test cell, curing time: 28 days.

Group Sample		Friction Angle ϕ' [deg]	Cohesion c' [MPa]
Aggregate Type			
	FT	33	0,27
	CT	43	0,22
	CW	44	0,93
Water/Solids Ratio			
	0,28	36	0,57
	0,33	38	0,42
	0,44	30	0,26
Binder Content			
	3 %	27	0,25
	6 %	27	0,34
	20 %	30	0,82
Binder Type			
	OP 1:2	27	0,30
	OP 1:1	34	0,23
	PBFC	35	0,52

All samples cured at 30 °C, tested at full saturation.
For abbreviations see Table 4.1.

shows a very steep increase in cohesion with reduced water content, whereas the cement content group virtually follows the average trend. As expected from the numbers of cementitious bonds and the deleterious effect of macro-porosity, the samples' cohesions reflect their unconfined, uniaxial compression performance.

Figure 4.30 shows the relationship between the backfills' friction angles and the cement/water ratio. Again, there is a general increase of friction angles with higher cement/water ratios, but the scatter is high and the relationship is not very satisfactory. The lowest friction angles are measured with fine grained tailings backfills of low cohesion; increasing cohesion raises the friction angle. Conversely, the highest friction angles are found with coarse grained backfills of high cohesion.

Figure 4.31 shows the results of a separate suite of tests, in which the increase of the friction angle with increasing compressive strength was demonstrated graphically. The samples consisted of full plant tailings, containing 10 % portland cement in dry mass, at a water/solids ratio of 0,39 (i e the control backfill composition). Increased cohesion was provided by the progressing formation of cement crystals in the samples, curing for up to 28 days. The figure shows the gradual steepening of the shear angles of the (200 kPa confinement pressure) specimens. The friction angles of the samples were then calculated, using the relationship:

$$\alpha = 45° + \frac{\Phi}{2} \qquad \text{(Eqn 4.1)}$$

where α is the observed shear angle (Vickers, 1983). The friction angle increased gradually from 24,6° in the 1 day old sample to 41,0° in the 28 day sample. In the absence of any other variable, the increased primary compressive strength (increased cohesion, reduced porosity) of the samples with increasing curing time must be responsible for this rise of the friction angle.

4.1.8 Loading During Curing

The hydration of binder components in cemented backfill gradually builds within the backfill a framework of calcium silicate hydrate crystals, which bond the mineral grains into a rigid structure. If the backfill components are properly mixed, crystallisation will commence more or less simultaneously throughout the fill and within a day provide a measure of rigidity to the material. If this cemented backfill is strained during the curing process, the already existing framework of CSH crystals undergos elastic deformation and, provided the yield point of the crystals is not exceeded, normal curing should continue unabated, protected by the existing rigid structure.

In a set of experiments (Lamos and Bradley, 1987 a), six cemented backfill specimens were loaded at a constant strain rate during the curing process, to determine whether the straining of the material had any deleterious effect on their primary compressive strength. Compared to the unstressed control samples, a 20 % deterioration of the compressive strength and the elastic modulus was measured, which was directly attributable to the straining of the backfill. In a subsequent test series, the constantly strained samples again showed much reduced strengths (30 % lower), but a parallel suite of samples, which were stressed with a static, constant load during curing (a creep test), which never exceeded the yield point, showed no deterioration at all.

From the strain rates and the normal strength development of the control samples, it was concluded that cemented backfills can indeed be loaded during curing without adverse consequences, provided that the yield point of the material is at no time exceeded. The elastic stressing of curing samples has no deleterious effect on final strength, as cement bonds continue growing within the structure - no bonds are lost. Once the material is stressed beyond the yield strength, however, cement bonds are physically broken and are irrevocably lost in terms of final cohesion.

Figure 4.32 shows a plot of the yield points (yield stress at yield strain) of a suite of cemented backfill samples of bench-mark composition (Section 4.1.4). With the curing time advancing from 3 to 90 days, the progression of the yield point can be followed up the graph. The upper limit of the observed strengths represents the lowest yield strains at any one time and therefore the safe limit strain for the backfill. Any stope closure profile to the left of this limit strain can be tolerated without deterioration of compressive strength. For this backfill material, this strain amounts to about 0,004 % per day, a minuscule rate, and it must be concluded that a backfill under normal closure rates will suffer some deterioration of strength.

In the light of these findings, two factors work in favour of in situ cemented backfill strength: backfill shrinkage, due to the drainage of water from the fill, effectively ameliorates the closure rate, and the fastest strength gain in cemented backfills occurs in the first few days, allowing the backfill a head-start in strength development.

4.1.9 Closure Rate

The effect of the strain rate on the strength of cemented backfill was determined in a series of experiments on standard 100 mm cube samples (Bradley, 1986). The backfill consisted of full plant tailings aggregate containing 10 % of a portland cement/fly ash 1:2 binder, at a water/solids ratio of 0,28. The samples were tested at strain rates of 1 %, 10 % and 1000 % per hour, after 3, 7 and 28 days curing, respectively. Table 4.4 lists the mean strength parameters of the tested backfill samples. Figure 4.33 summarises the test results.

Table 4.4 Unconfined, uniaxial compressive strengths of cemented backfill samples at a range of strain rates and curing times. Mean stress [MPa], strain [%].

	Curing Time [d]					
Strain Rate	**3**		**7**		**28**	
[%/h]	**Stress**	**Strain**	**Stress**	**Strain**	**Stress**	**Strain**
1	0,78	3,69	1,11	2,89	1,84	1,97
10	0,66	3,07	1,02	2,60	2,06	2,03
1000	0,71	3,47	1,15	2,69	1,88	1,[illegible]7

From the test results it was concluded that the compressive strengths of the cemented backfill cube samples were insensitive to the strain rate. The findings of this test series led to the adoption of the 100 mm/h strain rate, used in most of the subsequent testing of cemented backfills at COMRO.

In Chapter 2 it was stated that under rockburst conditions deep mining stope support is subjected to very high closure rates. Current stope support design recommendations provide for closure rates of up to 3 m/s, which would account for about three quarters of all rockbursts in deep gold mines (Ryder, 1988). At a closure rate of 3 m/s, a 1 m wide stope would close completely in 0,33 seconds, a closure rate three orders of magnitude higher than what was measured in the above test series.

To test how cemented backfill behaves under these loading rates, 50 mm high rib samples of classified tailings, containing 10 % OPC, at a water/solids ratio of 0,27, were cured for one day and then loaded at a closure rate of 0,15 m/s, the velocity equivalent to a closure rate of 3 m/s in a 1 m wide stope (1) . Figure 5.5 shows the arrangement of the model backfill ribs for testing. The purpose of this particular test arrangement will be explained in Chapter 5; for this aspect of the experiment, it is only necessary to know, th[illegible] identical control sample was tested at the standard 100 mm/h closure rate.

Figure 4.34 compares the traces of the three loading test results. It can be seen that despite a 5400 fold increase in the closure rate, the backfill specimens essentially reacted identically: all three specimens show the primary, binder-induced elastic deformation, similar primary compressive strengths and similar post-failure behaviour (2) . Had the backfill samples been saturated with water, their compression behaviour may have been different: given identical permeabilities, the slowly loaded samples may have been able to exude their pore water and compress in a controlled fashion, whereas the rapidly loaded backfills may just not have had the time to expel their pore water and explode under impact load. These aspects are important to the behaviour of backfill under rockburst conditions and merit further investigation.

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1) It is understood, that only the vertical closure velocity can be scaled in this way. The horizontal velocity, attained by the dilating fill, increases non-linearly with closure, the function depending on the shape of the backfill body. In a real, full-scale event, the inertia of dilating backfill could make it extremely dangerous.

2) The traces of the fast loading curves were extracted from stress and displacement curves recorded by the data logger of the TerraTek press. Since the trace shown represents a time span of only 50 milliseconds, the original traces of the recorded variables are quite jagged. The resultant curve of combining two of these originals is, therefore, even worse. There is no doubt about their validity, however.

4.2 Effect of Backfill Geometry

In the preceding section, hardened cemented backfills were viewed with an emphasis on material properties, particularly their primary compressive strengths, as they related to compositional and environmental factors. Once a backfill has been mixed according to a specific composition, and cured at certain environmental conditions, it will have developed reproducible material properties, but its load bearing capacity may still vary considerably, since this is strongly influenced by yet another variable: the shape of the backfill mass; that is the subject of this section.

In Section 4.1.2, it was stated that the loading response of a cured cemented backfill comprises three distinct phases: an initial elastic stage, in which the backfill is effectively deformed as a single, rigid body, an intermediary phase of binder crystal breakdown and backfill collapse, and a strain-hardening stage where the backfill compresses essentially as a cohesionless soil. Looking again at Figure 4.2, it has been established, that the primary compressive strength, (4), is produced by the binder in the backfill. The ultimate strain value, (8), is a result of inherent backfill porosity, as well as the contribution made by the residual cement crystal fragments, in reducing overall porosity. The domain controlled by the *shape* of the cemented backfill mass, is the intermediary post-failure region, indicated by (5).

In deciding on the experimental parameters for this test series, several choices were available on how to alter sample shapes for the tests, each of which would have furnished slightly different results. If the volume remained constant, to eliminate potential scaling errors, both, the width and the height of the samples would have to change, in practice producing some samples only a few millimetres high, where the particle size may begin to influence results. If the sample cross-section remained constant, their height and volume would change with different width/height ratios. Eventually, the third option was selected, whereby the sample height remained constant, and the cross-section was altered. It was felt that this most closely modelled real conditions of a more or less constant stoping width. Furthermore, it will be argued in the next section, that because of the constant sample height, the effect of increased sample volume on the tested backfill specimens is not as critical, as it may have been for other testing geometries.

4.2.1 Sample Volume

A large body of literature exists on the behaviour of geometrically scaled materials and structures, because of its fundamental importance to the design and modelling of engineering and civil structures. Detailed studies were undertaken for the testing and design of mine support pillars, of particular relevance being the studies in coal (Bieniawski, 1968, Madden, 1988) and in cemented backfill (Gonano, 1975). Bieniawski found that cube-shaped coal specimens of side length larger than 1,5 m (3,375 m^3), showed constant compressive strength. Below 1,5 m, down to about 6 cm side length, cube samples showed increasing strengths; whereas below 6 cm side length, cube strengths were again constant. In the cited reference, Bieniawski shows the sample volume to strength relationship, but does not offer an equation to predict strengths in this intermediate range below 3,375 m^3.

Natural rock and coal samples are, however, substantially different from homogeneous cemented backfill samples and their behaviour under load can strictly not be extrapolated. Besides a large difference in their compressive strengths (primary compressive strength in the case of backfill), rocks and coal contain natural flaws and discontinuities which were introduced either during their formation or during subsequent tectonic stressing. In comparison, cemented backfills, especially fine grained materials, are very homogeneous with a statistical distribution of minor flaws such as air voids.

Cemented rockfills can by their nature be expected to contain more substantial flaws and Gonano, working with cemented tailings backfills and cemented rockfills presents similar findings to Bieniawski's, in that he sets the minimum volume of backfill samples, above which strengths remain constant at between 2 m^3 and 5 m^3. But Gonano also shows that cemented hydraulic backfill is not very sensitive to the sample scale: for a million-fold increase in sample volume, the compressive strength changes from about 1,9 MPa for a 2 cm cube, to about 0,9 MPa for a 2 m cube.

If the results, presented in the following section (Figure 4.37), were corrected, applying Gonano's relationship, the effect would be to accentuate the findings somewhat, in that the compression curve of the smallest sample (the unit cube) would be lowered by about 15 % - at 8,2 % strain from 1,3 MPa to 1,1 MPa. All other curves would re-adjust somewhat between the new low curve and the present highest one.

The increase of the volume of the constant height samples, as applied in this study, is however not the same, as scaling a test cube. As the side length of a cube is increased linearly, geometrically more flaws are added into the potential shear zone, triggering failure at progressively lower stresses, until the statistically largest flaw size or discontinuity in the material determines the bulk strength at, say, 5 m^3 volume.

The lateral size increase of the constant height sample, on the other hand, can be interpreted as placing more unit cubes next to the original sample, thereby merely increasing lateral confinement. Even though the failure mechanism of a plate involves shearing, as does the failure of a cube, the flaws on the 'inner' shear planes of a plate do not trigger failure in the same manner. The centre of the plate does not fail because of increasing numbers of material imperfections, it fails by being forced out from between the platens, just the same way, as a perfectly flawless sample would. The shear planes appear at the edge of the plate in the same dimensions, as they occur in the single cube (Figure 4.35).

In practice, therefore, the increased *volume* of a tabular backfill body is only of minor consequence to the strength of the backfill body. Only if the sample size were decreased below the statistical distance between the minor flaws, would a slight increase in strength be expected.

4.2.2 Sample Cross-Section

In an attempt to determine the influence of the geometrical shape of the sample cross-section on their compressive strength, a set of width/height ratio backfill samples with square and circular cross-sections were tested (Bradley, 1987 a). From the results, summarised in Figure 4.36, it was concluded that there is no significant difference between square and circular samples of otherwise identical dimensions. This finding allowed the confident comparison of the standard cylindrical test samples (W/H 0,5), cylindrical confined compression samples (W/H 2,0) and the square and rectangular samples, described in Section 4.2.3 below.

4.2.3 Width to Height Ratio

Most research work on the effects of specimen volume on their compressive strength, was inevitably also concerned with the shape of specimens, commonly expressed in the form of the width to height ratio of samples. Bieniawski (1968) found that for coal specimens, of smaller than 1,5 m maximum dimension, the following relationship holds between compressive strength and width/height ratio:

$$\sigma = 0{,}3833 \frac{W^{0.16}}{h^{0.55}} \quad \text{(Eqn 4.2)}$$

For specimens larger than 1,5 m, with W/H $\geq$ 1,0, the following relationship applies:

$$\sigma = 0{,}1394 + 0{,}0767 \frac{w}{h} \quad \text{(Eqn 4.3)}$$

where σ is the specimen compressive strength in [MPa], and w and h are the specimen width and height, respectively (1). Again it must be stressed that because of natural discontinuities, coal samples are substantially different from essentially homogeneous cemented backfill samples and that Bieniawski's equations are not, strictly, applicable to cemented backfills.

Figure 4.37 summarises the results of the suite of cemented backfill samples tested to determine the effect of the width to height ratio on sample compression behaviour. The full plant tailings backfill contained 10 % portland cement, was prepared to a water/solids ratio of 0,33 and was tested after 14 days curing. The samples were

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1) The equations are modified after Bieniawski, who used imperial units.

48,8 mm high and the 'unit cube' thus had a volume identical to that of the standard 42 mm diameter test cylinders. Since the cross sectional areas of the samples changed, all results were re-calculated and normalised to the unit cube area.

It can be seen that, essentially, all constant height samples showed the same primary compressive strength (1). This result is in line with the discussion in Section 4.2.1, where it is stated that a lateral increase of a constant height sample should not strictly be interpreted as a direct increase of sample volume. Initial sample failure (yield) is dependent on the bulk cohesion level and is not significantly influenced by any introduction of increasing numbers of randomly distributed, failure-triggering flaws into potential shear zones.

In the post-failure region, i e after the collapse of the cohesive cement structure, the backfill samples effectively behaved as cohesionless granular materials. Specimens with width/height ratio of 1 (cubes) are crushed until sufficient frictional and gravitational confinement is provided for the centre of the sample to begin to take load by the compression of its porosity, as described in Section 3.2.7. As the widths of the specimens increase, progressively more effective confinement is provided and the samples can build up stress at increasingly lower strain values. This is indicated in Figure 4.37 by the level of compressive stresses generated in the samples at the arbitrary comparison strain of 8,2 % (4 mm closure). The samples of width/height ratios 5 and 7 approach the behaviour of confined compression, which, in effect, represents an infinitely wide sample.

For the tested cemented backfill, a width/height ratio of 1,5 delineates the geometry, above which the backfill body no longer shows a decline in the load bearing capacity with increasing strain - the load-deformation curve becomes monotonic. The tested rib samples showed marginally stiffer behaviour than the unit cubes, an expected result because of the slight confinement provided in one dimension.

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1) The scatter in the elastic region can be ascribed to the difficulties of loading large plate samples evenly.

It was explained in Section 4.1.6. that the high stress compression behaviour of permeable cohesionless backfills, specifically the maximum strain, or b-value (1), is directly related to the porosity of the material. Since all test specimens were composed of the same cemented backfill material, their porosity, i e the mass of solids per unit volume, is identical after any specific curing time. It stands to reason, therefore, that all specimens will reach the same stress level at the material-specific maximum strain. The differences observed in the 'post-failure' region are thus only of transient nature, eliminated as soon as a backfill specimen is compressed far enough to attain its confined compression width to height ratio of approximately 11 (2).

4.3 Summary and Conclusions

Chapter 4 was concerned with the load bearing properties of cured cemented backfill materials and how they relate to compositional, environmental and geometrical variables. The original, and largest part of the investigation, was carried out with Witwatersrand gold mine tailings and formed the basis of the guidelines for the design of cemented backfills, as it will be presented in the next chapter. Subsequently, the database was considerably enlarged with the incorporation of additional tests, including several suites of platinum tailings.

In the original investigation, twelve different cemented backfill compositions were tested to determine the influence of their material composition on their compressive strengths. This matrix was later extended to 22 different mixes, arranged in four main compositional groups, identified, *a priori*, as the most influential compositional variables on cemented backfill strength:

a) Aggregate type, i e aggregate particle size distribution,

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1) The definitions of the a- and b-values are given in Figure 2.9, the fitting functions for the uncemented backfill compression curves are given in Section 5.4.

2) In laboratory tests on cohesionless backfills it was established, that Witwatersrand backfills effectively reach confined compression conditions at a width to height ratio of 11,2 (Bradley, 1987 b, Briggs, 1985).

b) Water content, i e water/solids ratio,

c) Binder content and

d) Binder type.

Several non-compositional variables have been investigated; some were procedural elements of the investigations, others were studied in separate test series, they included:

e) Cemented backfill curing time,

f) Curing temperature and environment,

g) Backfill sample porosity and

h) Loading parameters.

Most tests in this first major investigation were concerned with the primary compressive strength peak of cemented backfills, as reflected by the unconfined, uniaxial compressive strength. This strength is indicative of a backfill's suitability for local stope and gully support duties.

From the results presented, it is evident that all of these variables influence the compressive strengths of cemented backfill materials. Under the various sub-headings in Chapter 4, as well as in the following applicable graphs, a large amount of data is presented, which has already proved useful in analyses and designs of practical mine backfill systems.

In a second major investigation, geometrical parameters were analysed for their influence on cemented backfill load bearing behaviour:

i) Backfill specimen volume,

j) Specimen cross-section and

k) Backfill sample width to height ratio.

If cemented backfills are loaded beyond the primary compressive strength peak, binder-induced cohesion is lost and the material compresses as a cohesionless backfill, albeit at reduced porosity. In this loading range, porosity and sample geometry become the dominant loading factors.

4.3.1 Material Composition and Environmental Factors

Aggregate type, aggregate particle size distribution. The findings indicate that the inherent backfill particle size distribution, which controls the porosity of the backfill, has a most significant influence on the strengths of the whole range of cemented materials. The relative strengths approximately follow minimum porosities of the uncemented backfill materials: the lower the inherent porosity, the higher the cemented backfill strength. The amount of ultra-fines ($\leq$ 0,010 mm) present in the fine backfill aggregate plays a very deleterious rôle: an excess of these fractions introduces very large surface areas to the backfill, with the associated high water demand and unfavourable cement/water ratio. At the same time, the permeability of the backfill is reduced by the ultra-fines, preventing efficient drainage.

Water content, water/solids ratio. Provided that a cemented backfill remains within practical pumping and placement limits, a reduction of its water content is comparable to an increase in binder content - it improves compressive strength significantly. The increase in strength can be ascribed to a decrease of the placed (macro-)

porosity of the fill, and to the more favourable cement/water ratio in the binder paste, substantially reducing the micro-porosity in the cement gel. An excess of water has a detrimental influence on all types of cemented backfills, regardless of binder type.

Binder content. Cemented backfill samples increase in strength approximately in proportion to their binder content. This is due to a proportionally increased number of cement crystals forming in the pores of the backfill material, resulting in increased adhesion and reduced porosity. The addition of 3 % binder to a backfill was found in practice to be about the lower practical limit, particularly with blen of portland cement and pozzolans. Below 3 %, proper mixing of the binder into the backfill becomes a problem and large pockets of uncemented fill were found in practice.

Binder type. Of those binders tested in this series, the portland cement backfill shows the fastest initial curing rate. All binders containing pozzolans set out at a slower rate and retained their relative strengths throughout the curing period. Backfill prepared with portland blastfurnace cement exceeded the portland cement backfill in strength after about a week and remained substantially stronger from then on. This good performance is probably due to the fine grind of it's 50 % OPC fraction, which increases it's reactivity. The dilution of portland cement with fly ash reduces its effectiveness approximately in proportion to the mix ratio. Different fly ashes have different pozzolanic activities, however, and the strengths of cemented backfills can vary widely with the type of ash used.

Curing time. Cemented backfills attain strength through the growth of cohesive cement crystals, this happens at rates dependent on curing temperature and binder type. For practical purposes, the fill can be regarded as fully cured after about 90 days.

Curing temperature. Elevated curing temperatures accelerate the gain in strength, i e the hydration reaction of cemented backfills; this is provided that full saturation is maintained and the temperature does not exceed a level, at which cement crystal growth is impaired by excessive evaporation of reaction water. For conventional South African cemented backfills, this limit is approached at about 50 °C.

Backfill porosity. The compressive strengths of cemented and uncemented backfills are inversely proportional to backfill porosity. Various factors control porosity, the most important being the particle size distribution of the backfill aggregate, the backfill water content and mechanical vibration. The hydration reaction of binders in cemented backfills results in the growth of cement crystals, which build water into their crystal structures, thus creating solids from a proportion of the water present. These cement crystals grow in backfill pores, chemically reducing porosity, which results in significant improvements of the regional support capabilities of cemented backfills.

Cemented backfill confined compression. The confined compression test reflects a cemented backfill's regional support capability. This test involves compression to very high stress levels, in this study about 170 MPa. At that scale, the effect of binder addition to the material can barely be seen in the initial rise of strength, but is very noticeable by the significantly reduced ultimate strain, or b-value, which, under ideal conditions, can be reduced by more than 25 %. For the regional support effort, the primary peak is insignificant in comparison to the potential benefits of the binder-induced reduction of porosity. By replacing 10 % of the tailings mass in a typical classified tailings backfill with portland cement, the backfill's porosity after full hydration is effectively reduced from 48 % to somewhere between 42,5 % and 45,3 %, depending on the compressibility of micro-pores in the gel.

Loading during curing. Cemented backfill specimens were loaded at a constant strain rate during the curing process, to determine the effect on strength development. Compared to the unstressed control samples, 20 % to 30 % deterioration of the primary compressive strength and the elastic modulus was measured, which was directly attributable to the straining of the backfill. Samples, which were stressed with a static, constant load during curing (a creep test), showed no deterioration at all, provided the stress in the material never exceeded the yield stress. From the strain rates and the normal strength development of the control samples, it was concluded that cemented backfills can indeed be loaded during curing without adverse consequences, provided that the yield point of the material is at no time exceeded.

Closure rate. Test results indicated, that the compressive strengths of cemented backfill cube samples are insensitive to strain rates between 1 % and 1000 %. The findings of this test series led to the adoption of the 100 mm/h strain rate, used in most of the subsequent testing of cemented backfills at COMRO. In a subsequent test series it was shown that despite a 5400 fold increase in the closure rate, cured cemented backfill behaved essentially identical to control samples, compressed at standard laboratory closure rates.

4.3.2 Influence of Backfill Geometry

Backfill specimen volume. There are significant differences between rock or coal samples and samples of cemented backfill. Whereas rock or coal samples contain natural discontinuities and planes of weakness, either from their formation or from subsequent seismic stressing, fine-grained cemented backfill samples contain only statistically distributed minor flaws and can, on the whole, be regarded as homogeneous. The lateral size increase of a constant height sample, such as was used in this study, can be interpreted as placing more basic units next to the original sample, thereby merely increasing lateral confinement; the centre of the plate fails in ordinary shear and not because of increasing numbers of material imperfections. In practice, therefore, the increased *volume* of a tabular backfill body is only of minor consequence to the strength of the backfill body. Cemented rockfills are expected to contain more substantial imperfections and their primary compressive strengths may have to be corrected according to Gonano (1975).

Specimen cross-section. It was concluded that there is no significant difference between square and circular samples of otherwise identical dimensions.

Backfill sample width to height ratio. Compression tests performed on constant height samples of varying width show that, essentially, all samples show the same primary compressive strength. Initial sample failure (yield) is dependent on the bulk cohesion level and is not significantly influenced by any introduction of increasing numbers of randomly distributed, failure-triggering flaws into potential shear zones. In the post-failure region, i e after the collapse of the cohesive cement structure, the backfill samples effectively behaved as cohesionless granular materials. As the widths of the specimens increase, progressively more effective confinement is provided and the samples can build up stress at increasingly lower strain values. The samples of width/height ratios 5 and 7 approach the behaviour of confined compression, which, in effect, represents an infinitely wide sample.

For the tested cemented backfill, a width/height ratio of 1,5 delineates the geometry, above which the backfill body no longer shows a decline in the load bearing capacity with increasing strain. Single width rib samples showed marginally stiffer behaviour than unit cubes, an expected result because of the slight confinement provided in one dimension. The different compression behaviour observed in the post-failure region are only of transient nature, eliminated as soon as a backfill specimen is compressed far enough to attain its confined compression width to height ratio of approximately 11, when the ultimate strain attained is porosity-, and not shape dependent.

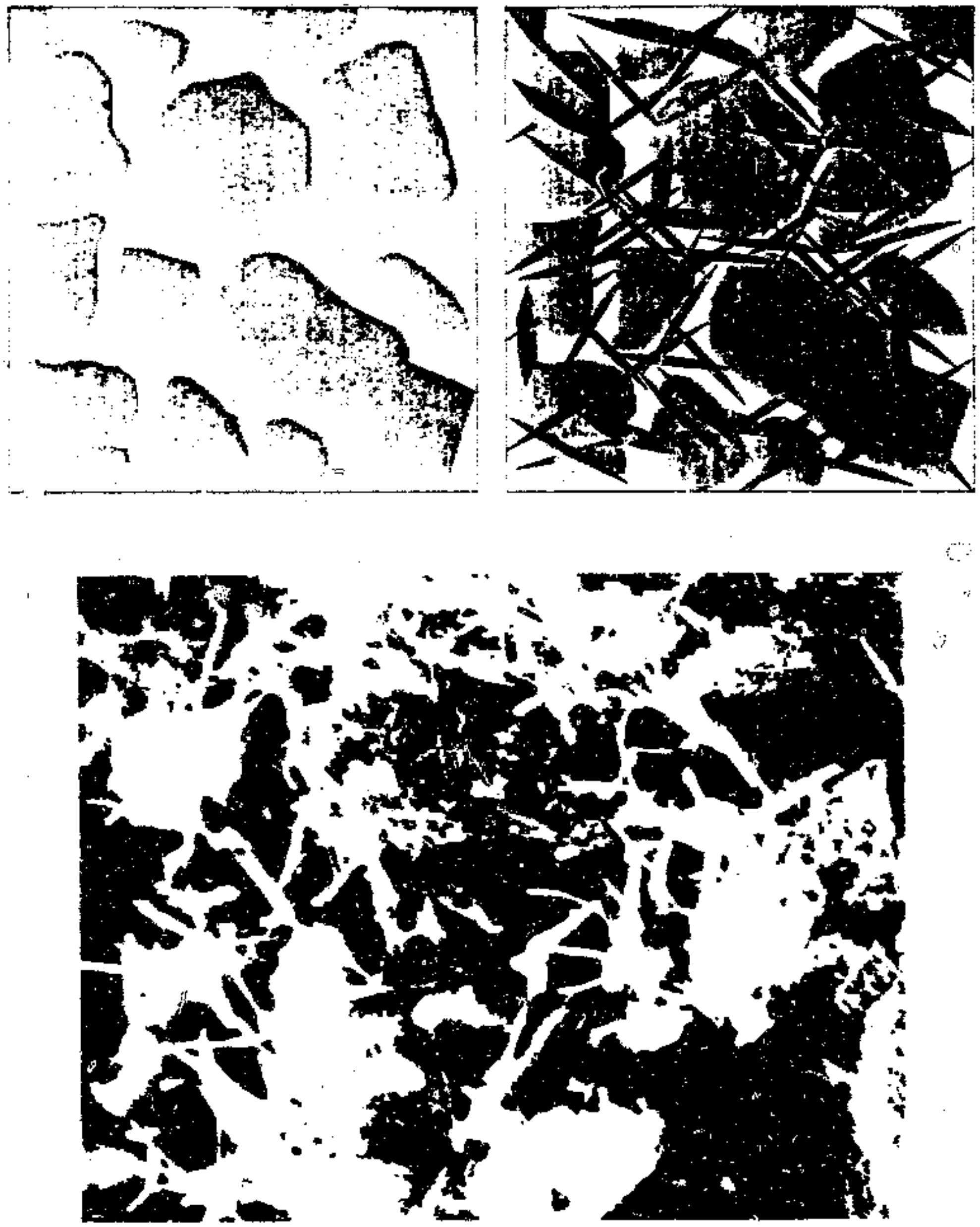


Figure 4.1 Schematic illustration and electron micrograph of backfill cementation.

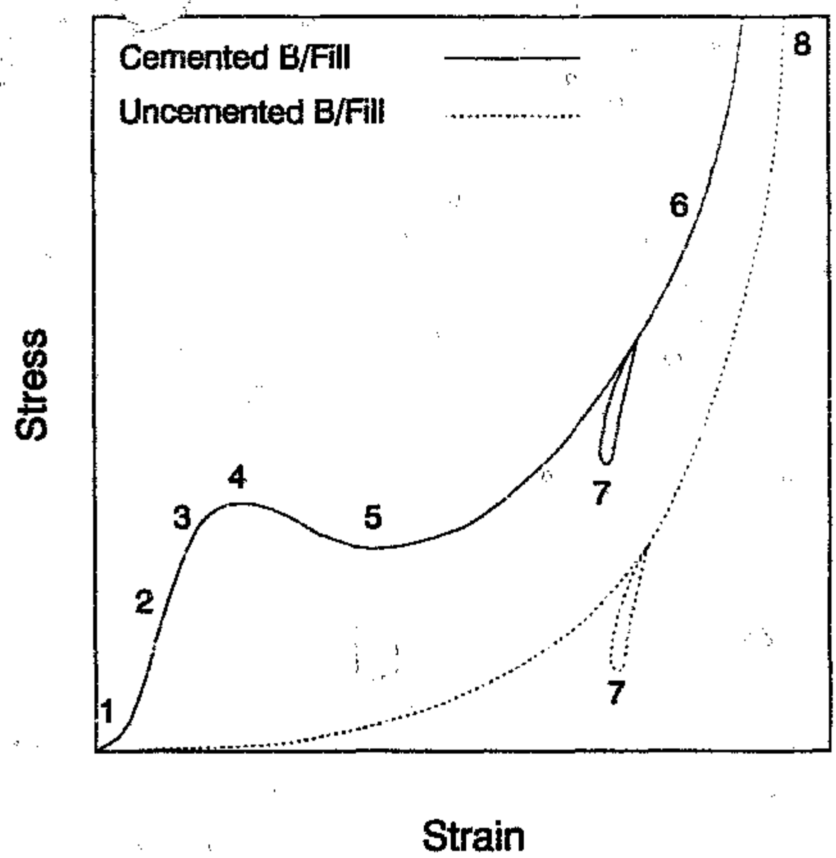


Figure 4.2 Schematic compression curves of cemented and uncemented backfills.

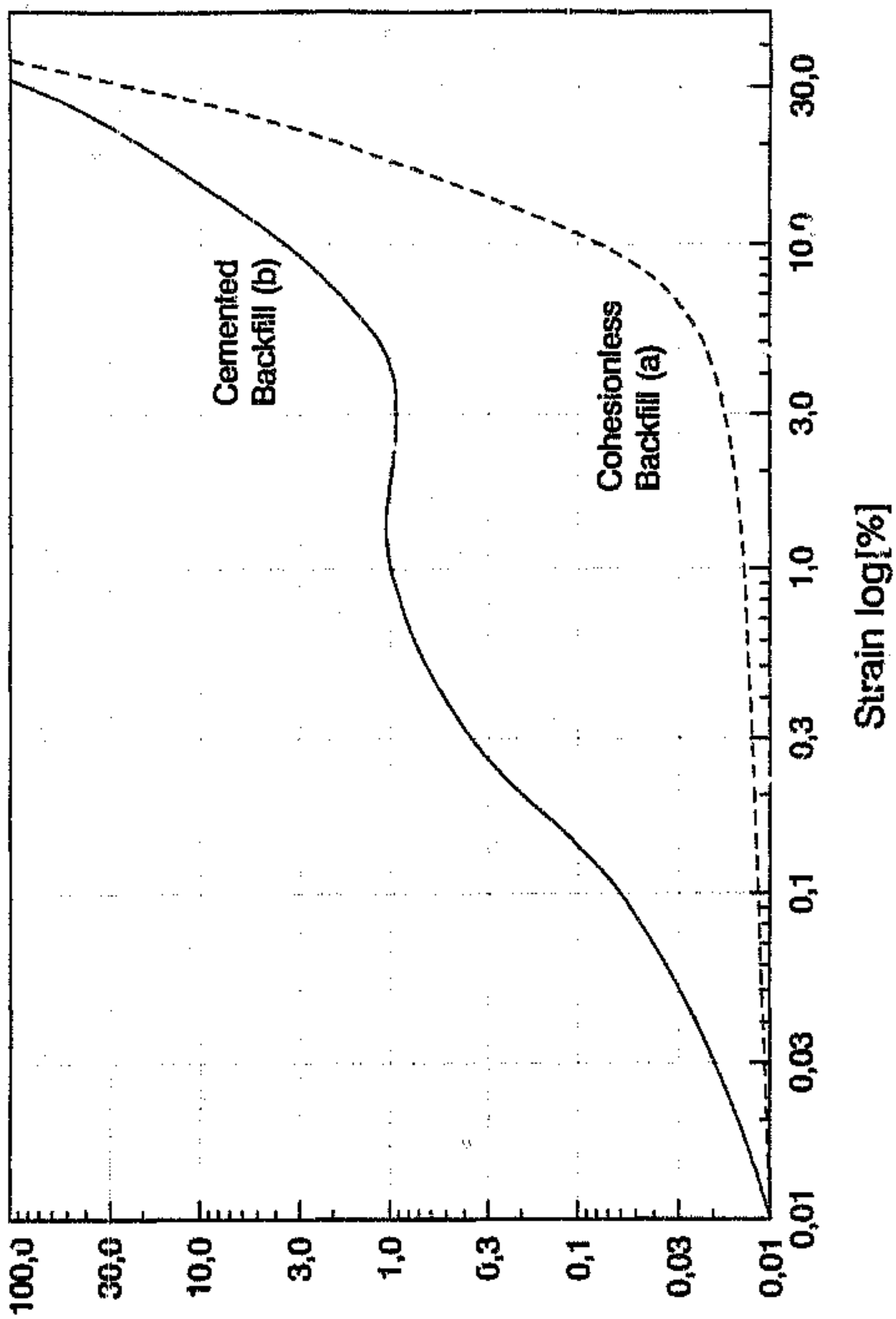


Figure 4.3 Compression curves of cemented and uncemented backfills, logarithmic scale.

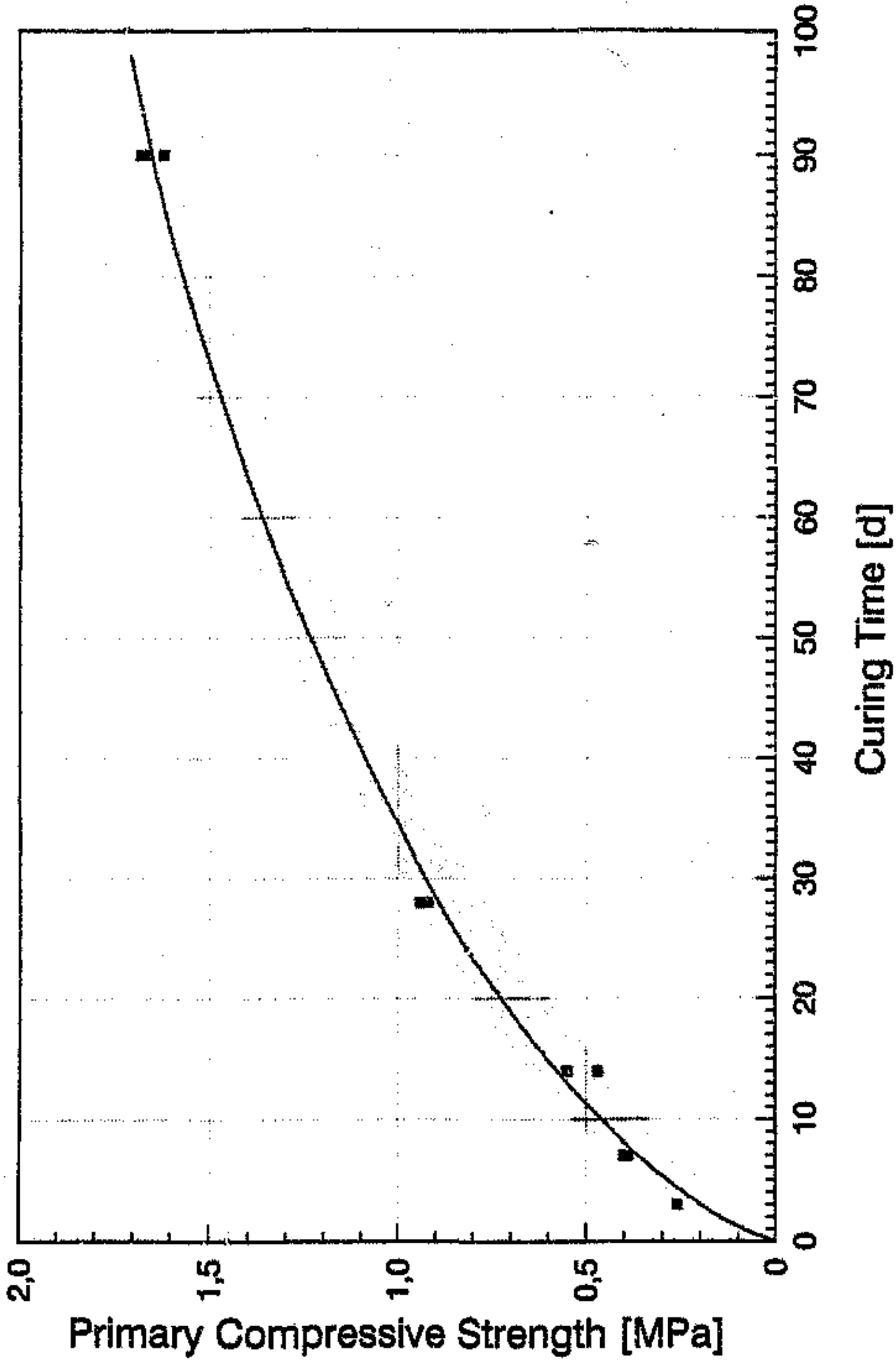


Figure 4.4 Range of primary compressive strengths of triplicate cemented backfill samples [MPa].

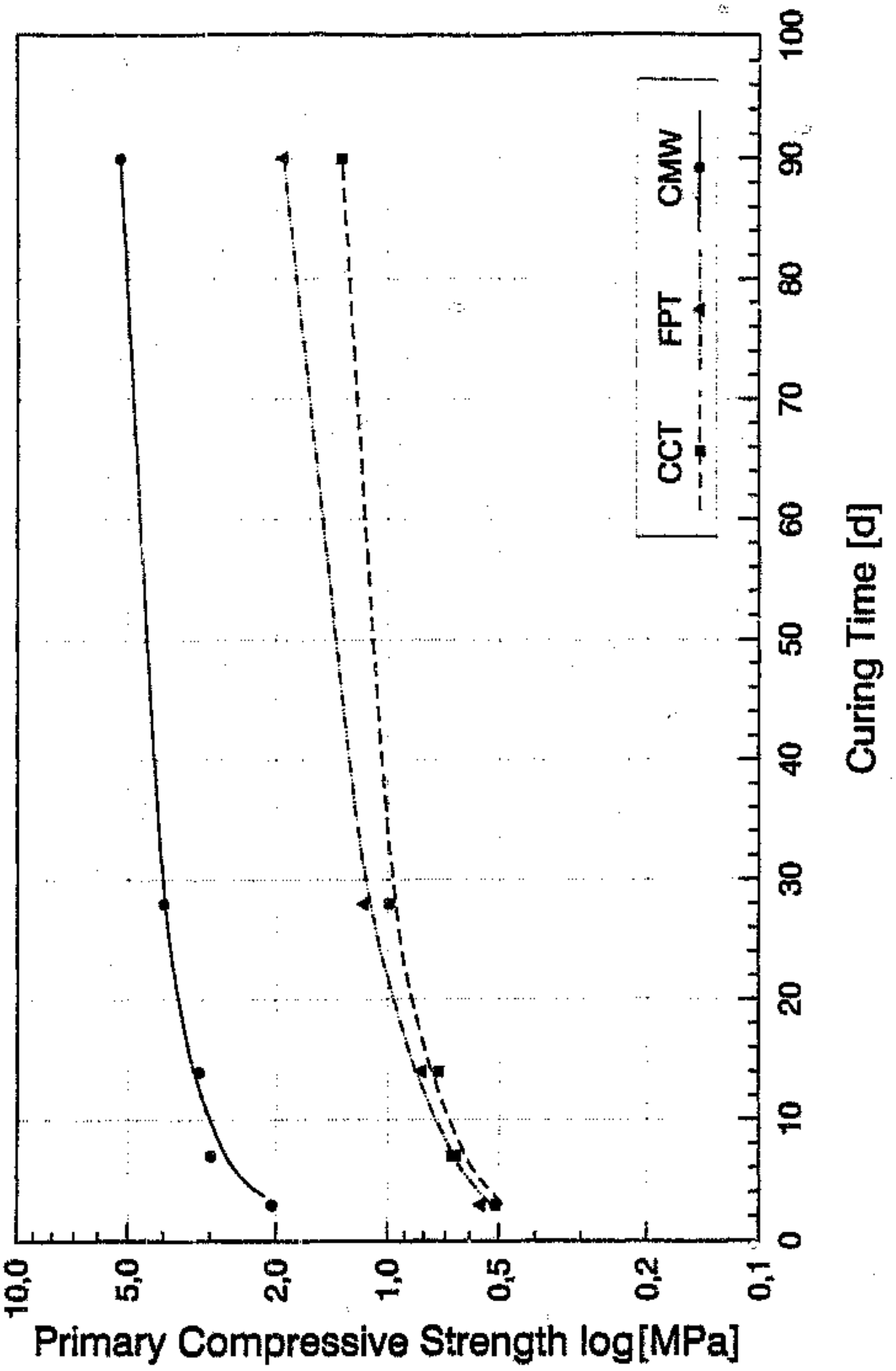


Figure 4.5 Primary compressive strengths of cemented backfills prepared with different aggregate types, full plant tailings (FPT), cyclone classified tailings (CCT) and crushed, milled waste (CMW), 10 % OPC, W/S 0,39 [log MPa].

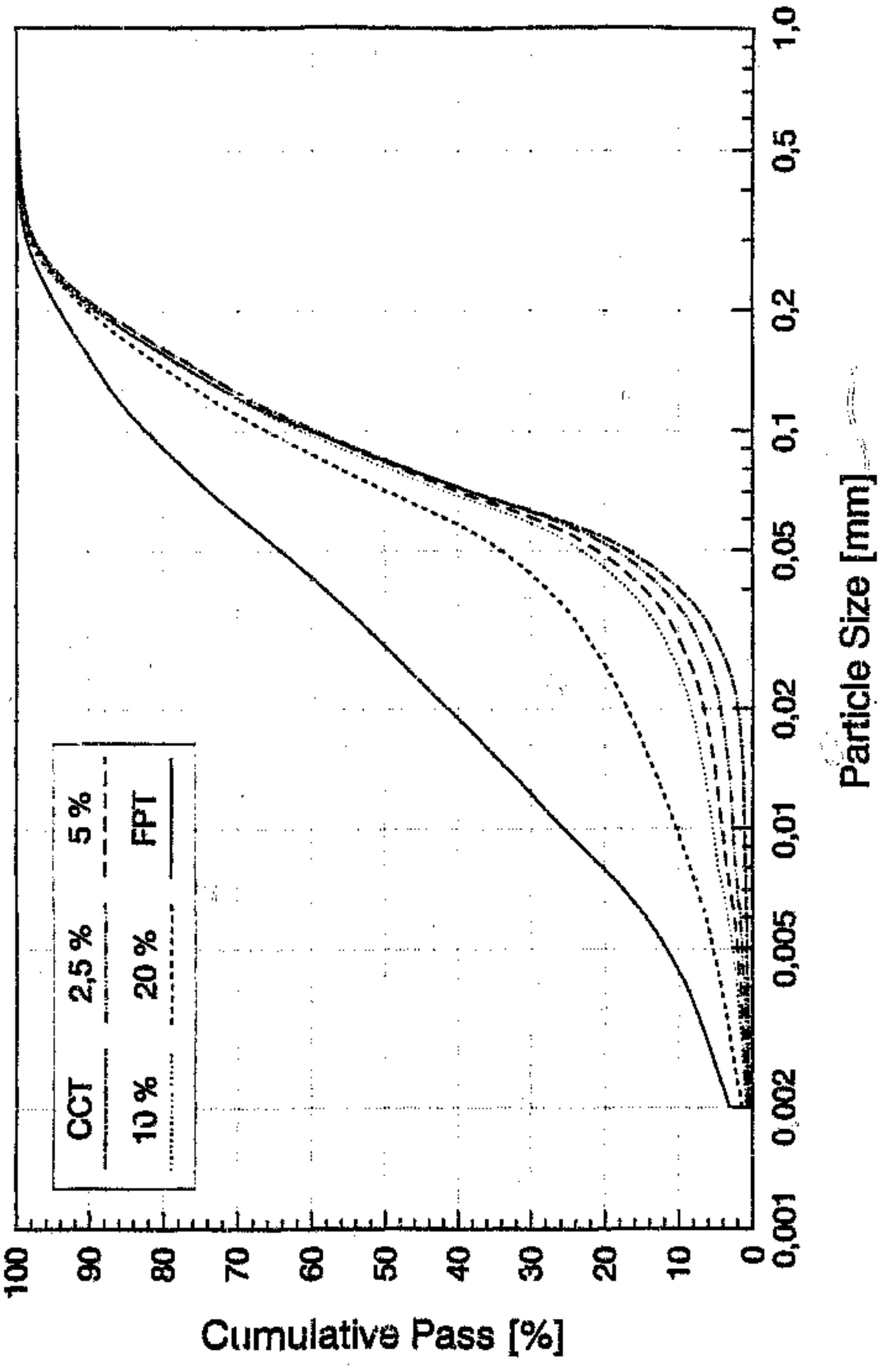


Figure 4.6 Particle size distributions of a full plant tailings sample (FPT) and a range of synthetic classified tailings (CCT) produced from it [Cumulative Pass %].

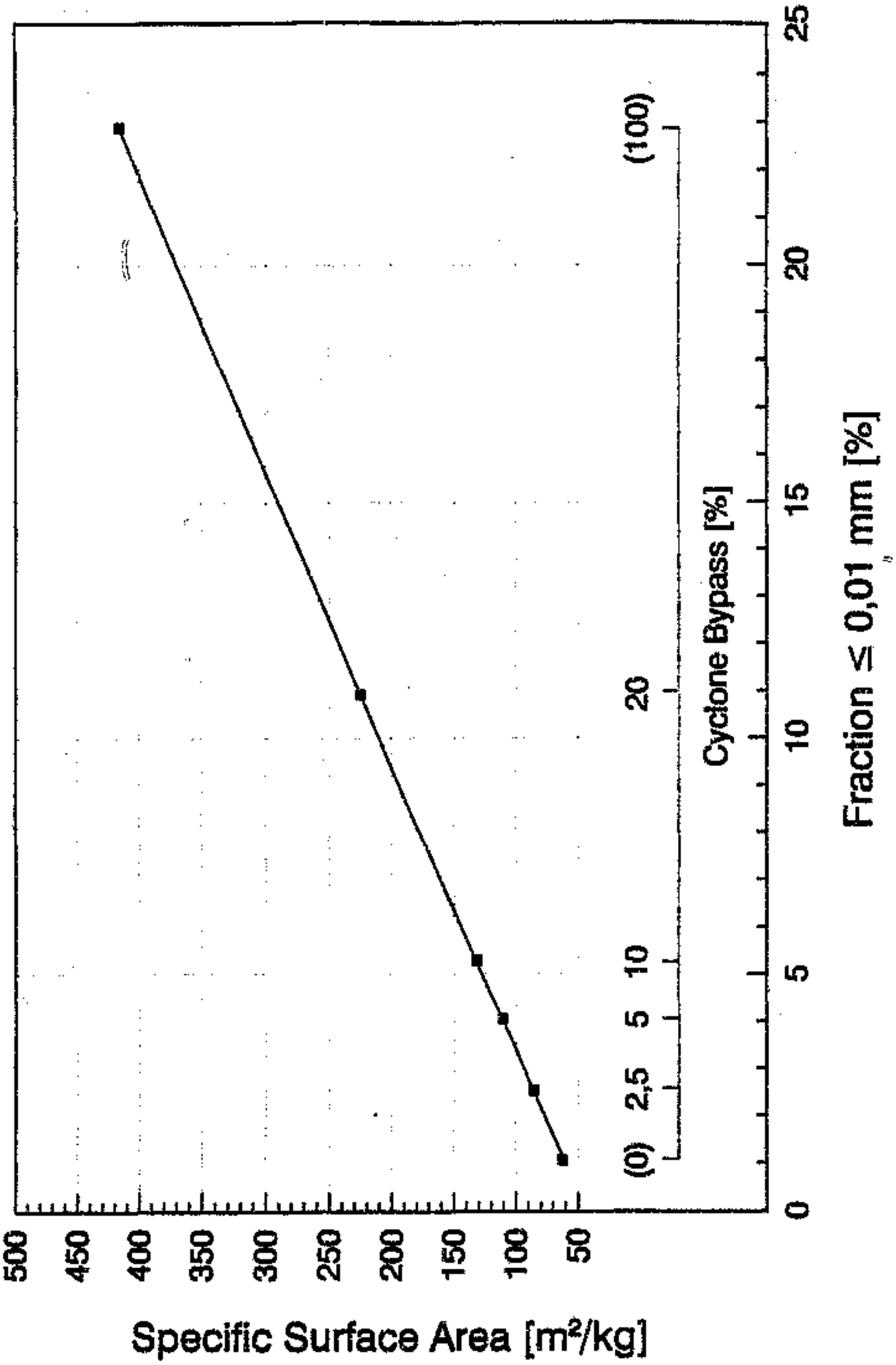


Figure 4.7 Specific surface area of hydrocyclone classified tailings products.

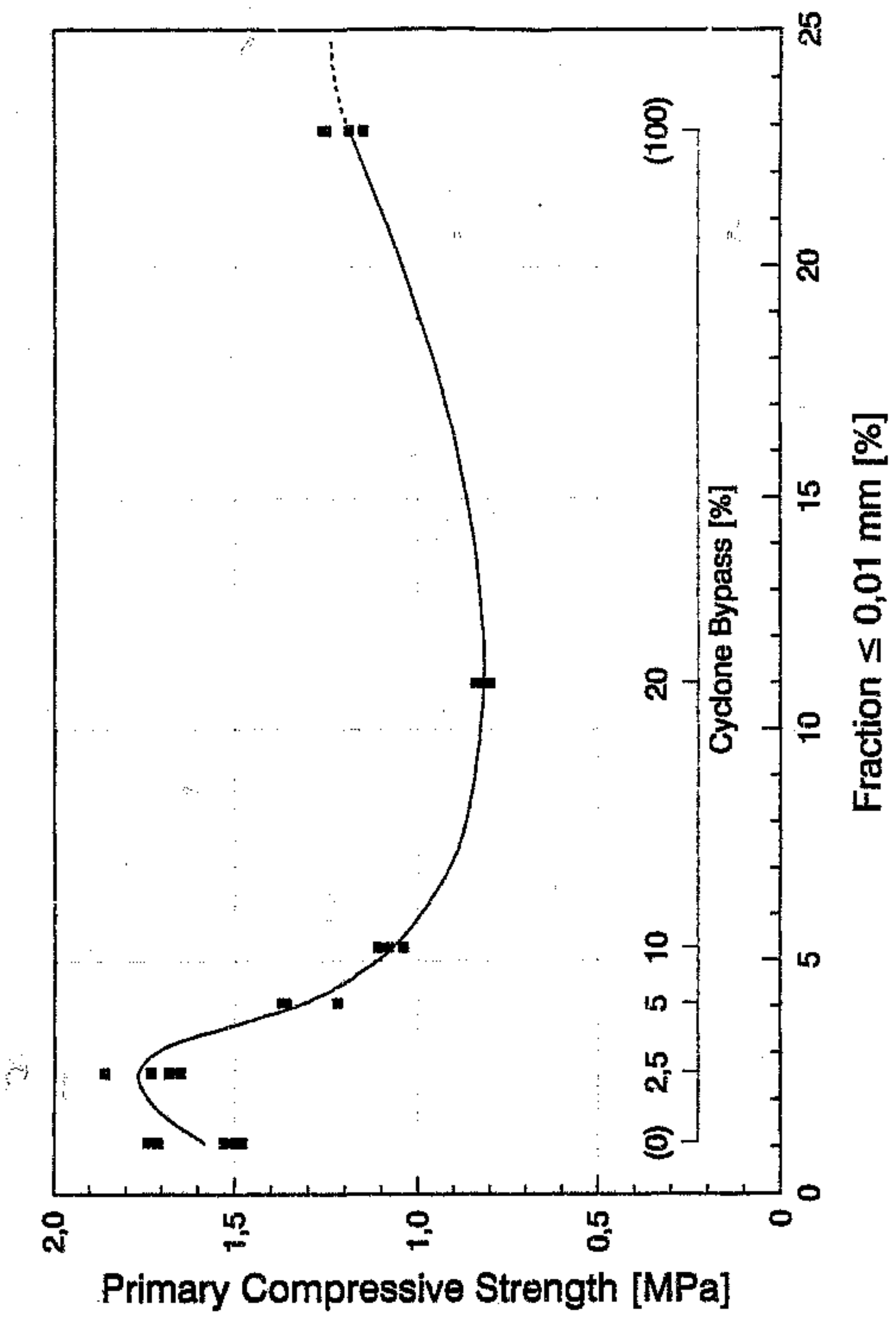


Figure 4.8 Effect of hydrocyclone classification on the primary compressive strength of cemented backfill.

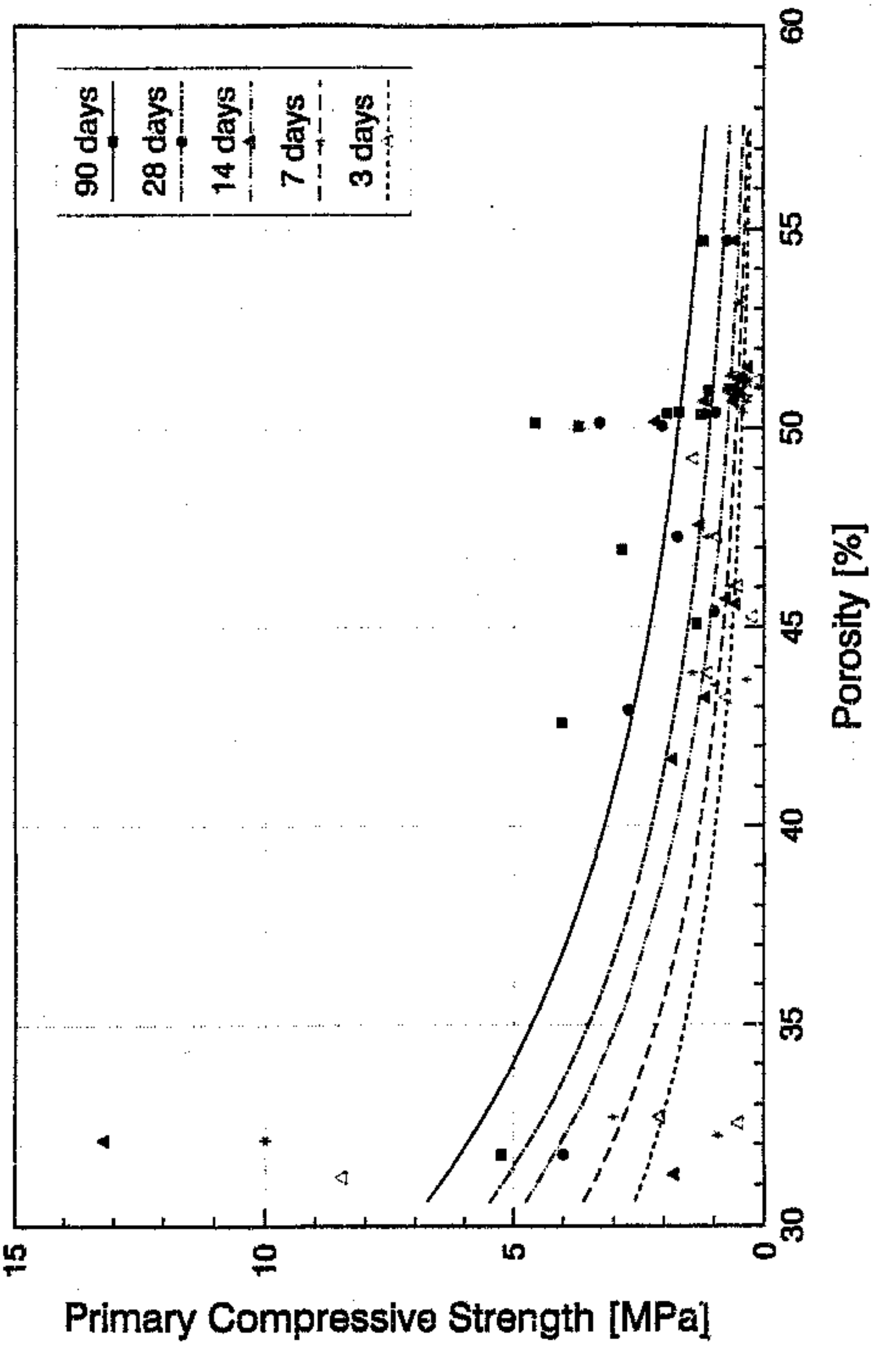


Figure 4.9 Relationship between primary compressive strengths [MPa] of cemented backfills and their porosity [%].

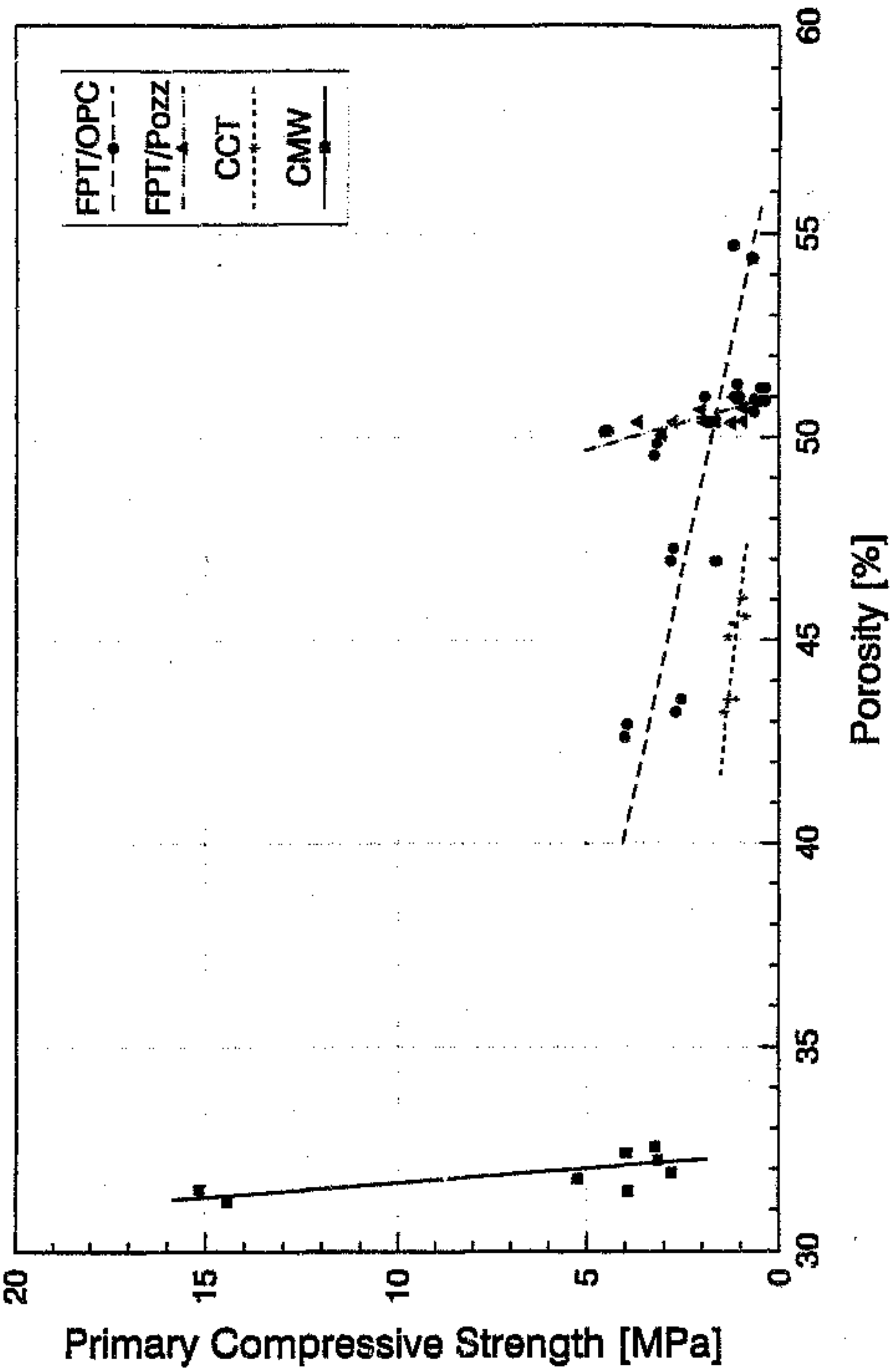


Figure 4.10 Relationship between primary compressive strengths [MPa] of cemented backfills and their porosity [%], differentiated according to aggregate type, full plant tailings (FPT), cyclone classified tailings (CCT) and crushed, milled waste (CMW).

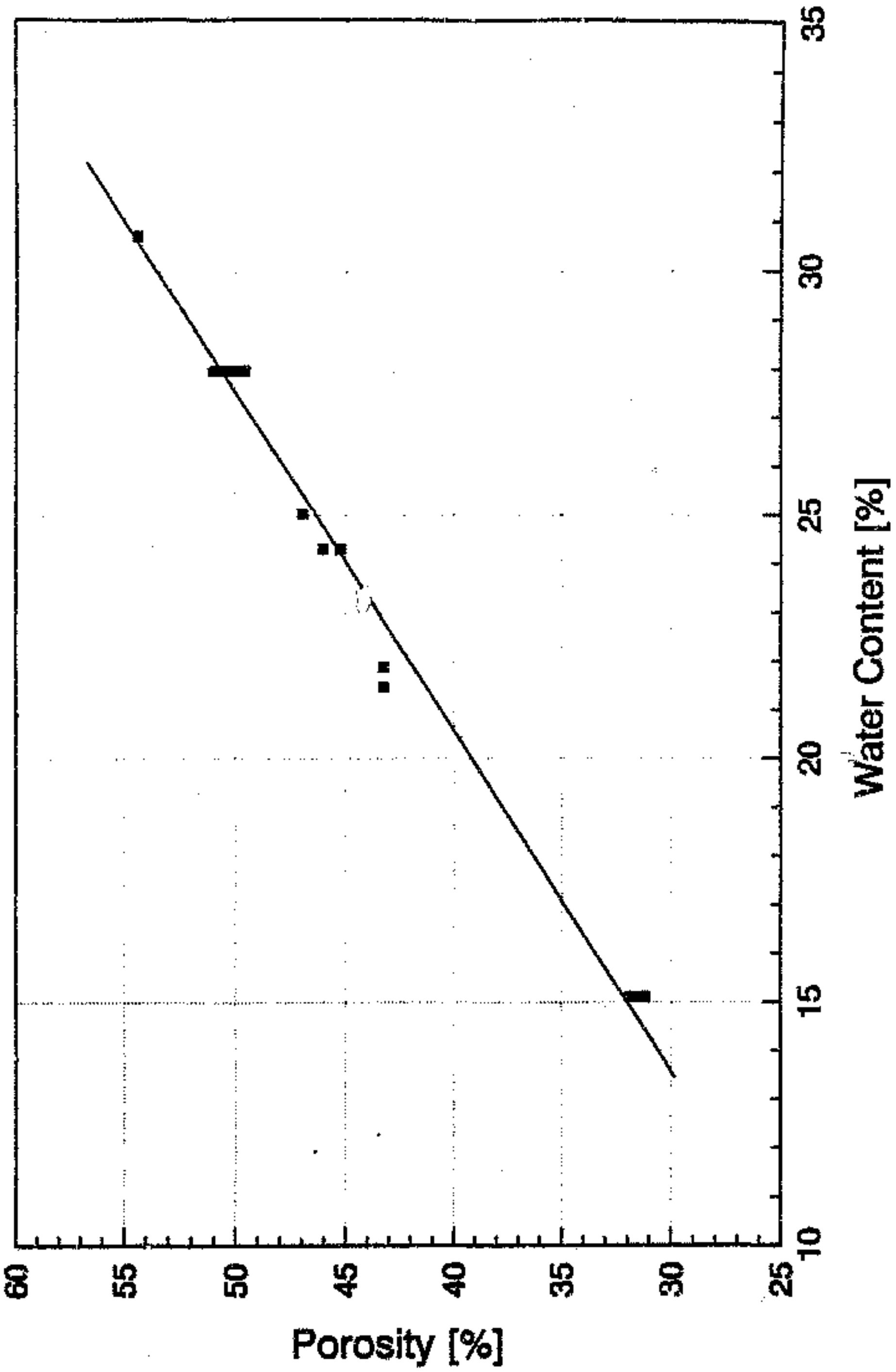


Figure 4.11 Relationship between water content and cemented backfill porosity, 28 days curing, different backfills.

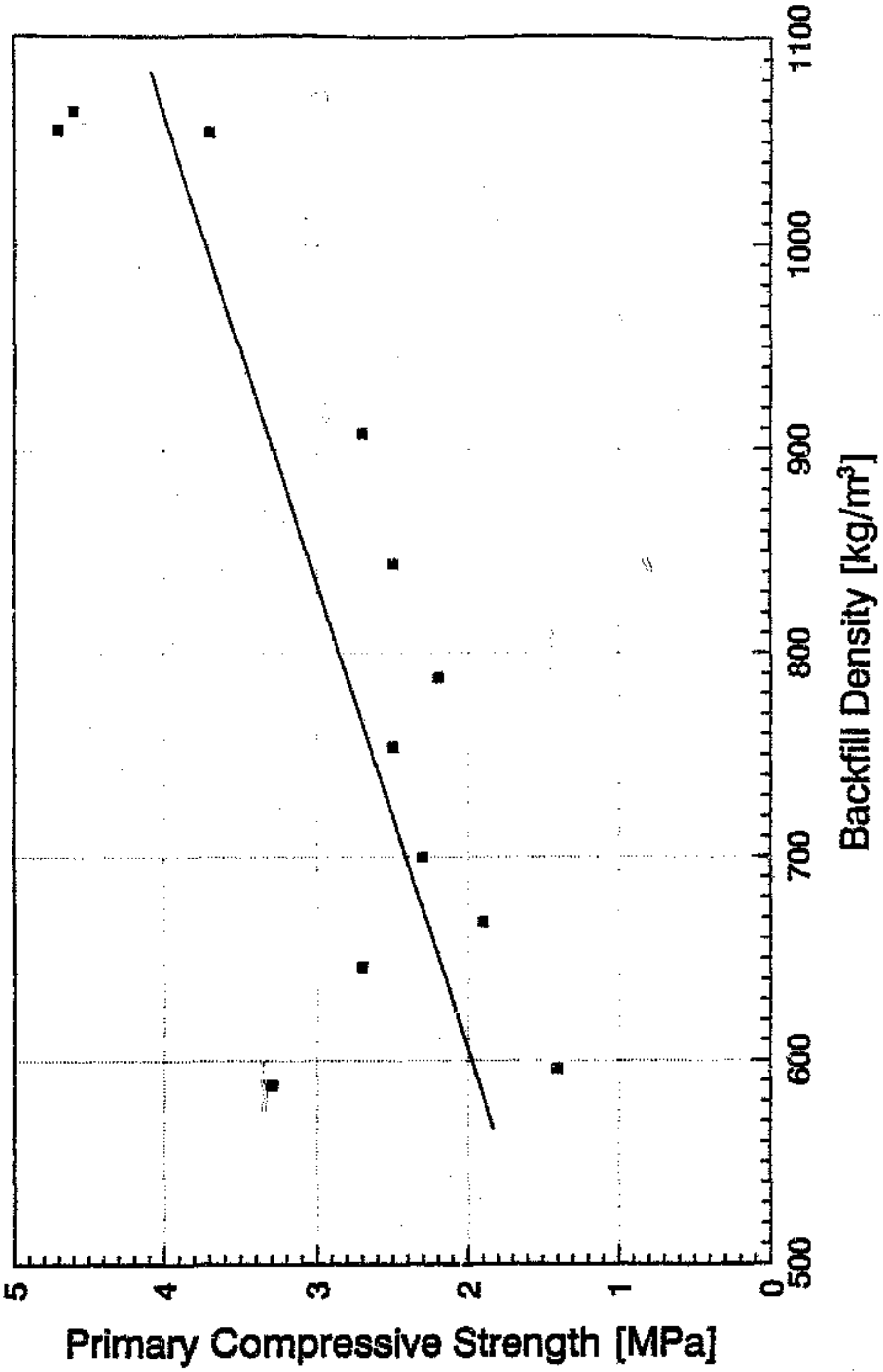


Figure 4.12 Foamed cemented backfill, strength dependence on overall density.

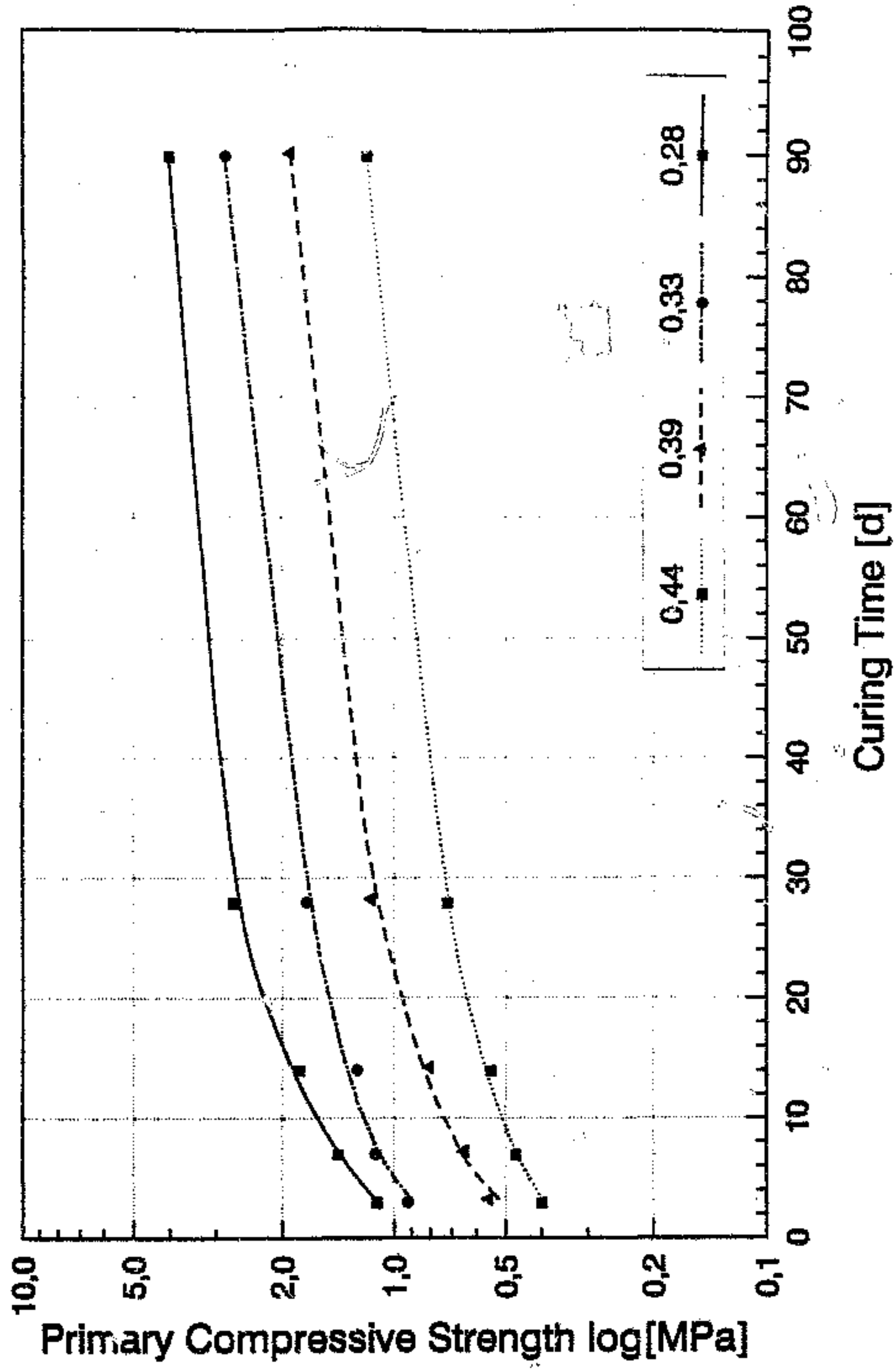


Figure 4.13 Primary compressive strengths of cemented backfills prepared to varying water/solids ratios, FPT, 10 % OPC [log MPa].

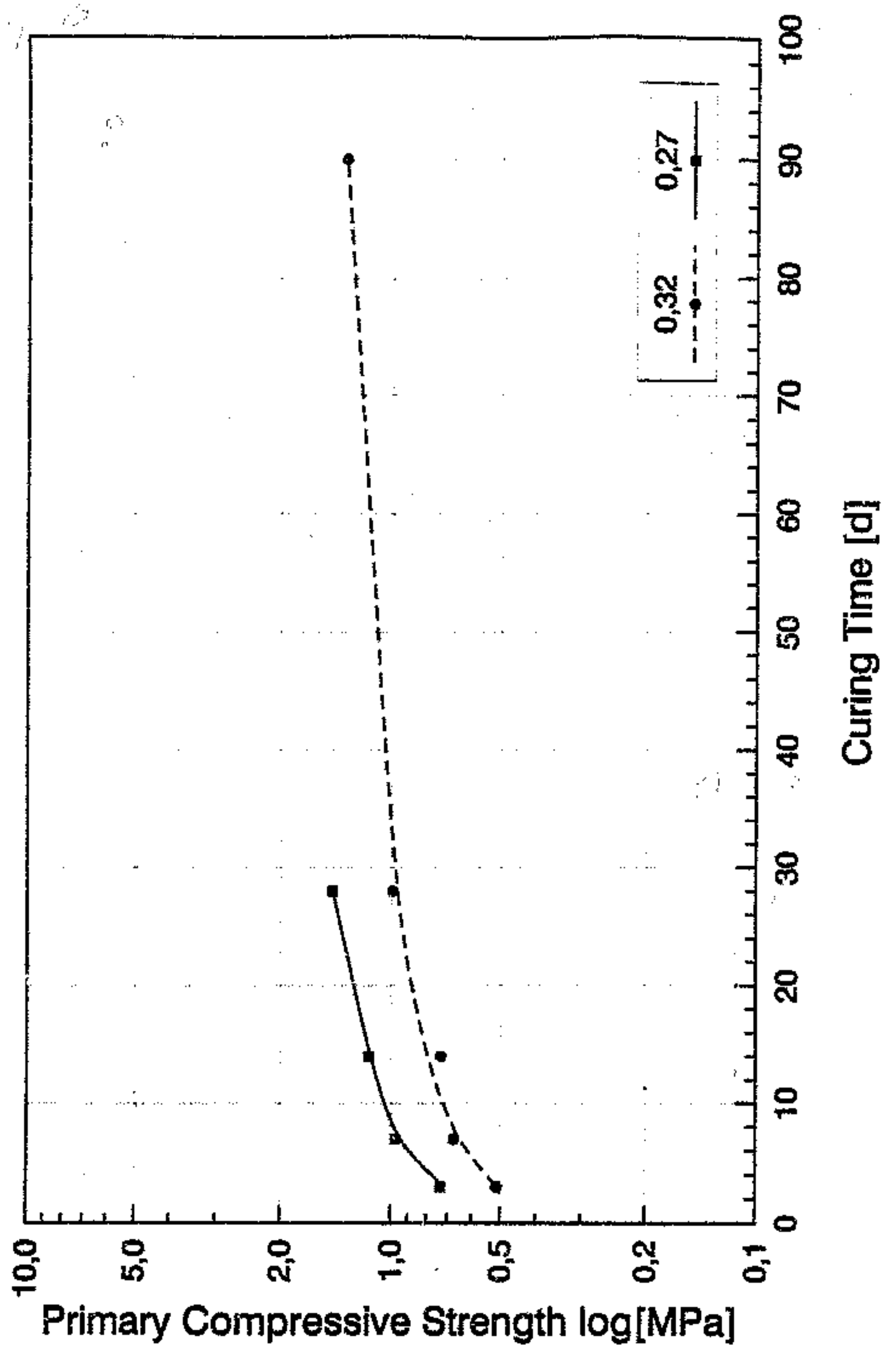


Figure 4.14 Primary compressive strengths of cemented classified tailings backfills prepared to different water/solids ratios, 10 % OPC [log MPa].

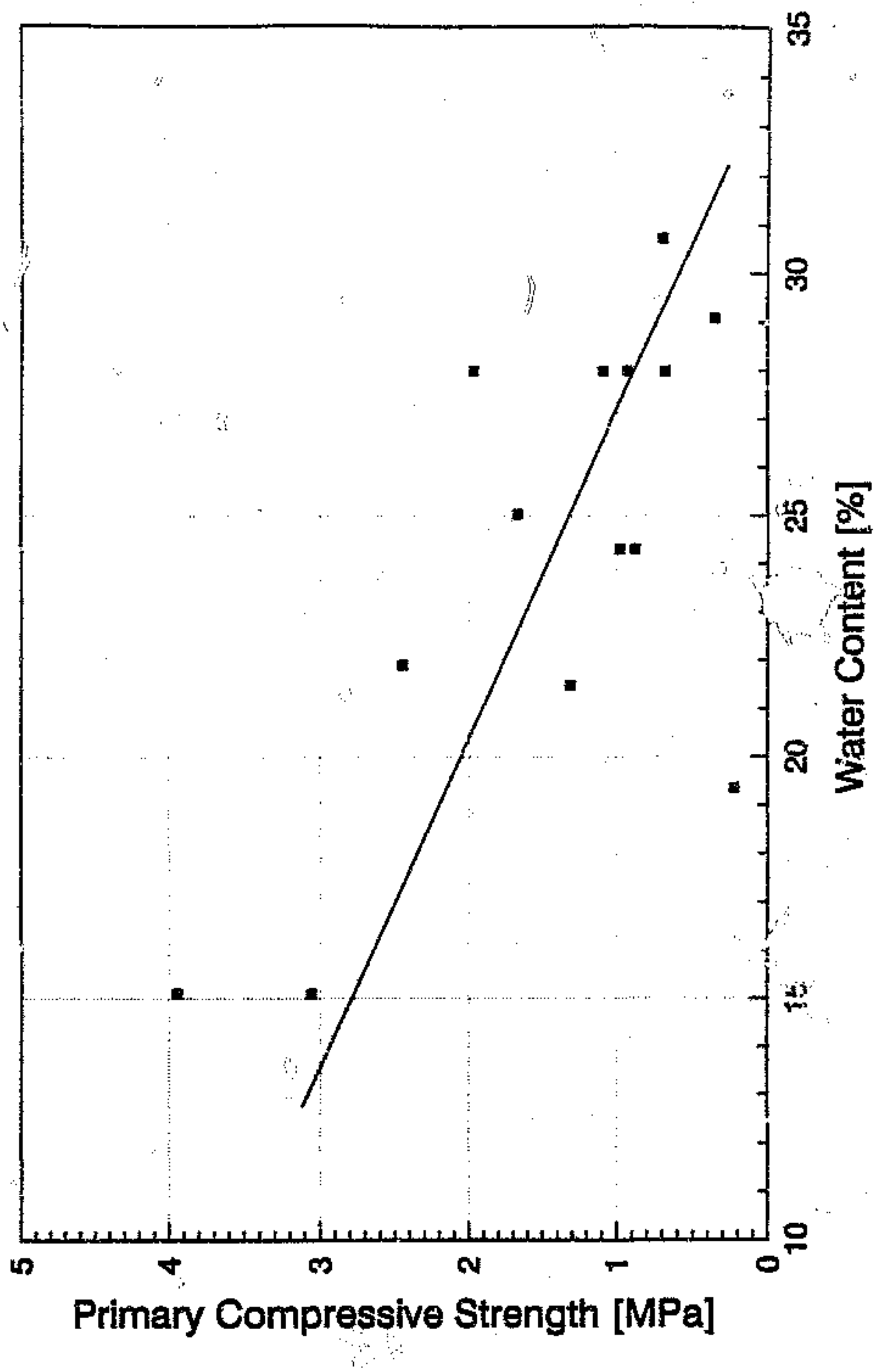


Figure 4.15   Decrease of cemented backfill strength with increasing water content.

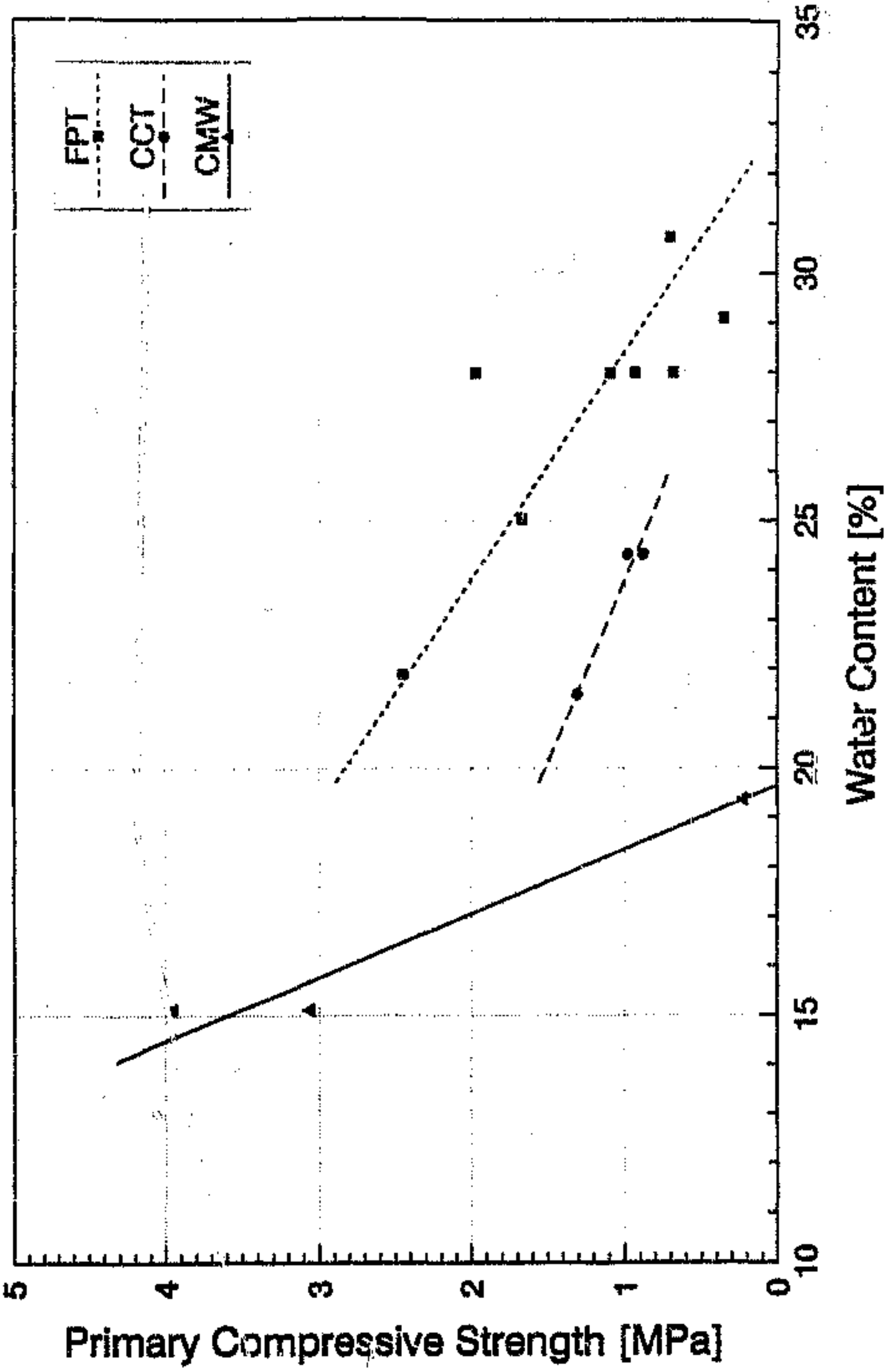


Figure 4.16 Cemented backfill strengths of different backfill types, varying with water content, full plant tailings (FPT), cyclone classified tailings (CCT) and crushed, milled waste (CMW).

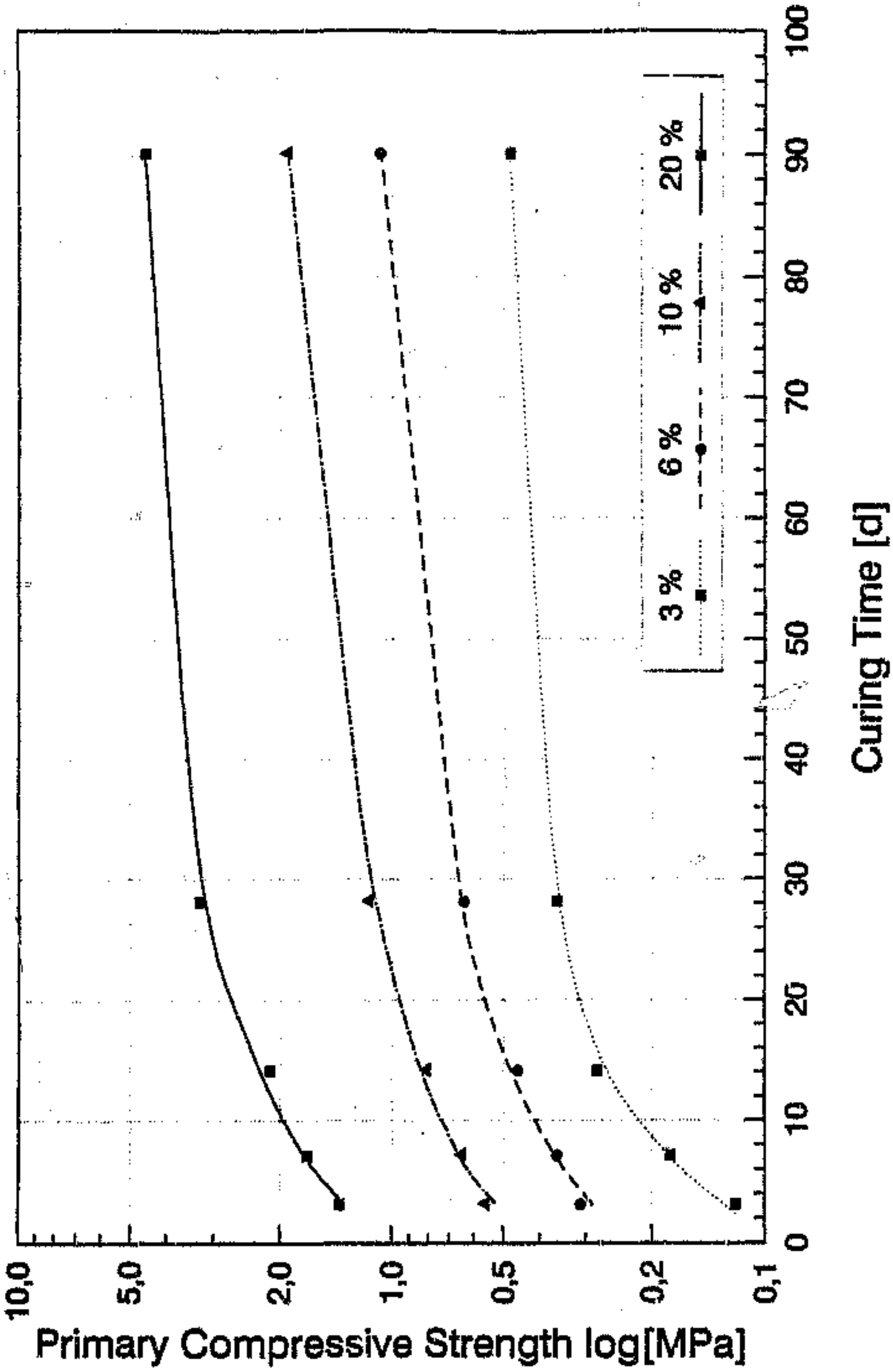


Figure 4.17 Primary compressive strengths of cemented backfills prepared with varying binder amounts, FPT, OPC, W/S 0,39 [log MPa].

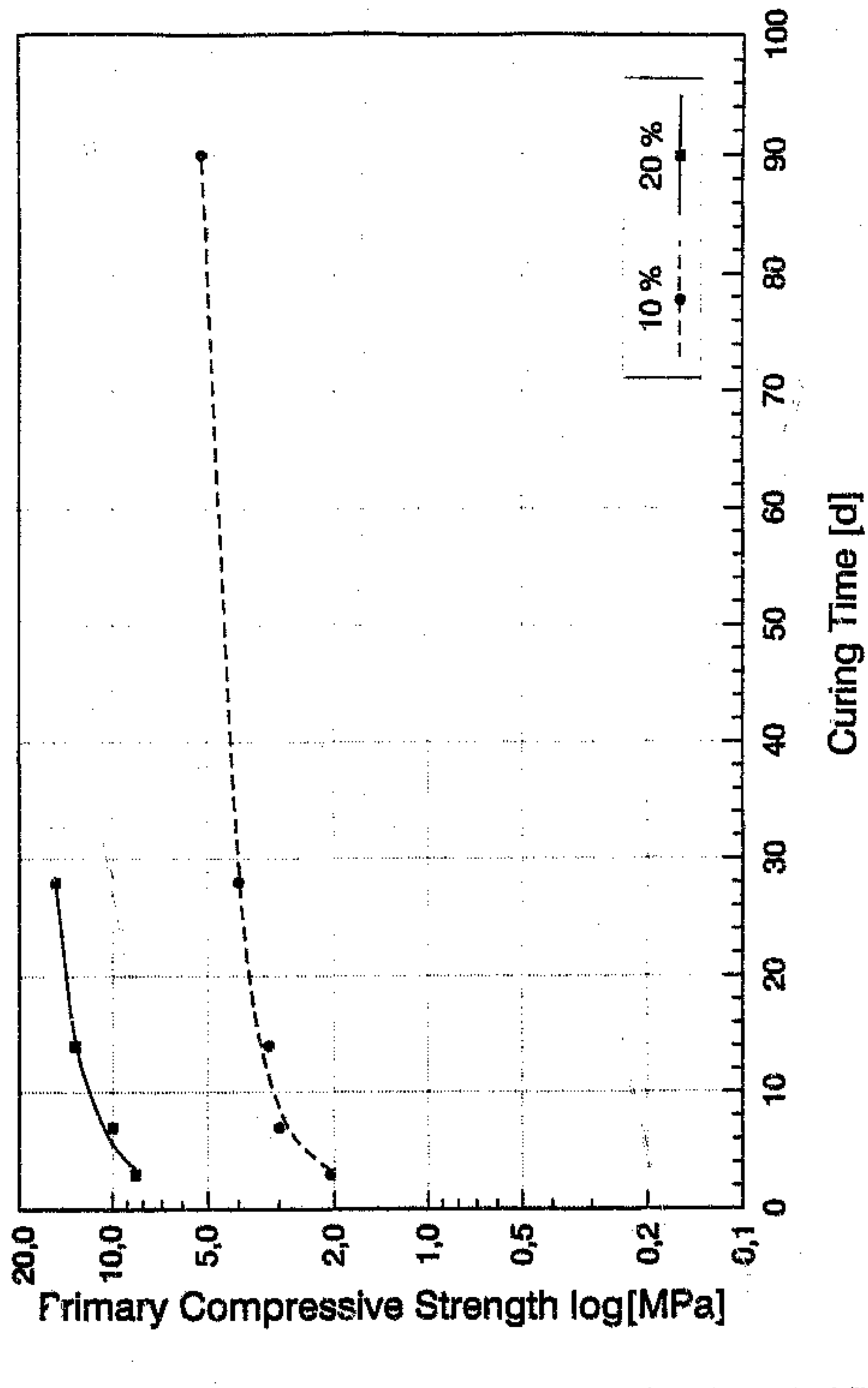


Figure 4.18 Primary compressive strengths of cemented comminuted waste backfills prepared with varying binder contents, CMW, OPC, W/S 0,39 equivalent [log MPa].

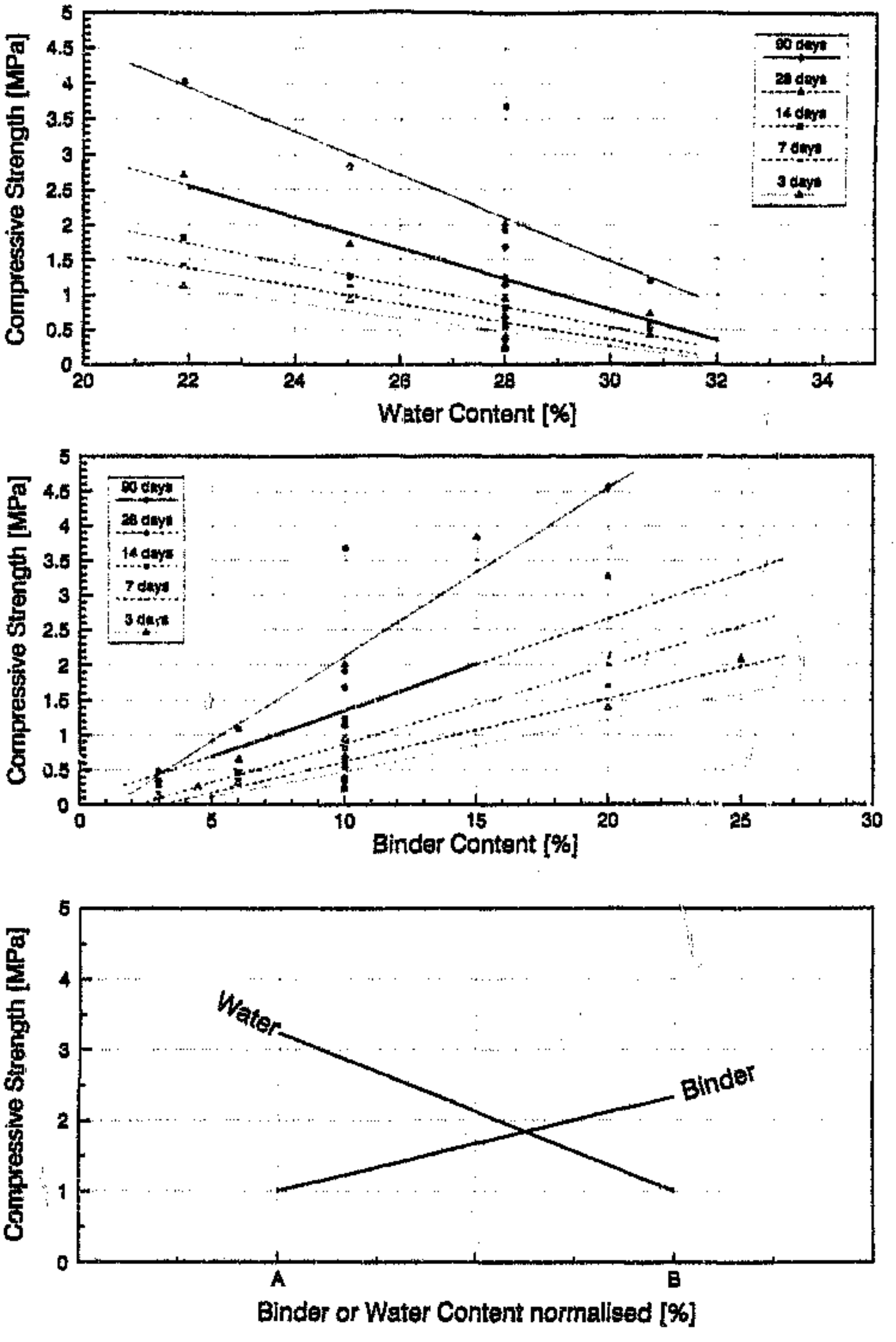


Figure 4.19 Relative influences of water content and cement content on the primary compressive strengths of cemented backfills.

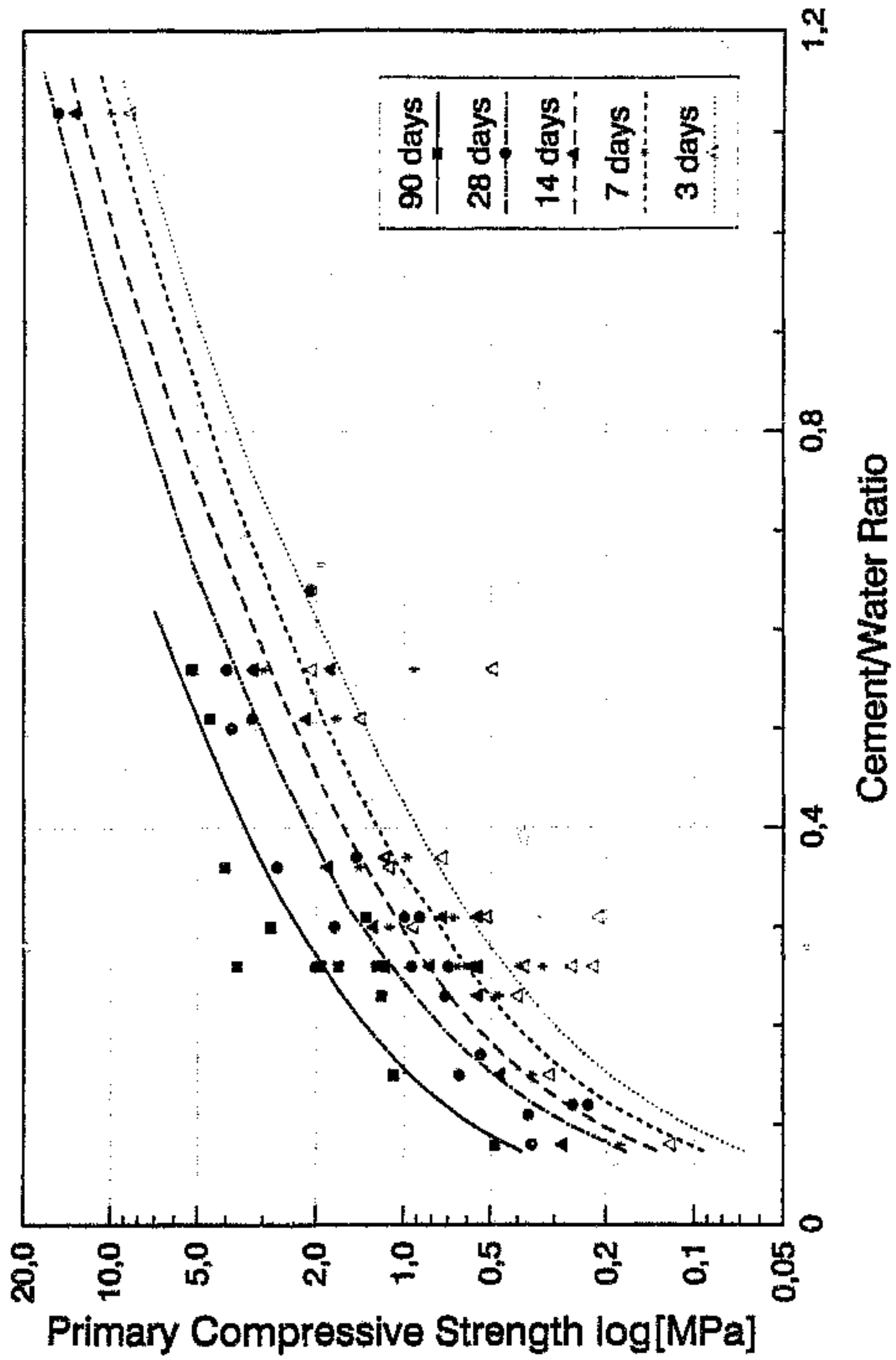


Figure 4.20 Relationship between primary compressive strengths [MPa] of cemented backfills and their cement/water ratio.

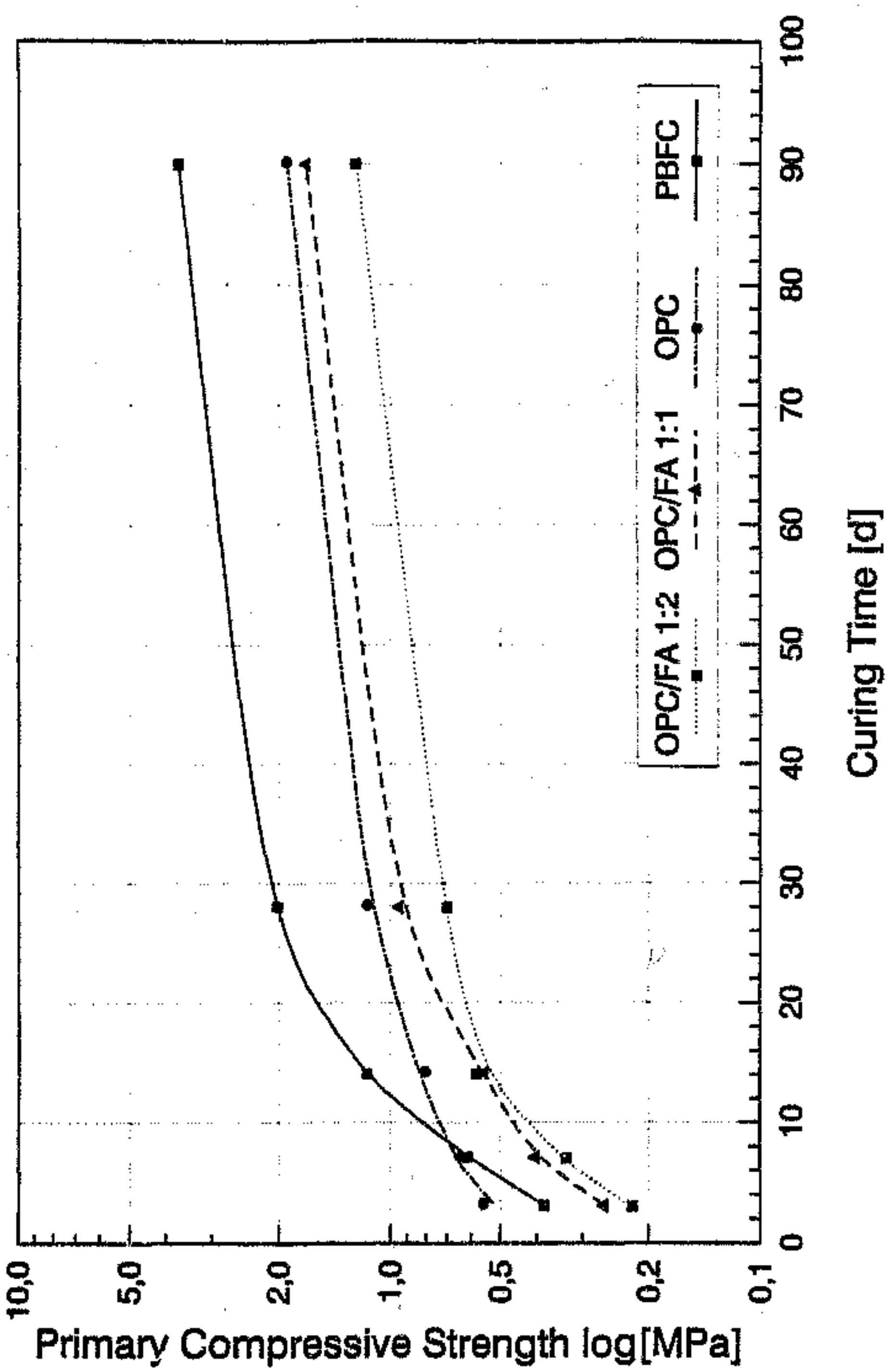


Figure 4.21 Primary compressive strengths of cemented backfills prepared with different binder types, ordinary portland cement (OPC), fly ash (FA) and portland blastfurnace cement (PBFC), FPT, 10 % binder, W/S 0,39 [log MPa].

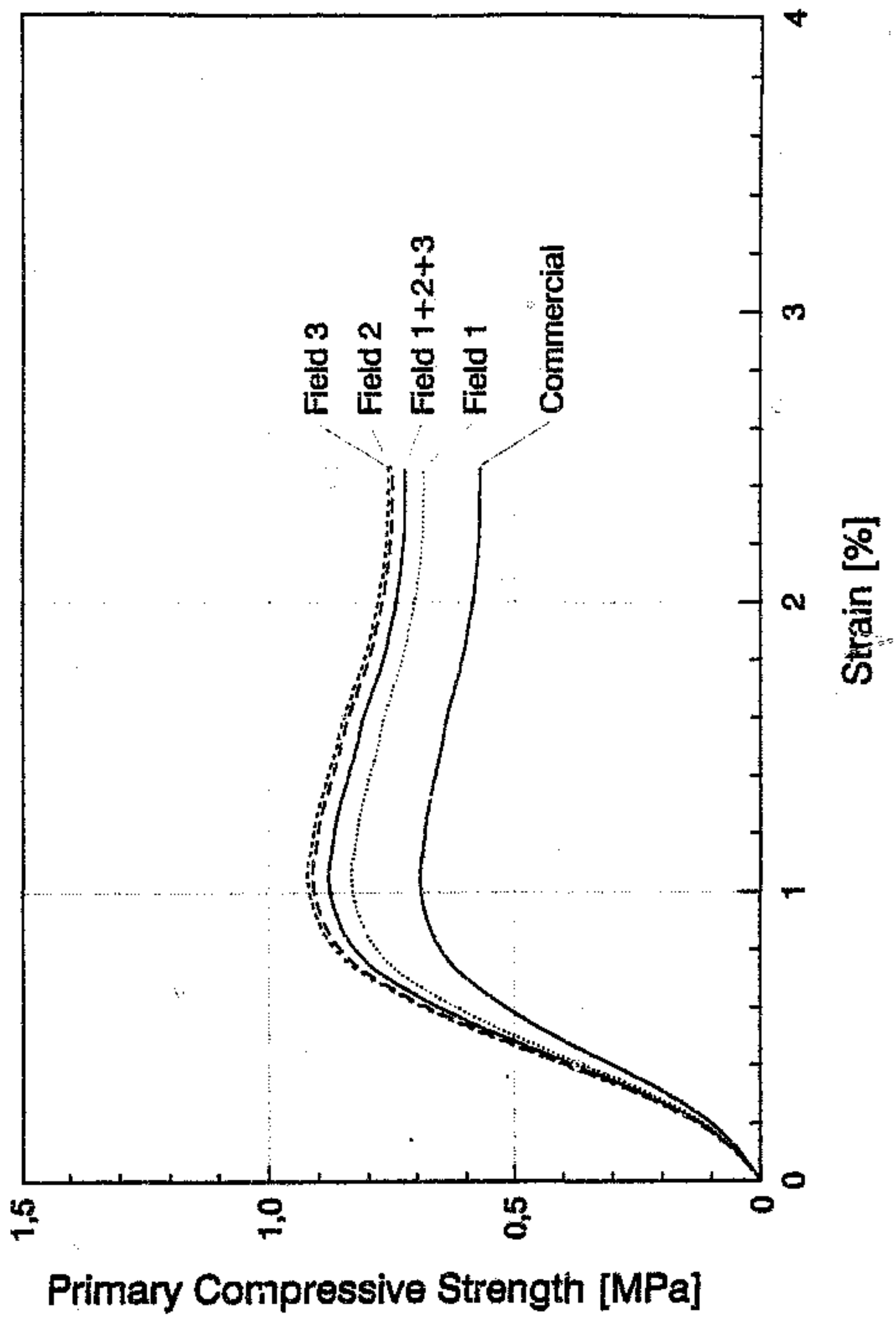


Figure 4.22 Primary compressive strengths of equivalent cemented backfills with different fly ash samples, FPT, 10 % OPC/FA 1:2 binder, W/S 0,39 , 28 d curing [MPa].

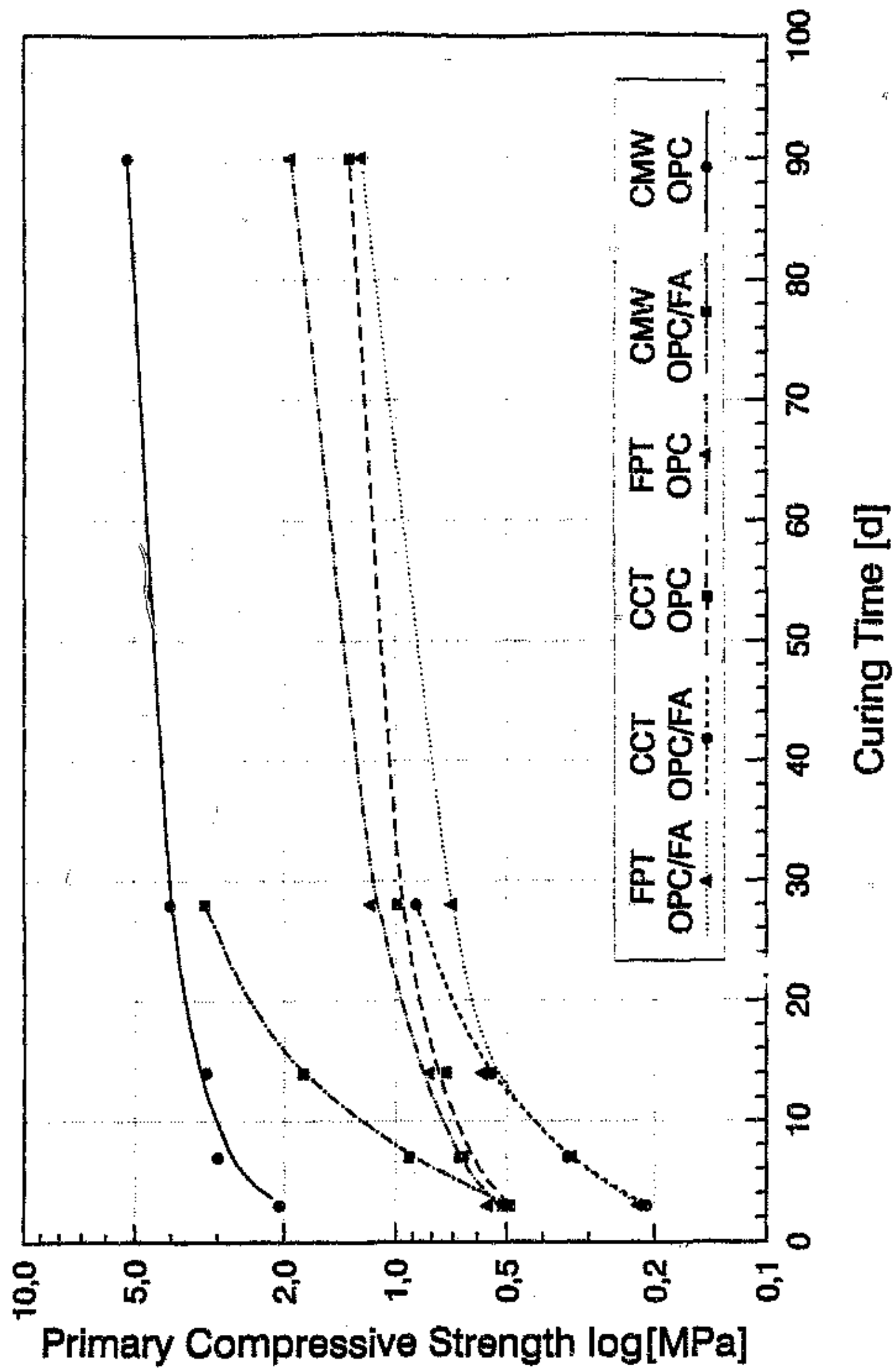


Figure 4.23 Effect of pozzolan addition on the primary compressive strengths of cemented backfills prepared with different aggregate types, 10 % binder, W/S 0,39 equivalent [log MPa].

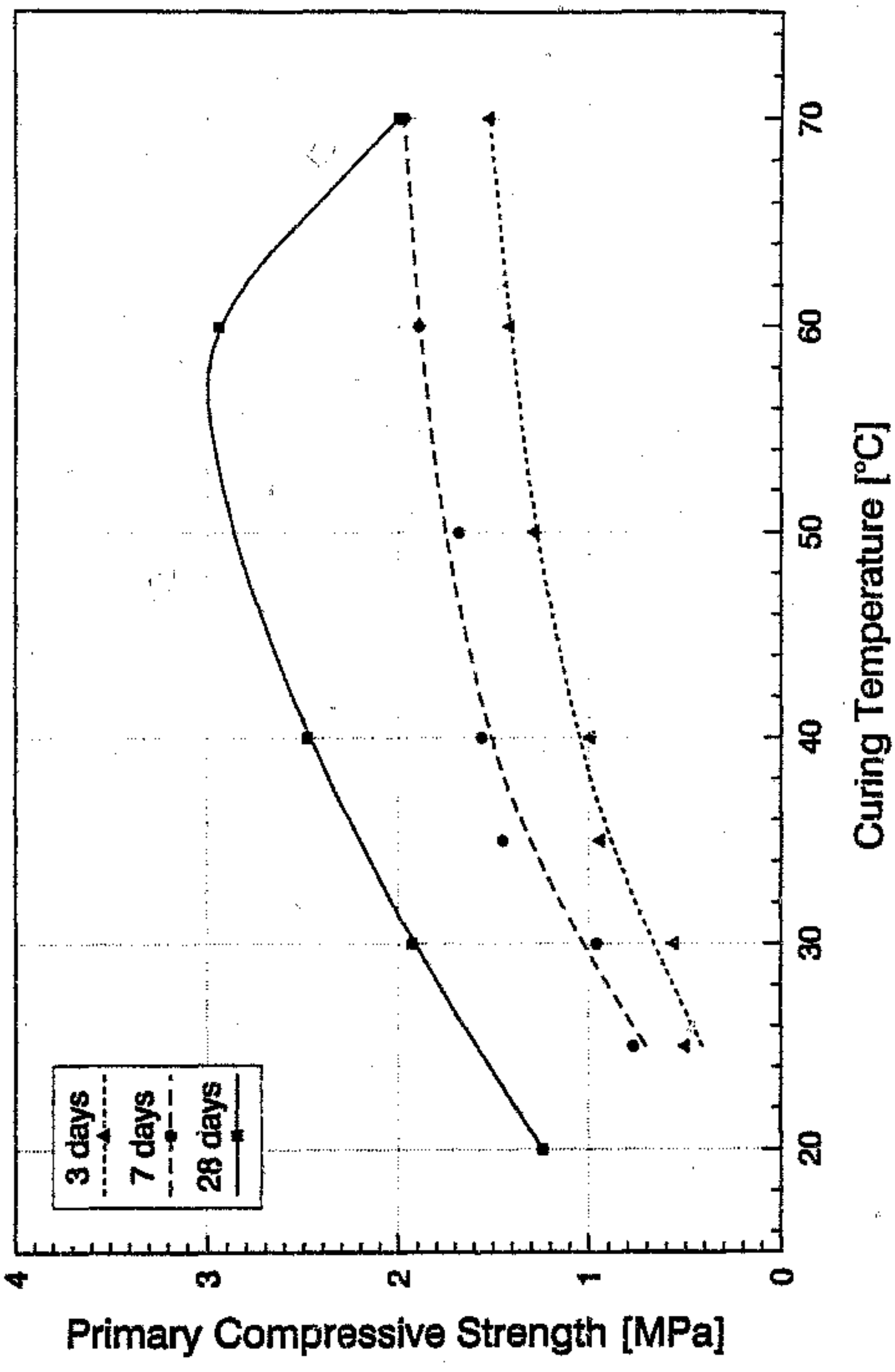


Figure 4.24 Effect of curing temperature on cemented backfill strength.

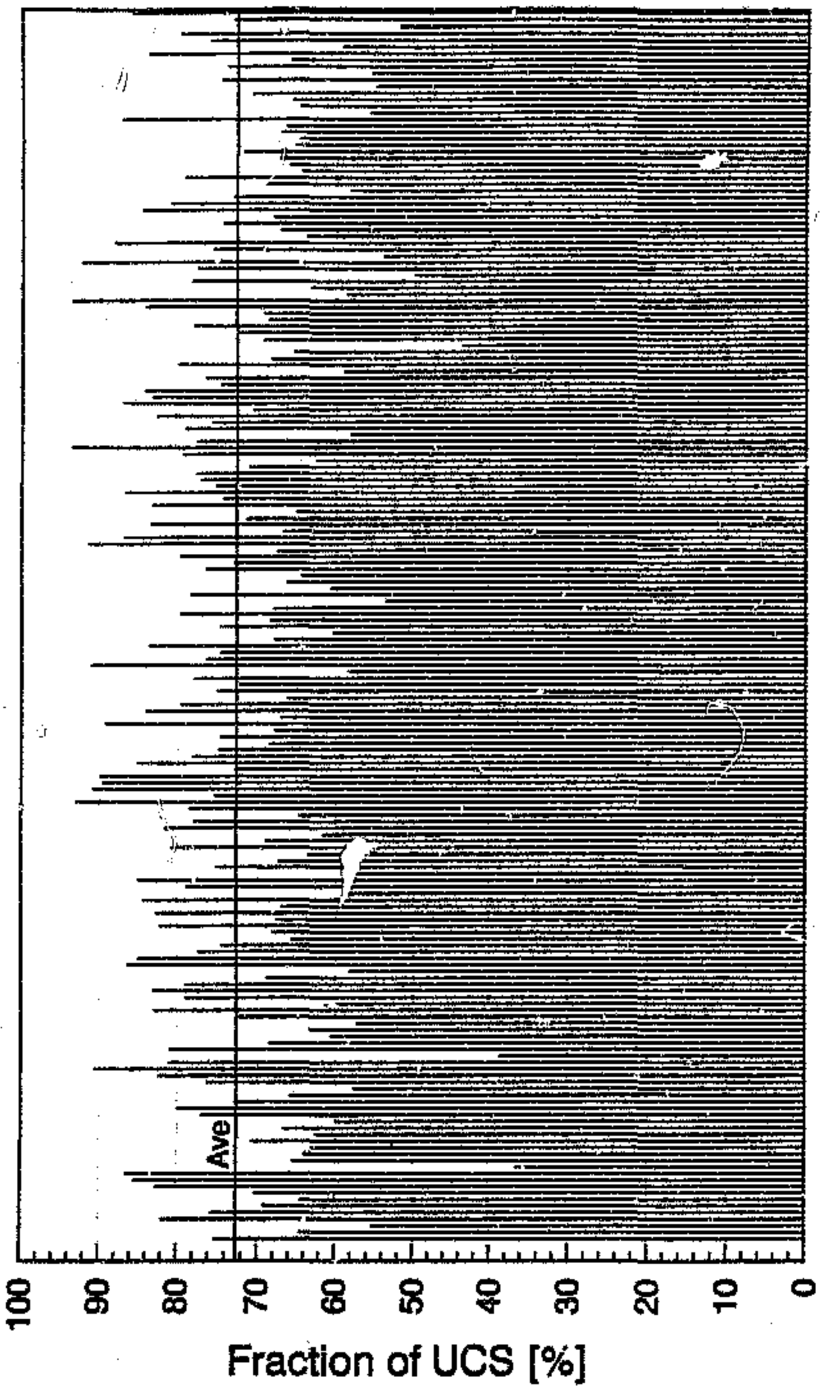


Figure 4.25 Cemented backfill yield strength as a proportion of primary compressive strength [MPa].

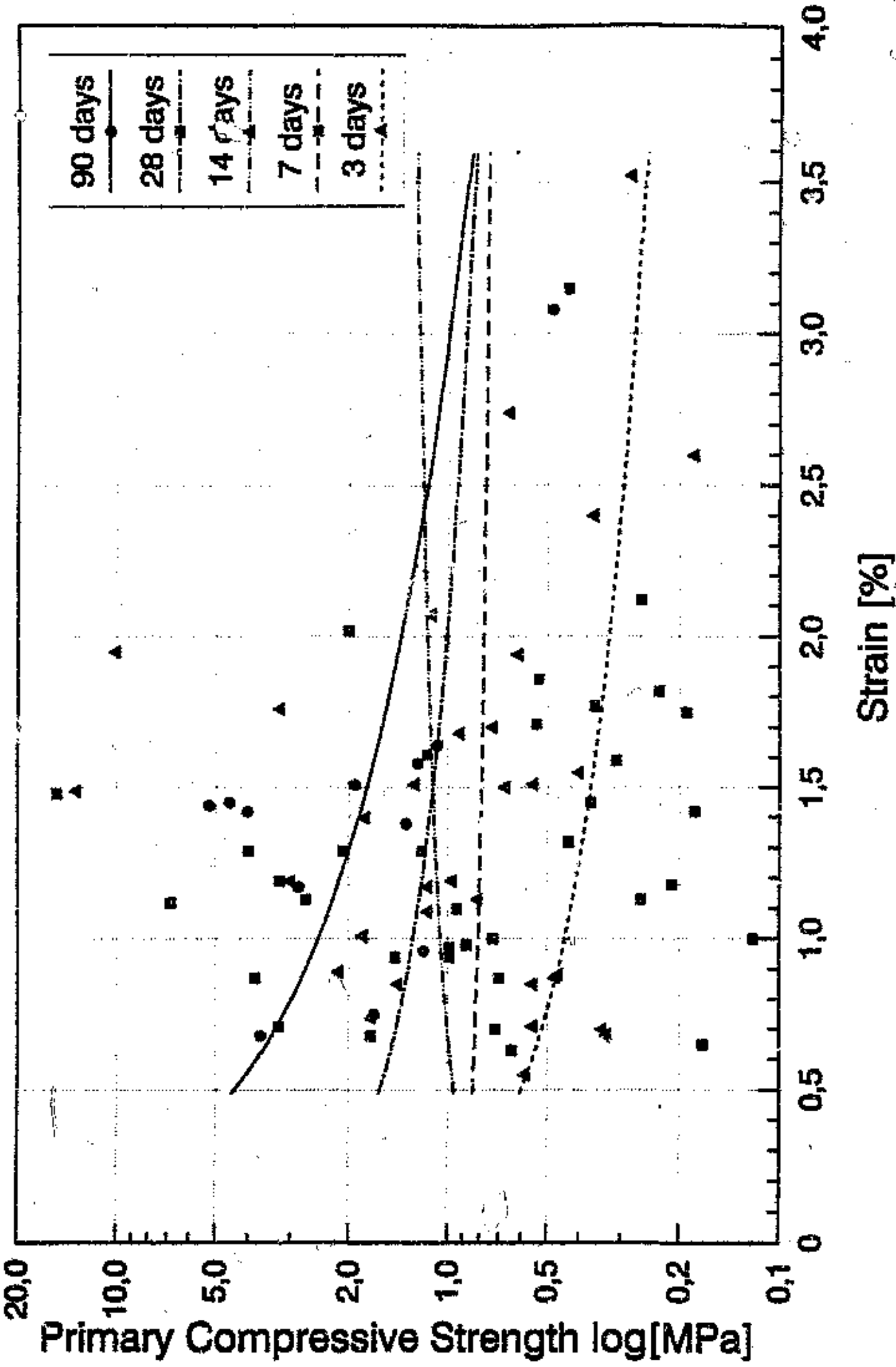


Figure 4.26 Relationship between primary compressive strengths [MPa] of cemented backfills and their deformation [%].

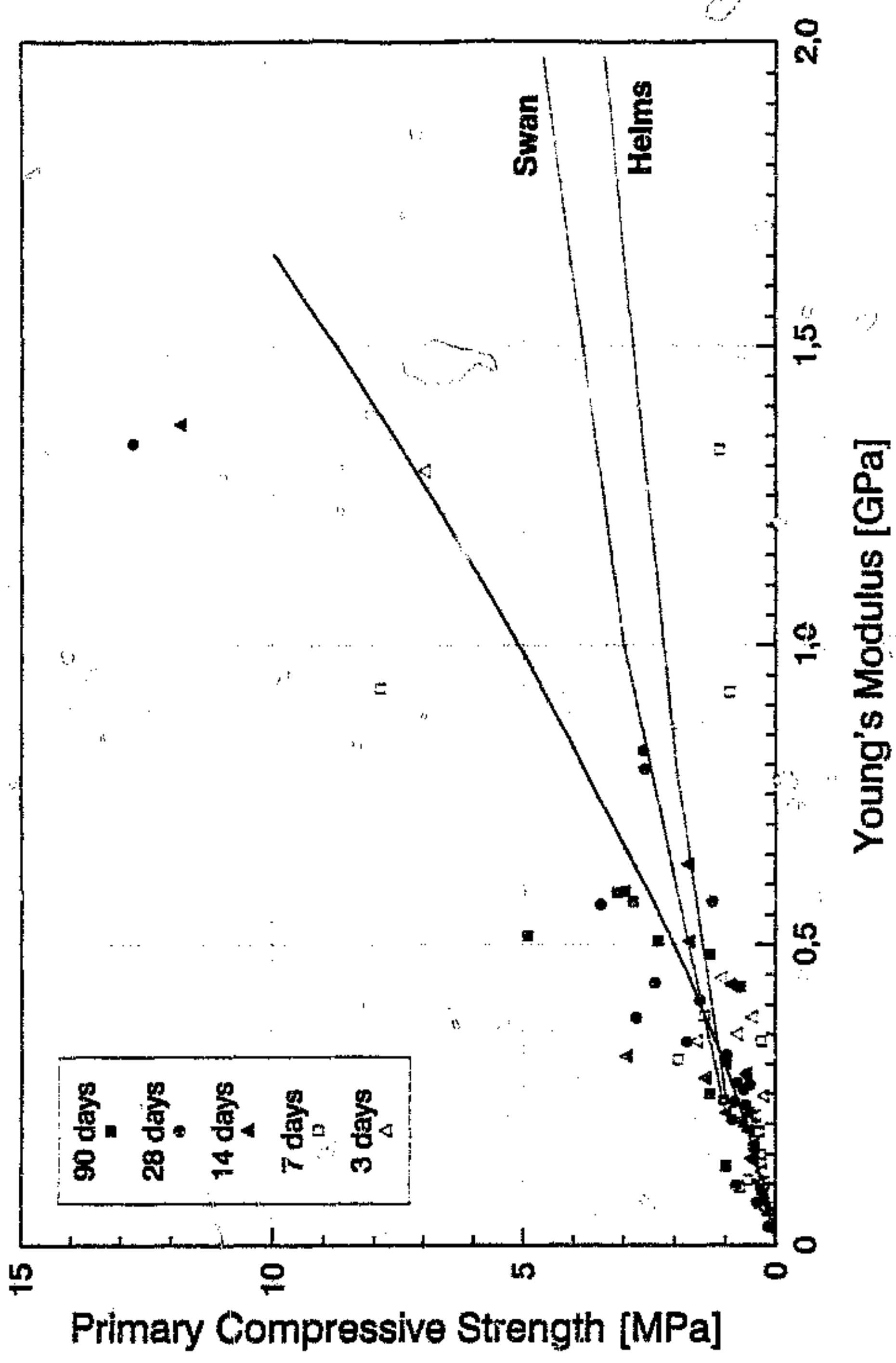


Figure 4.27 Relationship between primary compressive strengths [MPa] of cemented backfills and their elastic modulus [GPa].

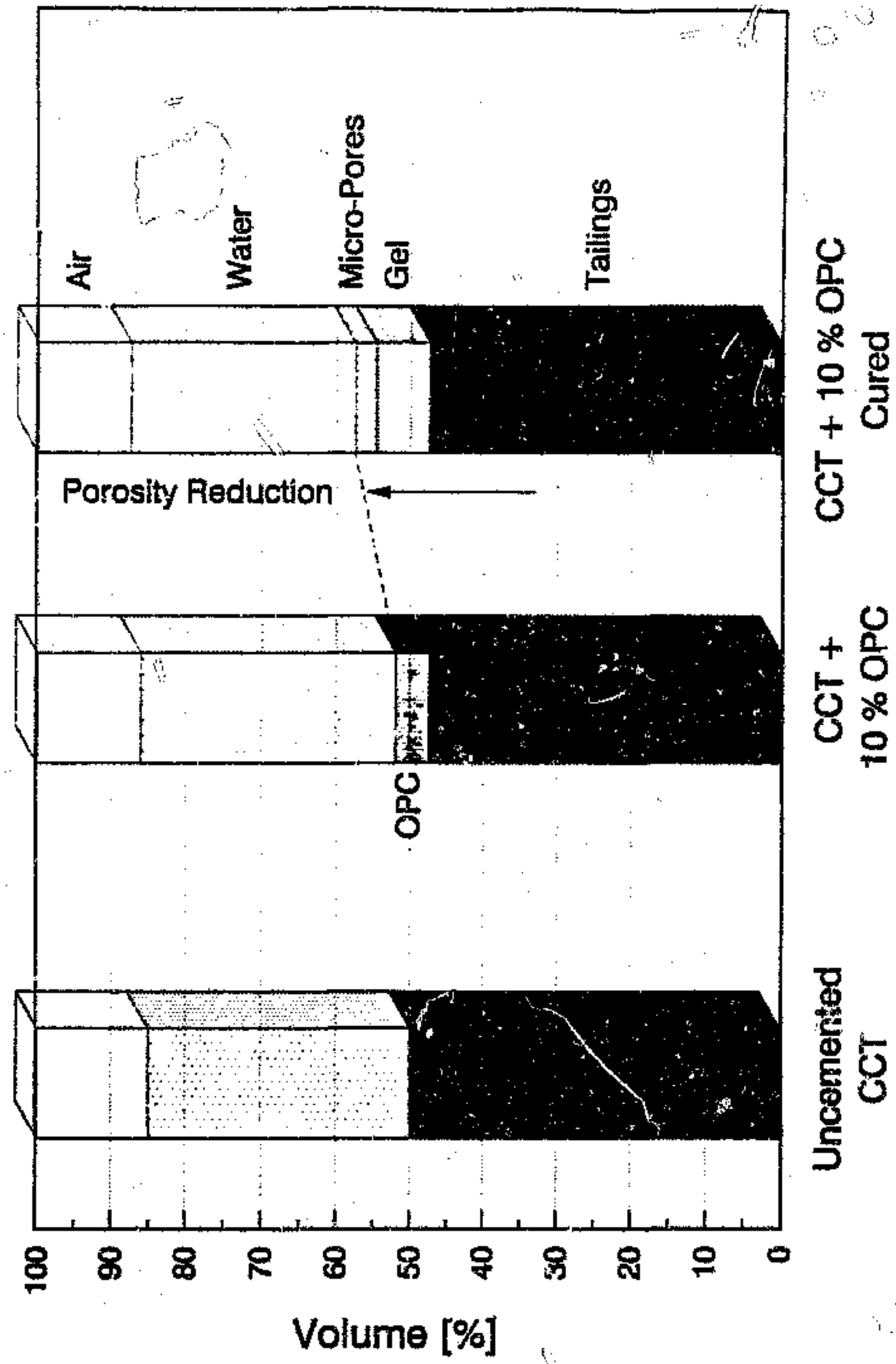


Figure 4.28 Cemented backfill porosity reduction with the hydration of binders.

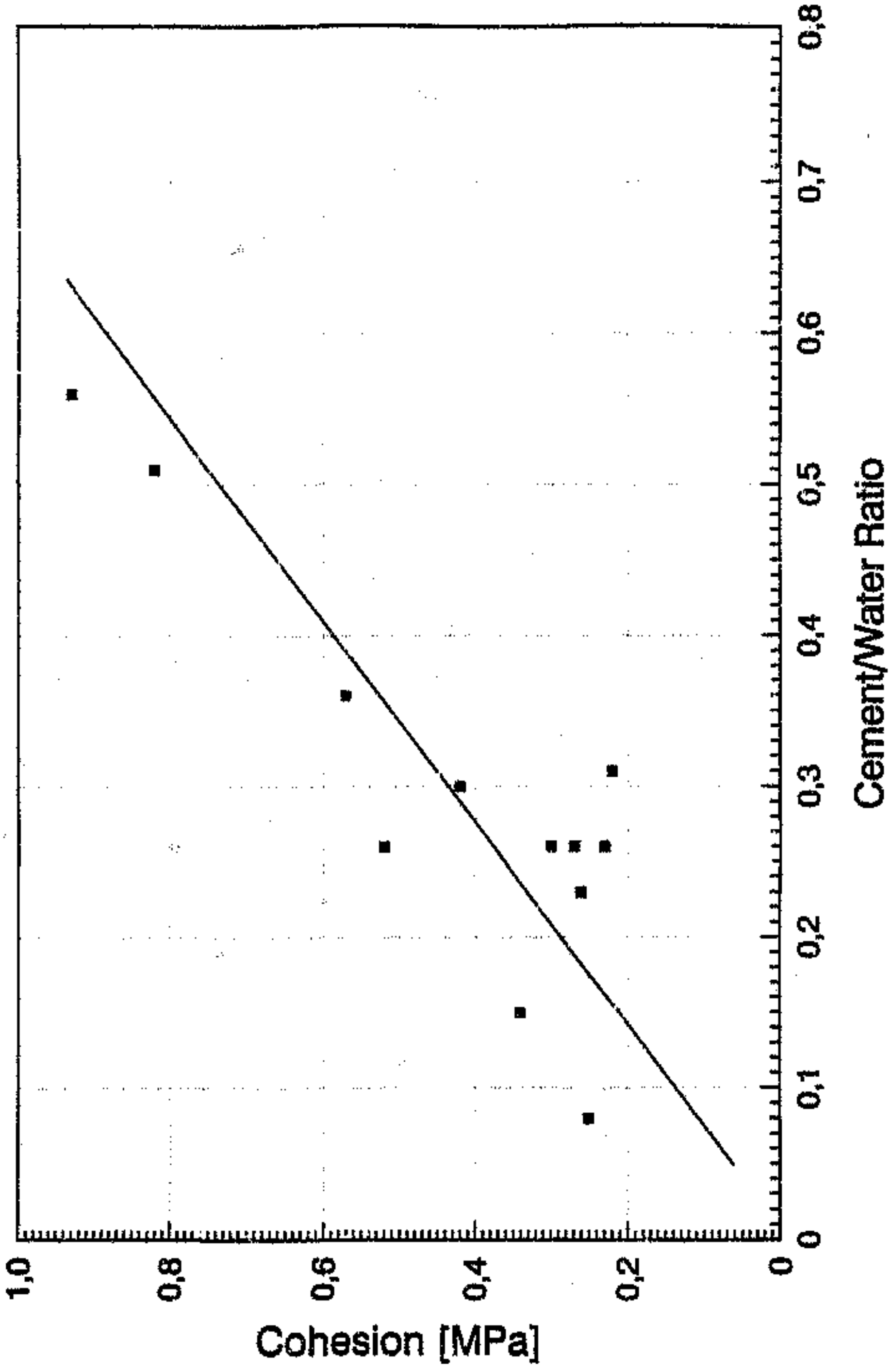


Figure 4.29 Increase of cemented backfill cohesion with increasing cement/water ratio.

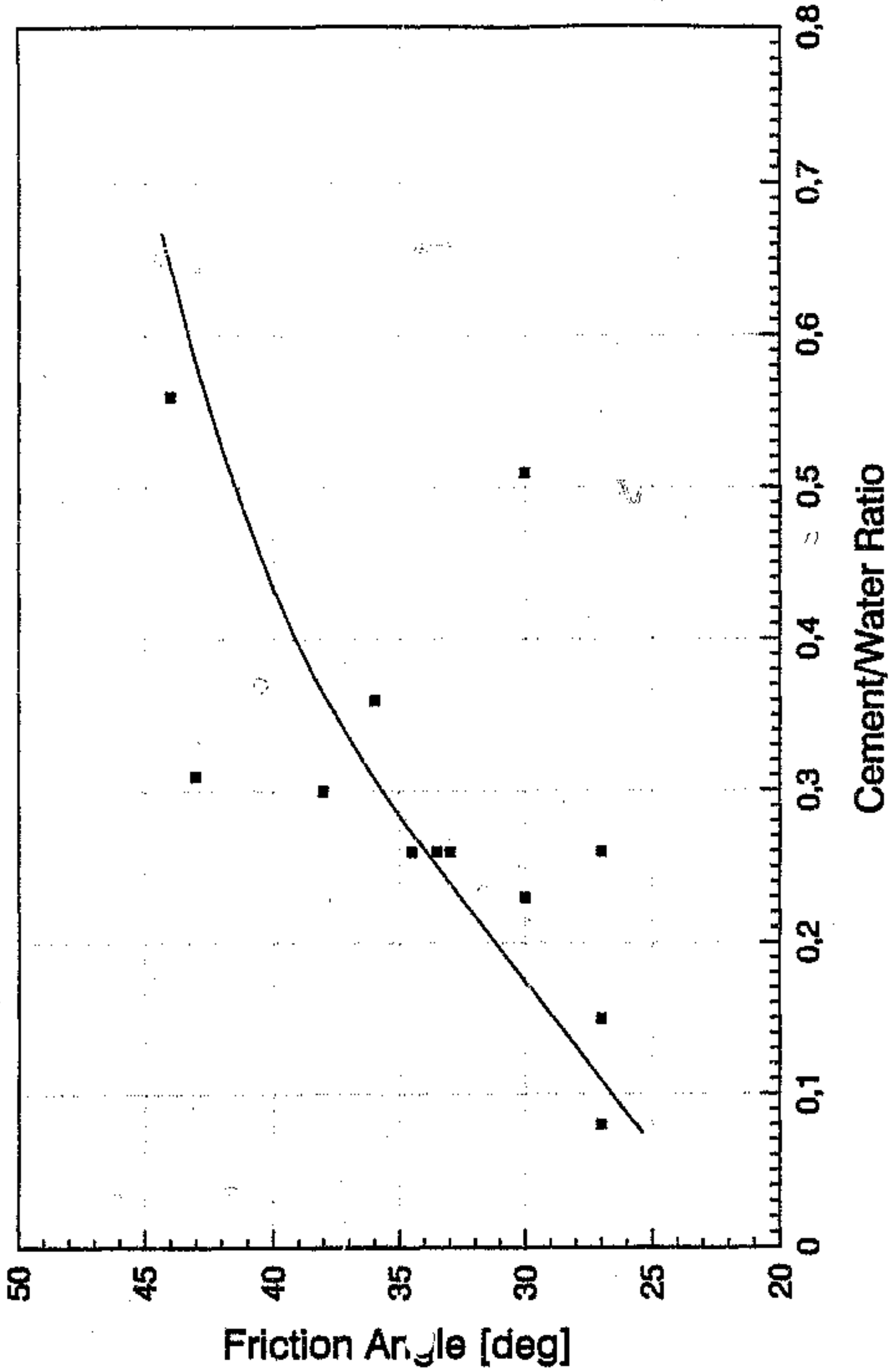
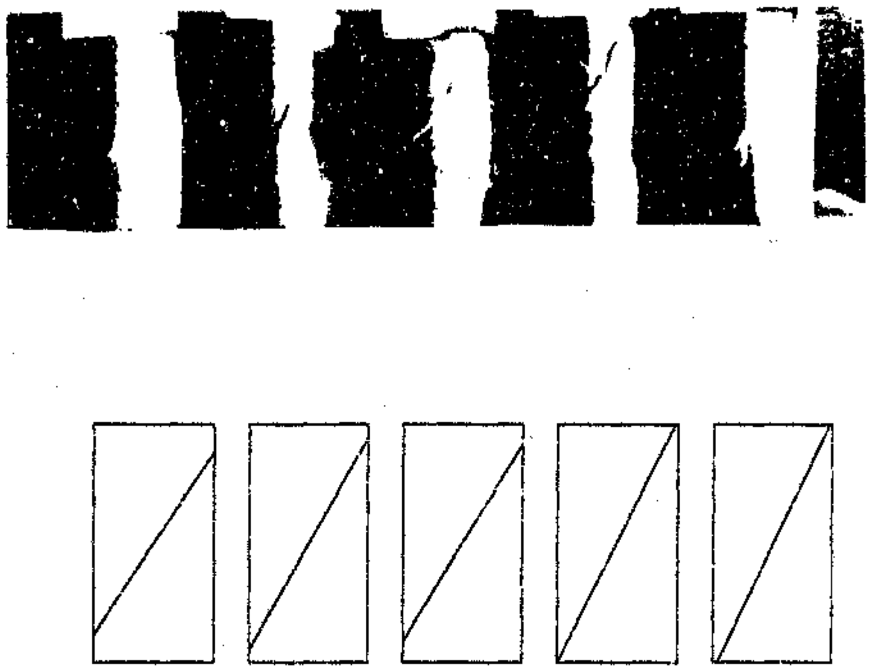


Figure 4.30 Increase of cemented backfill friction angle with increasing cement/water ratio.



Shear Angle	57,3	61,0	59,3	63,5	65,5
Friction Angle	24,6	32,0	28,6	37,0	41,0

Figure 4.31 Increase of shear angle and friction angle of cemented backfill with curing time.

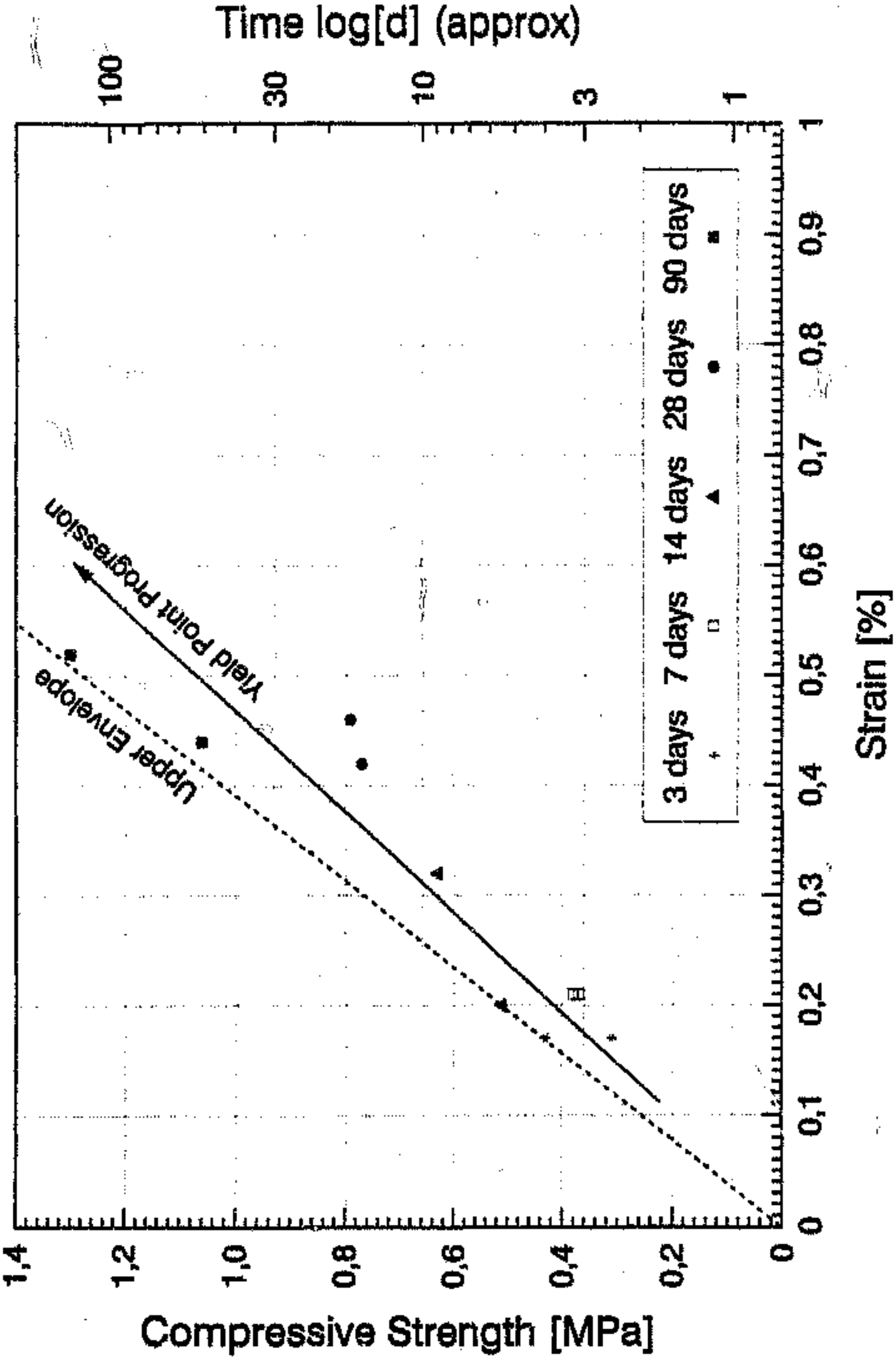


Figure 4.32 Safe loading of cemented backfill during curing.

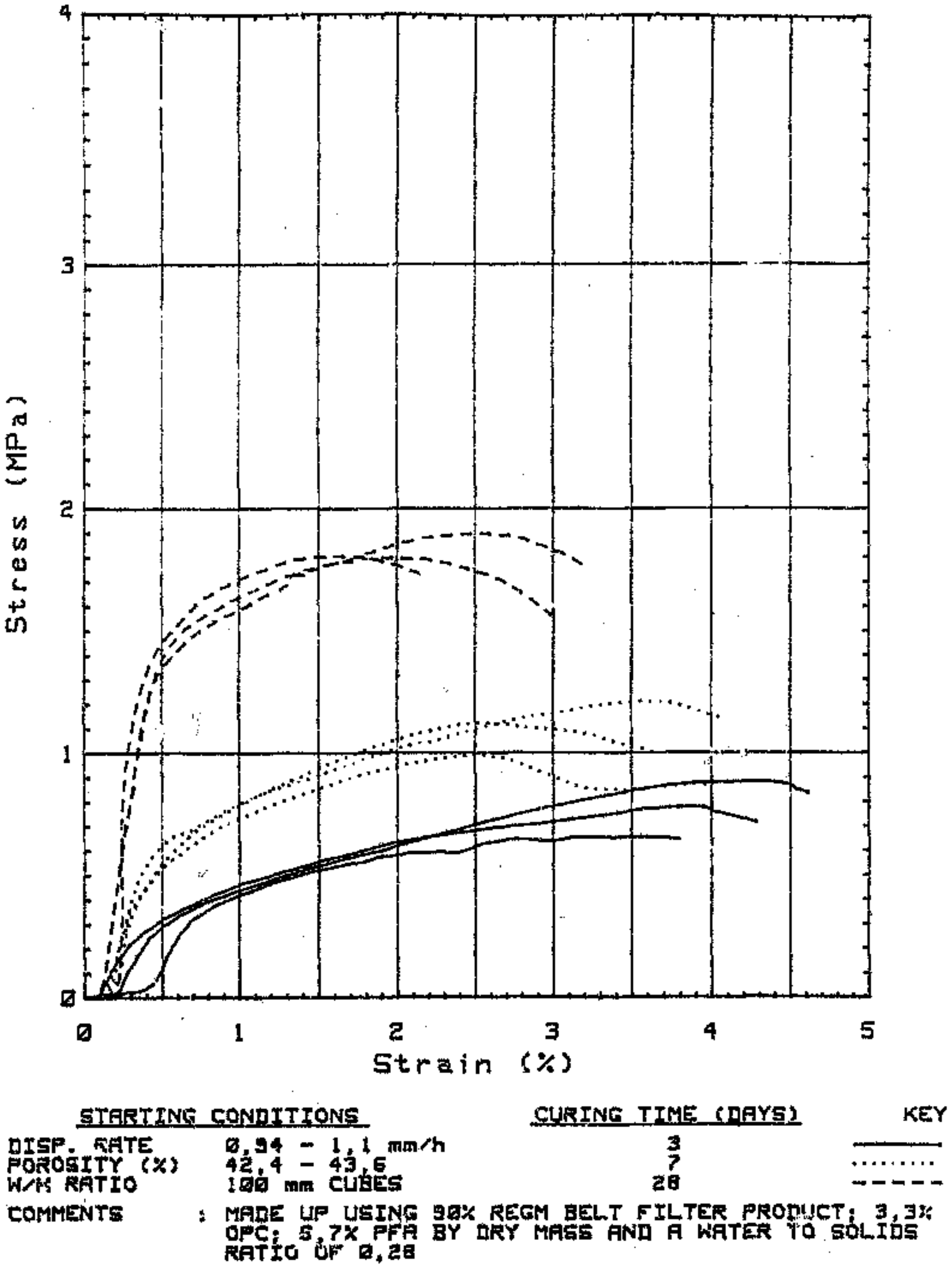


Figure 4.33 (a) Effect of strain rate on the primary compressive strengths of cemented backfills, 100 mm cubes, FPT, 10 % OPC/FA 1:2 binder, W/S 0,28, 1 % strain/h (after Bradley (1986)) [MPa].

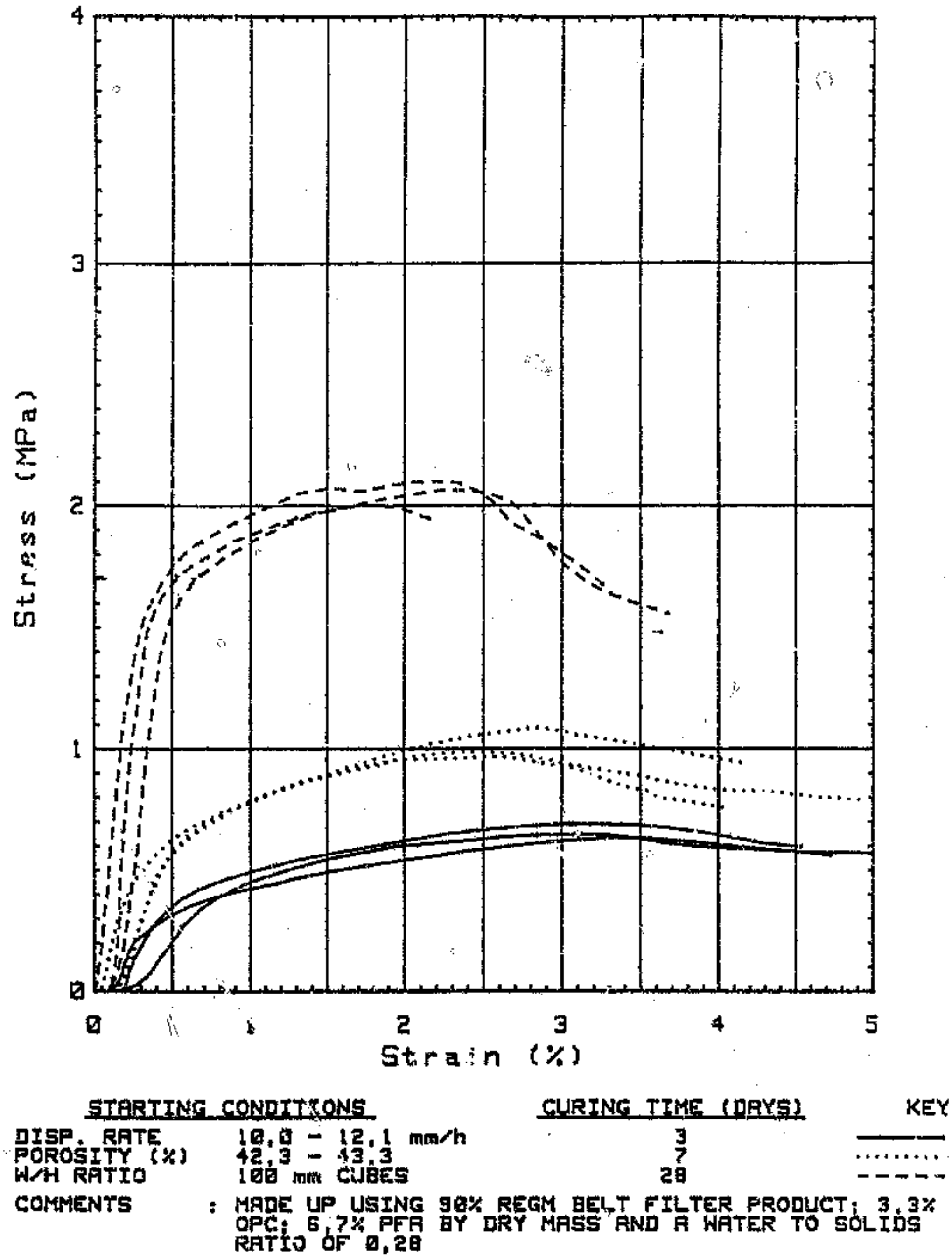


Figure 4.33 (b) Effect of strain rate on the primary compressive strengths of cemented backfills, 100 mm cubes, FPT, 10 % OPC/FA 1:2 binder, W/S 0,28, 10 % strain/h (after Bradley (1986)) [MPa].

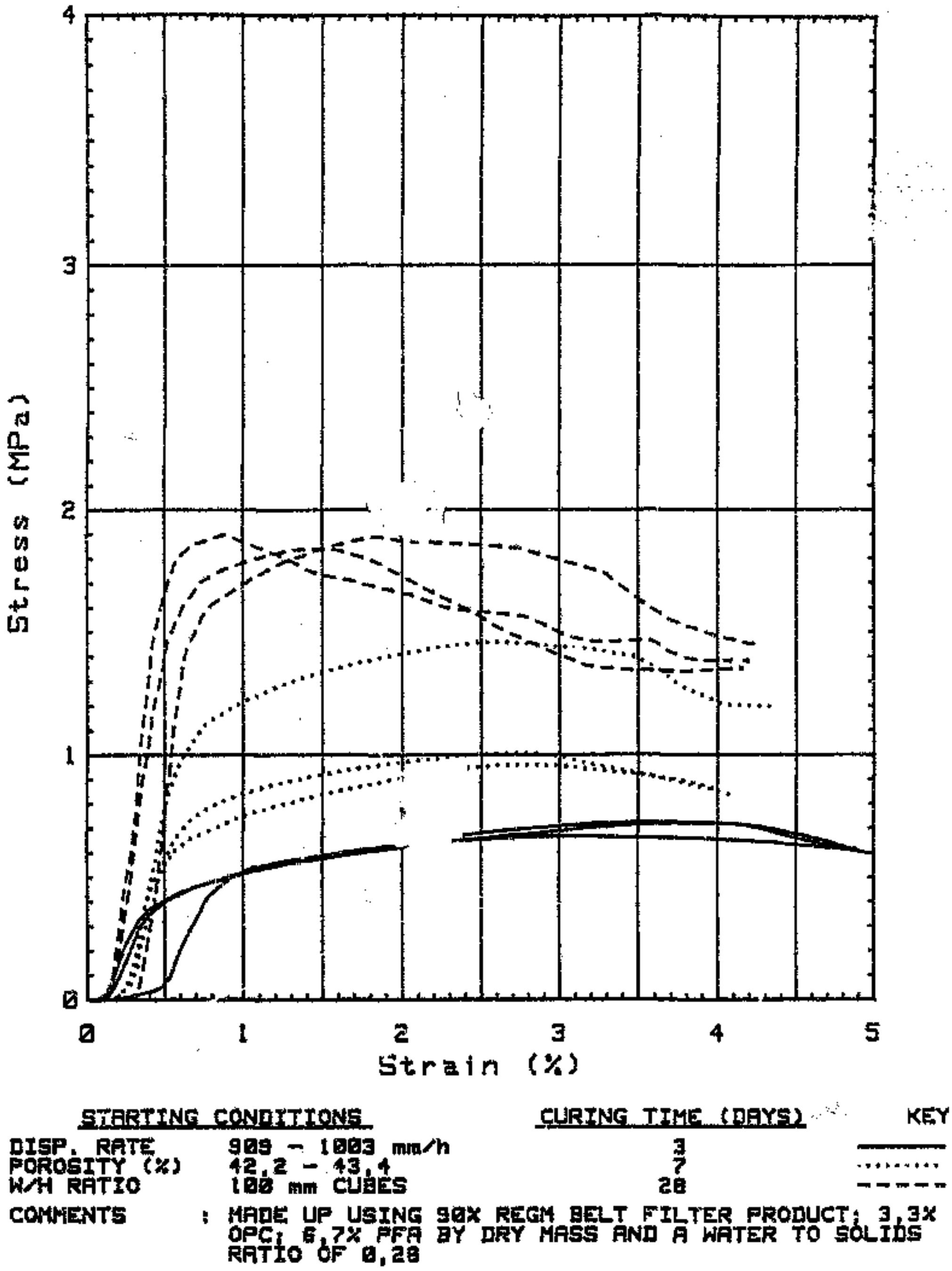
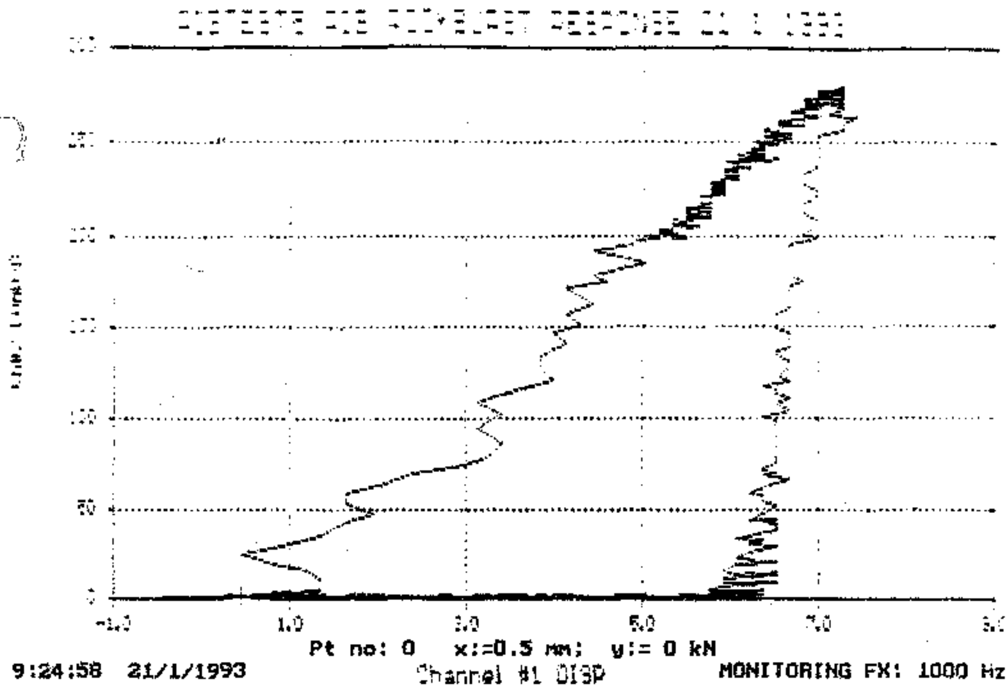


Figure 4.33 (c) Effect of strain rate on the primary compressive strengths of cemented backfills, 100 mm cubes. FPT, 10 % OPC/FA 1:2 binder, W/S 0,28, 1000 % strain/h (after Bradley (1986)) [MPa].



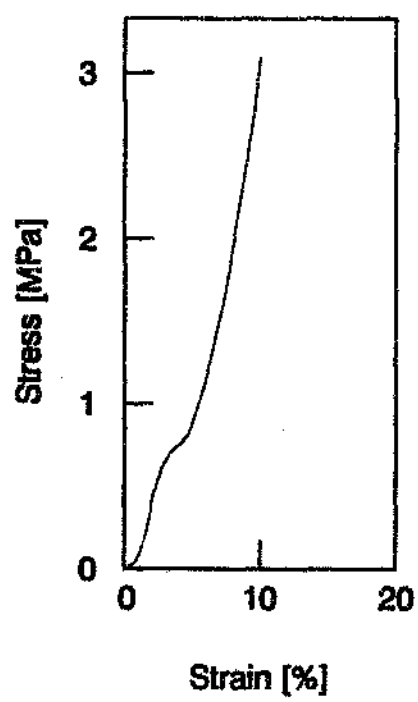
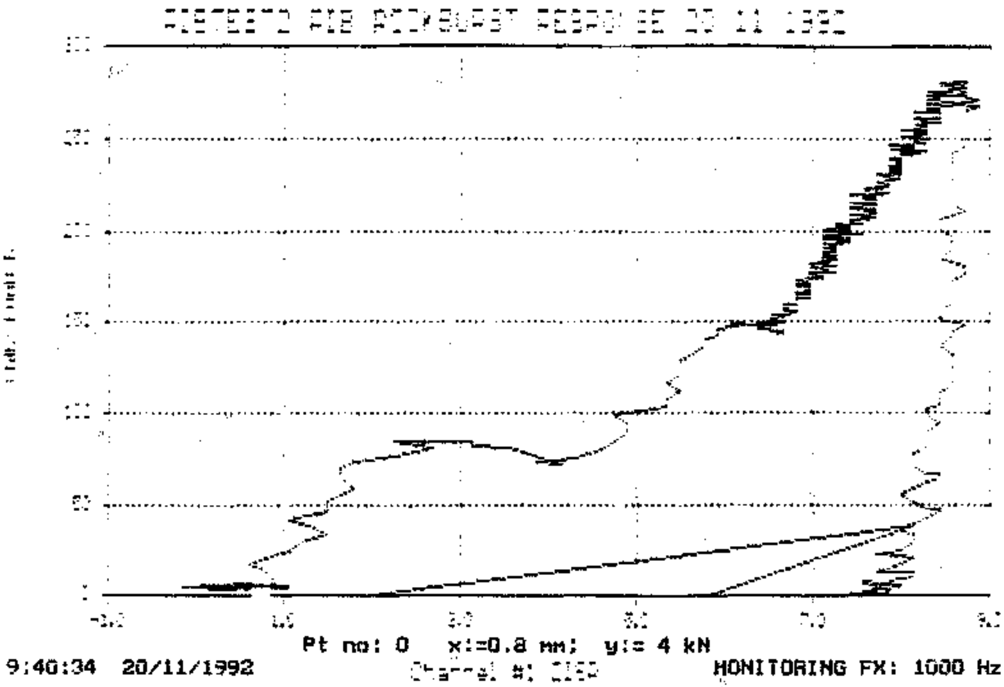


Figure 4.34 (a) Rapid loading compression tests, TerraTek press, 0,15 m/s closure, CCT, 10 % OPC, W/S 0,27 , 24 h curing, 2,5 mm rib spacing.



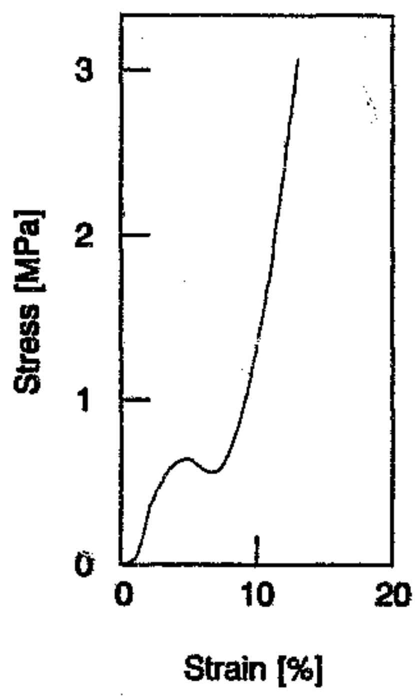


Figure 4.34 (b) Rapid loading compression tests, TerraTek press, 0,15 m/s closure, CCT, 10 % OPC, W/S 0,27 , 24 h curing, 5,0 mm rib spacing.

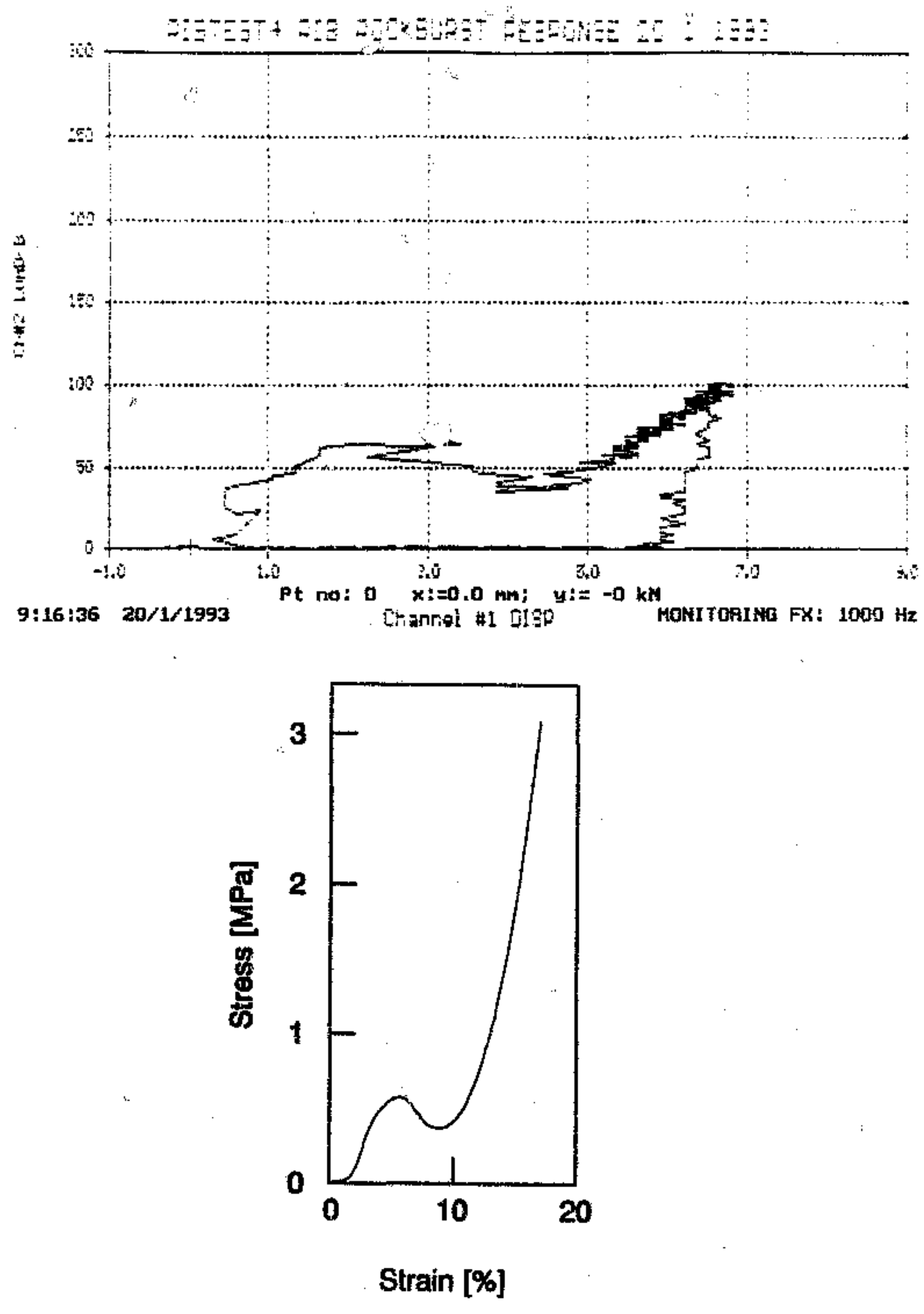


Figure 4.34 (c) Rapid loading compression tests, TerraTek press, 0,15 m/s closure, CCT, 10 % OPC, W/S 0,27 , 24 h curing, 7,5 mm rib spacing.

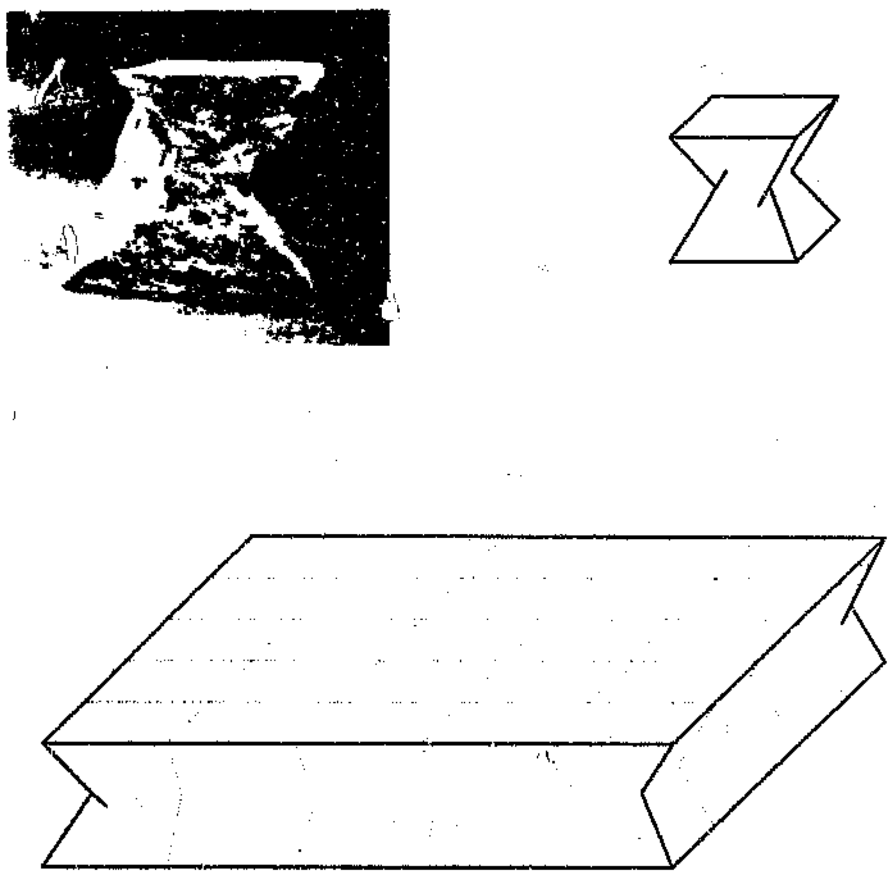


Figure 4.35 Shear failure at edges of cemented backfill samples of different width to height ratios.

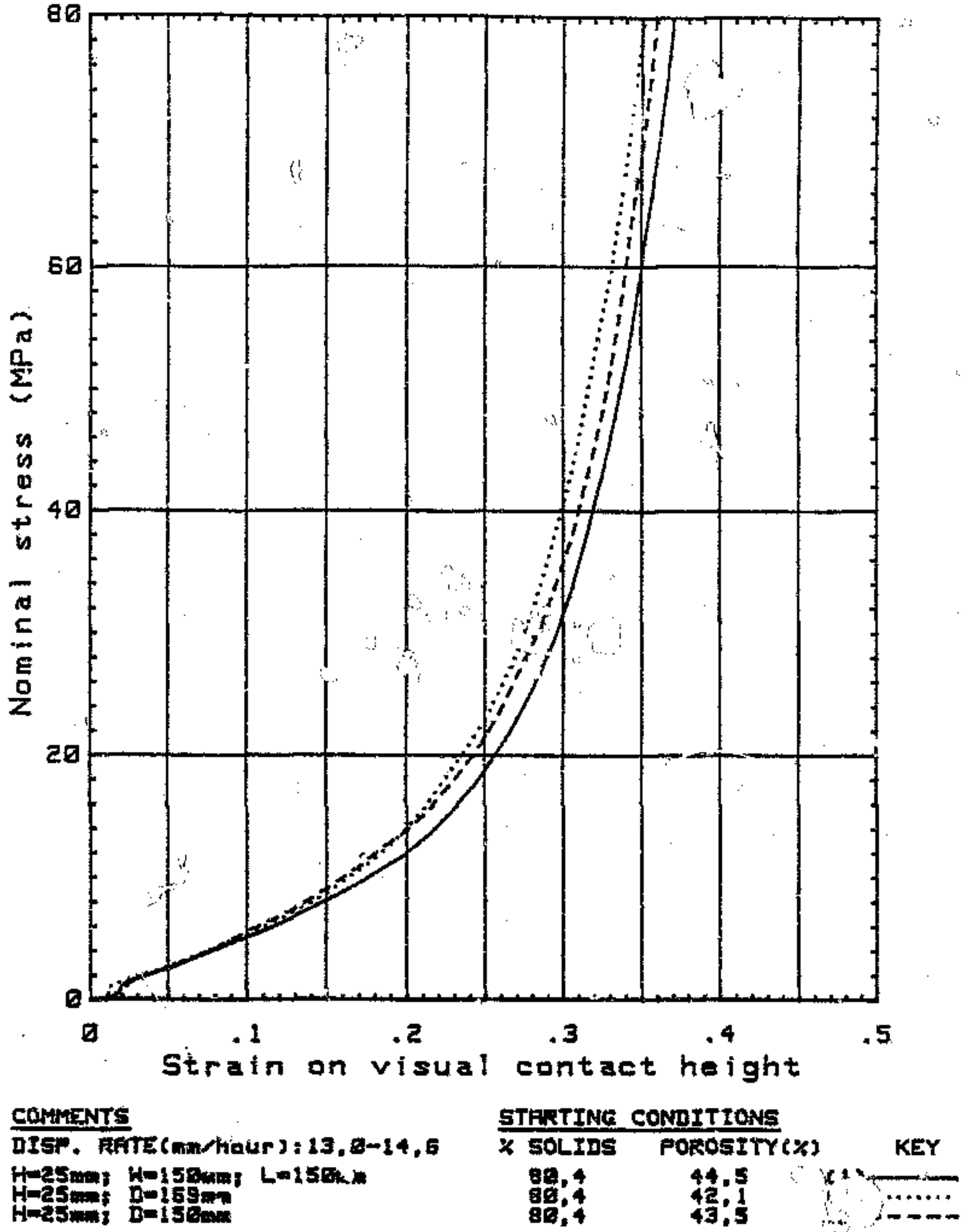


Figure 4.36 Comparison of compression curves of cemented backfill samples with square and circular cross-sections, FPT, 10 % OPC/FA 1:2 binder, W/S 0,28 , 7 d curing, [MPa].

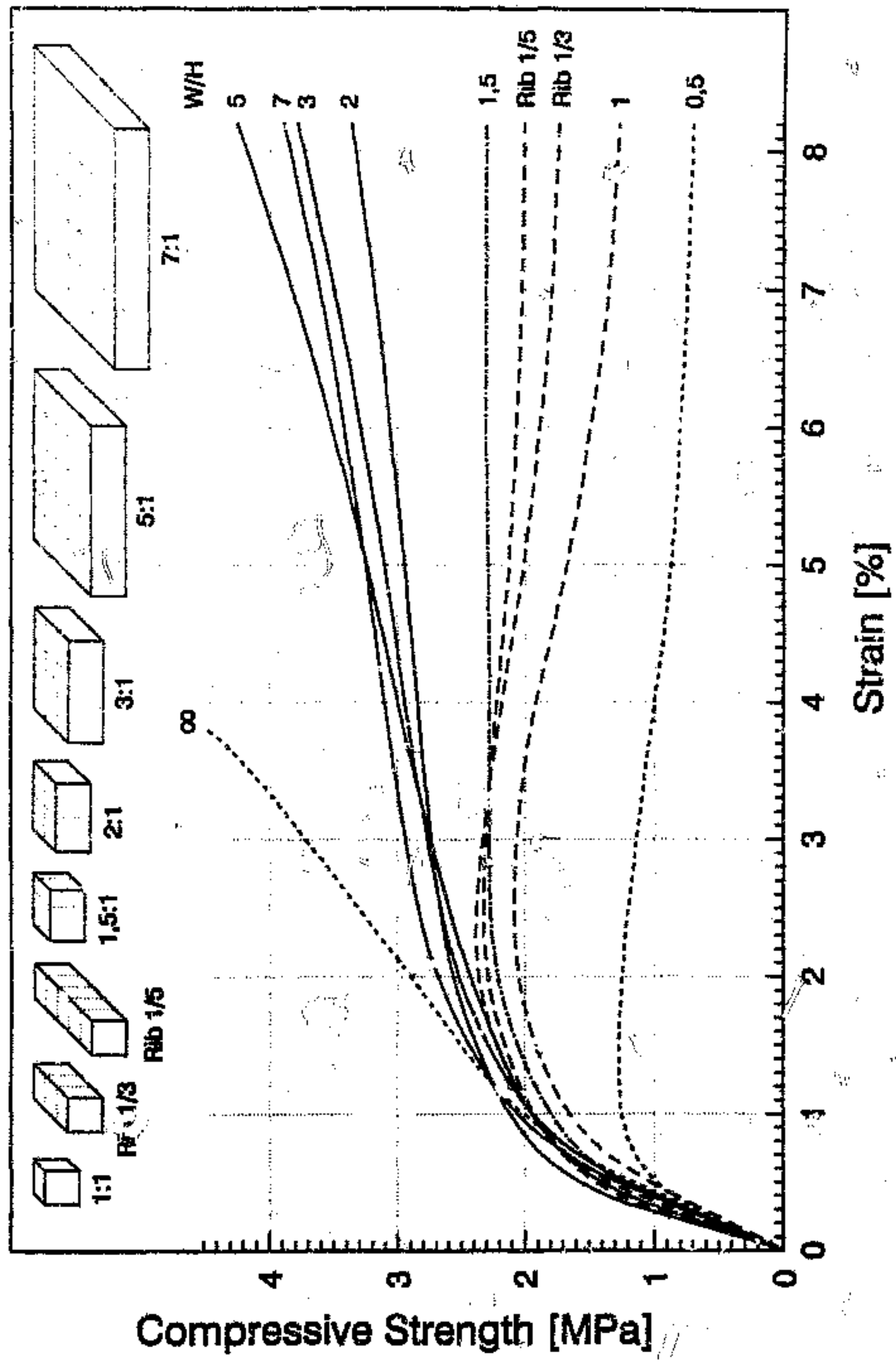


Figure 4.37 Post-failure behaviour of cemented backfill bodies of differing geometry.

5 DESIGN OF CONTROLLED LOADING RESPONSE CEMENTED BACKFILL

The previous two chapters described components which make up and identify cemented backfills in the widest sense. They described material constituents, engineering aspects, compressive strength parameters and loading behaviour patterns. In this chapter, all these components will be integrated in the design of cemented backfill with a predetermined, clearly defined load-deformation response.

A large variety of elements are available for the support of underground excavations. The suitability of any one element or system for a specific support requirement is determined by its load-deformation characteristic. In Chapter 2, it was shown that because of the high stresses and closure rates involved, as well as the possibility of violent stope closure due to rockbursts, deep level hard-rock mining places very strict demands on mine support. On account of their high support resistances, cemented and uncemented backfills have increasingly been utilised and have indeed become indispensable at great depth.

If the load-displacement characteristic of a hydraulic prop is considered, such as shown in Figure 2.7 (Chapter 2), it is realised, that the loading response of the prop is fully designed: the pre-load stress is set with an external pump and a non-return valve in the prop, the elastic response is a function of the stiffness of the relieve spring, the elasticity of the oil and the prop barrel, and the yield stress is set with the spring tension in the relieve valve. Within mechanical and fluid-dynamic limits, any one of those parameters can be changed at will. In many years of development and underground use, the loading response has been optimised and the prop has reached a high level of reliability.

In a similar, if less complicated manner, the loading response of timber packs has been improved and adjusted over many years to finally conform to the requirements of stope support. Eventually, with the incorporation of concrete blocks, the timber pack had developed into the composite pack, the loading response of which allowed it to take higher loads, faster, in a well defined compression response. Likewise, grout packs were modified with internal and external reinforcements to improve their behaviour under load in a controlled manner.

Until comparatively recently, the engineering and support behaviour of backfills was not well understood, presumably because backfill technology fell between the separate disciplines of mining engineering, rock mechanics and soil science. Only after about 1960 were soil science principles being applied systematically to mine backfill. Today, backfills are well understood by mining engineers and rock mechanics personnel, even though the modification of support characteristics by adjusting backfill material parameters and placement densities is still largely the domain of researchers and specialised consultants. Regrettably, the support functions of backfill are still poorly understood underground: many mine overseers and workers in deep South African mines do not yet perceive backfill as competent mine support, but rather as some cheap replacement for support.

This thesis deals with cemented backfill mine support and in this chapter, it is attempted to prove that cemented backfill can be an engineered material, that its performance and behaviour can be designed in detail, and that cemented backfill can be as sophisticated a support medium, as any other support type.

5.1 Principle of Controlled Loading Response Backfill Support

The loading response of typical cemented backfill was discussed in Section 4.1.2 and a general load/displacement curve was presented in Figure 4.2. If that cemented backfill compression curve is considered once more, three loading response phases, or regions, can be distinguished:

* the primary, binder-induced compression peak, consisting of the elastic rise, the yield point and the crest of the binder-induced peak, indicating the maximum primary compressive strength,

* the post-failure region, initially indicating the declining residual compressive strength imparted by the disintegrating remnants of the cement structure, a trough at the maximum cohesionless porosity, and the onset of the cohesionless stress rise, marking the beginning of the collapse of the material porosity, as well as

* the cohesionless strain-hardening region, reflecting the progressive collapse of the material porosity and the gradual crushing of solids at high stress.

Traditionally, backfills were composed and utilised either for the primary, binder-induced strength, or for their cohesionless strain-hardening qualities. As a consequence, the two types of backfill, i e cemented and uncemented, developed along separate paths. Both of these two loading regions are material controlled and backfill design, therefore, concentrated on the composition of materials. If backfill is viewed holistically, however, as a fully designed stope support medium, all of its loading characteristics should be controlled and applied to the task of support. Furthermore, the distinction between cemented and uncemented backfill should be limited to what it really is: backfill with, or without a cementitious additive.

Each of the three regions in the cemented backfill loading curve has a specific competence in terms of stope support and, depending on the total design of the support, can be varied in their relative magnitude. Moreover, the total response characteristic of the backfill can change with time - and can be altered after placement. A controlled loading response backfill is thus a backfill, which is designed to control all of its loading characteristics in accordance with stope or gully support requirements. This approach to backfill design will allow the optimisation of, both, the local and regional support capabilities of backfill in deep level, tabular mining conditions.

5.2 Design of the Primary Compression Region

The primary compression peak is of value principally in local stope and gully support, to prevent the disintegration of the fractured hangingwall and resultant rockfalls. It is also required in backfills, when stoping width exceed about 1,5 m and cohesionless fills would slump excessively under gravity. The primary compression peak is produced by the addition of a suitable cementitious binder to the backfill. Much of this thesis is devoted to the effects of compositional and environmental variables on this primary compressive strength. This is, in a way, justified, since the cost-effective control of this parameter is arguably the most intricate compositional problem of backfill design and, secondly, a comprehensive study on South African materials has not been conducted before. Since the relevant variables and their effects on material strength have been discussed in detail in the previous chapters, only the resultant design implications will be presented here.

5.2.1 General Backfill Composition

The shaded area in Figure 5.1 shows the domain of cemented backfills in the aggregate / cement / water ternary diagram. Aggregates form the major material components, of course, and usually the cement contents range up to about 10 %. The upper parallelogram delineates crushed and milled waste aggregates, the lower one shows the boundaries of tailings backfills. It can be seen, that the far right corner of the crushed waste parallelogram, where this material contains the most cement and the least water, overlaps somewhat with the weak side of the concrete field - low strength mass concretes. Uncemented backfills occupy the left edge of the diagram, at about the same aggregate content, but no cement addition. Further down that left edge, at about 60 % water content, one would find typical Witwatersrand gold plant tailings slurries as they are pumped to tailings disposal dams. At the bottom edge of the diagram, pure cement paste is located, the adhesive which binds concrete and cemented backfill aggregates.

5.2.2 Primary Strength Design Graphs

In Figure 5.2, the upper part of the ternary diagram is enlarged to show the immediate environment of the cemented backfill domain. It can now be seen, that the two parallelograms, delineating the two backfill compositions with crushed waste and tailings as aggregates, respectively, are effectively separated by their water contents. The two areas now also show compressive strength contours, as measured in the laboratory. The appropriate UCS values are plotted on the graph, with the decimal point indicating their actual positions in the diagram.

Two values are plotted outside of the contour areas, both of backfills containing 20 % portland cement. The two values illustrate graphically, how the reduced water content, or conversely, the increased solids content improves backfill strength: whereas the full plant tailings material at 28 % water/52 % aggregate (lower point) reaches only 3,26 MPa primary compressive strength, the crushed and milled waste backfill at 15 % water/65 % aggregate (upper point) achieves 15,17 MPa, nearly five times the strength. The much higher porosity and water demand of the full plant tailings would make it impossible in practice to mix a backfill composition with such a high solids content.

Another interesting fact is displayed in Figure 5.2: the slopes of the contour lines are different for the two backfills. The full plant tailings backfill shows nearly vertical contour lines, suggesting an approximately equal influence of binder content and water content on the primary compressive strength. The contour lines of the comminuted waste backfill, on the other hand, are rotated anti-clockwise by about 20°, indicating a higher sensitivity to water content, than to cement content. This is explained by the low water demand of comminuted waste; any excess water immediately reduces the cement/water ratio. The low strength of 0,23 MPa at the bottom of the CMW field attests to that. Like concretes, comminuted waste backfills are very unforgiving of excess water; this is true of, both, primary compressive strength and hydraulic transportation behaviour.

Ternary diagrams, like the one presented in Figure 5.1, are useful for the visualisation of the general relationship of the three main backfill constituents to each other, and of the position of backfills relative to other materials, such as concrete. If such diagrams are generated in sufficient detail, they can be used as design graphs. Since the proportion of any one of the three constituents can be predicted by the other two, however, simpler cartesian diagrams can be used: for cemented backfills, the more important water and cement contents are varied and the amount of aggregate is omitted; Figure 5.3 shows such a design graph. All percentages are now calculated in total slurry mass.

For the full plant tailings aggregate, which was used for this study, and ordinary portland cement, Figure 5.3 shows the contour map of the 28 day primary compressive strengths. Once again, the nearly perfect inverse relationship between water content and cement content for this material combination is reflected by the slope of the contour lines at close to unity. The curvature of the contour lines at low water and cement contents indicates that, as one would expect, a minimum of cement is required to develop any primary compressive strength and that below that minimum, no reduction of water content can, of course, produce cohesion.

Figure 5.4 shows a similar contour diagram, but the binder now consists of a blend of one part of portland cement and two parts fly ash. The primary compressive strengths are clearly lower than with neat portland cement, but the general trend is the same. There is, however, a cost implication in the use of this binder blend, compared to portland cement; this will be discussed in Section 5.5 further on.

5.2 3 Empirical Primary Peak Strength Predictions

There have been several attempts to predict the primary compressive strength from the composition of a cemented backfill [1], but they were mainly done on non-South African backfill materials. From the data gathered for the present thesis, empirical relationships were derived for the influence of backfill material composition, as well as curing time.

Compositional factors

If the relationship between the primary compressive strengths and the cement/water ratios of a wide range of cemented backfills is considered (Figure 4.20, Section 4.1.5), it is clear that this basic ratio is a primary element in the relationship. Unfortunately, however, the scatter of values is too wide to utilise the ratio as a predictive tool, other compositional factors had to be incorporated to reduce the scatter. Using regression analyses on the data, the following empirical relationship was established:

$$\sigma = e^{(P_1 + P_2(NCS/W) + (x \cdot y))} \quad , \qquad \text{(Eqn 5.1)}$$

1) For instance: Corson, (1970), Swan, (1985), Helms, (1988 a).

where

$$x = P_2\left[\frac{OPC}{W} + P_3\frac{FA}{W} + P_4\frac{PBFC}{W}\right] \quad .$$

$$y = 1 + P_5\frac{CCT}{NCS} + P_6\frac{CMW}{NCS} \quad .$$

σ = the primary compressive strength [MPa],

P_1 = -2,15 ,
P_2 = 5,65 ,
P_3 = 0,67 ,
P_4 = 1,60 ,
P_5 = -0,07 ,
P_6 = -0,34 ,
P_7 = 0,21 ,

and the following material masses in grams:

OPC	=	ordinary portland cement,
FA	=	fly ash,
PBFC	=	portland blastfurnace cement,
W	=	water,
CCT	=	cyclone classified tailings,
CMW	=	crushed, milled waste,
NCS	=	Non-cementitious solids.

The following limits apply:

Cement/water ratios between 0,10 and 0,50,
Total water/solids ratios between 0,20 and 0,50.

Within the given limits, the model allows the estimation of the 28 day primary compressive strengths of similar cemented backfills, cured at 30 °C and at 100 % humidity. The validity of the model has been tested by comparing the model strength predictions against observed values; this shows that the model predictions are statistically within ± 25 % of the observed strengths.

The poor prediction accuracy of ± 25 % is explained by the fact that all 28 day strength results were included in the calculation, i e aggregate materials ranging from full plant tailings to coarse crushed waste, binders including portland cement, portland blastfurnace cement and OPC/fly ash blends, as well as the full range of water/solids ratios and binder contents. It is clear that the scatter of values is far too high for any accurate strength predictions encompassing all types of cemented backfills. The model was, nevertheless, incorporated into a computer program, which was made available to the South African mining industry for the purpose narrowing the range of more detailed laboratory investigations.

The range of possible cemented backfill compositions is a very wide one, as could be seen in the previous chapter; there are numerous variables to consider and a single equation encompassing all these alternatives was bound to be inaccurate. Considering, furthermore, the site-specific choice of backfill materials, it must be concluded that a universal strength predicting equation is of little practical value. Design graphs such as those shown in Figures 5.3 and 5.4 are much narrower in scope, but are much more accurate indicators of cemented backfill strengths.

Curing time

It is well known that the primary compressive strengths of cemented backfills increase with curing time. Using all available data, Churcher (1989) modelled the time-dependent nature of the cemented backfills' primary strengths. The data suggested that either a square, or a power function could be used for the model. After detailed error analyses, the power function was selected:

$$Y = A\,X^{a} \quad , \qquad \text{(Eqn 5.2)}$$

where

Y = the primary compress... ength and

X = is the curing time in days.

The A parameter for the curve will be determined by:

$$A = \exp[\ln(Y) - B\ln(X)] \quad .$$

where

B = 0,394 ,

Y = the 28 day primary compressive strength and

X is the day number (28).

The primary compressive strength can be estimated for any curing time from 3 to 90 days. During the analysis of the data, it appeared, as if two different populations were present, weak and strong backfills, following separate curve parameters, but on closer analysis it was found that there was no statistically significant difference between weak and strong cemented backfill compositions.

5.3 Design of the Post-Failure Region

The post-failure region in the cemented backfill compression curve describes the behaviour of the fill after the break-down of the cohesion structure, provided by the binder. This part of the compression curve can be modified by the shape of the back-fill body, as described in Section 4.2.3, or by placing two or more backfill elements of a relatively low width to height ratio at some distance from each other. If properly arranged, this layout ensures that under compression, the dilating backfill bodies make contact with each other, rapidly consolidating themselves into a single, stiff body of large width to height ratio.

5.3.1 Backfill Model Rib Compression Tests

The control of the post-failure backfill behaviour was demonstrated in a series of compression tests, whereby five adjacent ribs of cemented backfill were spaced at increasing inter-rib distances. The sample ribs were 250 mm long and each was 50 mm high and 50 mm wide. The aggregate used was a quartzitic classified tailing, the binder consisted of 10 % portland cement, the water solids ratio was 0,28 and the samples were compressed at a closure rate of 100 mm/h. The samples were prepared in a custom-made mould and were cured for 24 hours before testing. Figure 5.5 shows the basic layout of the backfill rib arrangement. The traces of the test series are compared in Figure 5.6.

Figure 5.6 (a) shows the compression curve of the five ribs with no inter-rib spacing. At that width to height ratio of 5,0 , there is no indication of the binder-induced primary compressive peak and the sample reached the maximum test strength of 3,2 MPa at 8 % deformation. For each subsequent test, the inter-rib spacing was increased by 2,5 mm and the traces are shown as curves (b), (c), and (d), respectively. It can be seen that the binder-induced peak gradually appeared, and that the deformation required to reach the 3,2 MPa stress level gradually increased with every increase of the inter-rib spacing from the original 8 % to the final 18,6 % for the 7,5 mm spacing. The slopes of each of the compression curves at the 3,2 MPa level can be seen to be nearly identical, indicating equivalent positions on the material-controlled cohesionless strain-hardening curves of the backfills, except that the arrival at this position is delayed by the collapse of the inserted inter-rib space.

Figure 5.7 shows a further modification of the backfill compression curve: here the 7,5 mm spacing, one day compression curve is compared to a geometrically identical layout but with the same backfill cured for 14 days. The much increased primary strength is immediately apparent, along with a reduction of the deformation required to reach the 3,2 MPa strength; this strength was reached at only 14,6 % strain, compared to the one-day backfill's 18,6 %. This reduction can be ascribed to the cement crystals, which have grown inside the backfill for 13 more days. they reduced the porosity of the backfill sufficiently to account for the earlier 'take-off' of the cohesionless material.

Identical samples to those tested for this series were also tested at high closure rates (Section 4.1.9). It was concluded that at a closure rate of 0,15 m/s. 5400 times higher than the present tests, the backfill ribs behaved essentially in an identical fashion. The closure rate of 0,15 m/s is equivalent to a *closure* rate of 3 m/s in a one meter wide stope.

5.3.2 Implications for Stope Support

It can be appreciated that by utilising this 'inserted porosity' and by altering the width to height ratio of the individual backfill support elements, the capacity of the backfill to yield a specific amount under load can be regulated. This behaviour can be seen as analogous to the yielding of a hydraulic prop under load by releasing hydraulic fluid. Once the elements are firmly abutting, the large width to height ratio takes effect and the material builds up stress very rapidly. Once mining has progressed for some distance and the backfill has cured sufficiently, the inter-rib spacing should be filled with backfill to improve its regional support capability.

In the light of stope support requirements, as described in Chapter 2, particularly under rockburst conditions, this capacity of backfill support to yield and absorb rapid stope closure and then to decelerate the hangingwall by the rapid strain-hardening of the now large width to height ratio backfill mass, has the potential of substantially increasing mine safety in stopes and gullies at great depth.

It was mentioned in the previous section, that the models were also tested at closure rates of 0,15 m/s. Over a 50 mm sample height, this is the same closure rate, as 3 m/s is in a 1 m wide stope. It must, however, be borne in mind that at a closure rate of 3 m/s in a real stope, an unconsolidated backfill will dilate at a very much higher velocity than what is observed in a small-scale model; correspondingly, the inertia of the full-scale mass of material is much higher than what is attained in the model. In the event of a rockburst occurring in a stope containing unconsolidated backfill, spacing between backfill ribs could absorb some of that destructive inertial energy, in effect partially collapsing on itself.

5.4 Design of the Cohesionless Compression Region

The cohesionless compression region of a cemented backfill stress-strain curve directly describes the stiffness of the material at high compressive stresses. Its significance lies, therefore, primarily in the regional support capability of the backfill. Since cemented backfills, which are often introduced to control local support problems, eventually become regional support, this portion of the total compression characteristic should also be considered in the initial design of the backfill.

The compression behaviour of cohesionless backfills has been described in detail by several researchers, including Briggs (1985) and Clark (1988 c). The consensus is that the behaviour is material dependent and is, in fact, directly correlated to the porosity at closest packing of the cohesionless backfill. The relative compressive strengths of cohesionless backfills are compared in confined compression, with the materials prepared at their minimum porosities. To obtain input parameters for numerical modelling, cohesionless backfill compression behaviour is described by either one of the following two fitting functions (Ryder, 1988). The first, a hyperbolic equation:

$$\sigma = \frac{a\epsilon}{(b-\epsilon)} \quad ; \qquad \text{(Eqn 5.3)}$$

the second equation is hyperbolic-quadratic:

$$\sigma = \frac{a(\epsilon-\epsilon_r)^2}{(b-\epsilon)} \quad , \qquad \text{(Eqn 5.4)}$$

where σ is stress, ϵ is strain, and a, b and ϵ_r are constants determined from the stress-strain curve. The root mean square (rms) errors are considerably larger with the hyperbolic prediction, than they are with the quadratic function, which, therefore, appears to be the better choice.

Figure 5.8 shows the stress-strain curves of a range of different, uncemented backfill aggregate materials. The addition of a cementitious binder to the backfill material results in the growth of cement crystals within the pores of the backfill. Once the rigid structure so produced is broken down, the remnant crystals reduce the (now cohesionless) backfill porosity and effectively increase its stiffness. The quantitative reduction of the porosity can be estimated from Figure 4.28 and under ideal conditions can lead to a reduction of the b-value of up to 25 %. This effect was described in Section 4.1.6 and results were shown in Figure 2.9.

5.5 General Design Considerations and Cost Implications

The design of backfill stope support is to a large extent a site specific undertaking and always requires the simultaneous consideration of several critical factors; it requires decisions and compromises and, perhaps most importantly, an accurate assessment of the real support requirements. A list of relevant questions can provide guidance in selecting the most suitable and cost-effective backfill support, disregarding fashionable trends and obsolete customs. Such a list would comprise several tens of questions and their inter-connections, encompassing, among others, the areas of stope support requirements, material suitability and availability, possible cementitious binders, hydraulic transportation, as well as backfill placement and permeability. Most of the questions have, in fact, been addressed in this thesis and the list of contents gives a reasonable perspective of the scope of the design task.

The one important factor which remains to be discussed is the cost of backfill stope support, including the cost of placed uncemented backfill and the cost-effectiveness of cementitious binders. Only a few publications have been issued on the first costs of backfill systems and the actual placed costs of the different backfil types; one of the best is by Kramers and Skirving (1989) on the capital and working costs of a classified tailings backfill system. From this and similar studies at COMRO, the following (1992) costs of placed uncemented backfill can be assumed:

Table 5.1 Placed costs of various uncemented backfill types (1992) [R/t].

Aggregate	Placed Cost R/t
Cyclone Classified Tailings	30,00
Full Plant Tailings	32,00
Comminuted Waste	35,00

The working costs include labour, power and consumables for preparation, overland and underground hydraulic transportation, shaft storage, placement and drainage water handling. The costs also include an amortised depreciation scheme, assuming a system life of 15 years and an inflation rate of 15 %. Working costs are sensitive to system utilisation; for the above estimates, 75 % utilisation has been assumed [1]. It has been shown, that the highest working cost element is backfill placement, at 73 % of the total working costs. In an analysis of the mine ventilation savings incurred by the use of backfill, Haase (1992) has shown that under favourable circumstances, the savings in mine cooling can, in fact, outweigh the costs of backfill.

The relative efficiencies of cementitious binders can be compared when the cost per unit strength (1 MPa) is considered. Two backfills are being compared: one of 0,5 MPa primary compressive strength and one of 1,5 MPa (both strengths after 28 days curing). There is a choice of two binders: ordinary portland cement and a 1:2 blend of ordinary portland cement and fly ash. The aggregate is full plant tailings, the total solids content in the backfill slurry is set at 75 %. For the comparison, the (1992) costs of the binder components were assumed as:

1) At 100 % utilisation, for instance, the cost for the classified tailings backfill would reduce from R 30,00 per ton placed to R 25,00 per ton placed.

Table 5.2 Delivered costs of portland cement (OPC), fly ash (FA) and a 1:2 blend of OPC and FA (1992) [R/t] and [R/kg].

Binder or Binder Blend	Cost * R/t	Cost * R/kg
Ordinary Portland Cement	250,00	0,250
Fly Ash	87,00	0.087
OPC/FA 1:2 Blend	141,33	0,141

* Delivered cost at a Carletonville gold mine

From Figures 5.3 and 5.4 the binder contents required to achieve 0,5 MPa and 1,50 MPa primary compressive strength, respectively, after 28 days, at a slurry total solids content of 75 % can be read as:

Table 5.3 Binder requirements (OPC and OPC/FA 1:2) for backfills of 0,5 MPa and 1,5 MPa 28 day primary compressive strength, respectively [%].

	Binder Requirements, [%]			
	OPC		OPC/FA 1:2	
Composition	0,5 MPa	1,5 MPa	0,5 MPa	1,5 MPa
Full Plant Tailings + OPC	2,10	6,40		
Full Plant Tailings + OPC/FA 1:2			3,60	10,90

In Table 5.3 it can be seen that when the OPC/FA binder blend is used, substantially more binder is required for the specified backfill strengths; this binder is, however, cheaper than the straight portland cement. From the respective binder costs (Table 5.2) and binder requirements (Table 5.3), the total binder costs in two 100 kg batches of backfill of 0,5 MPa and 1,5 MPa strength, respectively, can then be calculated. From these values, the binder cost per unit strength (1 MPa) can be determined for both backfills which allows a direct efficiency comparison of the two binders (Table 5.4).

Table 5.4 Cemented backfill comparison in terms of total and strength specific binder costs.

	Backfill Binder Cost [R/100kg backfill]	
Composition	OPC	OPC/FA 1:2
Cemented FPT, 0,5 MPa	0,53	0,51
Cemented FPT, 1,5 MPa	1,60	1,54
	Backfill Binder Cost per 1 MPa [R/MPa/100kg backfill]	
	OPC	OPC/FA 1:2
Cemented FPT, 0,5 MPa	1,05	1,02
Cemented FPT, 1,5 MPa	1,07	1,03

It can be seen, that the OPC/FA 1:2 binder blend is more cost effective than pure portland cement in achieving, both, low and high primary compressive strengths at a given slurry relative density. It can also be seen, that it is marginally less cost-effective to produce higher primary compressive strengths, than lower ones.

5.6 Summary and Conclusions

In this chapter many of the findings of the reported investigations were integrated in a holistic approach to cemented backfill stope support design. It was shown that each of the three regions in the cemented backfill loading curve has a specific competence in terms of stope support and can be varied according to stope support requirements. The total response characteristic of the backfill changes with time in a predictable manner and can to some extent be altered after placement.

Ternary and cartesian type cemented backfill design graphs and empirical equations were described as aids for the design of the binder-induced primary compression region. They were derived from data established in an extensive investigation, most of which was presented in Chapters 3 and 4. The high stress compression behaviour of backfills, determining the strain-hardening region of the curve was reviewed and it was reiterated that this behaviour was largely controlled by the minimum porosity of the backfill aggregate. This porosity can, however, be significantly reduced by residual fragments of the broken down cement crystal structure; this reduction can at times lead to a lowering of the backfill's b-value by up to 25 %.

From the investigations presented in Chapter 5, it is concluded that cemented backfill support capabilities can be enhanced significantly by modifying the previously neglected post-failure region of the backfill load-deformation response. By spacing cemented backfill support elements at pre-determined distances of each other, their response to rapid, high stope closure can be controlled. Once the relatively soft dilating elements are firmly abutting, the large width to height ratio takes effect and the material builds up stress very rapidly. The collapse of the cemented backfill elements into the 'inserted porosity' under rapid stope closure can be seen as analogous to the yielding of a hydraulic prop under load by releasing hydraulic fluid. When, after some face advance, this safety configuration was not required, the inter-element spacing should be filled with backfill to improve the backfill's regional support capability.

In the light of stope support requirements, as described in Chapter 2, particularly under rockburst conditions, this capacity of backfill support to yield and absorb rapid stope closure and then to decelerate the hangingwall by the rapid strain-hardening of the, now, large width to height ratio backfill mass, has the potential of substantially increasing mine safety in stopes and gullies at great depth.

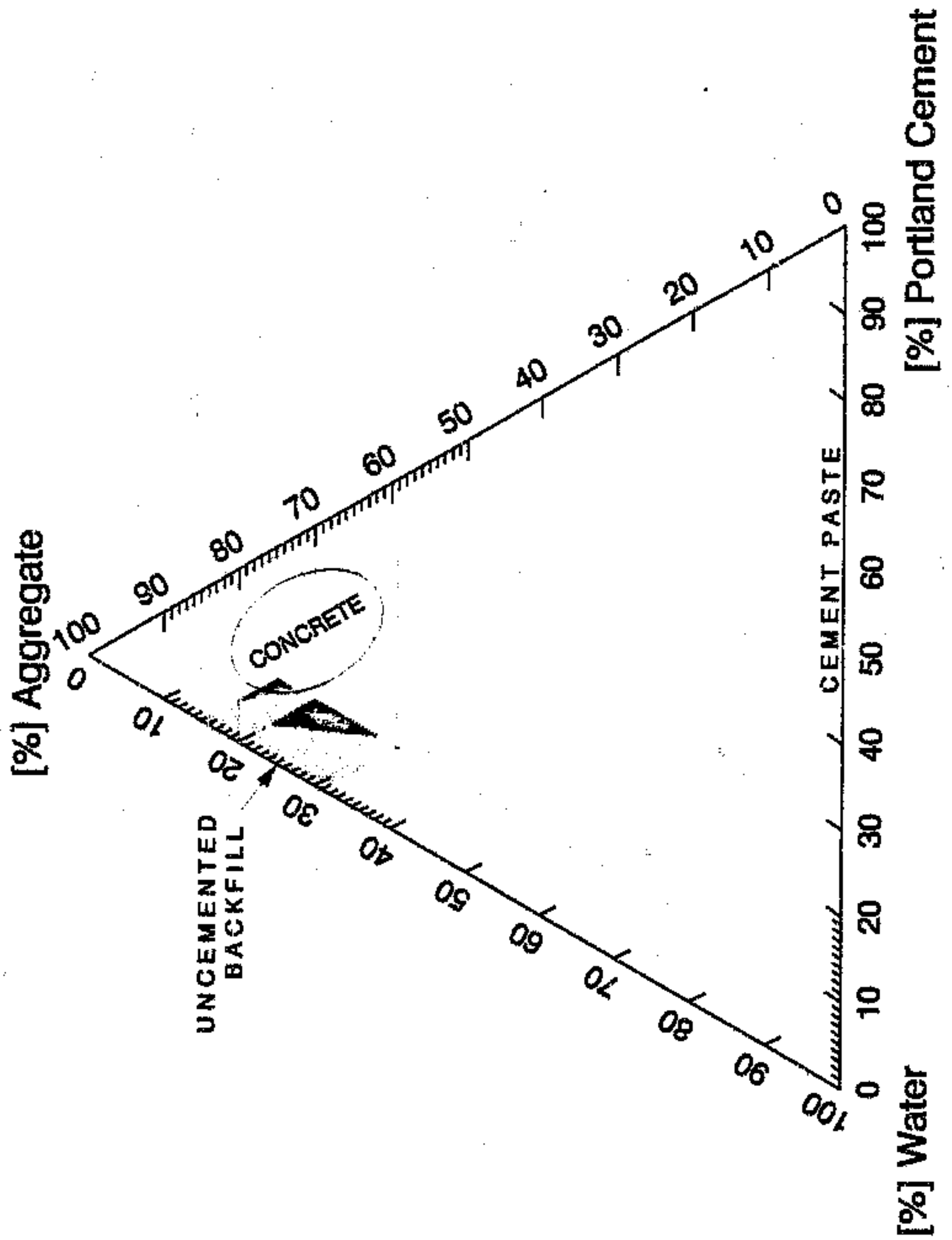


Figure 5.1 Ternary diagram of cemented backfill compositions.

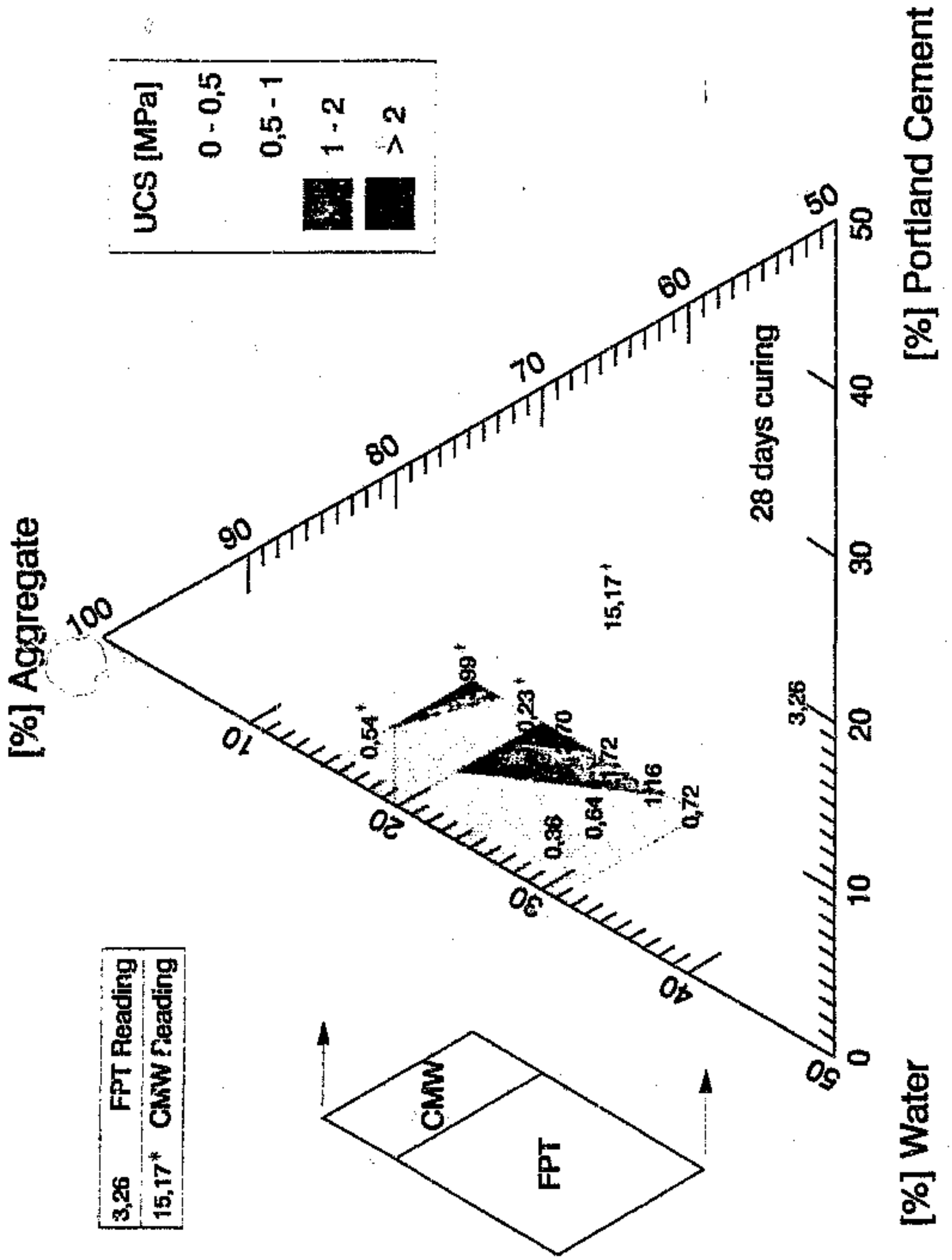


Figure 5.2 Ternary diagram of cemented backfill compositions and primary compressive strengths, enlarged, full plant tailings (FPT), crushed, milled waste (CMW).

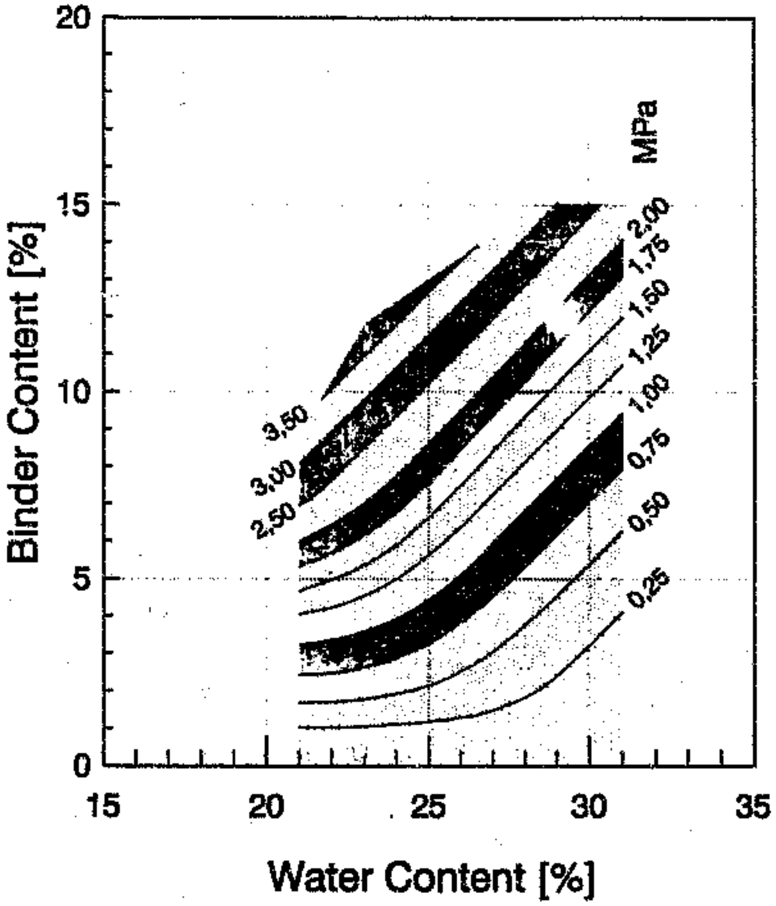


Figure 5.3 Cemented backfill design graph, FPT, OPC.

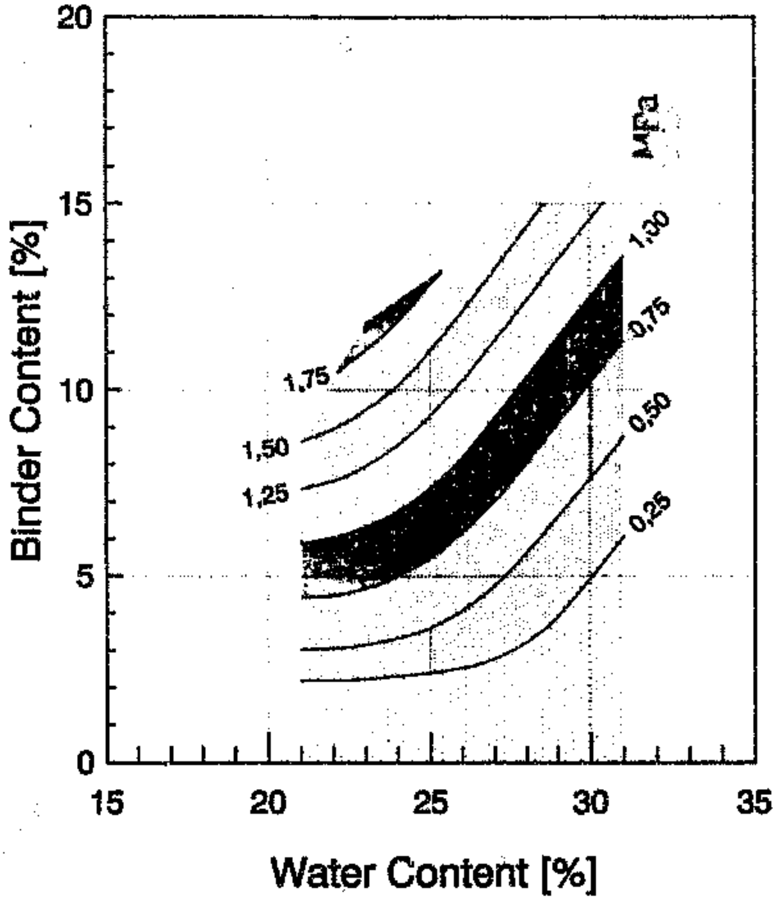


Figure 5.4 Cemented backfill design graph, FPT, OPC/FA 1:2.

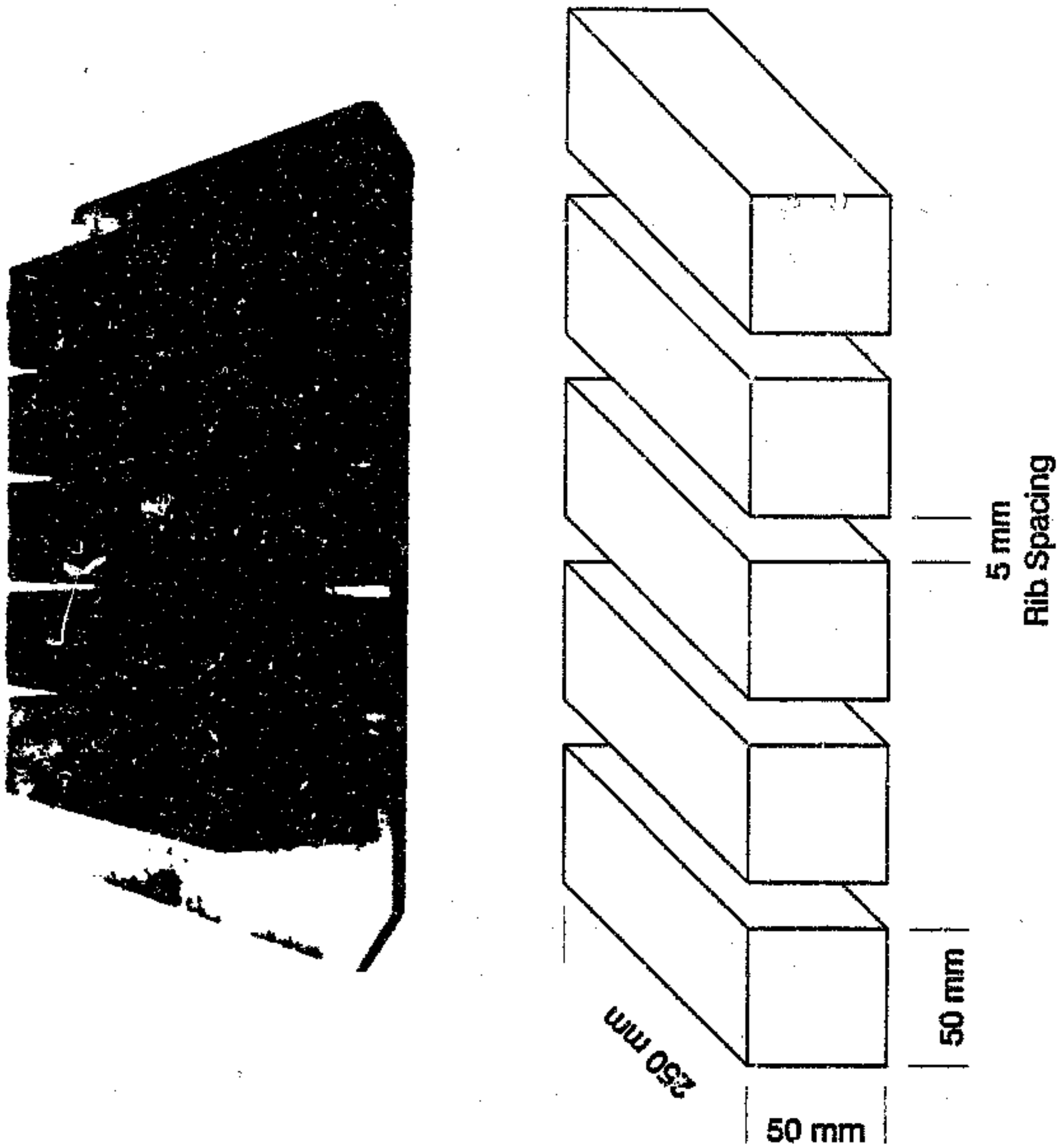


Figure 5.5 Layout of separated cemented backfill rib arrangement for compression testing.

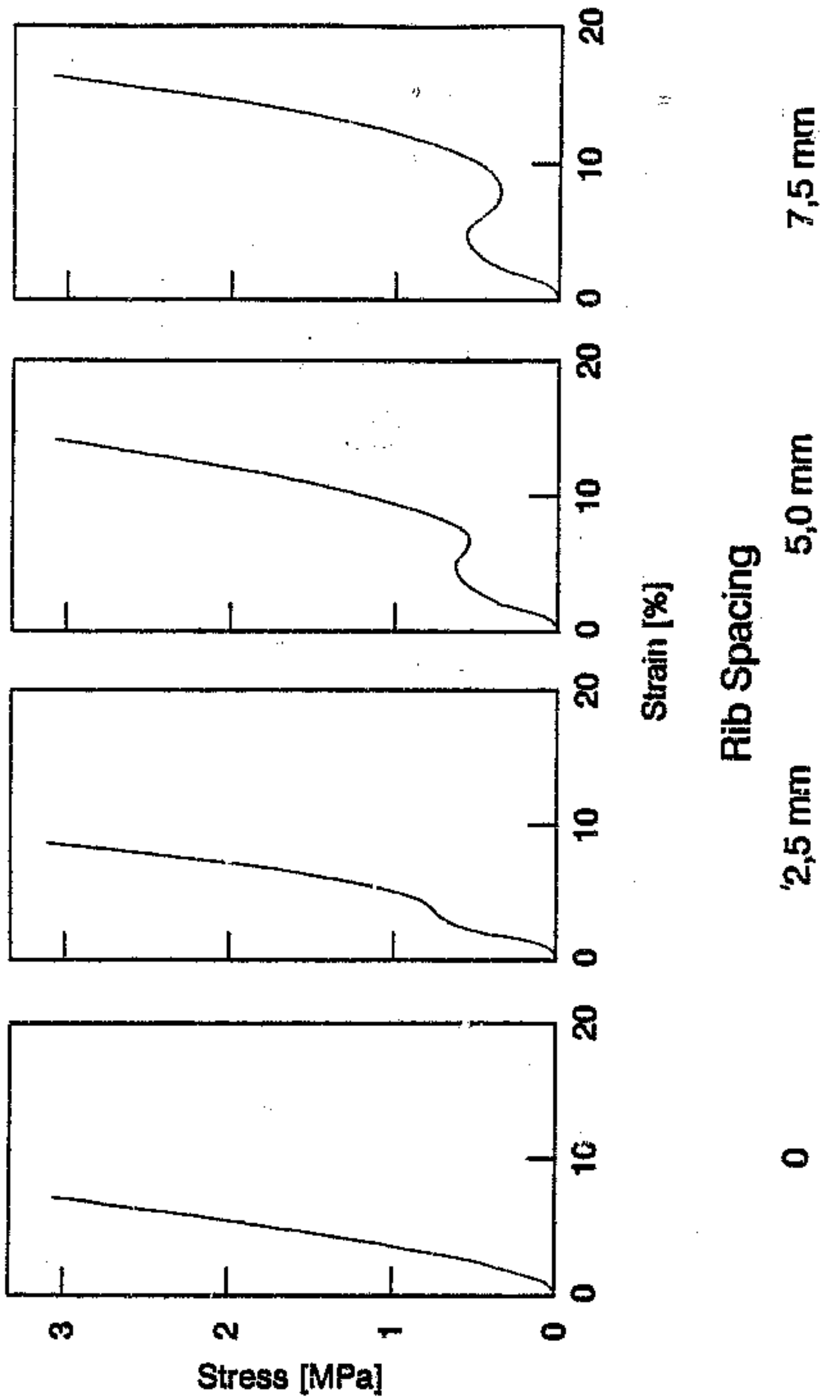


Figure 5.6 Compression curves of cemented backfill ribs at increasing inter-rib spacing, FPT, 10 % OPC, W/S 0.27 , 24 h curing.

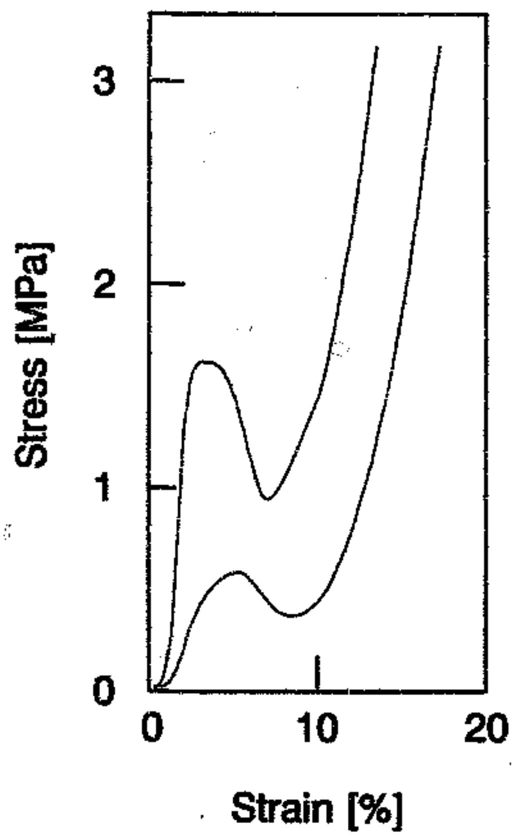


Figure 5.7 Comparison of two backfill rib compression tests, identical material, identical geometry, different curing times: 24 h, and 14 d, respectively.

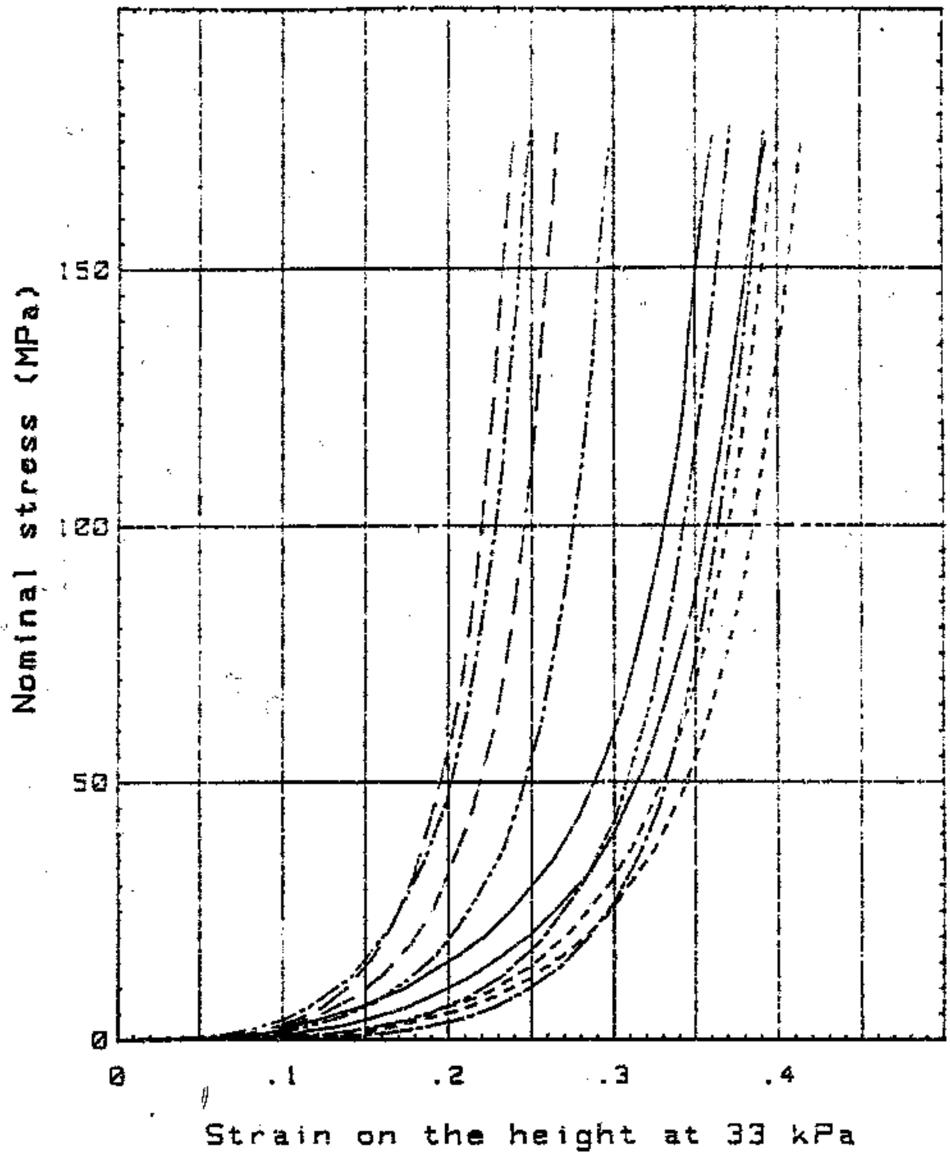


FIGURE 27 RESULTS OF CONFINED COMPRESSION TESTS ON 5 MATERIALS

EAST DRIEFONTEIN DESLIMED TAILINGS (91% SOLIDS)
VAAL REEFS DESLIMED TAILINGS (91% SOLIDS)
CHAMBER OF MINES ROD MILL PRODUCT (93% SOLIDS)
CHAMBER OF MINES RIB (WDL) (93%SOLIDS)
FREE STATE GEDULD BELT FILTER PRODUCT (87% SOLIDS)

Figure 5.8 Confined uniaxial load-deformation curves of a range of different, uncemented backfill aggregate materials.

6 SUMMARY AND CONCLUSIONS

A survey of the published literature showed that, to date, a comprehensive study on South African cemented backfill, considering material and structural aspects of cemented backfill support design, had not been undertaken. In the light of the increasingly stringent support requirements in deep South African mines, there was a need, therefore, to conduct a detailed study of cemented backfills with the objective of providing relevant information for the design of improved, cost-effective cemented backfill mine support; that was the main purpose of this study.

In total, close to 500 cemented backfill samples of about 35 significantly different compositions were tested during the course of this investigation. The work generated a large number of results and findings, all of which contributed to the aim of providing design information for narrow stope cemented backfill support. The most important findings are summarised in the following sub-sections.

From the investigations undertaken for this study, it is concluded, that by integrating the material design of cemented backfills with the spatial configuration of backfill support elements, it is possible to effectively control all phases of the cemented backfill loading curve. This implies the control of the large-scale yielding behaviour of backfill support, in addition to the binder-induced stiffness at low strains, and the improved stiffness at high loading stresses.

In practice, cemented backfill support elements, usually ribs, would be spaced some distance apart on placement. Under rapid closure, the ribs dilate and yield to a predetermined height, until they abut to adjacent ribs. At that moment, the effective width to height ratio of the support changes to a very much higher value and the support stiffens rapidly. In stable mining conditions, the gaps between ribs a few rows back would be filled with backfill to ensure the stiffest possible regional support.

This principle allows the construction of a backfill support system, which provides outstanding local and regional backfill support, under static and dynamic loading conditions. It is understood, that the bulky nature of hydraulically placed backfill support does not allow its deployment at the work face, neither is backfill support able to put active load against the stope hangingwall. Rapid-yielding hydraulic props, therefore, have to form the first lines of support at the face. By matching the load deformation characteristics of the adjoining backfill elements to those of the props, however, cemented backfill can form an excellent complementary support system.

In the light of stope support requirements, particularly under rockburst conditions, this capacity of backfill support to yield and absorb rapid stope closure and then to decelerate the hangingwall by the rapid strain-hardening of the now large width to height ratio backfill mass, has the potential of substantially increasing mine safety in stopes and gullies at great depth.

6.1 Backfill Stope Support Requirements

Three types of stope support are effectively differentiated in deep level, high stress mining environments: local face area and gully support under static loading conditions, local face area and gully support under dynamic loading conditions (i e rockburst conditions) and regional support. Backfills, both uncemented and cemented, have support properties which make them particularly well suited for deep mine support and are for that reason increasingly being used in South African mines. The following are the requirements of backfill stope support:

* Under static, or slowly changing stress conditions, a support resistance of about twice the carried deadweight is required for local stope support. Therefore, a minimum support resistance of 50 kN/m^2, applied as soon as possible, is typically recommended. Specific mining conditions may, however, demand significantly higher support resistance.

* The maximum acceptable stope closure under rockburst conditions is considered to be 0,3 m in narrow stopes. For a 3 m thick hangingwall beam to be decelerated within 0,3 m, at a ground velocity of 3 m/s, a support resistance of at least 200 kN/m^2 has to be provided. Little benefit is, however, gained by increasing the support resistance to over 400 kN/m^2.

* For the provision of adequate regional support, it is aimed to reduce the ERR to a value below 30 MJ/m^2, by restraining volumetric convergence through suitable stoping layouts, the provision of regional stabilising pillars, and with high quality backfills. Backfills with a low figure of merit, i e low porosity, placed early in the mining cycle to avoid excessive stope closure, can provide very effective regional support.

6.2 Backfill Constituent Materials

This chapter, by its nature, was mainly descriptive; it provided the necessary background information on the material properties of cemented backfill constituents, for the design of backfill support systems. The following paragraphs briefly summarise the main results:

* A detailed catalogue has been provided of the characteristics of the material components which make up hydraulically placed backfills in deep South African hard-rock mines. The materials include backfill aggregates, binders, and fresh backfill slurries.

* The most commonly used backfill aggregates are: full plant tailings, containing on average, about 20 % ultra-fines; classified tailings, with about 5 % ultra-fines; and crushed waste, characterised by a well graded size distribution and a large top grain size.

* Backfill binders typically consist of an active cement, such as ordinary portland cement, and a cement extender (or pozzolan), such as fly ash or ground, granulated blastfurnace slag. Two crucial compounds are produced in the cement hydration reactions: calcium silicate hydrate (or CSH), the main cementation compound, and calcium hydroxide, which makes possible the additional hydration of otherwise inactive pozzolans, such as fly ash.

* It was shown that pipeline pressure losses of classified tailings slurries are considerably lower than those of full plant tailings. This fact, combined with the vastly improved drainage rate and shear strength development of classified tailings, compared to full plant tailings, explains why nearly all mines practice backfill classification.

* The addition of binder to a backfill slurry increases its pumping pressure losses. This is contrary to a commonly held believe in the industry.

* In an attempt to clarify ambiguous backfill terminology, several backfill terms were defined for this thesis, including; *aggregate*, to be used for all non-cementitious backfill bulk materials; *fines*, to be used for backfill aggregate of grain size smaller than 4,75 mm; and *ultra-Fines*, to describe the backfill bulk material fraction smaller than 0,01 mm.

6.3 Load-bearing Characteristics of Cemented Backfills

Chapter 4 was concerned with the load bearing properties of cured cemented backfill materials and how they relate to compositional, environmental and geometrical variables. Twenty two different cemented backfill compositions were tested to determine the influence of material composition on their compressive strengths. They were arranged in four main compositional groups, identified, *a priori*, as the most influential compositional variables on cemented backfill strength:

* Aggregate type, i e aggregate particle size distribution,
* Water content, i e water/solids ratio,
* Binder content and
* Binder type.

Several non-compositional variables were investigated; some were procedural elements of the investigations, others were studied in separate test series. they included:

* Cemented backfill curing time.
* Curing temperature and environment.
* Backfill sample porosity and
* Loading parameters.

From the results presented, it is evident that all of these variables, except the loading parameters, influence the primary compressive strengths of cemented backfill materials. In a second major investigation, geometrical parameters were analysed for their influence on cemented backfill load bearing behaviour:

* Backfill specimen volume,
* Specimen cross-section and
* Backfill sample width to height ratio.

The following paragraphs summarise the behaviour of cemented backfill support materials:

* *Aggregate type, aggregate particle size distribution.* The inherent backfill particle size distribution, which controls the porosity of the backfill, has a most significant influence on the strengths of the whole range of cemented materials. The relative strengths approximately follow minimum porosities of the uncemented backfill

materials: the lower the inherent porosity, the higher the cemented backfill strength. The amount of ultra-fines ($\leq$ 0,010 mm) present in the fine backfill aggregate plays a very deleterious role: an excess of these fractions introduces very large surface areas to the backfill, with the associated high water demand and unfavourable cement/water ratio. At the same time, the permeability of the backfill is reduced by the ultra-fines, preventing efficient drainage.

* *Water content, water/solids ratio.* Provided that a cemented backfill remains within practical pumping and placement limits, a reduction of its water content is comparable to an increase in binder content - it improves compressive strength significantly. The increase in strength can be ascribed to a decrease of the placed (macro-) porosity of the fill, and to the more favourable cement/water ratio in the binder paste, substantially reducing the micro-porosity in the cement gel. An excess of water has a detrimental influence on all types of cemented backfills, regardless of binder type.

* *Binder content.* Cemented tailings samples increase in strength approximately in proportion to their cement content. This is due to a proportionally increased number of cement crystals forming in the pores of the backfill material, resulting in increased adhesion and reduced porosity. The addition of 3 % binder to a backfill was found in practice to be about the lower practical limit, particularly with blends of portland cement and pozzolans. Below 3 %, proper mixing of the binder into the backfill becomes a problem and large pockets of uncemented fill were found in practice.

* *Binder type.* Of those cements tested in this series, the portland cement backfill shows the fastest initial curing rate. All binders containing pozzolans set out at a slower rate and retained their relative strengths throughout the curing period. Backfill prepared with portland blastfurnace cement exceeded the portland cement backfill in strength after about a week and remained substantially stronger from then on. This good performance is probably due to the fine grind of it's 50 %

OPC fraction, which increases it's reactivity. The dilution of portland cement with fly ash reduces its effectiveness approximately in proportion to the mix ratio. Different fly ashes have different pozzolanic activities, however, and the strengths of cemented backfills can vary widely with the type of ash used.

* *Curing time.* Cemented backfills attain strength through the growth of cohesive cement crystals, this happens at rates dependent on curing temperature and binder type. For practical purposes, the fill can be regarded as fully cured after about 90 days.

* *Curing temperature.* Elevated curing temperatures accelerate the gain in strength, i e the hydration reaction of cemented backfills; this is provided that full saturation is maintained and the temperature does not exceed a level, at which cement crystal growth is impaired. For conventional South African cemented backfills, this limit is approached at about 50 °C.

* *Backfill porosity.* The compressive strengths of cemented and uncemented backfills are inversely proportional to backfill porosity. Various factors control porosity, the most important being the particle size distribution of the backfill aggregate, the backfill water content and mechanical vibration. The hydration reaction of binders in cemented backfills results in the growth of cement crystals, which build water into their crystal structures, thus creating solids from a proportion of the water present. These cement crystals grow in backfill pores, chemically reducing porosity, which results in significant improvements of the regional support capabilities of cemented backfills.

* *Cemented backfill confined compression.* The confined compression test to stress levels of about 170 MPa reflects a cemented backfill's regional support capability. At that scale, the effect of binder addition to the material can barely be seen in the

initial rise of strength, but is very noticeable by the significantly reduced ultimate strain, or b-value, which, under ideal conditions, can be reduced by more than 25 %. For the regional support effort, the primary peak is insignificant in comparison to the potential benefits of the binder-induced reduction of porosity. By replacing 10 % of the tailings mass in a typical classified tailings backfill with portland cement, the backfill's porosity after full hydration is effectively reduced from 48 % to somewhere between 42,5 % and 45,3 %, depending on the compressibility of micro-pores in the gel.

* *Loading during curing.* Cemented backfill specimens were loaded at a constant strain rate during the curing process, to determine the effect on strength development. Compared to the unstressed control samples, 20 % to 30 % deterioration of the compressive strength and the elastic modulus was measured, which was directly attributable to the straining of the backfill. Samples, which were stressed with a static, constant load during curing (a creep test), showed no deterioration at all, provided the stress in the material never exceeded the yield stress. From the strain rates and the normal strength development of the control samples, it was concluded that cemented backfills can indeed be loaded during curing without adverse consequences, provided that the yield point of the material is at no time exceeded.

* *Closure rate.* Test results indicated, that the compressive strengths of cemented backfill cube samples are insensitive to strain rates between 1 % and 1000 %. In a subsequent test series it was shown that despite a 5400 fold increase over the previous highest tested closure rates, cured cemented backfill behaved essentially identical to control samples, compressed at standard laboratory closure rates.

* *Backfill specimen volume.* There are significant differences between rock or coal samples and samples of cemented backfill. Whereas rock or coal samples contain natural discontinuities and planes of weakness, either from their formation or

from subsequent seismic stressing, fine-grained cemented backfill samples contain only statistically distributed minor flaws and can, on the whole, be regarded as homogeneous. The lateral size increase of a constant height sample, such as was used in this study, can be interpreted as placing more basic units next to the original sample, thereby merely increasing lateral confinement; the centre of the plate fails in ordinary shear and not because of increasing numbers of material imperfections. In practice, therefore, the increased *volume* of a tabular backfill body is only of minor consequence to the strength of the backfill body. Cemented rockfills are expected to contain more substantial imperfections and their primary compressive strengths may have to be corrected according to Gonano (1975).

* *Specimen cross-section.* It was concluded that there is no significant difference between square and circular samples of otherwise identical dimensions.

* *Backfill sample width to height ratio.* Compression tests performed on constant height samples of varying width show that, essentially, all constant height samples show the same initial elastic behaviour. Initial sample failure (yield) is dependent on the bulk cohesion level and is not significantly influenced by any introduction of increasing numbers of randomly distributed, failure-triggering flaws into potential shear zones. In the post-failure region, i e after the collapse of the cohesive cement structure, the backfill samples effectively behaved as cohesionless granular materials. As the widths of the specimens increase, progressively more effective confinement is provided and the samples can build up stress at increasingly lower strain values. The samples of width/height ratios 5 and 7 approach the behaviour of confined compression, which, in effect, represents an infinitely wide sample.

For the tested cemented backfill, a width/height ratio of 1,5 delineates the geometry, above which the backfill body no longer shows a decline in the load bearing capacity with increasing strain. Single width rib samples showed marginally stiffer behaviour than unit cubes, an expected result because of the slight confinement provided in one dimension. The different compression behaviour observed in the 'post-failure' region are only of transient nature, eliminated as soon as a backfill specimen is compressed far enough to attain its confined compression width to height ratio of approximately 11, when the ultimate strain attained is porosity-, and not shape dependent.

6.4 Backfill Design for Defined Loading Response

In this chapter it was shown that each of the three regions in the cemented backfill loading curve has a specific competence in terms of stope support and can be varied according to stope support requirements, by integrating the material design of cemented backfills with the spatial configuration of backfill support elements. In this way, the large-scale yielding behaviour of backfill support can be altered in addition to the binder-induced stiffness at low strains, and the improved stiffness at high loading stresses.

* Ternary and cartesian type cemented backfill design graphs and empirical equations were described as aids for the design of the binder-induced primary compression region. They were derived from data established in the extensive investigation presented in Chapters 3 and 4.

* A very important addition to a cemented backfill's support capabilities can be obtained, by modifying the previously neglected post-failure region of the load-deformation curve. By spacing cemented backfill support elements at pre-determined distances of each other, their response to rapid, high stope closure can be controlled. Once the relatively soft dilating elements are firmly abutting, the large width to height ratio takes effect and the material builds up stress very rapidly. The collapse of the cemented backfill elements into the 'inserted porosity' under rapid stope closure can be seen as analogous to the yielding of a hydraulic prop under load by releasing hydraulic fluid. When, after some face advance, this safety configuration is no longer required, the inter-element spacing should be filled with backfill to improve the backfill's regional support capability.

* The high stress compression behaviour of backfills, determining the strain-hardening region of the curve is largely controlled by the minimum porosity of the backfill aggregate. This porosity can be significantly reduced by residual fragments of the broken down cement crystal structure, this reduction can at times lead to a lowering of the backfill's b-value by up to 25 %.

6.5 Further Research

The possibility of determining the deformation response of cemented backfill support over the whole compression range through appropriate composition design and support element geometry, calls for an evaluation of such a support system under mining conditions. Such an evaluation would initially be carried out through computer simulation, using advanced mining simulation models. The objective of the work would be to determine the interaction of such a cemented backfill support system with the converging rock mass in a simulated stope. One of the difficulties of this study would be the suitable modelling of the complex loading curve.

In the laboratory, further research work is indicated to establish the detailed behaviour of separated cemented backfill ribs under rockburst conditions. One of the more important aspects would be the study of the internal collapse behaviour of uncured cemented backfill ribs, in an effort to minimise the ejection of backfill into workings in the event of a rockburst. The more complex nature of the load-deformation behaviour of separated cemented backfill ribs should also be incorporated into numerical computer models for theoretical evaluations on their effect on stope and gully support.

APPENDIX A

LABORATORY PROCEDURES

The main experimental methods employed in the study will briefly be outlined in this appendix; it commences with a list of major laboratory equipment utilised and then summarises the most important working procedures.

Major Equipment Used

Besides a fully equipped laboratory, the following major equipment was used for the investigations:

* A 2 MN hydraulic press, custom-built at COMRO, with accessories, computerised displacement control and data capture (Figure A.1).

* A 25 MN hydraulic press, custom-built at COMRO, with accessories, computerised displacement control and data capture.

* A TerraTek rapid loading hydraulic press, computerised, fully instrumented, capable of loading velocities of up to 3 m/s (Figure A.2).

* Wykeham-Farrance triaxial test equipment with accessories and computerised data capture.

* A Fritsch Analysette 22 Laser particle size analyser.

Laboratory Procedures and Techniques

All solid materials, used in the experiments were dried, mixed, and then sampled to determine their respective material properties, including size distributions, placement properties, as well as mineralogical and chemical compositions, as appropriate.

Size distribution measurements. The size distributions of all solids used in the experiments were measured using a set of SI standard wire-mesh sieves of the $\sqrt{2}$ series. The standard sieves ranged from 0,038 mm to 75 mm in opening size. All aggregate materials were wet-screened to obtain reliable 0,038 mm, 0,053 mm and 0,075 mm size fractions; subsequently, the coarse residues were dried and sieved in vibrating sieve-stacks. The masses of all size fractions were measured to 0,1 g accuracy and recorded. Particle size distributions below 0,038 mm were measured in a Laser diffraction particle size analyser (Fritsch).

Specific surface area. The specific surface areas of backfills and binder samples were measured in the Fritsch Laser particle size analyser.

Permeability. The permeabilities of backfill samples were measured in a constant-head permeameter and the coefficients were determined according to the following relationship:

$$k = \left(\frac{QL}{thA}\right) \quad \text{(Eqn A.1)}$$

where:

Q = the total volume of water which drained through the sample.

L = sample length.

A = sample cross-sectional area.

t = time elapsed and

h = the total head lost across the sample.

Relative density measurements. The relative density of granular solids was measured with the aid of a pyknometer. In the laboratory, slurry relative densities were determined either by weighing a measured volume of slurry, or by drying a measured mass of slurry and determine the moisture loss. In the field, slurry relative density was measured either with a nuclear density gauge, of with a Marcy scale.

Chemical and mineralogical composition. Most chemical and mineralogical analyses were done in commercial laboratories, using X-ray fluorescence (XRF) or X-ray diffraction (XRD) techniques, as appropriate.

Placement properties. The placement properties (-porosities) were measured according to Clark (1985): a 250 g sample of loose, dry material is vibrated at a set intensity and time in a measuring cylinder and the attained porosity is recorded. The material is removed from the cylinder and thoroughly mixed with a measured amount of water, returned to the measuring cylinder, vibrated, and its porosity recorded. This procedure is repeated until the minimum porosity is reached near the material's saturation point.

Cemented backfill sample preparation and curing. Four different sample types were produced for the various compression tests:

a) Cylindrical samples, 42 mm ø, 84 mm high, for the comparative material testing and the Hoek cell triaxial tests.

b) Cylindrical samples, 38 mm ø, 76 mm high, for triaxial testing in the Wykeham-Farrance cell.

c) Rectangular samples of various lateral dimensions, all 50 mm high, for width/height ratio tests and rapid loading experiments.

d) Cylindrical samples, 120 mm ø, about 60 mm high, prepared in steel cylinders for the confined compression tests at high pressures.

The dry components of the backfill were screened through a 1,18 mm screen to remove any lumps. The materials were then weighed out to 1 g accuracy and thoroughly blended for at least three minutes. The appropriate weighed amount of (tap-) water was gradually added to the blend and mixed in for at least two minutes, until a uniform consistency was achieved. Small samples were mixed by hand, larger batches were processed in a laboratory batch mixer.

The fill sample was then placed into a split stainless steel mould (Figure A.3) and vibrated on a vibrating table to remove entrapped air from the slurry. The full mould was sealed into a heavy-gauge poly-ethylene bag and placed into a laboratory oven for the material to cure at 30 °C for a minimum of 24 hours. After that period, the backfill had usually hardened sufficiently, for the specimens to be trimmed to their correct length and to be removed from the mould. For the rest of their respective curing periods, they were cured separately, re-sealed in double poly-ethylene bags, to prevent moisture loss. This sealing method was effective; no appreciable mass loss was recorded by the end of the curing periods. The curing periods were selected following standard practice; they were: 1, 3, 7, 14, 28 and 90 days; for most tests, the curing temperature was 30 °C.

Compression testing. Six different types of compression tests were done on cemented backfill samples, each with their own specific methods and equipment layout:

a) Unconfined, uniaxial tests on cylindrical samples for the materials' compressive strengths comparisons.

b) Unconfined, uniaxial tests on the rectangular width/height ratio samples.

c) Confined, uniaxial compression tests to determine the high pressure behaviour of the backfills.

d) Triaxial tests to determine the materials' friction angles and cohesions using the Hoek cell.

e) Triaxial tests using the Wykeham-Farrance equipment.

f) Unconfined, uniaxial tests of rib samples to determine rapid loading characteristics.

a) The unconfined, uniaxial compression tests of cylindrical samples were done in a 2 MN hydraulic press, using a 45 kN high precision load cell (Revere CSP 1-10-A); this ensured sufficient accuracy at low sample strengths (max ± 0,05 % non-linearity). The load cell was fitted with a spherical seat to facilitate uniform sample loading.

b) For the small rectangular width/height ratio samples a set-up identical to the one above was used; the larger samples were loaded on a 210 mm ø spherical seat, utilising the press's own 2 MN load cell.

c) The high pressure confined compression tests were done using floating carbon steel confining cylinders and drained pistons (Figure A.4). The samples were compressed in the 2 MN press, loading the backfill up to 175 MPa; some samples were loaded up to 440 MPa in the 25 MN press.

d) The triaxial compression response of a series of cylindrical samples was measured with a Hoek cell using the 45 kN load cell in the 2 MN press. The confining pressure was controlled with a hand pump.

e) Most triaxial tests were done using conventional Wykeham-Farrance equipment, operated according to standard soil testing procedures.

f) The rapid loading tests of backfill rib samples were conducted in the TerraTek rapid loading press at a closure rate of 0,15 m/s.

All unconfined, uniaxial and triaxial samples were loaded to beyond their primary maximum compressive strength, uniaxial confined samples were loaded to machine capacities.

Displacement rates. The displacement rate for the unconfined tests was kept constant at 100 mm/h. Displacement rates in triaxial tests were as low as 3 mm/h to allow for pore water pressure dissipation. The displacement rate in the TerraTek press was 0,15 m/s.

Measurement records and processing. The load/displacement curves of all tests using the hydraulic presses were recorded on paper with an X-Y recorder and subsequently converted to stress-strain curves by re-scaling the axes. The triaxial test measurements on the Wykeham-Farrance rig were captured into a personal computer.

From the stress-strain curves, the following values were extracted: the maximum compressive stress, the yield stress, the strains at these stresses and the tangent modulus of elasticity. The triaxial test results were processed according to standard soil testing procedures.

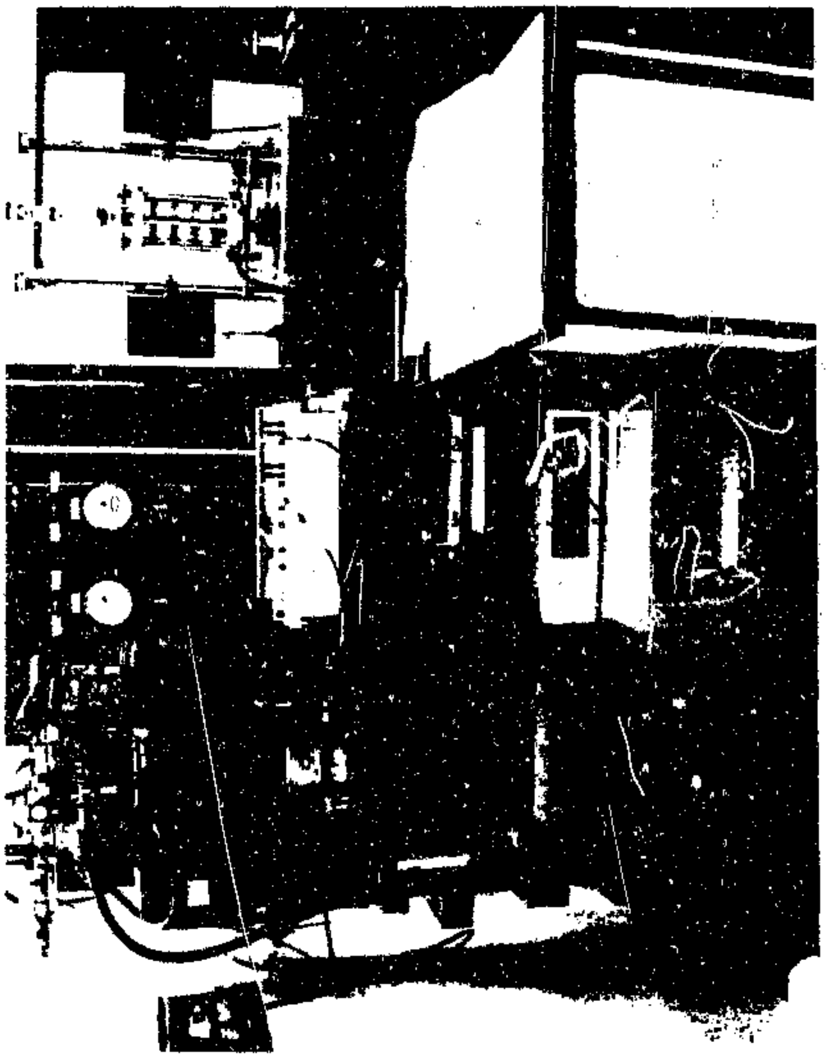


Figure A.1 COMRO 2 MN hydraulic testing press.

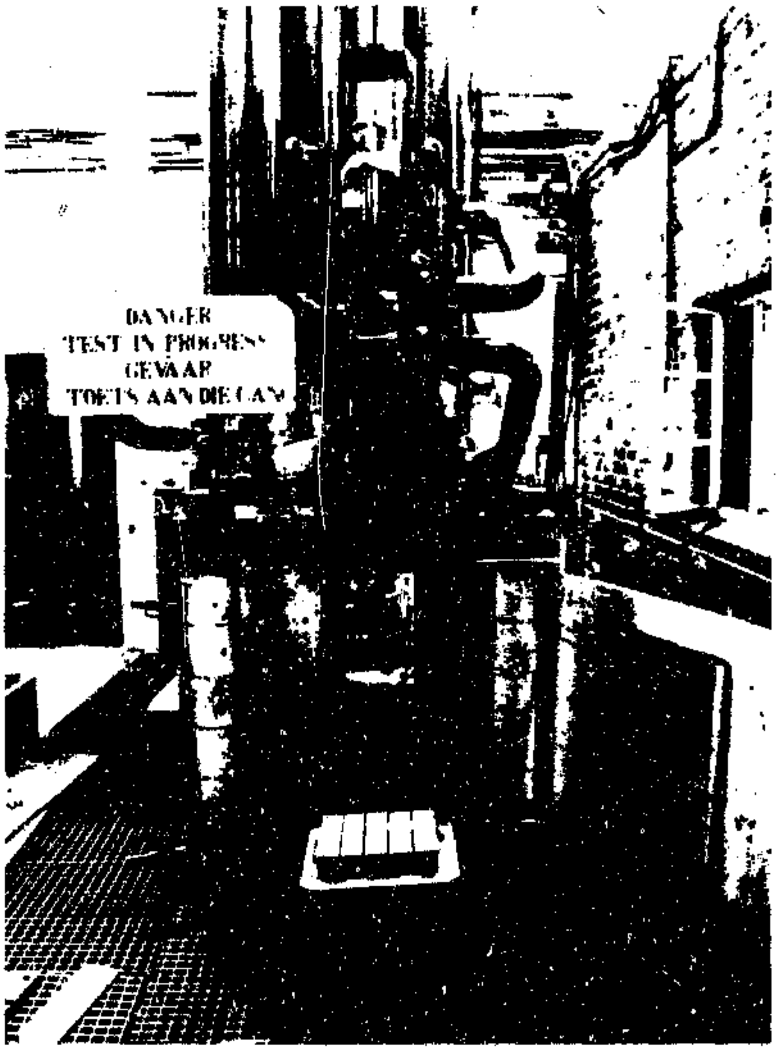


Figure A.2 TerraTek rapid loading hydraulic testing press.

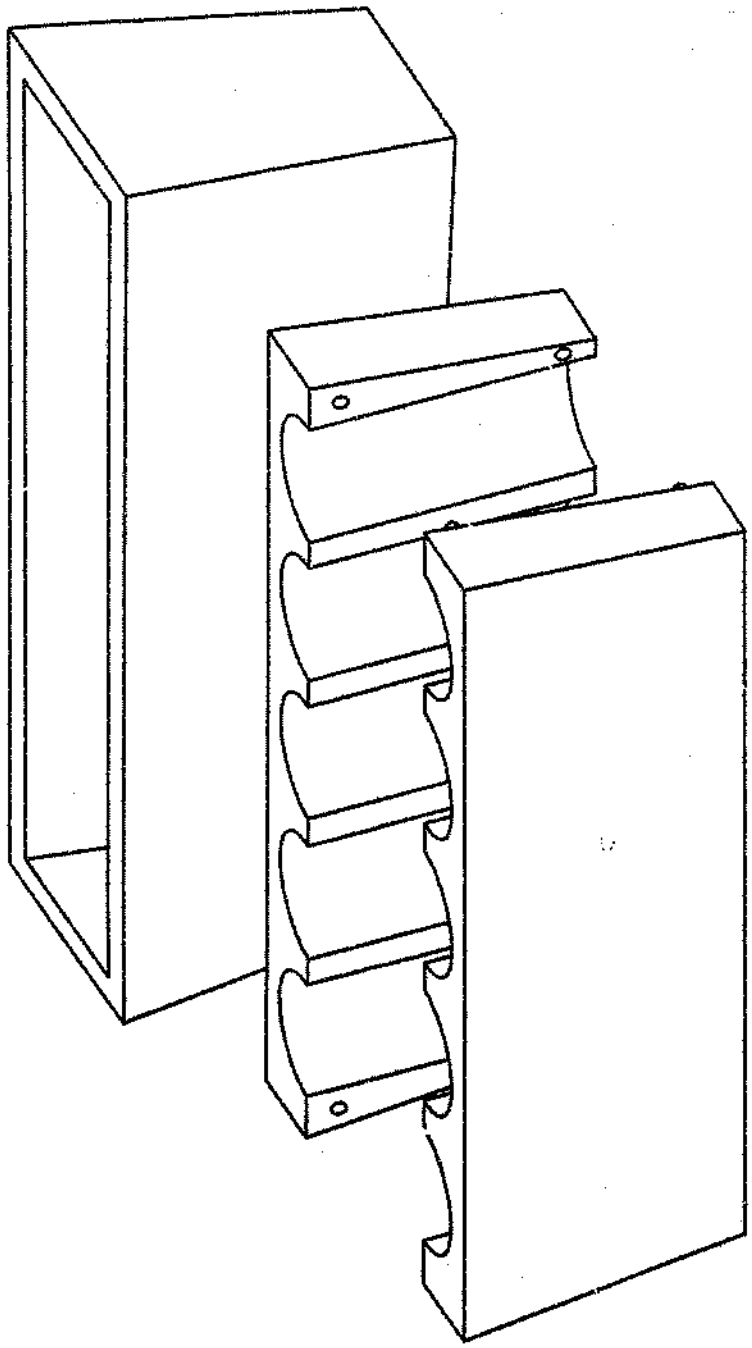


Figure A.3 Stainless steel cemented backfill sample mould.

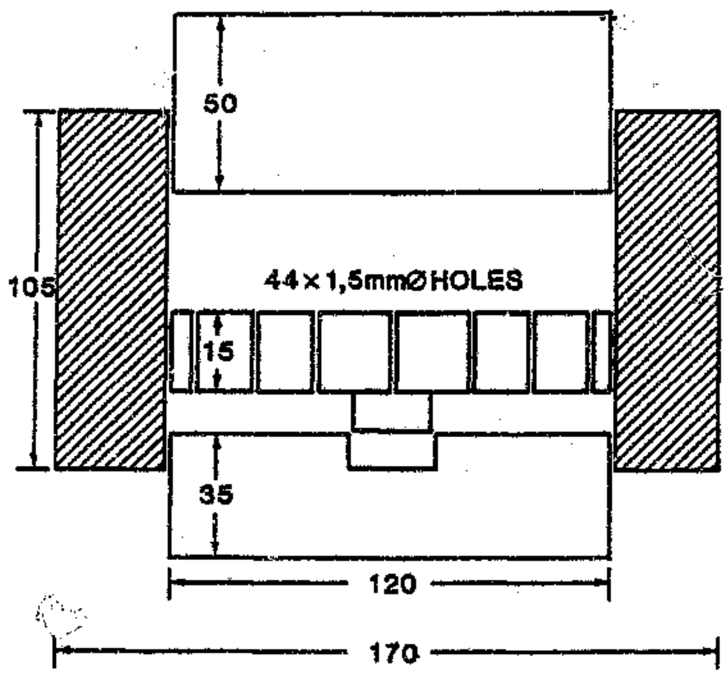


Figure A.4 Carbon steel confined compression oedometer with floating jacket and drained pistons.

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