

SIMRAC

Final Project Report

Title: MINING HIGHLY STRESSED AREAS

Part 2: Appendices (1 - 7)

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Research

Agency: CSIR: Division of Mining Technology

Project No: GAP 033

Date: December 1995

APPENDICES

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Appendix 1.
QUESTIONNAIRE - RESULTS SUMMARY

**SUMMARY AND ASSESSMENT OF QUESTIONNAIRE ON MINING AT
ULTRA-DEPTH AND IN HIGHLY STRESSED AREAS**

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Internal Note

CSIR Miningtek RE 5/94

February 1994

INTRODUCTION

A questionnaire related to mining at great depth and in very high stress conditions has been completed with the assistance of mine rock mechanics personnel on over twenty mines in all mining districts, and covering all deep level mines. This work forms part of the SIMRAC GAP033 project.

The objective of the questionnaire was to identify and where possible, quantify problems associated with deep level and high stress mining taking place at present as well as anticipated changes in the future. Results obtained will have a bearing on current research and may also motivate further research to be carried out under this project.

The information collected includes:

- Levels, locations and causes of seismicity.
- Faults, dykes & their tendency to be seismically active.
- Reef, hanging and footwall geology - especially any causing problems.
- Mining method, local and regional support details.
- Details of reefs mined - dip, depths, stope width & expected change in future.
- Mine layouts that cause problems.
- Any difficulties with off-reef excavations - haulages, chambers, rock passes...
- Types of high stress mining. Deep, ultra deep, remnant, shaft pillar, ...
- Approximate current and future production from such areas.

Results are summarised in this assessment for the entire industry and then for the different mining districts, but not for individual mines as information from the returns is confidential. Principal findings are summarised towards the end of the document. Production figures are based on 1992 data.

OVERALL RESULTS SUMMARY

1. Remnants

Perhaps the most surprising result is the high production percentage coming from remnants, isolated blocks of ground, high stress areas (excluding deep level mining) and difficult mining conditions. In total 26% of South African gold mines production or around 27 000 000 tons per annum are from such sources.

The breakdown by region is as follows:	Klerksdorp	44%	(KL)
	Far West Rand	20%	(FWR)
	Central Rand	20%	(CR)
	Orange Free State	25%	(OFS)
	Evander	25%	(EV)

Also somewhat surprising are the relatively few problems associated with mining such ground. Support and mining standards appeared adequate. Problems reported include:

Remnants adjacent to faults or dykes, and particularly at fault/dyke intersections (KL & OFS).

Access difficulties - need to re-raise or re-develop (KL, EV & OFS).

High remnant face stress problems (CR, FWR & KL).

Remnants bursting, sometimes repeatedly (KL & FWR).

Bursting when holing a remnant (KL & FWR).

Many mines have seismic data related to remnants which are available for further analysis.

In addition nine shaft reef pillars are currently being mined, with associated concentrations of stress, and with the need to protect many important excavations. Early extraction of the reef area in a new shaft is being done increasingly, with four such areas being mined at present.

2. Ultra-deep mining

Ultra-deep level mining is increasing, principally taking place in the Far West Rand and Central Rand areas. In total 16% of South African gold mines production or around 16 500 000 tons per annum are from such sources.

The breakdown by region is as follows:	Far West Rand	50%
	Central Rand	16%
	Klerksdorp	2%

Future ultra-deep mining will be concentrated in the Far West Rand and Central Rand with limited contributions from Klerksdorp and the Orange Free State. Mining depths on a number of current mines are or will increase to ultra-depth. In addition several new or planned mines will operate at such depths.

Associated with ultra-deep mining are a number of severe problems:

How to design suitable regional support? Pillars and/or backfill (FWR, CR, KL).

How to mine in the presence of seismically hazardous geology (FWR, CR, KL).

Small faults and dykes - mine or bracket? (FWR).

Falls of ground sympathetic to nearby seismicity.

Pillar foundation failure problems (FWR & others?).

Protection of off-reef excavations in relation to geology, pillars,... (FWR, CR & KL).

Need for design criteria applicable for such depths (FWR, CR).

3. Seismicity, faults & dykes

A high degree of geological control in levels of seismicity is indicated. In all areas, with the exception of the Far West Rand, almost all seismicity can be associated with geological features, mainly faults and dykes.

Fault/dyke intersections are particularly hazardous - large events (KL & OFS).

Seismicity associated with some faults and dykes (OFS, KL, CR & FWR).

Why are some geological structures hazardous and others safe?

Need better layout design in vicinity of geological hazards (OFS, KL, CR & FWR).

Pillars or adjacent rock failing gives rise to significant seismicity on some mines (FWR).

Highly stressed remnants cause considerable seismicity on some mines (FWR, CR).

Reduced seismicity reported on one mine corresponding to reduced rockmass strength (CR).

4. Off-reef excavations

Great difficulty is experienced with support and stability off-reef development in the Klerksdorp area, even at moderate depths. Despite a high initial level of support, many tunnels and cross-cuts need to be re-supported or rehabilitated following seismic damage. These problems will become more severe in the new, deeper level mining in this area. In other regions problems with off-reef structures are principally related to geology, poor ground condition, high stress levels (pillars & abutments) and seismic damage. Support of haulages traversing faults and dykes is very difficult at great depth.

5. Geology

The influence of geology on mining conditions at depth, both favourable and unfavourable are listed below.

Very weak reef horizon correlates with few stress problems encountered on reef? (KL).

Seismic damage moderated by near reef soft rock (OFS, FWR, EV).

Greenbar in hanging can cause breakout problems if exposed (FWR).

Difficulty in carrying and supporting shale hangingwall (OFS).

6. General

Several mines (>4) are or will be mining some areas of wide reef orebody at depth - little is known about such mining.

KLERKSDORP

Seismicity

Much seismicity - many events, with larger event magnitude 2 - 5.
All mines use regional ISS network, with some additional local coverage.
Seismic data relating to remnants are available on all mines.
Much seismicity associated with faults and dykes.
High stress on a geological feature often gives a seismic problem.
Holing the top of a X-mas tree remnant frequently gives rise to a large event.
Not much movement observed on structures.
Seismic damage mainly recorded off-reef.

Faults and dykes

Many smaller faults with throws of 10 - 30 m.
Few large faults having throws of hundreds of metres.
Much seismicity associated with faults and dykes.
Many dykes, though not all are dangerous.
Hard, strong dykes most dangerous.
Some dykes known as hazardous.
Difficulty in mining fault/dyke intersections, "lock up" for big event?

Reefs & mining method

1. Vaal reef

Stope width: 0.7 m - 2.5 m, average 1.3 m.
Dip: Generally 10° - 25°, local exceptions 0° - 60°
Depth range - current: 1200 m - 2900 m
Depth range - future: 1200 m - 3000 m (5000 m for Moab far in future!)
Mining method:
 Scattered mining.
Local support:
 Much use of 20/40 RYHP, sometimes with headboards.
 20 - 30 % use 75 x 75 mat packs in stopes, larger packs on gullies.
 Profile props in rapid closure area as RYHP get locked in.
 Sometimes mesh and lace strike gullies.
 Caving used on one mine.
Regional support:
 Geological features, remnants and fault losses.

Backfill in "problem" areas, though sceptical of its benefit.

Caving - reported local & regional benefit.

Packs (Durapack better in a rockburst?)

Deeper (2500 - 3000 m) some use of 25 m barrier pillars, 150 m apart.

Bracket pillars used to clamp problematic faults.

Geology:

Hangingwall generally 200 MPa, though weak in some patches, can be very friable.

Footwall weaker, 160 - 180 MPa.

Reef generally very weak - may be favourable in dissipating higher face stresses.

Well defined bedding planes often with gouge.

2. VCR

Stope width: 1.5 m - 3.0 m.

Dip: Generally 10° - 15°

Depth range - current & future: < 1000 m

Local support:

Much use of grouted twist bar.

High stope width areas - room and pillar mining.

Regional support:

Squat pillars - occasionally get massive areas collapsing.

3. 'C' reef

Depth range - current: 900 m - 2000 m

Local support:

Similar to Vaal reef - use elongates.

Geology:

Reef geology difficult - weak 150 MPa footwall.

Strong 250 MPa hangingwall matrix of large strong pebbles gives support difficulties.

High stress

Deep orebody:

Highly stressed Strathmore area of Buffelsfontein. (3000 m)

Moab or Eastvaal - deep, highly faulted orebody, currently sinking the shaft. (>3000 m)

Shaft pillars:

2 shaft pillars being mined at present.

2 shaft pillar extractions planned for the near future.

Remnants:

Some remnants burst for no apparent reason?

Only deeper remnants give problems in extraction.

Some remnants burst repeatedly during extraction.

Some evidence for remnants becoming easier to mine with age.

Access to remnants becomes difficult due to high closures, may need to re-raise & ledge.

Many remnant areas abandoned because of stability/seismic problems.

Weak reef & footwall makes remnant mining easier.

Biggest problem is access and development of a remnant.

Many remnants left on fault/dyke intersections - difficult to know how to mine them.

Seismic history, experience & modelling assist in planning remnant extraction.

Some remnants left because of fires.

Seismicity on holing X-mas tree remnant may cause mining to stop.

Isolated blocks in multiple fault sets, reef dragged down, can be very highly stressed.

Off-reef excavations:

Many problems with off-reef structures.

Support with bolting, mesh, lacing, long anchors, wire rope and even gunnite.

Much tunnel rehabilitation required for old and seismically damaged tunnels.

"Anything off reef is a problem"!

Damage relates to high stress, proximity to remnants and geology.

Geology plays an important role.

Weak footwall, high horizontal stress. Much plastic deformation.

Need to re-support recurs despite good support.

Big problem: haulages crossing dykes and faults. Increase support at intersection - yielding, long cables etc.

High stress > 120 MPa - haulages under abutments or through dykes - severe damage to haulages.

Bad tunnel problems - ledge out sidewall & insert packs.

Haulages deep 70 - 100 m in the footwall.

Large chambers & cross cuts difficult to support.

Box holes and rock passes spall badly.

Production figures

Klerksdorp area mines around 19 000 000 tons of ore per annum, or 18 % of SA production.

Around 44 % of this is from remnants, isolated blocks of ground or difficult areas.

Amount of remnant mining will increase slightly in the future.

Klerksdorp problem areas

1. Geologically complex - much seismicity associated with faults, dykes and remnants.
2. Seismic damage mainly off-reef.
3. Patches of very weak, friable hangingwall.
4. Generally very weak footwall.
5. Some remnants burst repeatedly.
6. Access difficulties associated with remnants.
7. Remnants on faults and dykes - how to mine?
8. Stability of all off-reef excavations is a problem, especially through dykes.

FAR WEST RAND

Seismicity

Most mines report high levels of seismicity, including large events of magnitude 2 - 5.

Some consider event signature more important than magnitude only.

ISS, PSS and Goldfields systems used.

Pillar associated events reported on several mines.

"Slip" and "crush" events reported.

The majority of mines report seismicity associated with geology: faults and dykes.

One mine found seismicity on longwall remnants exceeded that on pillars or geology.

Remnant data available for analysis on several mines.

There is evidence of problem features on several mines, on which repeated large events occur.

Strong evidence for geological control on seismicity.

Faults and dykes

Many dip faults 1 - 25 m throw, mined through with many problems.

Fault seismicity on a number of mines, though not all.

Some large dykes, used for regional support.

Many dykes - seismic.

Weak dykes on one mine cause problems.

Dyke/faults - many with seismic problems.

Reefs & mining method

1. VCR

Stope width: minimum 1.0 m, average 1.5 m, exceptional maxima - 3.5, 6.0 & 13.0 m.

Dip: Generally 12° - 30°, local exceptions 5° - 50°

Depth range - current: 700 m - 3100 m (most mines 2000 - 3300 m deep)

Depth range - future: 700 m - 4000 m

Mining method - very varied due to grades and stoping widths:

Mostly longwalls with stabilizing pillars.

Scattered mining.

100% extraction longwalls with stiff fill.

Sequential grid scattered with pillars and backfill.

Room & pillar.

Local support:

20/40 RYHP & water jet cleaning.

Backfill, silicated in high stope width areas.

Elongates.

Packs (solid, grout, Hercules, composite).

Regional support:

Strike stabilizing pillars, 40, 45, 50, 60 m wide.

Some dip stabilizing pillars.

Backfill.

Unpay remnants from scattered mining.

30 m dip stabilizing pillars with backfill.

Geology:

Hangingwall lava - generally competent, tuffaceous and weak (100 MPa) in first 5 m of hangingwall.

Weak jointing & infilling in lava.

Lava flow planes in hangingwall may be weak, with low cohesion.

Shale in footwall weak and may moderate seismic damage.

2. Carbon Leader

Stope width: 0.85 m - 2.5 m, average 1.2 m.

Dip: Generally 16° - 28°

Depth range - current: 1400 - 3600 m, most around 2700 m.

Depth range - future: - 4000 m, most around 3000 m.

Mining method - very varied due to grades and stoping widths:

Mostly longwalls with stabilizing pillars.

A little scattered mining.

Local support:

Backfill with RYHP, elongates and packs.

RYHP & pipe sticks.

2/3 rows RYHP sometimes with headboards.

Use of waterjet cleaning is increasing.

Some gullies use tendon support.

Solid timber or Hercules packs on gullies.

Mechanical props on face for temporary support - removed to clean.

Regional support:

Strike stabilizing pillars, 40 m wide, 85 % extraction.

Backfill (classified tailings).

Geology:

Competent hangingwall & footwall quartzite.

Green bar close to reef in hangingwall, can give breakout problems if exposed.

Some regular jointing.

3. Deelkraal:

Mined using mini-longwalls, local support mainly using packs, & some RYHP.

4. Kloof:

Underlies VCR, 0 to 60 m middling.

Some stress problems undermining VCR stabilizing pillars.

Generally de-stressed competent rock - no other problems.

5. Wide reef:

Either several reefs mined as a package or wide reef VCR.

Mined either as double cut, or multiple cut using RYHP, tendons and backfill.

Wider reefs using conventional cut and fill.

Large pillars on geology or backfill provide regional support.

High stress

Deep orebody:

How to design regional support? Pillars and/or fill?

Small faults & dykes - mine through, bracket pillar ? - a big problem!

Foundation failure of stabilizing pillars - sensitive to length of pillar and rate of advance?

In general there is a need for models with suitable design criteria that are easy to apply.

Shaft pillars:

4 shaft pillars are currently being mined, with many planned for the future.

This includes some early mining of shaft reef areas.

Remnants:

Approaching longwalls prior to holing , longwall remnants - high ERR , stress and seismicity.

Long thin longwall remnants give problems on several mines.

In general current support and standards for remnant mining suffice.

Principles needed to form longwall remnants suitable for safe extraction.

Longwall remnants reach a stage that they can be safely mined. (Fractured through?)

Off-reef excavations:

Many support problems are site specific.

Major problem with rock-pass stability , bad scaling on 3 mines.

Some mines considering lining rock-passes at depth.

Problems with stability of ventilation cross-cuts through stabilizing pillars.

Cross-cuts difficult if a panel lags badly.

In some areas, haulages and cross-cuts need to be very heavily supported.

Haulages in bad ground - increase support density & use yielding elements.

Haulages under pillars or abutments often supported with cable anchors, or yielding support.

Areas of weak lava as low as 50 MPa - difficult to support excavations.

Production figures

Far West Rand mines surveyed produce around 23 530 000 tons per annum.

Far West Rand mines produce around 27 200 000 tons of ore per annum, or 25% of SA production.

Around 20 % of this is from remnants, isolated blocks of ground or difficult areas at present.

Around 50 % of production is currently from deep level mining.

Amount of deep level mining will increase significantly in the future, from both existing and new mines.

Far West Rand problem areas

1. Seismicity - slip, crush and pillar system failure events.
2. Longwall remnants seismically active.
3. Strong evidence of geological control on seismicity.
4. Many faults - mine through or bracket?
5. Tuff & greenbar in immediate hangingwall gives breakout/FOG problem.
6. Need for better regional support design.
7. Pillar foundation failure.
8. Scaling & spalling of rock passes.
9. Some problems with haulages and cross-cuts associated with bad ground or poor layout.

CENTRAL RAND

Seismicity

Yes, 5 - 7 damaging events per month.

PSS monitoring.

Most seismicity associated with faults and dykes.

Record reduced damaging seismicity as hangingwall & footwall rockmass gets weaker.

Seismic records available for analysis.

Can only identify one incident of foundation failure with certainty.

Anticipate increased seismicity with depth.

Faults and dykes

Dykes - some seismically active.

Many faults - some seismicity associated with faults.

Faults difficult to negotiate.

Reefs & mining method

South Reef (Sometimes composite with Main & Main Reef Leader).

Stope width: 1.1 m fairly uniform.

Dip: Generally 10° - 22°

Depth range - current: 0 m - 3460 m

Depth range - future: 0 m - >4000 m (by year 2000.)

Mining method:

Scattered mining up to 2500 m if grades not consistent enough for longwall.

Longwall at 80 % extraction, maintain ERRs low.

Local support:

3 rows 40 ton hydraulic props.

Backfill being considered for the new areas.

Regional support:

Stabilizing pillars 50 or 60 m wide, at 80 % extraction.

Try to incorporate geology in pillars to reduce gold loss.

Geology

Footwall weaker than the hangingwall - causes no problems.

High stress

Deep orebody:

Plenty of ore-reserves at reasonable grade, BUT at depths of 4000 m and deeper.

Shaft pillars:

1 shaft pillars being mined at present.

Shaft reef area pre-extracted - 2 extracted with third being planned.

Remnants:

Few problems experienced in mining remnants, occasional events cause damage.

Remnants are mined with standard rockburst precautions.

Drilling difficulties are experienced at high stress.

Off-reef excavations:

At depth most haulages are overstoped.

Some connecting haulages not overstoped - ongoing, severe support and stability problems.

Attempted to pre-condition a dangerous dyke. Seismicity started some distance away, when haulage approached. High stress made it difficult to drill.

Production figures

Central mines surveyed produce around 11 450 000 tons per annum.

Central Rand mines around 14 400 000 tons per annum or 13.5% of SA production.

Around 20% of this is from remnants, isolated blocks of ground or difficult areas at present.

This amount will increase to around 30% in the future.

Around 2 400 000 tons per annum produced at depths between 2500 & 3500 m at present.

Both this amount and the mining depths will increase in future.

Central Rand problem areas

1. Damaging seismicity associated with faults & dykes.
2. Reduced seismic damage noted as footwall rockmass gets weaker.
3. Anticipate higher seismicity as depths increase.
4. Faults difficult to negotiate.
5. Depth of reserves.
6. Connecting haulages - stability & support difficulty especially near geology.

ORANGE FREE STATE

Seismicity

Plenty of events ranging from small to large reported on all mines visited.

Several large events per month.

PSS, ISS and Gentel seismic systems in use.

Most seismicity (95%) associated with geology: faults, dykes and sills.

Many FOG problems associated with seismicity.

Fault/dyke intersections particularly hazardous.

Several mines report low ESS values from modelling and little evidence of shearing on structures.

One mine reported shearing of intact rock.

Several mines have data available for remnant studies.

Faults and dykes

Geologically very disturbed area.

Many faults with throws from a few metres to over 1000 m.

Some large fault losses.

Bedding plane faults or sills present on some mines.

Faults different to other regions - generally softer.

Faults responsible for some seismicity.

Many dykes - responsible for most seismicity!

Some mines mine to dyke until $\sigma_v \geq 0.75$ UCS using MINSIM, then stop leaving bracket pillar.

Remnants along dykes give major problems!

Plenty of problems with fault/dyke intersections.

Hard dykes intersecting faults at high angle tend to "lock up" stress with large event potential.

Seismic history may help to orientate the face more favourably in latter cases.

Horizontal bedding faults undulate, so shear movement can cause unstable "domes" and large FOG.

Reefs & mining method

1. Basal reef

Stope width: 1.1 m - 2.0 m, average 1.5 m.

Dip: Varies mine to mine - 10° - 12°, 30° - 35°, 0° - 60°, 0° - 90°

Depth range - current: 600 m - 2700 m

Depth range - future: 1400 m - 3500 m

Mining method:

Mini-longwalls on breast, under/overhand.

Scattered mining, 20 m or 30 m panels.

Problems with steep layouts.

No problem with high face advance up to 18 m per month, because of shale hangingwall?

(Does the shale help or does the quick mining mean that the shale is not allowed to fail and cause problems?)

Local support:

Wide range of local support is used including:

Extensive use of packs - different sizes, compositions and spacing.

In bad ground, very dense pack spacing is used - still can get fallout between units.

Small areas of backfill with binder.

Some ineffective backfill placed far from the face.

1 - 3 rows 400 kN hydraulic props + Camlocs for temporary face support.

Pencil sticks, pipe sticks.

Pre-stressed elongates.

3 rows of 20/40 RYHP used in seismically hazardous areas.

Regional support:

Heavy pack support on gullies.

Mostly geology, dykes and fault areas.

Remnants, many unpay areas - mostly unplanned.

Geology:

Quite a lot of local variation in rock conditions.

Khaki shale + dominant bedding, almost horizontal faults.

Reasonable hangingwall - some shale, some weathered quartzite.

Some areas have no shale.

Shale as weak as 60 - 70 MPa in hangingwall varies in thickness from 0 to 25 m.

Some mines try to undercut the shale, and carry a thin quartzite hangingwall, mined with RYHP.

Some mines extract the shale in preference to carrying it as a hanging.

Soft shale layer may be beneficial in reducing stress and allowing aseismic release of energy.

Some areas have very strong footwall, 240 - 260 MPa.

Problems associated with hard horizontal sill?

2. 'A' reef

Stope width: 1.5 m - 4.0 m, average 1.8 m.

Dip: Generally 5° - 15°

Depth range - current : 1300 - 1500 m

Depth Range - future: 1300 - 3300 m . (+/- 120 m above Basal)

Local support:

Packs only or packs and tendons depending on stope width.

Geology:

Competent 2 m thick hangingwall quartzite.

3. Leader reef

Stope width: 1.0 m - 3.0 m, average 1.5 m.

Dip: Generally 8° - 16°

Depth range - current & future: 430 m - 1500 m. (+/- 20 m above Basal)

Mining method:

Not extensively mined - nibble at high grade.

Mini-longwalls .

Depending on middling, A, B, Leader mined as a package.

Local support:

Packs, pipe sticks, Camlocs, 400 kN hydraulic props.

High stress**Deep orebody:**

Only Target could be ultra-depth in the long term.

Shaft pillars:

2 shaft pillars being mined at present.

3 shaft pillar extractions planned for the near future.

Remnants:

Some remnants burst repeatedly during extraction.

Off-reef excavations:

No problem with haulages - use standard support based on modelled stress levels.

Production figures

Mines surveyed produce around 17 382 000 tons per annum.

OFS area mines around 38 000 000 tons of ore per annum, or 35 % of SA production.

Around 25 % of this is from remnants, isolated blocks of ground or difficult areas at present.

Orange Free State problem areas

1. Seismicity associated with sills, faults, dykes and fault/dyke intersections.
2. Geologically very disturbed.
3. Bracket pillar design?
4. Remnants on dykes are dangerous - severe bursting.
5. Seismically induced FOG.
6. Shale/weak hangingwall gives FOG problem.

EVANDER

Seismicity

Very limited seismicity. Seismicity monitored using one Gentel station.

Seismicity increasing slightly with depth.

Seismicity associated with only Kinross fault.

Long ago some seismicity attributed to one dyke

Faults and dykes

Plenty of faulting having throws from 2 m to 240 m.

Reefs & mining method

Kimberly reef

Stope width: 0.9 m - 2.2 m.

Dip: 12° - 38°

Depth range - current: 800 m - 2200 m

Depth range - future: 800 m - 2500 m

Mining method:

Scattered mining

Local support:

Mat packs & 0.9 m long end anchored tendons.

Geology

Some problematic hangingwall areas of strong cross-bedding and interlacing of argillaceous quartzite and shale with little or no cohesion.

Some regular joint sets.

High stress

Shaft pillar::

A shaft pillar to be mined in the future.

Remnants:

20% production comes from remnants.

No stress problems related to remnant mining.

Remnant support based on modelled stress & in situ conditions which proves adequate.

Remnants become more crushed with age, and access becomes a problem.

Production figures

Evander mines surveyed produce around 2 310 000 tons per annum.

Evander area mines around 4 300 000 tons per annum or 4.1% of SA production.

Around 25% of this is from remnants, isolated blocks of ground or difficult areas at present.

Amount of remnant mining will come down in future.

Evander problem areas

1. Access problems in some remnants.
2. Areas of strong cross-bedding in the hangingwall with very low cohesion.

PRINCIPAL FINDINGS

1. High percentage production in all mining regions from remnants, isolated blocks & difficult to mine areas.
Problems are: bursting - related to high stress, rock type or geological features in the vicinity.
access - difficulty in getting and maintaining access to remnants.
layout - proximity to geological features, other remnants, crosscuts and haulages.
2. There is a marked trend of increasing mining depths, with associated problems across all mining districts.
3. A considerable, and increasing percentage production on many mines in the Far West Rand, and some in the Central Rand regions is taking place at deep & ultra-deep levels,
Problems are: what regional support?
mining near faults and dykes - layout, bracket or mine through?
protection of connecting haulages and advanced development.
high stoping widths - how to mine and support?
4. Seismicity associated with pillar system failure, geological features and high stress.
5. High percentage production from scattered mining.
Problems are: taking place at greater depths.
use of pillars?
protection of advanced development.
how to mine safely adjacent to many faults and dykes?

TABLE OF MINES INCLUDED

Mine	Group	District	People
Blyvooruitzicht G.M.	Rand Mines	West Rand	M. Cross, P. Brenchley
Buffelsfontein G.M.	Genmin	Klerksdorp	P. van der Heever
Deelkraal G.M.	Goldfields	West Rand	D. McGregor
Driefontein Consolidated	Goldfields	West Rand	B. van der Kevie
Elandsrand G.M.	AAC	West Rand	A. Lagerweij, A. Banning, D. Olivier
ERPM	Rand Mines	Central	M. Spengler
General	Genmin	OFS	M. Pretorius
General	AAC	OFS	Dr G. van Aswegen
Harmony G.M.	Rand Mines	OFS	R. Tucker
Hartebeestfontein G.M.	Anglovaal	Klerksdorp	D. Arnold, J. Blignaut
Kinross G.M.	Genmin	Evander	J. Keen, S. Steyn
Kloof G.M.	Goldfields	West Rand	A. Haile
President Steyn G.M.	AAC	OFS	D. Minney
Randfontein Estates	JCI	Central	M. Keen
Saaiplaas G.M.	AAC	OFS	G. Brinch
South Deeps G.M.	JCI	West Rand	J. James
St. Helena G.M.	Genmin	OFS	G. Lambrecht, W. Tucker
Vaal Reefs G.M.	AAC	Klerksdorp	D. Booyens, J. Laas
Western Areas G.M.	JCI	West Rand	J. James
Western Deep Levels	AAC	West Rand	Dr T. Hagan, N. Henderson

TABLES SUMMARISING MAJOR FINDINGS BY AREA

TABLE OF REEF INFORMATION

	% production from remnants & isolated ...	% production less > 3000 m & problems	Depth range present	Depth range future	Dip	Stope width range & average	Principal mining method	Reefs mined
Klerksdorp	44%	2%	0 - 2900 m	0 - 3000 m 5000 long term	10° - 35° exceptions 0°- 60°	0.7 - 3.0 m average 1.3 m	Scattered	VAAL VCR 'C' reef
Far West Rand	20%	50%	700 - 3600 m	700 - >4000 m	12° - 30° exceptions - 50°	1.0 - 3.0 m average 1.4 3 - 13 m exceptional	Mostly longwall some scattered	VCR Carbon Leader Deelkraal Kloof Wide reef
Central Rand	20%	16%	0 - 3500 m	0 - >4000 m	10° - 22°	1.1 m except where composite	Mostly scattered some longwall & pillars	South Main Reef & Main reef leader
Orange Free State	25%	-	600 - 2700 m	1400 - 3500 m	Variable by area 0° - 90°	1.1 - 4.0 m average 1.6 m	Scattered mini-longwall pillars	BASAL 'A' reef & Leader
Evander	25%	-	800 - 2200 m	800 - 2500 m	12° - 38°	0.9 - 2.2m	Scattered	Kimberly

TABLE OF REMNANT INFORMATION

	% production from remnants & isolated ...	Remnants, isolated blocks, difficult to mine ...									
		Bursting	Access problems	Faults & dykes	Longwall seismicity	Face stress	Seismic data avail.	# pillars present	# pillars future	Easier with time	
Klerksdorp	44% increasing	X	X NB	X		X	X	2	2	X	
Far West Rand	20% constant	X			X	X	X	4	many	X	
Central Rand	20% increasing					X		1	some		
Orange Free State	25%		X	X			X	2	3		
Evander	25% increasing		X						1	X	

TABLE OF SUPPORT INFORMATION

	Mining method			Regional support					Stope support					
	Scattered	Longwall	Room & pillar	Backfill	Stab. pillars	Geology	Remnant	Bracket pillar	Caving	Bolts	RYHP	Elongates	Backfill	Packs
Klerksdorp	X		X	X	X	X	X	X	X		X	X	X	X
Far West Rand	X	X	X	X	X	X	X	X		X	X	X	X	X
Central Rand	X	X	X	X	X	X	X			X			X	
Orange Free State	X	X	X			X	X	X			X	X	X	X
Evander	X					X	X			X				X

TABLE OF SEISMIC & OTHER INFORMATION

	Seismicity					Off-reef problems			Geology & other problems		
	Magnitude	Assoc. faults	Assoc. dykes	Assoc. pillars	Assoc. remnant stress	Assoc. stress	Rock passes	Large chamber	Tunnels & X-cuts	Geology	General
Klerksdorp	5	X	X		X	X	X	X	XXX	Weak footwall Fault/dyke intersect	High ↔ stress Plastic deformation Tunnel rehabilitation
Far West Rand	5	X	X	X	X	X	X		X	Greenbar breakout Weak Tuff in h/w	Haulage layout wrt pillars & geology Small faults/dykes?
Central Rand	4	X	X			X			X		Connecting haulages
Orange Free State	5	X	XX							Sills cause problems Shale in h/w - breakout Fault/dyke intersect	Remnants on dykes! FOG problem
Evander	Small									h/w cross bedding - very low cohesion	

Appendix 2.

LITERATURE REVIEW ON MINING AT ULTRA-DEPTH AND IN HIGHLY STRESSED AREAS

LITERATURE REVIEW ON MINING AT ULTRA-DEPTH AND IN HIGHLY STRESSED AREAS

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SIMRAC PROJECT: GAP033

Research Report

CSIR Miningtek April 1994

February 1994

INTRODUCTION

This literature review has been completed as part of the SIMRAC, and identifies a number of shortcomings with current mining practise, as well as several promising layout and support ideas for ultra-deep mining that warrant further investigation. Cook and Salamon (1966) motivate the introduction of stabilizing pillars for deep-level stope support as follows: "As mining extends to greater depths, the frequency and severity of rockburst is likely to increase. It appears that a substantial reduction in the rockburst hazard can be achieved only by radical changes in the present stoping method." They further point out that such mooted radical changes need only apply to mines that are unable to achieve adequate improvements using current support. As mining at ultra-depth is considered, both "radical alternatives" and improved use of current support methods need to be examined and evaluated.

Salamon (1985) suggests that there are two essential ingredients to reduce unstable rockmass behaviour at depth - firstly, to limit the change in stress and energy due to mining by restricting volumetric closure and secondly, to design layouts in the vicinity of geological features to reduce large rockburst potential. They indicate that pillars and backfill provide the most promising ways of achieving these objectives, and that these coupled with current improvements in local support would lead to greater safety. Wagner (1986) supports this viewpoint that the only two known methods for effective rockburst control at depth are stabilizing pillars and backfill.

Wagner (1987) states that gold mines are, and will increasingly be working where stresses exceed the strength of the rockmass. The challenge for rock mechanics is to lay out mines and sequence excavations so as to obtain a most favourable fracture zone from a support point of view.

Other than on-the-reef considerations of regional and local support, are problems related to mining through or in the vicinity of geological features and their influence on mine stability, as well as location and protection of a range of off-reef excavations during normal mining or rockburst conditions.

This literature review has been divided into several sections each addressing different aspects of mining under high stress conditions, where stresses regularly exceed half the uniaxial compressive strength of the rock. These are highly stressed remnants, high stress and ultra-depth mining, stabilizing pillar systems and backfill, off-reef excavations, shafts and shaft reef pillars and pre-conditioning. The section on pillars and backfill is included in some detail as this regional support is working well at deep/ultra-deep-levels on some mines, while causing considerable difficulty on others. An assessment is included at the end of each section. Principal findings from this review are summarised in the conclusions.

HIGHLY STRESSED REMNANTS

Remnants are small unmined sections of reef surrounded by extensive mining, and which may be very highly stressed. These high stresses, the many accidents associated with remnant mining and

the incidence of bursting in remnants motivates its inclusion in this study on high stress mining. Records of remnant mining show that this can be a hazardous operation even at moderate depths. Berrange (1956) reports fairly severe pressure bursts, associated with remnants, occurring on West Rand Consolidated at depths no greater than 1200 m. Hinds (1962) on Doornfontein describes "sudden releases of energy" or rockbursts when scattered mining reached remnant stages at depths of around 1000m. van der Wal and Macauley (1968) also at West Rand Consolidated describe experiences of remnant mining at 1500 m depth. On this mine all blocks of ground where pressure was a consideration were classified remnants. They note that pressure bursting was most likely to occur within a few blasts of re-starting a remnant that has stood for a long time. Breton (1956) at Grootvlei under similar conditions finds no incidence of bursting in remnants, which he attributes to the soft shale footwall which allows the stable release of stresses as they build up. Roberts et al (1953) mining the carbon leader reef at Blyvooruitzicht at depths below 1500 m found very rapid closure rates often accompanied by great pressure and sometimes severe rockbursting. They found benefit in mining such remnants as quickly as possible. Early experiences in mining remnants indicated that the North face method held some advantages. Pretorius (1968) of Rose Deep and Berrange (1956) found that North facing a remnant reduced bursting, improved hangingwall conditions, provided better escape routes and increased face advance. Roulliard(1956) found no improvement in rockbursting, but preferred the method for its practical advantages. van der Wal and Macauley (1968) found that up-dip mining of remnants with waste packing most successful. They report reduced intensity and frequency of bursting over a number of years.

Hill (1946) found that over 90 % of pressure bursts at East Rand Proprietary Mine were associated with remnants. Firstly he recommends the use of longwall mining wherever possible to reduce the number of remnants and so reduce pressure bursts. After a detailed study of 36 remnants he concludes that bursting in a remnant can't be stopped by support, mining direction or rate of face advance, though their effects may be moderated. He does find that the final shape of the remnant should preferably be long and thin rather than round. He noted that pressure bursting started with a large remnant and increased as the remnant was mined.

Hinds (1963) describes the use of hydraulic props and caving for stope support for mining of remnants on Doornfontein. After overcoming early problems, this method contributed to markedly improved hangingwall conditions. The cave appeared to reduce the pressure on the face improving the face hanging and making mining easier. In cases where the cave didn't occur, conditions deteriorated, necessitating inducing of the cave. He advocates the use of these techniques for rockburst conditions and for "ultra-deep" mining. Roberts et al (1952) present methods for mining two different types of remnants. They emphasise the importance of planning the remnant long before remnant stage, so as to make its location, orientation, shape and proximity to other excavations and mining as favourable as possible. Wilson (1970) suggests that as one approaches remnant holing, support density in the working faces should increase, and care taken to install support close to the face. He notes that few accidents were associated with remnant extraction, and attributes this to additional care and support in these cases. Davel and de Jongh (1972) found that

longwall spans need to be reduced significantly as mining depth increases. This in turn creates more remnants, with their associated problems.

van Antwerpen and Noble (1987) report experiences of both scattered mining and longwall remnants on Blyvooruitzicht at depths around 2 000 m. This mining resulted in island remnants left between raises which proved difficult and often impossible to extract, and were frequently the site of serious rockbursts. Serious problems in advance haulages were experienced as overtopping took place. Again at Blyvooruitzicht, Rorke et al (1990) describe the use of pre-conditioning in the successful mining of a small up-dip remnant. They report severe drilling problems and concluded that the centre of the remnant was unfractured and highly stressed. Pre-conditioning appeared to significantly steepen the hangingwall fractures which improved local conditions. The remnant was successfully extracted without damaging seismicity despite a number of moderate events.

More O'Ferrall (1981) in an excellent paper on remnant mining concludes that stoping of remnants needs always to be very carefully considered because, firstly it may be very highly stressed ground with large amounts of stored energy, and secondly because remnants often stabilise nearby geological structures, and their mining may give rise to very large rockbursts. Remnants should not be seen in isolation, but with other remnants, geology and unmined ground. Numerical modelling may be used to give indications of the effect of mining one remnant on nearby remnants and features. One needs to be careful of any remnant mining near a fault even if there is no history of seismicity on the structure. Full remnant support should be used for all such mining. Be aware of this mining's effect on nearby footwall developments. All old support should be regarded as suspect, as timber rots and tendons rust. Spengler and Ortlepp (1981) add that a remnants "history" should always be examined, as remnants are often left for good reasons. Coggans (1986) experience of mining remnants at great depth was that this operation was accompanied by rapid closure and strong hanging/footwall fracturing. The nature of the hazard depends on the local conditions, though the use of rapid yield hydraulic props is usually indicated.

Legge and Spottiswoode (1987) monitored a deep-level remnant some 2160 m below surface at Vaal Reefs. They found that the Gutenberg-Richter 'b value' changed significantly at pre-remnant and remnant stages of mining. They (as does Hill, 1946) note that the remnant stress and hazard increases with mining only up to a point after which the remnant sheds load as the fracture zone coalesces and the core of the remnant fractures. At this stage the mining hazard is considerably reduced. Haile (1993) observed that much of the seismicity at Kloof gold mine relates to approaching longwalls in pre-remnant and remnant stages.

The industry guide (1988) sets out a number of principles for mining remnants. Care should be taken as to the position and shape of remnants. Long, thin remnants should be avoided as they are prone to rockfalls and rockbursts. (This differs to findings of Hill, 1946). This may depend on the orientation of the long axis of a remnant in relation to geological structures. An underhand mining layout is preferred as the final remnant is left adjacent to solid rather than mined out area. Mining of remnants leads to increased stress, increased seismicity and changes to stresses on nearby

structures and mining. There should be a preference to mine towards the largest, closest solid area. One should mine away from or obliquely ($>30^\circ$) to geological hazards. Except for shaft pillars it is better not to split a remnant, but to mine in one direction. Panels in a remnant should not approach one another. Avoid large leads and lags, maintain two or more access ways, blast faces regularly and upgrade support density and quality in high stress areas, particularly as geological structures are approached.

Assessment

The highly diverse experiences related to mining remnants described in the literature indicates that remnant mining varies considerably with local geology and layout among many factors, which emphasises the need to consider local conditions. Despite the fact that much seismicity and many rockbursts are associated with remnant mining it appears that relatively few accidents are recorded because of careful adherence to remnant standards for layout and support. Remnants nevertheless remain the site of much damaging seismicity - potentially hazardous and disruptive. The 'b' value findings of Legge & Spottiswoode and possibly other seismic parameters may prove valuable indicators of remnant seismic risk, and motivate further work.

HIGH STRESS and ULTRA-DEPTH MINING

In this section general results relating to deep-level, high stress conditions are surveyed, and, as far as possible extrapolated to the ultra-depth situation. Details on regional support, off-reef excavations and shafts follow in separate sections.

High stress

Mining in high stress is likely to encounter problems including many listed below from the Guide (1988). These are likely to become more severe as depths increase.

- The trend of increasing accident and fatality rates with increasing depths.
- Difficulty in negotiating faults and dykes.
- Stabilizing pillar failure, punching or foundation failure.
- level of seismicity, and rockbursting.
- rapid closure rates reduce effectiveness of local support, with the exception of backfill.
- scattered mining difficulties like high abutment stress, advance development and remnants.
- tunnels, rock passes and off-reef excavations subject to high ambient stress and stress change making support more difficult.
- gullies adjacent to abutments or pillars are difficult to support.

Work done by Wilson (1970) and Roberts and Jager (1992) indicate the more hazardous areas of mining based on an analysis of accident statistics. The stope face, gully/face intersection, gully intersections and unsupported development ends accounting for a high percentage of fatalities. Coggan (1986) reinforces this by saying the gully/face area is one of particular danger, and its layout and support needs to be re-thought. Wilson indicates four factors contributing most to accidents: face and gully, strike gully, falls of ground in vicinity of faults and dykes, and high stope width. He advocates a critical review of gully support, every effort be made to reduce stope width and particular

care to increase stope and gully support when intersecting a fault or dyke. With Wilson having identified these "trouble spots" over 20 years ago, the newer analysis of Roberts and Jager is disturbing, indicating either too little attention has been paid to the problems, or that the problems are inherently difficult to solve, or that the mining environment has become more complex as depths have increased and mines have become more worked out. They point to the need for improved temporary face support and the way it is included in the mining cycle. Often this is removed or dislodged during cleaning and subsequently not replaced or incorrectly installed. Many accidents point to non-adherence to mine standards rather than problems with the standards per sé. Support requirements depend not only on depth and span, and the influence of stress, but also on rock conditions - the reef itself and immediate geology as well as proximity to more dominant geological features (Waldek, 1986).

Several authors (Spengler & Ortlepp, 1981 and Spearing, 1990) indicate the value of previous experience of the same or similar conditions on a mine. Mine and site specific local knowledge of support, geology and rock behaviour shouldn't be overlooked.

Ultra-depth

Spearing (1993) defines ultra-deep mining as that taking place in excess of 4 000 m depth, and having a regional average Energy Release Rate (ERR) of over 60 MJ/m². Ryder (1993) would put the transition from deep-level to ultra-deep-level at between 3 500 and 4 000 m below the surface depending on the strength of the host rock. Current problems (Lenhardt, 1992) being experienced in mining in the transition zone between deep and ultra-deep suggest that mining methods be investigated for 3 500m and below. Roberts and Jager (1991), in an assessment of accidents in the industry find that problems identified will be exacerbated as depths increase.

Several mines are currently mining in this depth range (Western Deep-levels, Kloof, East Rand Proprietary Mines) and several more are being developed or planned (South Deep, East Vaal or Moab, Leeudoom, Sun, Target). The ever increasing depths of identified gold ore-reserves in South Africa indicates that the long term future of the industry may hinge on finding viable technology for mining at ultra-depth.

The deep-level mining conditions listed in the Chamber of Mines (COM) Industry Guide (1988) apply and will be exacerbated as depths increase to ultra-deep-levels. These primary considerations are as follows:

- high stress levels and high ERR gives rise to highly fractured rock.
- very rapid stope closure gives rise to rapid, high stress regeneration in the back area.
- seismicity which is often associated with geological discontinuities.

The guide lists a number of strategies for mining at deep-levels, and while these may not necessarily translate to the ultra-deep conditions, they at least provide a starting point for further work.

- mining sequences should be modified to reduce average ERR and stress levels and minimise deleterious effects of remnants (longwall or scattered)

- regional support (pillars or backfill) should be used to reduce rockburst hazard.
- procedures for negotiating geological features (bracket pillars) need to be devised.
- longwall mining with follow-on haulages preferred.
- local support to cope with highly fractured rock, rapid closure rates and seismic loading.
- leads and lags kept to a minimum in high stress areas.
- keep gully headings to a practical minimum.
- pillars may be required in very high stress/deep-level scattered mining layouts.
- gully support should include yielding tendons and lacing in problem areas.

A range of proven stope support techniques should prove adequate at greater depths, provided principles for their use are adhered to. Convincing evidence of benefits of backfill as a local support in conjunction with RYHP has been reported by many authors including Gürtunca and Gay (1993). Most current problems with the use of backfill for stope support can be attributed to poor placement, low % fill and a long fill to face distance. Modelling indicates that considerable regional support benefits should result if extensive areas of backfill are placed.

Gürtunca et al (1993) document the use of silicated fills for mining in high stope widths, concluding this to be suitable up to depths of 3000 m. Spearing (1993) describes a double cut method of mining high stope width regions, using RYHP for the first cut and silicated fill thereafter. Several of the new mines starting or being planned have areas of high stope width, up to 40 m in extreme cases (Gay et al, 1993) which will aggravate the seismic risk and reduce the effective rockmass strength. Gürtunca and Gay (1993) indicate how different types of fill be matched to differing underground conditions. Squelch and Gürtunca (1990) believe that many current backfill operations are not effective because of a poor mining cycle and poor placement. Three criteria are set for improving the placement: Fill must not be more than 6 m from the face; support resistance in face area of at least 50 kN/m² (static) and 200 kN/m² (rockburst), and the use of RYHP, eliminating timber support. Kirsten and Stacey (1989) argue for the use of backfill because it provides continuity of support with stiffness increasing exponentially and can stabilise hangingwall blocks so maintaining its integrity. They find that fill is not likely to reduce seismicity, but may stabilise the stope area under dynamic load.

Noble (1990) feels that pillar geometries working adequately at current depths on mines in the Rand Mines group don't translate well to greater than 3500m, as both Absolute Pillar Stress (APS) and face ERR's exceed recommended limits. At depths between 3000 and 4000 m combinations of pillars and fill are indicated. Davel and de Jong (1972) assess longwall pillars as follows: with 85 % extraction at 2400m depth rockbursting will cause problems if spans exceed 260m, while at 3700m the inter pillar span reduces to 105m. They predict a maximum extraction of 60 - 65 % for depths of 4400m required to keep ERR < 41 MJ/m² and that such pillars could only be removed at the risk of severe rockbursting. Despite the above, Spengler (1993) at ERPM is confident of being able to mine to 4000m using similar extraction percentages and only slightly increased pillar dimensions based on current experience at mining depths around 3700 m. Lenhardt and Hagan (1990) describe dominant pillar failure mechanisms at Western Deep-levels, and indicate these to be largely independent of pillar dimensions, and likely to worsen as depths increase. Smith and York (1993) motivate the use

of concrete pillars because current regional support strategies will not allow "safe" (30 MJ/m²) mining at depths greater than 4000 m.

Gay et al (1993) report that increased event magnitudes are expected on all the major mining districts resulting from gradually increased mining depths, associated with geological structures and influenced by increasing areas mined. Spearing (1990) indicates that mining at 4000 m will lead to increased seismic hazard and increasingly complex shaft systems. A new approach or significant refinement of current mining and support techniques is required to be even relatively safe. For the present, the highest quality stope and gully support be used at deeper levels. Atkins (1987) describes three options that permit deeper level mining, namely: stope width reduction, stabilizing pillars and backfill. Roberts and Brummer(1988) specify support requirements for rockburst conditions. The stope recommendations are that the support be able to absorb energy, that it must maintain at least 200 kN/m² resistance, and that a suitable headboard be used on hydraulic prop support units. Roberts (1991) finds that elongate timber support is generally not suitable for rockburst conditions due to its limited yielding range and because units fail after a relatively small amount of closure. In cases of closures greater than 10 - 20 mm per day there is effectively no back area support using timber. At closure rates of 30 - 40 mm per day timber support is generally not suitable. Such closure rates are likely at ultra-depth. Every effort should be made to prevent falls of ground in the stope face area as this tends to weaken the hangingwall beam making further falls of ground likely. To minimise this risk especially in weak, friable hangingwall conditions, support should be kept as close to the face as possible, and suitable headboards used. Regular barring down and inspection of the face hanging is essential. Rapid yield hydraulic props (RYHP) should be used provided the stope width is not too high.

A significant disadvantage of longwall mining with stabilizing pillars is that it leaves little scope for selective extraction in areas with very variable grade. Thus the average grade for the full area needs to be payable. In highly faulted ground, scattered mining is also indicated. The whole layout for scattered mining at depth needs to be re-thought, and may possibly incorporate pillars and follow-on development.

Mechanisation holds advantages for ultra-depth mining as the work complement in the more hazardous face area is limited. Pickering (1987) reports on significant developments at COMRO on non-explosive mining methods and Spearing (1990) describes diamond wire cutting developments. The use of non-explosive mining as having numerous advantages as mining depths increase, which include reduced workforce in the face area and a less fractured hangingwall. Espach (1990) supports this view, setting out a complete mining and support system to be applied to a new deep-level mine. This includes extensive use of mechanisation. Wagner(1987) states categorically that " effective rockburst control is a pre-requisite for successful and meaningful face mechanisation". Gay et al (1993) discuss eight strategies for enabling safe and productive mining at all depths, of which several relate particularly to deep-level, high stress mining: the need for improved regional support layouts; improved understanding of seismicity and rockbursts towards developing counter measures; support

strategies for high stress, high stope width areas. They further point to the advantage of reduced risk with increased mechanisation.

Assessment

Most problems at deeper levels relate to increased stress and stress relative to the strength of the rock. The indications of increased levels of seismicity at increasing depths points to the need to control total energy released and associated seismicity, for which a suitably designed regional support system is essential. Such solutions are unlikely to be universal, needing to be designed for particular conditions. Backfill, concrete & rock pillars as well as combinations of these provide the basis for such regional support development, though modified or largely new mine layouts may be required if these are to be effective. The difficulties associated with mining near geological discontinuities points to the need for improved layouts and support practices in such areas. Advance development on or off-reef may be needed to pre-examine the orebody, and points to the need for significantly improved or new tunnel layout and support techniques.

STABILIZING PILLARS and BACKFILL

This section on stabilizing pillars and backfill is included not so much for completeness, but rather because these two support elements are currently used as regional support for most deep-level mining, with mixed success. Ultimately regional support is to reduce back area closure and thereby reduce both face stress and seismic potential. Nevertheless, these seem most likely to continue as principal regional support elements for ultra-deep mining.

Stabilizing pillars

Cook and Salamon (1966) calculated a ten fold reduction in ERR using stabilizing pillars at 3900 m depth and 80 % extraction. To achieve a 20 fold ERR reduction at 3200 m only 74 % extraction was possible. They also state that a 20:1 pillar cannot fail, but that hanging or footwall failure is possible and that this may be violent. Salamon and Wagner (1979) proposed the use of 70 m wide stabilizing pillars motivated by reduced volume of closure and hence reduced ERR. They found that the minimum principal stresses in both the hanging and footwall must remain compressive to avoid punching. A design formula suggested by them is as follows $\delta_1 - k \delta_3 < \delta_C$, $6 < k < 10$. In contrast to the findings of Cook and Salamon above, Brummer(1987) found that 20 m wide stabilizing pillars were too small at a depth of 3000 m, being extensively fractured, though too large to act as a crush pillars. He cites a foundation failure criteria: $\delta_{ff} < N\delta_C$, $N = 3$ for stabilizing pillars.

Ortlepp (1988) reports significant success with the introduction of stabilizing pillars mining at 80 % extraction at ERPM using 50 m wide pillars and at Blyvooruitzicht using 45 m pillars. These resulted in significantly reduced fatality rates over a number of years. The reduced seismicity at ERPM corresponded well with predicted reductions based on reduced ERR levels. Spengler (1986) describes mining at ERPM using stabilizing pillars at depths from 2500 - 3600 m with no undue problems. Bayley and Hagan (1987) report a linear relationship between ERR and damaging rockbursts experienced at WDL. They found that both pillar stability and an ERR < 10 MJ/m² can be

obtained at 85 % extraction. Noble and van Antwerpen(1987) on neighbouring Blyvooruitzicht found longwall mining attractive because of follow-on development. This method could lead to longwall peninsula remnants with increased seismic hazard, and these should be avoided. They lists two principal advantages of stabilizing pillars, first they reduced closure and hence ERR, and second, to clamp the hangingwall reducing the extent of propagation of fractures along the face. The flattening of the fractures in the vicinity of the strike pillars has led to support difficulties in those areas.

Atkins (1987) expresses two concerns related to stabilizing pillars, firstly, the transition from intact to highly fractured pillar under higher stresses may be sudden, violent and unstable, and secondly, the problem of foundation failure. Lenhardt and Hagan (1990) identify four dominant failure mechanisms for stabilizing pillars, which they term "crush", "shear", "punch" and "simultaneous shear". Further they find that violent failures on pillars recur if counter measures are not taken. Damage most often manifests itself as additional closure, and damage to nearby footwall drives. They found event magnitudes independent of pillar size from 20 m to 60 m, and add that stabilizing pillars have effectively moved the seismicity from the face to the pillars. Ozbay and Ryder (1989) found, based on laboratory and numerical modelling, that damage to pillar foundations reduces the pillar load carry capacity and hence also its effectiveness as a support element. The pillar foundation failure and accompanying large volumetric closures leads to larger than predicted ERRs (Diering, 1987). They conservatively estimate that at 4000m depth pillar foundation damage can increase ERRs by 30% over the unfractured case. Leach and Lenhardt (1990) report that some seismic events have caused "catastrophic" damage among pillars, yet these phenomena are poorly understood. Their study of pillars indicates that changing the pillar width from 20 m to 60 m will have no bearing on the magnitude of pillar associated seismicity, because such seismicity is not due to pillar failure but to the footwall heaving violently into adjacent excavations. Such pillar associated seismicity increases with depth. The violence of foundation failure relates to the footwall geology. If there is shale less than 20 m below the pillar, stable punching of the footwall into the shale occurs. This becomes less stable as the distance of the shale below the pillar increases, and at greater than 40 m, violent failure of the footwall quartzite occurs. Bayley and Hagan (1987) found that despite reduced seismicity levels, the pillars increased the incidence of hangingwall fracturing resulting in increased fall of ground (FOG) problems at lower seismicity levels. Jantzon et al (1990) make a number of points relating to stabilizing pillars at WDL. They report foundation failure at just 2000 m depth on VCR. Foundation failure is strongly dependent on local rockmass properties, largely independent of pillar width, and failure increases with increasing depth. This pillar failure problem needs to be addressed if these are to be used for support at greater depths. Ozbay and Ryder (1990) report that despite the reduction in rockbursts brought about by stabilizing pillars, a number of less desirable consequences occur. Reef is locked up (perhaps permanently), layouts are restricted and pillars themselves become sources of seismicity - these are driving the industry to look for alternatives. Spearing (1990) argues against extending stabilizing pillars deeper because of problems including reduced face availability, ventilation difficulties and foundation failure and the seismicity associated with this.

Despite numerous problems with stabilizing pillars described above, one should not discount the use of such pillars for regional support at ultra-depth. Most of the problems described emanate from only

one mine, and a number of mines are using such pillars with few problems at similar depths. It is interesting to note that despite severe problems experienced with stabilizing pillars, some mines are successfully mining with stabilizing pillars at depths exceeding 3 500 m, and plan to mine to 4000 m and below with some modifications to the same methods. Wagner (1987) suggested that dip pillars should be considered as an alternative to strike stabilizing pillars.

Backfill

The use of backfill for regional support at depth has been advocated by a number of authors, and indicates a number of wide-ranging benefits.

Gay et al (1993) anticipate regional benefit of backfill through reduced volumetric convergence and reduced ERR. Theoretical studies indicate lower ERRs, lower face stresses and reduced seismic potential. Jantzon et al (1990) indicate that the use of fill is promising both for local and regional support either alone or with pillars, and has many benefits over and above pure rock mechanics. Spearing (1990) promotes the use of fill as a regional support at depth saying that widespread use of fill will dissipate energy in numerous smaller events in preference to large rockbursts. Spearing (1993) proposes replacing stabilizing pillars with backfill, only leaving pillars to clamp adverse geological structures. Espach (1990) proposes extensive use of backfill for local and regional support. Backfill placed initially as a local support later provides regional support benefits. Grtunca and Gay (1993) review the benefits of backfill to the industry. Significant local support benefits of backfill have been proven. Long-term regional support benefits are indicated from theoretical studies, though field observations are inconclusive. The relative softness of backfill compared to pillars means one can't achieve postulated regional benefits in the short term, but over the long term, with much larger percentage placement. The underground performance of backfill as a regional support has been inconclusive (Hemp, 1993), because of relatively low percentage placement, the difficulty in finding similar sites, and the strong influence of local geology. Mine seismicity appears to be most controlled by geology and layouts with backfill having little effect. Extensive filling would reduce the seismic moment, accompanied by a reduction in the number of large magnitude events, and an increase in the number of smaller events. Perhaps more significant is that extensive fill reduces peak accelerations and velocities hence reducing the potential for seismic damage. Vibration times and development of low frequency surface waves would be reduced resulting in less damage.

Ortlepp (1988) argues against the use of backfill as a substitute for stabilizing pillars because the convergence initiates well in advance of the face , so even good quality fill cannot prevent the accompanying relaxation of clamping forces.

Combinations and other

Atkins (1987) motivates the reduction of stoping width as an option to permit deeper level mining. The stoping width option has limited scope for improvement, though, wherever possible stope width should be reduced to a minimum for it holds both economic and rock mechanics advantage.

Lenhardt and Hagan (1990) infer that the pillar problems at WDL are likely to worsen as mining depths increase, necessitating the use of additional regional support. Backfill is suggested to supplement the fill, to reduce the load and hence also the failure potential of the pillars. Piper (1987) considers the use of stabilizing pillars and backfill alone or in combination for regional support at depth. He found that backfill alone can replace stabilizing pillars up to 2500 m depth. Backfill can be used in combination with stabilizing pillars at greater depths to reduce ERR, provided spans are sufficient to allow backfill to take load. This has the potential to reduce the amount of gold locked up in pillars by up to 30 % up to 3000 m depth. Backfill applied to ultra-depth mining may require the addition of a binder to increase its initial stiffness (Spearing, 1990). Wagner (1987) suggests that the introduction of stabilizing pillars in mines has reduced rockbursts, but the problem appears to have shifted from a rockburst problem to a fall of ground problem. The area requiring particular attention is the 20 m or so on the immediate up dip side of the pillar at the face intersection. In looking for a solution, the use of backfill close to the up dip side of the pillar has been proposed to mitigate against the bad hanging in this area. A pillar/fill combination may also soften the effect of foundation failure. Diering (1987) evaluates the performance of 20 and 40 m wide stabilizing pillars on WDL, and concludes that the theoretical low ERR values are not being achieved in practise. He reports dramatic improvement in hangingwall conditions on introducing backfill. In looking to the future, he believes that backfill alone can be used to depths up to 2700 m maintaining the ERR below 30 MJ/m². At 3000 m depth, 95% extraction with 40% backfill can do the same. To extend this to depths of 4000m, 90 m wide pillars and only 68% extraction are indicated. He concludes that at depths of 4000 m and below it is not a case of backfill or stabilizing pillars, but both. Hagan(1988) writes that additional stabilisation mechanisms will have to be introduced for stabilizing pillars as depths increase, and certainly below 4000 m. Again a combination of pillars and fill is proposed. Gay et al (1993) suggest that combinations of backfill and crush pillars be investigated as a means of providing a cheap stiff support.

Lenhardt (1992) found that the process leading to large events is governed by time dependant behaviour of the rockmass surrounding the excavation, and that this creep effect should receive more attention in numerical model development. He also concludes that the large events of magnitude greater than two are independent of the production rate. He identifies a number of different types of seismic events and their characteristics. These include shear type events, non-shear events, fault slip, dyke contact slip, abutment failure, pillar foundation failure and crush type events. Further work on event classification may assist in better addressing these problems. The slip on a geological feature may best be controlled by either stabilizing the feature (bracket pillars and layouts) or de-stabilising it (triggering). Backfill doesn't appear to stabilise features, though does have significant potential to combat pillar foundation failure.

Adams et al (1989) describe layouts making use of concrete pillars as an alternative to stabilizing pillars. These may well hold benefits applied to the ultra-depth case, though require further modelling and underground assessment. Smith and York (1993) propose a method of extraction at great depth involving a radical departure from current mine layouts to include the use of concrete pillars placed in headings for regional support. The economic viability depends on the grade and

geometry. Concrete pillars are likely to have a much more stable, non-violent, crushing type failure. Gay et al (1993) cite unpublished work by Ryder in which he indicates that even using a 1 GPa concrete, safe ERR levels could be maintained, 100 percent extraction is possible and foundation failure seismicity eliminated.

Assessment

This section has reviewed the performance and experience in deep level mines with stabilizing pillars and backfill as regional support elements. Despite the problems experienced, many benefits were observed. These support elements should not be seen in isolation but in conjunction with the reef and immediate surrounding rockmass. It seems that a better understanding of their in situ behaviour will include the need to design support to suit specific local conditions.

SHAFTS and SHAFT PILLARS

Shaft pillars are included in this write-up on high stress mining because exceptionally high stresses may be encountered in shaft pillars even at moderate depths, either statically or during shaft pillar mining. This section is split into two parts: extraction of shaft pillars, and layout and protection of the shaft and associated excavations at ultra-depth.

Shaft pillar extraction.

The extraction of many shaft pillars on South African gold mines have been documented. According to More O'Ferrall (1987) some 12% of shaft pillars have been extracted and around 170 pillars remain that could be extracted (1987 figures). As shaft pillar mining is usually left to near the end of the life of the shaft or mine, high stresses may be encountered even at moderate depths. Several problems relating to this high stress have been recorded and are summarised below.

Stress: Both the shaft itself, haulages, rock-passes and large excavations are subject to substantial increases in what may already be a high stress, followed by a rapid reduction in stress. Support of these structures needs to be upgraded to cope with the high stress levels and later change in stress as well as possible seismic loading. The protection of ancillary excavations such as pump and hoist chambers, dams and haulages, in addition to the shaft itself needs to be considered in laying out the excavation sequence (Wagner and Salamon, 1973). Apart from the excavation sequence, the use of stiff support, backfill, crush pillars and a minimum stoping width all help to keep the effects of stress to a minimum. High stresses will concentrate on any reef not extracted, geological loss, unpay ground or final remnant and one should anticipate associated problems. Isaac and Simms (1974) state that care should be taken in positioning and mining final remnants as very high stresses will be encountered and bursting is likely.

Seismicity: High stress concentrations on geological features like faults or dykes leads to a high probability that seismicity will be experienced. Indeed, most authors report seismicity associated with shaft pillar extractions. van Emmenis and More O'Farrell (1971) describe two severe seismic events associated with extracting the Stilfontein TONI shaft pillar some 1400 m below surface. Movement on a fault caused significant damage to the shaft and lining. Faces should be angled to at least 30°

when approaching geological features in the shaft pillar. Stope support suited to remnant or rockburst conditions should be used. Bruce and Stilwell (1990) anticipated seismicity around the shaft due to two dykes in the shaft pillar at West Driefontein.

Perhaps most significant is that a shaft pillar being extracted at depth, may quite closely resemble stope and tunnel conditions at ultra-depth. The stresses encountered, the depth of fracturing and the interaction with geological features may be similar. Much may be learned about ultra-depth conditions from detailed descriptions of a number of shaft pillar extractions that have been well documented. Further, a shaft pillar being mined affords the opportunity to test on and off-reef layouts and support in conditions similar to those expected at ultra-depth.

Shafts at ultra-depth.

McKinnon (1989) clearly indicates that conventional shaft pillars cannot adequately protect shaft systems at depths exceeding 3 000 m below surface. With a number of shafts significantly deeper than this at present this is not a future problem. He lists a number of alternatives for shafts at depth including stabilizing pillars, satellite pillars, differential stoping width and backfill, all based on the early extraction of the reef in the shaft area. Detailed assessment of these options is presented for both the 3000 m and 4000 m cases. Several new shafts have been designed based on these principles. Particular care needs to be taken in positioning stiff shaft area support, be that backfill or any form of pillar, to ensure that high stress build-up or regeneration does not damage the shaft or associated excavations. Some of the backfill surrounding the shaft barrel in the inner pillar area of the ERPM far East sub-vertical shaft had to be removed to counter possible shaft damage as a result of high stress regeneration in this fill (Spengler, 1993).

More O'Ferrall (1983) suggests the even if the shaft pillar is not extracted (at depths where this is suitable) there is benefit in mining and supporting the inner pillar at an early stage. In addition, the shaft should be designed so that guides are not attached to the shaft lining in the vicinity of the reef intersection, and that this reef intersection should be midway between stations. Willemse (1990) describes the early extraction of the VCR reef around a shaft at Western Deep-levels, at close to 3000m depth. The area was supported with backfill, and a number of advantages were anticipated such as production advanced by 18 months and destressing of major excavations occurred. Shaft lining support and steelwork were designed to accommodate anticipated movements. Raffield et al (1993) describe the design of a deep-level shaft system for South Deep mine. The reef around the shaft position (radius of 150 m) is to be extracted prior to sinking the shaft with access from a neighbouring mine. Backfill is used to limit closures in this area and again expected shaft deformations are designed into the shaft steelwork and concrete from the outset.

Pre-extraction of the reef for deep-level shaft systems will still result in stress regeneration taking place later in the life of the mine either due to total closure or through the backfill. These regenerated stresses will remain below virgin stress, but may be as high as 100 to 120 MPa, and sufficient to require careful support of the shaft and associated excavations. If stabilizing pillars or satellite pillars are used, these will become extremely highly stressed as mining proceeds. Current

problems with pillars at depth indicates against their use in favour of backfill. Pre-conditioning of these pillars may allow them to dissipate stress in a stable manner. Care needs to be taken to site excavations away from the influence of high stresses on these pillars. Pillars have an advantage over backfill in that high stresses are concentrated in the vicinity of the pillars and not everywhere as in the case of backfill. In all cases the possible effects of seismicity on the shaft system needs to be considered.

Assessment

The shaft-reef areas of several new deep level shafts have been, or are being extracted, and there appears no doubt that pre-extraction of the reef in the shaft area is required at greater depths. Later extraction of the shaft reef area requires careful assessment and site specific design. The similarity between mining of a shaft pillar at intermediate depths and ultra-deep mining conditions should be examined as such areas provide opportunity to simulate the deeper level conditions.

OFF-REEF EXCAVATIONS

A section on tunnels and off-reef excavations is included in this survey on mining in high stress because it appears there may be significant problems associated with such structures as mining depths increase. In geologically complex areas, and where grades are highly variable the scattered mining method is generally used, exposing pre-developed haulages to high field stresses, extreme abutment stress and large stress changes. In such cases tunnels may have been sited close to or have to traverse faults and dykes with accompanying risks. Longwall mining accounts for only about 20 % of tunnels developed, with much of the remaining 80% developed in advance of the mining, and subject to problems described above, though not all subject to such high stresses (Wojno et al, 1986). In longwall mining at great depth, even follow-on development is likely to experience significant stress regeneration, at times approaching virgin stress levels. At greater depths with the early shaft-reef area extracted, the protection of numerous large excavations (pump and hoist chambers, dams, rock passes, etc.) in the shaft complex need to be protected from very high stress regeneration either due to backfill or total closure which may occur very rapidly.

Wiseman (1977) found, based on extensive underground observation, that in the absence of geological features stress was the major factor in tunnel condition, however in close proximity to intersecting geological features - faults and dykes - these had a significant negative effect on tunnel stability, far more than stress. Wagner (1983) describes the dominant failure mechanism for tunnels at deep-levels. Parallel slabbing normal to the major biaxial stress direction occurs at around 40% of the rocks UCS, with the thickness of the slabs dependant on the rock properties. Fracture formation is exacerbated by parting/bedding planes, and in the case of more steeply strata, requiring particular care in the siting of tunnels in such circumstances. He sets out principles for tunnel support under static and dynamic conditions. In essence one should not try to support the rock, rather to use support to stabilise the fractured rock around the tunnel. For support under dynamic load conditions, he distinguishes tunnels intersected by faults and dykes and associated shear type movement from tunnels subject to high ground velocities from nearby seismic events. In the former case, severe damage over a short section of the tunnel is anticipated, while in the more common latter case,

damage may extend over a much larger area. The effectiveness of support under dynamic load relates to the work done by the support as it yields with the dilating rock, pointing to the need for yielding support elements.

Wagner and Godfrey (1976) write that tunnel support costs become unduly high as stresses exceed 2/3 of the sidewall UCS. At depths greater than 3000 m this is likely to be the rule, and tunnels will need to be developed in destressed ground. Hepworth and Gay (1986) describe an experimental tunnel at depth and sited such that the stress environment was similar to depths of 3000 to 4000 m. Sections of the tunnel were supported with different support systems for comparative purposes. They found that shotcrete, meshing and lacing with 3 m ropes or 2.2 m grouted smooth bar most effective. A greater support density reduced sidewall dilation. Ortlepp and Gay (1984) monitored a deep-level tunnel on ERPM at a depth of over 3200 m and sited around 50 m above an abutment/stabilizing pillar. Stress levels, as stoping proceeded, rose to above 230 MPa over the pillar. The tunnel sidewall spalled to assume a more or less elliptical shape along much of its length, with this shape appearing to stabilise itself for the prevailing stress conditions. All the different support units tested seemed to be adequate, indicating the potential to support even more highly stressed tunnels. Many seismic events were recorded over the duration of monitoring, up to magnitude 3.1, and caused no more than superficial damage. More O'Ferrall and Brinch (1983) believe, based on their experience at Buffelsfontein GM, that large cross section tunnels can be maintained at depths of 5 000 m, even if developed in advance! They found the shape of the tunnel to be important with an elliptical shape both practical and stable. While confident that such tunnels are viable, they pose several unanswered questions requiring further research.

Ortlepp (1983) sets out support requirements for rockburst prone tunnels. He makes the point that in severely faulted ground where, scattered mining is unavoidable, the possibility exists for very severe rockburst damage to tunnels. He stresses the need for yieldability of tunnel support elements stating that it is better to sacrifice some load bearing ability to gain a measure of controlled yieldability, without which the total loss of a support element is possible. He surveys the support characteristics of a number of commonly used tendons as well as lacing and meshing. A largely empirically based tunnel support design suitable for rockburst conditions is presented, which includes tendons, meshing and lacing. Wagner (1987) notes that tunnels are currently being developed at depths around 4000 m and are planned to go deeper, which presents a rock engineering challenge - to design an excavation sequence and geometry so as to obtain a most favourable fracture zone from a support point of view. This can be aided by excavation shape. An effective and affordable support system needs to be found. He notes that the presence of softer material layers with brittle rock, compounds the tunnel support problem because of additional deformation in the soft rock, and the shear damage to the brittle rock at the interface. Like several previous authors, he prefers an elliptical tunnel shape. He concludes that it is possible to design and support excavations for field stresses approaching the UCS of the rock. He advocates more formal back analysis of recorded tunnel behaviour with ever improving numerical models. He is at variance with several previous authors believing that seismic loading of properly designed and supported excavations is not a serious problem. The industry guide (1988) sets out standard and special guidelines for the support of

tunnels under differing conditions. These requirements are primarily driven by three factors, namely the condition of the rockmass, the initial and anticipated state of stress, and the rockburst problem. The guidelines include layout considerations to minimise the risk of damage. In deep-level or high stress conditions, the stresses will dominate, requiring positioning of tunnels in overstoped ground wherever possible. Tunnels should be positioned so as to avoid undue proximity to seismically hazardous structures by at least 50 m. These should preferably not intersect potentially active features at an acute angle or run parallel for any distance. Rather swing the tunnel to intersect the structure as close to normal as possible. Shape of tunnels is discussed, with elliptical or "T" shapes holding advantages, though in cases of strongly developed bedding, the shape should conform to the bedding to minimise the formation of unfavourable blocks. The use of modelling and Rock Condition Factor (RCF) for layout of tunnels and for assessing tunnel support requirements have been suggested (Noble, 1990).

Jager et al (1990) discuss tunnel design suited to high initial stresses (150 - 200 MPa) and subject to subsequent increase in stress of 25 to 50 MPa, followed by later relaxation (typical for deep-level scattered mining). The stability of such tunnels under seismically active conditions is addressed. They make the point that some 20 % of the annual tunnel development has to be rehabilitated annually at huge cost to the industry. This figure is likely to increase with increasing mining depths. The overriding consideration in tunnel support is the stress induced fracturing and associated deformation of this fracture zone. This deformation process is usually slow but continuous. However, it can be dramatic under seismic conditions, where large peak ground velocities of at least 5 or 6 m/s are possible. They describe new developments in tunnel support such as yielding tendons, and the use of fibre reinforced shotcrete. The use of preconditioning may be required for extreme stress conditions. The first strategy for tunnel support is always to pay attention to layout in relation to pillars, abutments and geology. They prioritise principles for tunnel layout and support for high stress conditions, indicating that such techniques need routine application at stresses exceeding 150 MPa. By way of "warning" they note that tunnelling at extremely high stress faces formidable problems, which, if not overcome, may affect the viability of mining at 4 500 m!

More O'Ferrall and Brinch (1983) indicate that at great depth or in very high stress conditions large excavations should not be developed using more than one cut because the fracturing associated with each cut will give rise to very poor sidewall conditions, initially making the next cut difficult and affecting the long term stability of the structure. Jantzson et al (1990) state that care must be taken to site longwall follow-on footwall access tunnels well away from stabilizing pillars to avoid damage. Kersten et al (1983) found that much care should be taken in developing large chambers at depth because they are usually required for the life of the shaft if not the mine. They conclude that tunnel support are not necessarily suited to supporting much larger excavations. In cases of soft host rock, and of well defined bedding, large displacements of the excavation sidewall should be anticipated. The guide (1988) extends tunnel support criteria to large excavations, while pointing out the need for quality assurance and anti-corrosion measures because of the long life expectancy. Roberts and Brummer (1988) set out principles for tunnel support for rockburst conditions. Support resistance of

at least 60 kN/m², yielding support, able to absorb energy, while maintaining the integrity of the fractured rockmass, assisting the rockmass to support itself.

Assessment

Extending current tunnel support knowledge to suit deeper levels and higher stress conditions poses a number of challenges. Current design knowledge and techniques available are not always employed to full advantage. There is a need to improve design and support of tunnels to withstand very high static and dynamic conditions for advanced development and un-protected connecting haulages. Such improvements in design need to be incorporated as an integral part of the haulage development. Quality of installed support needs to be maintained. There is the need for tunnels in advance of stoping, either in scattered mining or to pre-explore the orebody. Improved layout of tunnels is required to minimise risk, particularly in geologically complex and scattered mining layouts. Rock engineering expertise needs to be applied to correctly identify those sections of tunnels most at risk and that appropriate support be installed in these regions.

PRE-CONDITIONING

Pre-conditioning techniques may be applied in a number of ways that could improve mining conditions at great depth and in highly stressed areas. Stope face conditioning may be used to transfer the high stress concentration further ahead of the face by fracturing the stope face area. Such changes potentially move the source area (of a seismic event) further away from the face so reducing its damage potential. Brummer (1985) in a literature review on pre-conditioning reports that preconditioning has proved successful in both local and overseas hard rock applications, and that high costs have been the principal factor limiting its use on South African mines. Stope face pre-conditioning was used in the 1950's at ERPM for pre-conditioning of highly stressed areas. Roux et al (1956) and Hill and Plewman (1957) report on the success of this work, and that a very significant reduction in the incidence of rockbursting was achieved. Rorke (1990) describes successful mining of a small remnant at Blyvooruitzicht Gold Mine using near reef conditioning. At this site, as at ERPM, considerable difficulty was experienced with drilling. Adams et al (1993) report on the layout of faces and pre-conditioning holes to successfully condition faces. They also report on increased production using pre-conditioning. Pre-conditioning of the reef for mining highly stressed conditions continues as a research topic at CSIR MINING TECHNOLOGY.

Brummer (1985) also describes several successful applications of pillar pre-conditioning on overseas mines. Spearing (1990) suggests that stabilizing pillars be pre-conditioned at ultra-depth to reduce the incidence of pillar failure, punching, foundation damage and related seismicity. He further mentions that such pillars may be possible to extract at a later stage.

Pre-conditioning ahead of a tunnel to help negotiate a hazardous geological structure has been successfully used on at least one South African mine.

The latter two applications would require much more work to prove their viability in deep-level, high stress conditions.

Assessment

Further work is required in this area to improve the implementation of these techniques as part of the standard mining cycle, and to better identify the areas that will benefit from pre-conditioning.

CONCLUSIONS

1. It is clear that the future of the South African gold mining industry depends to a large extent on finding safe, feasible and economically viable techniques for mining at ultra-depth.
2. Mine layouts and sequences should be designed to minimise the formation of remnants, and where remnants are formed, care should be taken to make their shape, size and location with respect to mining, geology and off-reef development as favourable as possible for later extraction. In remnant extraction the history of remnants in the area should be considered, and generally rockburst precautions taken. Principles for such mining have been well described in the Guide (1988). Further work on developing seismic indicators of the burst potential of a remnant seems warranted.
3. Mining at ultra-depth will exacerbate current deep-level mining problems which include seismicity and rockbursting, rapid closure rates, high face stresses and fractured rock, pillar and abutment failure and difficulty in negotiating faults and dykes. Mine layouts should be designed to minimise the above problems. Work on procedures for negotiating geological features is underway. Current local support including RYHP, headboards & backfill should prove adequate for increased depths provided standards for use are strictly adhered to. Very rapid closure rates argue in favour of backfill and RYHP. Support in the face/gully area and problems related to temporary face support and cleaning need further work. Scattered mining at depth is required in highly faulted areas and where grades are highly variable. Such conditions exist on some of the new ultra-deep mines being planned. Significant changes to adapt current scattered mining layouts for such high stress conditions need to be examined.
4. Good regional support is essential for ultra-deep mining to reduce the seismic potential and to reduce face stresses, and remains a high priority area for further research. Stabilizing pillars are working well on a number of mines, and in such cases should translate well to deeper levels. A better understanding is required of the geological and other conditions that lead to violent pillar failure and work on this problem continues. In the longer term, alternatives to pillars should be investigated as economics may require 100 percent extraction. Suitably designed backfill or pillar/fill combinations, tailored to match a particular mining configuration may replace or supplement stabilizing pillars. The use of concrete ribs and crush pillars hold promise, and should be further investigated.
5. Pre-extraction of the shaft reef area is indicated for all deep/ultra-deep-level mines. Support alternatives include stabilizing pillars, satellite pillars, crush pillars, backfill and combinations. Alternatives and details relating to shafts have been well covered by McKinnon (1990). The long-

term stability of shaft associated excavations including large chambers, rock passes, haulages and the shaft itself is of concern as stresses will approach virgin stress, and further work should address the location, proximity to on-reef pillars, support and long term stability of these vital structures.

6. Haulages and off-reef development are another priority area and should, wherever possible, be developed in de-stressed ground, where virgin stress levels are an upper limit for stress. Support should be designed to suit local ground conditions, but must offer aerial coverage, must be able to yield, and maintain a support resistance of at least 60 kN/m². Layouts should be designed to avoid unfavourable proximity to pillars, abutments, faults and dykes. Traversing beneath pillars, or through geological features should be minimised, should be close to normal to the feature if they do intersect such features, and supported appropriately. Development ends should be well supported. Work should be done to improved layouts and on using follow-on development for deep-level scattered mining. Methods of developing and protecting advance development and long-term connecting haulages in areas of high stress should also receive attention. There seems no doubt that tunnels at depth, and in advance of mining will be required, either for cross-connections or for pre-exploring the orebody.

7. Work is currently being carried out on pre-conditioning applied to mining highly stressed reef areas. This could be applied to pre-fracturing of stabilizing pillars to reduce violent foundation damage and possibly ahead of deep-level haulages approaching highly stressed ground or geological features.

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Appendix 3.

INDUSTRY WORKSHOP - TOPICS, DISCUSSION & FINDINGS

WORKSHOP DISCUSSION QUESTIONS

A tentative list of brainstorm/discussion questions for GAP033 workshop. [*Suggested possible work directions.*]

1. Investigate the use of crush pillars as stiff initial stope support alone or in conjunction with backfill or hydraulic props. [*underground & non-linear modelling*]
2. Document a safe procedures for recovering a bad hangingwall - undercutting a brow. [*Survey, consolidate and evaluate current practice*]
3. Investigate the premise that faces are likely to be less highly stressed at ultra depth than shallower, due to stress peak moving further away as a result of intense face fracturing. Does this make mining easier & safer? What about stability under strong ground motion? [*underground & non-linear modelling*]
4. Increasing depth => increasing closure rates => lessens effectiveness of much commonly used stope support (except backfill). What modifications to the mining system can lessen this influence? [*MKR*]
5. Pillar support systems, squat, stabilizing, crush - can changes be made to make these better at depth (rock mechanics, practical & economic considerations)? [*Improve pillar modelling, underground...*]
6. Serious consideration needs to be given to improving the layout and support of off reef excavations (large chambers, haulages...) for both static and dynamic conditions. [*Follow on, higher stope width overstope, backfill stress regeneration, limiting off reef development, use of on-reef access?*]
7. "ledging" cut or "T" shaped sidewall protection for tunnels [*underground & non-linear modelling, practical assessment*]
8. Negotiating geological features on & off reef! [*Collate current experience, bracket pillars (DAH), better tunnel support (LW), triggering & conditioning (NL), improved criteria - back analysis*]
9. Use of high stress foams to supplement backfill? [*contract, underground, non-linear modelling*]
10. Investigate means of protecting rock passes at ultra depth. Considerable difficulty with spalling & self mining is reported. [*optimal layout modelling, survey, lining...*]
11. Combining classic rock mechanics criteria with improving seismic observations. [*SMS, Mendecki*]
12. Where are we with seismic predictions? AJSS alert alarm scam?
13. Mine layouts incorporating pillars and fill? Often suggested, not thoroughly investigated. [*underground & non-linear modelling*]
14. Modifications to mining method & layouts to reduce the workforce in identified dangerous areas.
15. What enables a certain mining method to "work" well in some high stress environments and not others. [*Identify geological & other factors - generalise so that method can be matched to the conditions. e.g. Caving.*]
16. Optimum stope sequencing in fault/dyke areas?
17. Remnants - how to identify the remnants that cause trouble? Support systems used to good effect. [*Survey accident stats., summarise current knowledge for remnant mining - positive & negative, General guidelines for mining remnants...*]
18. Wide & multi-reef mining at depth? [*Review local & overseas experiences, model*]

GAP033 - DETAILS ON WORKSHOP DISCUSSION TOPICS

Discussion topics for the GAP033 workshop, including a short motivation and description of the work indicated, is set out in order of priority within several sections. Topics to be discussed in this order, omitting the shaded ones which will be discussed at the end if time permits. The person introducing the topic is indicated.

Regional support

Good regional support is essential for deep/ultra-deep level mining as it reduces the stress at the face and seismic potential. Problems with pillar stability and the inconclusive benefit of backfill for regional support motivate further work in this area.

1. The use of crush pillars alone or in conjunction with backfill. (RAJ)

Crush pillars with a width to height ratio of between two and four will be fractured throughout, and in the post-failure range of behaviour. Despite this, crush pillars can provide very stiff support right at the face, which will limit ERR and face stresses. Their advantage over stabilizing pillars is that foundation failure is eliminated and gold locked up in pillars is substantially reduced. The high initial stiffness should provide more effective local support than backfill alone. Combinations of backfill and crush pillars should be considered.

[Both underground investigation and numerical modelling are required to quantify the behaviour of crush pillars alone and in combination with backfill, to evaluate mining layouts for both static and rockburst conditions.]

2. Improved pillar system design. (DJA)

It may be possible to modify the stabilizing pillar concept to improve its effectiveness as a regional support element with or without backfill such that failure is reduced, seismicity and seismic damage minimised and extraction percentage maximised. This is a far more open-ended motivation which includes many diverse alternatives: pre-conditioning of stabilizing pillars as they are cut to reduce unstable failure and foundation failure, and possibly enable secondary extraction; two stage mining with large pillars and concrete ribs followed by complete or partial pillar extraction; replace stabilizing pillars with concrete pillars and backfill; grid pillar variations as an alternative to scattered mining in areas of highly variable grade... a few of many that could be pursued.

[Initially concentrate on "lateral thinking" related to pillar layouts including rather "radical" alternatives and their impact on practical mining, rock mechanics and economics.]

3. Stabilizing pillar/backfill combinations. (RAJ)

Combinations of stabilizing pillars and backfill are currently being used effectively for regional support at depth on several mines. Can suitably designed and placed backfill be effective in eliminating, or at least reducing the incidence of pillar foundation failure? This would reduce ERR, seismicity and damage related to such failure.

[This work would complement current research on stabilizing pillars, and may involve modelling (initially) and if indicated, underground trials to confirm modelling results. Work is also needed to make such pillar/fill combinations suitable for practical mining.]

4. Wide reef mining at depth. (DJA)

Several of the proposed new deep level mines have wide tabular deposits, or several reefs suitable for mining as a package. VCR on some mines has areas of wide reef, and in some instances a 2m cut is taken and the rest left behind because of mining difficulties. High stope widths will reduce the effective rockmass strength and this could cause problems as stresses increase. Multiple cut, and cut and fill methods are indicated, though these haven't been tried at such high stress levels.

[Review overseas experiences of similar stope widths. Try to infer areas that could give problems at higher stress levels.]

Layouts to minimise adverse seismicity.

Current trends of increasing seismicity with increased mining depth are cause for concern relating to several areas including safety of workers and costs associated with rehabilitation of seismic related damage.

1. Negotiating faults and dykes. (RAJ)

Most of the deep/ultra deep level mines have many faults and dykes which are seismically active. As depths and stresses increase, seismicity associated with geological structures is expected to increase. How should one best negotiate such features on and off the reef? Not all geological structures are problematic - how can one identify "safe" geology in advance? Are there techniques which one can use to locate such features more accurately without advance development? Can a re-think of mine layouts offer improvements? e.g. Mine away from an identified hazard. Are bracket pillars still the answer?

[Collate current knowledge, improve criteria, bracket pillars... D.A. Hemp, design tunnel support suitable for traversing faults and dykes (L. Wojno), pre-conditioning (N. Lightfoot), design and evaluate alternate layouts.]

2. Design Criteria. (DJA)

Most current rock mechanics design criteria are based on stress, rock properties and empirical knowledge. Improved seismic systems are more widely available and cover all deep/ultra-deep mining. More advanced analysis techniques allow one to characterise seismicity and relate it to a particular type of event, failure mechanism or phenomenon. Can one couple such seismic indicators to "classic" rock mechanics criteria to give more reliable results? Perhaps this is best illustrated by example: ERR is commonly used to give a measure of risk in mining a remnant; the seismic 'b' value may indicate when a remnant is fractured through, effectively de-stressed, and could be mined with little chance of bursting. Can a combination of these be developed to give a more reliable day-to-day measure of the risk in mining a remnant? What progress with alert, alarm, scram system? Are there any pre-cursive signals that could be used when mining in dangerous area? Is it possible that these may apply only to subset of seismic problems? It may also be possible to turn the "prediction of hazard" idea around, to rather predict when a hazard is no longer a problem? One could think of this applied to remnant mining, developing through a fault... Should seismic "predictor" work be pursued?

[Seismologists and rock engineers to work together on improving criteria, which need be assessed by back analysis. Can this "safety indicator" be applied even to a very limited subset of mining situations to which more can be added as techniques improve?]

3. Mine layouts for fault/dyke intersections (RAJ)

A number of mines particularly in Klerksdorp and OFS at deeper levels experience difficulty in mining remnants or isolated blocks of ground adjacent to the intersection of a fault and a dyke. Such intersections seem to "lock each other up", and may give rise to large events during mining. Without knowing which is the more hazardous structure, which way does one orientate the mining? Is there any way of identifying which is the more dangerous? Can mine planning be used to minimise the formation of such remnants?

[Back analyse using seismic records and numerical modelling. Re-think layout alternatives both for such isolated blocks, to minimise formation of such blocks, and to position final remnants away from such areas.]

4. How to identify dangerous remnants. (DJA)

It is clear over most mining districts and mines over a number of years that remnant bursting is a problem. Some work has been done over the years to relate such bursting to the position, size, shape, stress, direction & rate of mining, .. with limited success. What allows one remnant to be mined without difficulty, while a nearby remnant in seemingly similar conditions causes considerable problems? Which are the controlling factors? Relatively low accident statistics related to remnants indicates special area support applied is generally suitable. Are there seismic parameters that can indicate when a remnant is safe to mine? What is the influence of geology - adjacent faults and dykes - on remnant mining and stability? When should one consider pre-conditioning?

[Survey accident statistics related to remnants. Summarise current knowledge both positive and negative related to remnant extraction. Back analyse a number of remnants using seismic records and numerical modelling assessing possible criteria such as 'b-value' (MSc student). Summarise into guidelines for remnant extraction.]

5. Longwall remnants (RAJ)

As mining depths increase, the longwall method is preferred. While this reduces the number of remnants, a number of problems with longwall remnants are reported. Approaching longwalls form long thin remnants having high stress, and deteriorating mining conditions, and concentrations of seismicity in face areas.

[Assess current layouts against practical alternatives varying panel lengths, leads and lags, overhand or underhand, overall span and stope support - most favourable alternatives. Similar to 2 above.]

6. Factors affecting mining (DJA)

A mining method and layout may work well in one deep level, high stress situation, yet may work very poorly in another. There are several examples of this in current deep mines. Does this reflect only subtle differences in layout and geology? While it is possible that this simply confirms the need to identify and follow a mining method or layout suited to the local conditions, it may be that there are inherent problems associated with a method and that certain circumstances moderate the problems inherent in the method. Perhaps this is best illustrated by an example. One mine has considerable difficulty with pillar foundation failure, while another has no problems at similar depth. Does this simply reflect differences in geology, layout, extraction ratio ... between the two, or is it that pillars are unsuitable for mining at much greater depths, and that local conditions on one mine simply reinforce this?

[This is a wide topic, requiring an understanding of what factors influence... Current work on stabilizing pillars and bracket pillars will provide a better understanding for current mining depths, which can then be translated to deeper conditions.]

Off-reef excavations.

The safety of workers while maintaining mining operations depend on the long term stability of a number of off-reef excavations including: the shaft complex, rock passes, pump & hoist chambers, haulages and cross cuts. Despite the fact that many of these are protected from the immediate effects of stress by suitable mining layouts, many are damaged through secondary stress regeneration or by seismicity. As mining depths increase, increased stress levels will occur accompanied by increased seismicity making off-reef structures more susceptible to damage.

1. Shaped haulages (RAJ)

Several authors have motivated the shaping of haulages in high stress/weak rock areas to accommodate local jointing as well as to assume a more stable shape from a stress concentration point of view. Elliptical, "T" shapes and ledging the sidewall to move stress concentrations further from the edge of the tunnel. Is this worth doing? How wide should one ledge? What support should be used? What is the effect of shaping under dynamic load? Can one integrate such a shaping or ledging to be concurrent with primary development?

[Initial survey of mines that have tried shaped tunnels. Numerical modelling and perhaps field trials of "difficult" tunnels.]

2. Minimise off reef development (DJA)

There may be advantages from both an economic as well as a safety viewpoint to modify mining methods and layouts to minimise the amount of off-reef development. e.g. Have more on-reef access.

[Lateral thinking around alternate layouts & assessment of those with promise.]

Protection of large excavations (RAJ)

At ultra-depth large excavations can no longer be protected by the shaft pillar as pre-extraction of the reef area is essential. Initially these will be de-stressed, and will be subject to increasing stresses which may eventually approach virgin stress levels. (At 4000 m depth this may exceed 100 MPa.) Support needs to accommodate such stress increases. Care needs to be exercised in placing these if satellite pillars, stabilizing pillars or crush pillars are used. It is possible to mine selected reef areas at a higher stoping width to protect important long-term excavations including the shaft itself from the full increase in stress levels. Several mines at depth report severe damage to rock-pass systems, with damage evident even where shaft reef area has been pre-extracted and with limited mining. Scattered mining will have to be used at depth in cases of very faulted, geologically complex ground, and where average grades don't warrant complete

extraction. Of concern here is to develop suitable layouts for follow-on or on-reef haulages for use with scattered mining as advance development is likely to be severely damaged and hazardous unless sited far in the footwall which may still be difficult to support and bring practical mining difficulties.

[Investigate and summarise support and layout principles required to supplement work done by McKinnon. Study rock pass location, size, shape, support ... and the possibility of using linings to reduce damage. Survey deep-level passes, location, depth, geology,... to assess the extent of the problem.]

4. Improve haulage and cross-cut layouts. (DJA)

Despite the fact that haulages are sited in distressed ground in longwall mining, improvements in haulage layout may be possible and offer advantages. How far below the reef should haulages be located? Too close - in heavily fractured ground. Too far - either influenced by abutment stress or cause practical difficulties being too far behind the face. How far should haulages be from stabilizing pillars to avoid damage? How should faults and dykes be negotiated? Parallel to geology - how far away? How should these be supported in such areas?

[Modelling + summary of experiences of current layouts at depth can be combined into guidelines to use as a starting point for design.]

Local support.

Local support requirements are generally well defined, and many suitable support elements are available to satisfy each application. Local support remains of high priority because of the concentration of workers in the stope face area and because most rockburst/rockfall related accidents occur in this area. Several remaining problem areas as well as promising support alternatives have been identified and are set out below.

1. Improved aerial support at the face. (RAJ)

"Spray on" supports have a very high shear strength & may assist in stabilizing the hangingwall, providing improved aerial coverage in the face area. Could this be applied to the face itself in areas prone to face bursting? Short end anchored bolts are being used to supplement face support to good effect on some mines - should this be investigated with possible application in deep level & high stress areas? How do face support requirements change in wide reef deep level conditions?

[These are new techniques and need to be integrated into current stope support program, or possibly done by sub-contract.]

2. Under-cutting a brow. (DJA)

The need to recover from a lost hanging, for under-cutting a brow to re-establish a competent roof, is relatively common in some areas. These overbreak conditions have a negative impact on stope stability & safety, production and economics. Most mines have ways to recover the hanging - though these seem poorly documented. At greater depths these problems may be more frequent, and more difficult to rectify. Several deep mines are using temporary or permanent crush pillars, or variations of hydraulic prop systems, for this with good results. How much of a problem is this?

[Survey current hangingwall recovery procedures for deep level mines and effectiveness of such methods. Document as guidelines applicable to differing conditions. Can improved support and mining reduce the incidence of such overbreak?.]

3. Reduce stope workforce. (RAJ)

Is it possible to make changes to the mining method or layout that would reduce the workforce required in identified dangerous areas such as the stope face, the gully intersections and the gully/face area? Water jet, vacuum cleaning and diamond-wire cutting are such examples.

[How can such dangerous work be minimised? Can such work be done remotely? Can one minimise the time in the dangerous area? Can such work be performed more safely, or done in another safer way? Mechanical mining, diamond wire saw, remote drilling rigs are examples.]

4. Support suited to very rapid closure rates. (DJA)

As mining depths increase, so too do closure rates, which may be as high as 100 mm per day at around 4000 m depth. Such rapid closure rates quickly reduce the amount of energy rapid yield hydraulic props (RYHP) and most stope support can absorb under dynamic load. ERR and economic considerations demand that the stope width be kept to a minimum compounding the problem. Several mines report damage to props and units being "locked in" by rapid closures at current depths. Such rapid closure reduces the effectiveness of most stope support with the exception of backfill. Can changes to the mining system, layout, support strategy or support units themselves minimise the negative effects of such closure rates, or use them to advantage?

[M.K.Roberts, crush pillars? small areas of silicated fill,... What are the effects of such changes to mining and on the overall support?]

5. Use of high stress foam. (RAJ)

The effectiveness of backfill initially as a local support and ultimately for regional stability is dependent on many factors such as quality, density, fill/face distance, closure rate... as well as fill height in the stope and slumping/shrinkage. It may be possible to address the fill height in the stope and slumping/shrinkage by pumping in a suitable high stress cementitious foam (1MPa) between the fill and the hangingwall.

[Testing of foams for laboratory strength and other parameters. Modelling to assess the influence of the foam on the stope. If promising - take further with a collaborator.]

REGIONAL SUPPORT

Good regional support is essential for deep/ultra deep level mining as it reduces the stress at the face and seismic potential. Problems with pillar stability and the inconclusive benefits of backfill for regional support motivate further work in this area.

1 The use of crush pillar alone or in conjunction with backfill (RAJ)

Crush pillars with a width to height ratio of between two and four will be fractured throughout, and in the post-failure range of behaviour. Despite this, crush pillars can provide a very stiff support right at the face, which will limit ERR and face stresses. The advantage over stabilizing pillars is that foundation failure is eliminated and gold locked up in pillars is substantially reduced. The initial high stiffness should provide more effective local support than backfill alone. Combinations of backfill and crush pillars should be considered.

[Both underground investigation and numerical modelling are required to quantify the behaviour of crush pillars alone and in combination with backfill, to evaluate mining layouts for both static and rockburst conditions.]

LATERAL THOUGHT - IDEAS ...

- 1 Use existing platinum mines type configuration with a strike crush pillar below the gully or somewhere in the stope and fill either side of it. This seems to eliminate the face burst problem.
- 2 Existing regional pillars could be preconditioned and surrounded by backfill to get rid of foundation failure and seismicity problems.
- 3 Crush pillars below gully and fill below that. May need pre-conditioning at the face.
- 4 Strongly dependant on relative post-failure properties of the hangingwall and pillar material.
- 5 Post-infuse "soft backfill with accelerator (CaCl₂) locally to stiffen selected areas away from gully sides.

POSITIVES

- 1 Could aid local support, reduce convergence and could eliminate regional pillars.
- 2 If you had sufficient crush pillars in the backfill you would create an extremely stiff backfill which is right at the face and be a major regional support. You may be able to eliminate regional pillars.

- 3 With high stoping widths of 3-4 m existing regional support pillars don't work. Crush pillars to reduce convergence may be suitable for high stope width areas.
- 4 Could delay closure but would need to be proved.
- 5 Potential to reduce face busting.

NEGATIVES

- 1 Crush pillar design is not a science but an art, so we could not forecast behaviour of the pillars.
- 2 Crush pillar needs to be designed "just right" or could contribute to face bursting/seismicity on face.
- 3 May not limit volumetric closure.
- 4 Not a regional support on its own.
- 5 Could influence stability of access ways and lead to possible damage to gullies.
- 6 The mining system could prove to be difficult.
- 7 Position in panel
 - below gully and backfill ~ cleaning?
 - in panel ~ cleaning and blasting.
 - paddock backfilling?

POTENTIAL IMPACT

- 1 Not great.
- 2 The need for regional pillars would be greatly reduced and there would be reduced foundation failure. Would there also be reduced extraction?
- 3 If they could be used instead of regional pillars you would solve the problem of crossing regional pillars with footwall development, because stress effects of a crush pillar dissipate very rapidly.

INDICATED RESEARCH

- 1 Accurate design methods and parameters for designing crush pillars by means of numerical modelling would be required. General rules of thumb would be no use in this application. Modelling parameters required.
- 2 Understand crush pillar behaviour with/without backfill
 - field study and measurements
 - cable system proposed to measure rapid expansion.
- 3 It would be necessary to decide how it would affect the extraction rates.
- 4 Detailed field surveys, some theoretical work and limited laboratory study would be required.
- 5 The post failure behaviour of pillars would need to be studied as well as design criteria.
- 6 Dr Spearing said that studies at Randfontein on a 3 m wide crush pillar indicated a residual strength of 20 MPa. Dr Gürtunca suggested that the backfill be strengthened by leaving crush pillars inside the backfill.

PRIORITY

- 1 Priority ratings of two and nine were given.

2. Improved pillar system design. (DJA)

It may be possible to modify the stabilizing pillar concept to improve its effectiveness as a regional support element with or without backfill such that failure is reduced, seismicity and seismic damage minimised and extraction percentage maximised. This is a far more open-ended motivation which includes many diverse alternatives: pre-conditioning of stabilizing pillars as they are cut to reduce unstable failure and foundation failure, and possibly enable secondary extraction; two stage mining with large pillars and concrete ribs followed by complete or partial pillar extraction; replace stabilizing pillars with concrete pillars and backfill; grid pillar variations as an alternative to scattered mining in areas of highly variable grade... a few of many that could be pursued.

[Initially concentrate on "lateral thinking" related to pillar layouts including rather "radical" alternatives and their impact on practical mining, rock mechanics and economics.]

LATERAL THOUGHT - IDEAS ...

- 1 A lot depends on the strengths and properties of the pillar, the hanging and footwall. As a lateral idea, drift and fill methods with cemented backfill could be considered. This assumes a soft footwall.
- 2 5:1 squat pillars preconditioned with ionic water infusion; this is a long-term effect.
- 3 Sequential grid mining techniques at Elandsrand moving away from shaft pillar using dip pillars and fill instead of a scattered mining approach.
- 4 We should apply current technology instead of just discussing it. Time and production constraints!
- 5 Take panel up- and down dip of pillar at very low stope width.
- 6 Change mine methods: large leads and lags used to engineer concrete pillars in a low closure environment.
- 7 A way should be found of surrounding regional pillars with backfill on both sides, or on the down dip side.
- 8 Reassess pillar dimensions and try to reduce them.
- 9 Pack blocks of concrete fill around pillars.

POSITIVES

- 1 As there seems to be no direct relationship between ERR and seismicity in the Klerksdorp area, this is a positive benefit because although ERR increases with depth it does not follow that seismicity will automatically increase as well.
- 2 High extraction ratio.
- 3 Reduced foundation failure.
- 4 The benefits from the pillars become apparent immediately.
- 5 Improved safety.

NEGATIVES

- 1 Negotiating faults will be a major problem.
- 2 Ventilation control could be a problem.
- 3 A >2 m stope could be a problem.
- 4 Complex mining sequence.
- 5 Stabilizing pillars are probably not applicable at great depths.
- 6 Foundation failure and the lock up of a large volume of reef is a negative aspect.

POTENTIAL IMPACT

- 1 Enabled mining at great depth.
- 2 It could reduce the magnitude of large events.
- 3 It could reduce seismicity and the size of seismic events.
- 4 It could be possible to tailor the ERR to the required figure.

- 5 Improve safety.

INDICATED RESEARCH

- 1 Non-linear modelling would be a required.
- 2 It would be necessary to study squat pillar behaviour and friction properties.
- 3 It would be necessary to look at design parameters and define them precisely taking note of geotechnical factors. Gain better understanding of squat pillar behaviour.
- 4 There would be a need to look at replacement options for pillars - concrete was suggested.
- 5 Investigate lead/lag concept for concrete.
- 6 Mr Kersten emphasized the need for non-linear modelling research for Ultra Depth Studies, particularly as we already had insufficient knowledge of this subject for normal depth.
- 7 In response to a question Dr Grtunca said that Wits University had been asked to study the relationship between ERR/ESS and seismic activity. Wits University had shown a reluctance to undertake the task this year so there was a possibility that CSIR Miningtek would do it.

PRIORITY

- 1 Priority range was from seven to ten.

Layouts to minimise adverse seismicity.

Current trends of increasing seismicity with increased mining depth are cause for concern relating to several areas including safety of workers and costs associated with rehabilitation of seismic related damage.

1. Negotiating faults and dykes. (RAJ)

Most of the deep/ultra deep level mines have many faults and dykes which are seismically active. As depths and stresses increase, seismicity associated with geological structures is expected to increase. How should one best negotiate such features on and off the reef? Not all geological structures are problematic - how can one identify "safe" geology in advance? Are there techniques which one can use to locate such features more accurately without advance development? Can a re-think of mine layouts offer improvements? e.g. Mine away from an identified hazard. Are bracket pillars still the answer?

[Collate current knowledge, improve criteria, bracket pillars... D.A. Hemp, design tunnel support suitable for traversing faults and dykes (L. Wojno), pre-conditioning (N. Lightfoot), design and evaluate alternate layouts.]

LATERAL THOUGHTS - IDEAS ...

- 1 The most lateral thought was to suggest gluing or fusing the fault with a suitable (unidentified) compound (increase cohesion).
- 2 The injection of ionic fluids in the hope of having long term effects. Pillars on faults for short term effect.
- 3 Strike pillars at right angles to the face could be tried.
- 4 Use GPR to assist in undertaking an intense geological survey ahead of the face.
- 5 Sequential grid mining could be tried but one would need to know of major faults in advance by using GPR, long drilling probes, seismic tomography or advance development.
- 6 Change to standard longwall systemt.
- 7 Use crush pillars next to features designed to crush at a given distance of mining from the feature.
- 8 Stope away from the feature.

- 9 Use face shape index -> develop this idea further

POSITIVES

- 1 Knowing the structure of a mine in advance will offer safer conditions and lead to higher extraction rates.
- 2 Always reduce stress transfer onto difficult areas being mined.
- 3 Mining away from a fault is a cheap method of triggering/controlled fault slip.
- 4 Safety is automatically improved if you mine away from a fault.
- 5 Can improve mine design - optimise.
- 6 Stabilization of geology.

NEGATIVES

- 1 There are no precise methods of indicating features ahead of the face, so intensive studies will be required.
- 2 The procedures suggested are expensive and demand expenditure before obtaining returns.
- 3 Off-reef development has to be secure - increases costs?
- 4 Pre-supposes flexibility of mining system.
- 5 Sequential grid at 4 km?

POTENTIAL IMPACT

- 1 Hazards could be reduced; pre-define hazardous features.
- 2 Good for safety, good for finance.

- 3 Optimize mining sequence with respect to structures.

INDICATED RESEARCH

- 1 How to delineate structures in advance and measure properties at a distance?
- 2 It will be necessary to initiate friction angle study. Is friction angle and strength a function of the throw of a fault? How does slip actually take place?
- 3 GPR will be essential together with numerical modelling, ERR and ESS studies.
- 4 More research will be needed in the bracket pillar studies.
- 5 It will be necessary to model bracket pillars and to decide what really happens if you mine away from the fault.
- 6 It will be necessary to study the physical characteristics of the faults, dykes and the tectonic environment.
- 7 Studies on the way the energy is contained within the rock or structure should be initiated.
- 8 How far is stress transferred in practise
- 9 Not everybody was convinced that mining away from a fault produced the desired effects and it was suggested that a case study was needed to resolve the question.

PRIORITY

- 1 The priority figure ranged from seven to ten.

2. Design Criteria. (DJA)

Most current rock mechanics design criteria are based on stress, rock properties and empirical knowledge. Improved seismic systems are more widely available and cover all deep/ultra-deep mining. More advanced analysis techniques allow one to characterise seismicity and relate it to a particular type of event, failure mechanism or phenomenon. Can one couple such seismic indicators to "classic" rock mechanics criteria to give more reliable results? Perhaps this is best illustrated by example: ERR is commonly used to give a measure of risk in mining a remnant; the seismic 'b' value may indicate when a remnant is fractured through, effectively de-stressed, and could be mined with little chance of bursting. Can a combination of these be developed to give a more reliable day-to-day measure of the risk in mining a remnant? What progress with alert, alarm, scram system? Are there any pre-cursive signals that could be used when mining in dangerous area? Is it possible that these may apply only to subset of seismic problems? It may also be possible to turn the "prediction of hazard" idea around, to rather predict when a hazard is no longer a problem? One could think of this applied to remnant mining, developing through a fault... Should seismic "predictor" work be pursued?

[Seismologists and rock engineers to work together on improving criteria, which need be assessed by back analysis. Can this "safety indicator" be applied even to a very limited subset of mining situations to which more can be added as techniques improve?]

LATERAL THOUGHT - IDEAS ...

- 1 Surface tomographic studies and GPR could be tried in an attempt to measure high stress areas and rock quality.
- 2 Energy measurements (by seismic system) on faults and dykes should be attempted in order to assess their hazards potential.
- 3 Use of velocity to determine areas of high stress.
- 4 Criteria for risk management model (factors to incorporate in model) to be specified.

POSITIVES

- 1 Integrate MINSIM/MS with actual closure rates and seismic system, to identify high risk areas.
- 2 The measurement of creep and stress on structure could be included.
- 3 Design accordingly - better knowledge of orebody -> improve safety.
- 4 Can seismic system include probability of damaging event?

- 5 Identify dangerous areas and trends. Positively identify high risk areas.

NEGATIVES

- 1 Don't be too optimistic! A long way to go!
- 2 We do not have a thorough understanding of the problems and more research is needed.
- 3 There are disadvantages of the "SCRAM" alert because the workforce will not enter the area if the figure is high.
- 4 Validity of ERR and hazard measure?

POTENTIAL IMPACT

- 1 Improved safety.
- 2 A system that could quantify the risk situation for management has great potential in that it could be used to plan the direction of exploration.
- 3 Design according to pre-knowledge of ore-body.

INDICATED RESEARCH

- 1 Some of indicated work being done under SIMRAC at present.
- 2 Creep, ride and closure should be monitored and studied after an event has taken place.
- 3 Research seismic parameters and usage.
- 4 Dr Grtunca said that the "ALERT and "ALARM" systems were receiving favourable publicity but in reality very little progress had been made in seismic prediction. General "NO" to SCRAM".

PRIORITY

- 1 The range of priority was from six to nine.

Off-reef excavations.

The safety of workers while maintaining mining operations depend on the long term stability of a number of off-reef excavations including: the shaft complex, rock passes, pump & hoist chambers, haulages and cross cuts. Despite the fact that many of these are protected from the immediate effects of stress by suitable mining layouts, many are damaged through secondary stress regeneration or by seismicity. As mining depths increase, increased stress levels will occur accompanied by increased seismicity making off-reef structures more susceptible to damage.

1. Shaped haulages. (RAJ)

Several authors have motivated the shaping of haulages in high stress/weak rock areas to accommodate local jointing as well as to assume a more stable shape from a stress concentration point of view. Elliptical, "T" shapes and ledging the sidewall to move stress concentrations further from the edge of the tunnel. Is this worth doing? How wide should one ledge? What support should be used? What is the effect of shaping under dynamic load? Can one integrate such a shaping or ledging to be concurrent with primary development?

{Initial survey of mines that have tried shaped tunnels. Numerical modelling and perhaps field trials of "difficult" tunnels.}

LATERAL THOUGHT - IDEAS ...

- 1 The idea of the profiled tunnel has already been done successfully.
- 2 A sacrificial tunnel can be developed near to the main tunnel. It was suggested that this idea had been found to work.
- 3 Locate tunnel so maximum anticipated ground velocity along tunnel length.
- 4 Tunnel borers or mechanical means of excavating tunnels should be considered.
- 5 Design lifetime of tunnel according to need.
- 6 Create fractured zone around tunnel - precondition prior to development.

POSITIVES

- 1 "T" or elliptical tunnels are known to be more stable but for the greatest effect the shape must be inclined in the direction of the maximum stress. May not "fit" geology or bedding.

- 2 Shaped tunnels have proved to be stable and the support cost is reduced. Safer in seismic event.

NEGATIVES

- 1 The "T" shape is not completely satisfactory and the preferred shape is the ellipse, however, the production of an elliptical footwall is difficult.
- 2 It is difficult to develop the shape and the alignment in the direction of maximum stress is difficult to achieve in practice.
- 3 The haulages are slow to develop.

POTENTIAL IMPACT

- 1 Shaped haulages can be placed in high stressed areas.
- 2 Reduce support costs and increase stability.
- 3 Maintain access ways ahead of the face in high stress/deep level.

INDICATED RESEARCH

- 1 Studies would be required to determine the optimum positioning of a shaped tunnel and what support it would require. Non-linear modelling would be essential.
- 2 Research appropriate tunnel shape for specific areas.
- 3 Monitor existing high stress tunnels.
- 4 Mechanics of dynamic tunnel failure?
- 5 How good is natural breakout profile?
- 6 Dr Gürtunca said that we had sufficient information to start modelling investigations immediately.

PRIORITY

- 1 A priority rating of seven to eight was given.

2. Minimise off reef development. (DJA)

There may be advantages from both an economic as well as a safety viewpoint to modify mining methods and layouts to minimise the amount of off-reef development. e.g. Have more on-reef access.

[Lateral thinking around alternate layouts & assessment of those with promise.]

LATERAL THOUGHT - IDEAS ...

- 1 It may be worth considering the bord and pillar approach used in coal mining. (squat/crush pillars)
- 2 Use of a continuous scraper should be considered, as well as more efficient material/ore transport in general.
- 3 Mr Johnson presented a technique using raise-boring on reef with linked tunnels.
- 4 Develop smaller tunnels.
- 5 In high stope width - use 90% on reef! Why not in low stope width areas?

POSITIVES

- 1 On-reef development could lend itself to mechanization possibilities.
- 2 On-reef development could generate less waste rock.
- 3 Eliminate traversing stress peaks; reduce high stress tunnels in solid.
- 4 Evaluation of ore/geology as you develop.

NEGATIVES

- 1 Stope closure, negotiating faults and the rolling of the reef would cause severe problems and could require new mining methods.
- 2 A much greater use of backfill would be required to control closure plus a detailed knowledge of the local geology.

- 3 All development would need to negotiate faults - need to know geology much better.
- 4 Transport to and from the stope could become a severe problem because of the need to follow the reef.
- 5 Alternate access difficult.
- 6 Lack of ore storage.

POTENTIAL IMPACT

- 1 Off-reef development would be minimized.
- 2 Hydraulic transport of the ore may well become a necessity.
- 3 This method of mining may be less flexible than the present approach.
- 4 Safer, cheaper and less waste rock.

INDICATED RESEARCH

- 1 3-D non-linear modelling would be required, including the studies of strain softening and dynamic creep.
- 2 Alternate ore transport methods. Belts
- 3 Study past and present experiences.
- 4 Best balance of on and off reef access.
- 5 Can we keep a tunnel open at great depth?

PRIORITY

- 1 A priority from four to six was given for this task.

Appendix 4.

AAC SUB-CONTRACT

A possible stoping method and support system for ultra-deep tabular reefs.

SYNOPSIS OF REPORT

“A POSSIBLE STOPPING METHOD AND SUPPORT SYSTEMS FOR TABULAR
GOLD MINING AT ULTRA - DEPTH”

BY
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GAP033

JULY 1993

1. **INTRODUCTION**

It is known from geological drilling on the Witwatersrand Basin that relatively high grade gold orebodies exist at ultra-depth (over 4 000m below surface). Whether these deposits can ever be economically and safely exploited is much in debate at present, due mainly to:-

- i) The unacceptable rock related accident rate at current mining depths (safety).
- ii) Rock mechanics consideration in general.
- iii) Ventilation considerations.
- iv) A basically constant gold price in a relatively high inflation rate economy.

Stope support on macro (regional support) and micro scales (face, gully and internal support) is the most vital consideration for successful mining at ultra-depth, in addition to an effective rockburst management system.

2. **CURRENT METHODS**

The current stope support systems were initially studied in this report to investigate the success of such systems and their general applicability at ultra-depth. It was found that :-

- I) The common regional support on deep mines, involving the leaving of in situ strike stabilising pillars, has not been very successful and would not be appropriate at ultra-depth due also to financial constraints as outlined in table1. This is due mainly too sismicity associated with pillar foundation failure, and the relatively low overall extraction rate that would be achieved.

**TABLE 1 : THE FINANCIAL EFFECTS OF LEAVING IN SITU REEF
PILLARS ON A SPECIFIC GOLD MINE**

% PILLARS	TOTAL PRE-TAX PROFIT (1992 MONEY TERMS) R_x 10⁶
0	2767
5	2709
10	2656
15	2604
20	2513
25	2338
30	2146

ii) Timber packs as internal support are generally adequate at current mining depths but would not be adequate or cost effective at ultra-depth. Backfill would provide a better support systems that could also act as a regional support, and effectively reduce the heat ingress into the working (thus reducing the quantity of cooling needed).

iii) Face support is generally too far from the stope face at present and is sometimes inadequate. This can be seen by the unsatisfactory safety record of the gold mining industry, at the stope face in particular (where the bulk of the rock related fatalities occur). This implies that the current systems in use will be totally inadequate at ultra-depth due to the probable increase in general seismicity. The use of diagonal blast barricades and water jet assisted scraping would be necessary at ultra-depth to ensure that the rapid yield hydraulic props (with headboards) could be installed and maintained close to the advancing stope face.

iv) The general areal coverage at the face currently is also a cause of concern and would be totally inadequate at ultra-depth, and hence the need for correctly designed long axis headboards on props in the stope face area.

v) The gully shoulder support consisting usually of long axis packs is generally acceptable at present mining depths but may not be the most cost effective solution at ultra-depth. A relatively high early strength backfill appears a better support systems for gullies.

vi) The support of the gully hangingwall at present needs to be improved and current standards employed will be totally inadequate at ultra-depth.

This review of current stope support systems also identified a need for a standard method of calculating support resistance and such a method is proposed by the author in the report. The need for this arises because the performance of different support systems need to be compared, and when back analysing previous fatalities or accidents, the performance of the nearby support elements must be considered

3. **A POSSIBLE MINING METHOD AND SUPPORT STRATEGY AT ULTRA-DEPTH**

The investigation subsequently undertaken focused on the regional (and stope) support and made extensive use of elastic and inelastic rock mechanics computer codes. In all cases the results obtained were compared qualitatively and tested against underground observations. It is very difficult, and often misleading to use the results from such computer analyses in a quantitative manner due mainly to the nature of such codes and the rockmass input data that varies considerable underground and is difficult to estimate reliably. In addition, energy release rates were compared, as currently, this parameter is extensively used in the industry as limiting design parameter.

The use of backfill was identified as a major component in any ultra-depth mine. Silicated backfill was found to be the most performance and cost effective fill type, supplemented by concrete fill ribs (at relatively high stoping widths only). A feasible conceptual distribution systems has been designed with the necessary infrastructure. The silicated backfill system has been already introduced on some deep mines and the results to date have been encouraging.

The extensive use of bracket pillars on adverse geological features was also identified as a necessary from of regional support in addition to the backfill. The need for bracket pillars would be identified mainly by past experience with the same feature (if applicable) and /or the use of a sophisticated and robust digital seismic system. Not all major geological features would require bracket pillars, as some could be mined through, and other triggered using blasting. In this regard, the role of the seismic system and the interpretation of the data from the system becomes very important.

The design of such pillars requires more research and depends on the nature and properties of the feature and the surrounding rock mass.

The optimum gully shoulder support was found to be silicated backfill supplemented by an active elongate type of support. This has also been tested on a deep mine and found to be cost effective and safe at relatively high stoping width.

In high stoping width areas at ultra-depth, a double cut mining method must be adopted. This is where a narrow cut is taken on the top reef contact and the remainder of the reef extracted in a destressed environment behind the face using trenching. This not only makes the face support more effective but also limits the number and extent of hazardous face bursts.

Where the stoping width suddenly increases due to a massive fall of ground, a more efficient, safer and cheaper option of re-establishing the stope panel has also been proposed by the author, ass the traditional method of updipping in front of the collapsed breast panel, with a wide end raise, can be relatively hazardous and time consuming.

Finally, cost comparisons were undertaken to compare the support cost of the method proposed and those commonly in use on deep mines at present. It was found that the costs were similar as shown in table 2, and hence there was no reason for support costs at ultra-depth to be significantly higher than those on deep mines. This could be achieved by moving the emphasis away from expensive and labour intensive pack based systems, to backfill with relatively intensive face and gully support. With the use of diagonal barricades and water jet assisted cleaning, productivity and safety could be realistically improved even at ultra-depth.

TABLE 2 : TOTAL STOPE SUPPORT

SUPPORT AREA	SUPPORT COST (R/M ²)	
	TYPICAL DEEP MINE	PROPOSED ULTRA-DEEP MINE
Face Support	22,12	25,68
Internal Support	78,11	66,07
Gully Support	15,17	24,69
Total	115,40	116,44

It is therefore concluded that the rock mechanics problems of mining at ultra-depth can be successfully and cost effectively resolved.

4. FUTURE RESEARCH NEEDS

Areas were identified in the report that needed additional research :-

- i) Differential velocity measurements must be made (during seismic events) between the hangingwall and the footwall to determine an optimum rapid yield velocity for a RYHP and the need for any rebound facility in the prop. The need for rebound will also depend on the frequency of the differential movement.
- ii) Drilling techniques and equipment must be designed to allow for drilling holes in highly stressed rock and being able to charge and detonate them effectively.
- iii) The reliability of the rockburst management strategy - Alert, Alarm, Scram-needs to be tested in the field for a period of time to establish a data base of seismic event precursors (if possible).

iv) A design code for bracket pillars needs to be establishing based on local geology, practical experience in the field, inelastic computer studies and seismic data. This could also require fairly extensive in situ stress measurements near geological features and rock property specimen tests.

v) The performance of concrete pillars at high stress needs to be investigated with particular reference to whether violent foundation type failure can in fact occur (it appears not from the computer based study).

vi) Dynamic inelastic rock mechanics computer codes need to be calibrated and refined using back analyses to assist in the longer term mine layout and support design in a specific mining area.

vii) The feasibility of re-establishing a stope panel after a major fall as described in principal (using shutters and cementitious foam) needs to be tested in the field.

Finally it should be noted that the detailed report is available from Mr D Adams of Miningtek (telephone : 011-7263020) or the author (telephone : 011 6382718) should more detail be desired.

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wp5.1\ajss2.45\wlv
28 July 1993

Appendix 5.

IMPROVED SEISMIC SOFTWARE AND TECHNIQUES

Automatic location of seismic event doublets by cross correlation criteria

Planes of location in mine induced seismic clusters

AUTOMATIC LOCATION OF SEISMIC EVENT DOUBLETS

BY CROSS-CORRELATION CRITERIA

By A.M. Milev, S.M. Spottiswoode and R.D. Stewart

ABSTRACT

A procedure for automatic location of seismic event doublets on the basis of cross-correlation analysis has been developed. The procedure was implemented in the following stages: (i) cross-correlation of P- and S-wave envelopes: use the calculated peak correlation value to identify possible doublets and the time differences for fine-tuning of correlation windows; (ii) cross-correlation of P- and S-phases: determine the event doublets and attempt high resolution relative location using the time differences of individual phases. In the relocation routine, the concept of "progressive master event" is introduced. This allows the growth of "families" of similar master events.

The procedure has been tested on mining-induced seismic events recorded underground at three deep-level gold mines and one platinum mine in South Africa. A large number of the mining-induced seismic events showed a high degree of similarity and, on average, about 50 per cent of the events in the four data sets studied were relocated reliably, the success rates varying according to specific mining conditions. It was found that the location accuracy of the procedure exceeded the accuracy obtained by other location methods.

INTRODUCTION

Mining-induced seismicity is characterised by a large number of seismic events which might be observed underground. The location of each event is generally based on manually selected P- and S-wave arrival times. Automatic arrival picking is not always reliable, for the following reasons: the presence of emergent arrivals caused by high noise levels, the presence of head waves, the difficulties in S-wave identification and the superimposition of

multiple events (particularly during the blasting time). These effects often result in a significant decrease in location accuracy. On the other hand, the mining-induced seismic events are often distributed in small seismogenic volumes with relatively homogeneous velocity structures and have similar source mechanisms (Spottiswoode, 1984). These events usually generate similar waveforms and can be categorised as doublets (Poupinet et al, 1984; Frechet et al, 1989). This phenomenon can be used to enhance location accuracy by the relocation of one event relative to the other. A number of studies on seismic event doublets in terms of high-resolution location have been reported in the literature (Ito, 1985; Thorbjarnardottir and Pechmann, 1987; Stewart, 1991; Deichmann and Garcia-Fernandez, 1992; Hartwig and Joswig, 1993). This study aims to explore the method further and to provide robust automatic locations.

We present a procedure for automatic location of similar events by the application of cross-correlation criteria. The procedure starts with the computation of the envelope functions for the entire seismogram, including both P- and S-waves, and cross-correlates them in the frequency domain. The calculated correlation coefficients and time lags are used for pre-selection of possible event doublets and for precise adjustment of correlation windows. The P- and S-waves are then cross-correlated and the calculated time lags are used in the relocation procedure, which is similar to the Joint Hypocentral Determination (JHD) method (Douglas, 1967; Pujol, 1992). In the JHD method, one (subsidiary) event is located with reference to the other, previously located (master) event. The advantage of using the correlation time lags is that we do not need to know the exact arrival times of P and S phases for the relocated event.

Normally, a single master event is applicable only to a specific region around its source volume and to a particular propagation path. To characterise a more extensive region - for example, one with typical deep-level mine dimensions - more master events are required (Harris, 1991). In this study, the concept of "progressive master event" was introduced. The principle of the "progressive master" means that we consider any previously

automatically located event as a potential master, as long as its correlation with the subsidiary event is sufficiently good. This procedure was tested on a large quantity of digital seismic data reflecting various geological and mining conditions.

DATA

We used data from three deep-level gold mines and one platinum mine in South Africa. The recording networks used were the Portable Seismic Systems (Patrick et al, 1990) developed at the Chamber of Mines Research Organisation (now the CSIR Division of Mining Technology). The system operates with various configurations, specific to any particular mine area, and utilises uniaxial or triaxial geophones with digital grade sensors. The geophones are grouted to the ends of boreholes drilled about 10 m into the rock mass from tunnel sidewalls. The arrays have a 16-channel capacity, although not all of the channels produce data of sufficient quality. The bad channels were rejected from further processing on the basis of poor signal-to-noise ratio. Sampling rates range from 1000 up to 20000 samples per second, depending on the network extent. The seismic events are identified and the system triggered using short term average/long term average ratios. The signals are amplified and telemetered to the Data Acquisition Unit (DAU), where they are digitised by a 12-bit analogue-to-digital converter. The seismograms, consisting of 2048 digital values per channel, are then transmitted from the DAU - which is usually placed underground - to a PC on the surface, where they are stored for further analysis.

Descriptions of the particular mines and related seismic arrays are given below.

Mine I:

The microseismic array at Blyvooruitzicht gold mine, near Carletonville, consists of 12 geophones placed at six sites, covering an area of 200 m by 200 m at about 1900 m below

surface. The array has been designed to record mining seismicity associated with the extraction of a stabilising pillar with the aid of precondition blasting (Adams et al, 1993). In this study, we used 202 seismic events with magnitudes in the range $-2 < M < 1$.

Mine II:

The array at the East Rand Proprietary Mines consists of nine geophones grouped into eight sites, covering an area of 3000 m in dimension at depths ranging from 1800 m to 3300 m. The array monitors seismicity occurring around longwalls and stabilising pillars. The data set from the area used consisted of 240 events with magnitudes in the range $0 < M < 2.5$.

Mine III:

The Orangia shaft network on Buffelsfontein gold mine consists of 15 geophones grouped into seven sites on two different levels at about 2200 m to 2300 m below surface. The planar dimension of the network is 400 m by 500 m. The data set used for analysis consisted of 316 seismic events with magnitudes in the range $-2 < M < 2.5$.

Mine IV:

The seismic array on the Impala platinum mine consists of 15 geophones grouped into five sites, covering an area of 200 m by 200 m at a depth of about 1000 m below surface. The seismic array was installed to study seismicity associated with five-metre wide crush pillars that are used for local and regional support. The seismic data set included 108 seismic events with magnitudes in the range $-2 < M < 0$.

The four areas studied are mined by blasting many small charges, each comprising about 0.5 kg of explosive, distributed along the stope working face. Access tunnels are developed using fewer, but larger blasts. Numerous blasts are recorded by the smaller

networks and these events typically have very similar waveforms when the blast configurations are similar. Data from mines III and IV consisted mostly of recorded regular production blasts.

METHOD

As discussed above, a large number of mining-induced seismic events generate nearly identical waveforms. One example of two events recorded at Buffelsfontein gold mine is given in Figures 1(a, b). It can be seen in the figures that the two sets of seismograms are quite similar. The small differences in their waveforms should produce only a small effect in the correlation analysis. This suggests applying the cross-correlation analysis for quantitative measurement of the degree of similarity between corresponding waveforms.

Cross-Correlation Analysis

Let us consider some theoretical concepts relating to the cross-correlation technique before describing the procedure.

The correlation $corr_{gh}(\tau)$ between two signals $g(t)$ and $h(t)$ in the time domain can be expressed by:

$$corr_{gh}(\tau) = \frac{\int_{-\infty}^{\infty} g(t)h(t+\tau)dt}{\sqrt{E_g E_h}} \quad (1)$$

where

$$E_g = \int_{-\infty}^{\infty} g^2(t)dt$$

and

(2)

$$E_h = \int_{-\infty}^{\infty} h^2(t) dt$$

represent the "energy" of each signal. The correlation is a function of τ , which is called the lag. If we replace $g(t)$ and $h(t)$ by the Fourier transform in the frequency domain, we can write:

$$Corr_{GH}(f) = \frac{G(f)H^*(f)}{\sqrt{E_g E_h}} \quad (3)$$

where $G(f)$ and $H(f)$ are the corresponding Fourier transforms of $g(t)$ and $h(t)$ and $(*)$ denotes the complex conjugate. The total "energy" in the signal is the same whether we compute it in the time or frequency domain. This result is a statement of Parseval's theorem and for E_g and E_h we can write:

$$E_g = \int_{-\infty}^{\infty} G^2(f) df$$

and

$$E_h = \int_{-\infty}^{\infty} H^2(f) df \quad (4)$$

Note that the equation (3) is the Correlation Theorem in the frequency domain. Equations (4) are useful when the seismograms are filtered. Numerically, $g(t)$ and $h(t)$ are the digitised seismograms or envelopes and $G(f)$ and $H(f)$ are the corresponding numbers obtained using the Fast Fourier Transform (Press et al, 1986).

The peak position of the correlation function on the time axis is the lag time between the two waveforms. Note that the correlation function $corr_{gh}(\tau)$ has a value of 1.0 at $\tau = 0$ when $g(t) = h(t)$.

This analysis provides two very useful pieces of information: (i) the peak value of the correlation function is a measure of the similarity between the two events, and (ii) the lag

of the peak is a measure of the time difference between the two events, with reference to the recording windows.

The cross-correlation coefficients were calculated for each P- and S- wave as well as for their envelope functions. This was done in a two-stage procedure. Firstly, we determined the average correlation value for P- and S-wave envelopes over all of the active channels. Secondly, we calculated the cross-correlation coefficients for the P- and S-waves.

To evaluate the degree of similarity between two events i and j , we used the M (matching) - factor, calculated as:

$$M_{ijkl} = \sum_{k=1}^m \sum_{l=1}^n R_{ijkl}^2 \quad (5)$$

where R is the cross-correlation coefficient, k is the channel index, l is the phase index, m is the number of sites and n is the number of phases. M -factors were determined for values of R exceeding a specified cross-correlation threshold.

The length of the correlation window was chosen to incorporate a single phase correlation and was set to half the difference between the P and S onset positions on the time scale. The precise adjustment of the correlation windows for the seismograms was done by using the time lags obtained from the envelope cross-correlation. Therefore, it was not necessary to select arrivals for the second event.

Care was taken, while performing the envelope cross-correlation, to allow neither the close (large-amplitude) records nor the distant (noisy) records to dominate the results. To achieve this, the envelopes were weighted according to the square roots of their amplitudes.

Seismograms from mining-induced events typically contain most of their energy above the corner frequency (Spottiswoode, 1993), but this energy has random amplitude and phase. To eliminate this effect, the signals were filtered by a low-pass filter.

Seismic event doublets

In searching for seismic event doublets, the cross-correlation method was applied consecutively to both P and S waveforms and their envelopes. The correlation coefficients were calculated for pairs of events in the data sets by two calculation loops, where the outer loop was used for the later (subsidiary) events and the inner loop, for the earlier (possible master) events. In this scheme, each event was compared with the previous 100 events. Thus, any particular data set studied here comprised about 10 000 event pairs.

The following criteria for regarding the events with similar waveforms as doublets were established:

i) Cross-correlation cutoff value: we used a cutoff value of 0.7 for the separation of well-matched and poorly-matched seismograms and a value of 0.5 for separation of the envelopes. Similar cutoff values were reported by Pechmann and Kanamori (1982), Thorbjarnardottir and Pechmann (1987) and Israelson (1990).

(ii) Event separation distance: Geller and Mueller (1980) suggested that the optimum event separation distance for a good correlation is less than one quarter of the dominant wavelength. More recently, it has been shown that the optimum event separation distance varies in different regions up to a few times λ_{\max} , where λ_{\max} is the largest wavelength for the considered event pair (Thorbjarnardottir and Pechmann, 1987; Hutchings and Wu, 1990; Harris, 1991). We used the average wavelength obtained from the spectral peak as follows:

$$\bar{\lambda} = \frac{1}{N} \sum_{i=1}^N (V_{ci} / f_{ci}) \quad (6)$$

where N is the number of arrivals and, therefore, $N-1$ is the number of redundant arrivals available for the location, V_{ci} is the phase velocity obtained from calibration blasts and f_{ci}

is the peak in the smoothed velocity spectrum for phase i . For location error, we have used the following restriction:

$$\text{Location error} < \bar{\lambda} / 2 \quad (7)$$

To analyse the cross-correlation value as well as to decide whether or not to use a correlation peak in the relocation attempt, we included the following additional criteria: (i) the maximum in the correlation function should exceed the specified threshold maximum; (ii) the maximum in the correlation function should exceed any other peak value by at least 20 per cent; (iii) the correlation peak should be located in the central half of the window (i.e. a lead or lag of less than 25 per cent). Thus, we ensured that the correct correlation peak was used by the relocation procedure.

High Resolution Relative Location (HRRL)

We relocated all well-matched seismic events using a procedure similar to the JHD method, in which one or more "subsidiary events" are relocated relative to a previously selected "master event". In our approach, we relocated the subsidiary events with respect to the master events using the time differences obtained from the cross-correlation analysis. The P- and S-wave arrival times were corrected for the envelope lags and for the lag of each particular phase. In this way, we enhanced the accuracy of the calculation of P- and S-wave arrival times and obtained better locations. The relocated subsidiary events were rewritten into the initial data set and considered further as possible masters. The arrival times of the current master event in the hierarchical tree were calculated as:

$$\begin{aligned} \text{Arrival(latest event)} = & \text{Theoretical arrivals(original master)} + \\ & \text{Sum of time differences(over the number of generations)} \end{aligned}$$

Figure 2 shows the inter-dependence among a family of six events. The event 46, considered a basic master event, was related directly to events 48 and 52 from the first generation and indirectly to events 49, 51 and 53 from the following generations.

We call the principle of re-using autolocated events as possible new master events the principle of "progressive master events" and extend it further to the creation of a hierarchy or family of related events. It is suggested that the grouping of events in families is useful for identifying a cluster of events with similar mechanisms.

The procedure can be summarised as follows:

1. Calculate the envelope function for each event, channel and phase.
2. Loop through the 100 previous events to find the best envelope cross-correlation ($R \geq 0.5$).
3. Correct the P and S arrival times and the position of the correlation windows for the envelope lag.
4. Loop through the 100 previous events to find the best P- and S-phase cross-correlation ($R \geq 0.7$).
5. For the good fits, perform HRRL and rewrite the new locations into the initial data set.

RESULTS AND DISCUSSION

Two examples, with error analysis, of the application of this procedure to mining-induced seismic events are given in this section.

A plan view of part of Buffelsfontein gold mine and of the geometry of the geophone sites is given in Figure 3. The boxed area in the figure will be used for illustration of the relocation procedure.

Figures 4(*a*, *b*) illustrate the relocation of six development blasts recorded at Buffelsfontein gold mine. The events were located by P- and S-wave manual arrival-time picking (Figure 4*a*) and by the HRRL procedure (Figure 4*b*). The closer clustering of the blasts is clearly seen in Figure 4*b*. The blasts group in an aperture of about 3 m, which corresponds to the width of the tunnel. The spread of the locations in Figure 4*a* is due to the poor resolution capabilities of the P and S manual location method.

The spatial dependence of the cross-correlation quality factor M is shown in Figure 5 for the Buffelsfontein data set. It is easy to see that the correlation value falls with increase in the inter-event distance, in most cases, and that the higher values lie within three times $\bar{\lambda}$. A similar distance-dependency was found for the other data sets.

Figure 6 compares the RMS location errors obtained by the HRRL procedure with the errors obtained by the JHD method and the P and S manual location method. The data for this example originated from Buffelsfontein gold mine. It is clear that the errors from HRRL are markedly less than those from JHD and from P and S manual location. Note that the errors are normalised by the average hypocentral distance, which varies with the network extent. In that way, we reduced the effects of network geometry and of velocity variations on location accuracy. The distributions of errors from the other mines are very similar.

Figures 7(*a*, *b*) show an area with seismic events recorded at Blyvooruitzicht gold mine. These events were clustered into two groups: a group associated with the changes in rock mass conditions in the pillar ahead of the mining faces (the biggest group on the figure) and a group of development blasts. The locations of the events in both groups after relocation using the HRRL procedure are shown in Figure 7*b*, where the blasts' clustering is much more evident. The persistent dispersion of locations in the first group is due to the events' originating within a region of intense fracturing occupying the width of the pillar for some distance ahead of the mining faces.

We obtained the following percentages of successfully relocated events: 38 per cent from Blyvooruitzicht gold mine, 27 per cent from East Rand Proprietary Mines, 63 per cent from Buffelsfontein gold mine and 62 per cent from Impala platinum mine. The high values for the last two data sets can be attributed to the relatively high number of blast events in these data sets. The low value for the East Rand Proprietary Mines data set resulted from the larger average inter-event distance for this area: in many cases, the inter-event distance exceeded ten times $\bar{\lambda}$.

Possible improvements to this procedure would include (i) the use of ground displacement, rather than velocity, to reduce the effect of the arbitrary filter above the corner frequency and to increase the effective wavelength of comparison, and (ii) the window lengths used for cross-correlation of individual phases should be chosen to match the body-wave pulse widths.

This procedure has significant implications for all seismic studies in which high accuracy of source location is necessary. The benefit derived from the routine application of the procedure, in terms of the reduction in manual processing time, is likely to be significant. The procedure may lead to increasing our understanding of the behaviour and distribution of mining-induced seismic sources. In combination with any statistical routine, such as Principal Component Analysis (Alexander et al, 1993), the procedure may enhance the use of seismicity measurements in such applications as the mapping of active fracture orientations. The procedure also is likely to be particularly useful in such applications as the investigation of source migration problems, the indirect discrimination of mining blasts, the identification of clusters of events with a similar source mechanism, and the determination of planes, joints, or faults on which events locate. This analysis could confirm the apparent increase in granularity in Figure 7 *b* compared to Figure 7 *a*.

CONCLUSIONS

The results described above illustrate the capabilities of the cross-correlation method for the identification and precise automatic location of recorded seismic event doublets.

The degree of waveform matching between two events is a function of the inter-event distance and decreases for distances greater than several times the average wavelength $\bar{\lambda}$.

The precise adjustment of the correlation windows by using the time lags from the envelopes increases the accuracy of P- and S-phase cross-correlation.

The relative location accuracy for well-matched events obtained by HRRL is far superior to that obtained currently from the P and S arrival-times location method and is even superior to that obtained from the JHD method.

Using the "progressive master event" concept improves the creation of hierarchies and families of related events, which is very useful in the study of sources with similar mechanisms.

A large number of the mining-induced seismic events showed a high degree of similarity ($R \geq 0.7$) and about 50 per cent of the events, on average, in the four studied data sets were relocated reliably. Different degrees of success can be attributed to differences in specific mining conditions.

ACKNOWLEDGEMENTS

This study was supported by the CSIR Division of Mining Technology and took place as part of the research of the Rock Engineering Programme under the SIMRAC project GAP033. Permission to publish the results is gratefully acknowledged. Thanks to the Managers and Managing Directors of Blyvooruitzicht gold mine, East Rand Proprietary Mines, Buffelsfontein gold mine and Impala platinum mine for the permission to publish

this paper. Thanks to Mr N.L. Cook, Dr R.J. Durrheim and Mr O.D. Goldbach for providing us with seismic data for analysis. Thanks also to Dr N.C. Gay and Dr R.J. Durrheim for their review and suggestions which improved our manuscript.

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FIGURE CAPTIONS

Fig. 1(*a, b*). Set of seismograms for a doublet recorded at Buffelsfontein gold mine:

a) Event 641: 28/4/93, $t_0 = 12:17:41$, $X = 79912$, $Y = 18748$, $Z = -2247$, $M = -0.2$;

b) Event 642: 28/4/93, $t_0 = 12:18:13$, $X = 79911$, $Y = 18746$, $Z = -2252$, $M = -0.4$.

Fig. 2. Scheme of "progressive master event" generation illustrated by six blast events recorded at Buffelsfontein gold mine: 46, 48, 49, 51, 52 and 53 are the numbers of the events.

Fig. 3. Plan view of part of Buffelsfontein gold mine and of the seismic network. The boxed area was used for illustration of the relocation procedure (see Figures 4 *a, b*); U1V, U2V, U3V, T1V, T2V, T3V AND T4V are the PSS site locations.

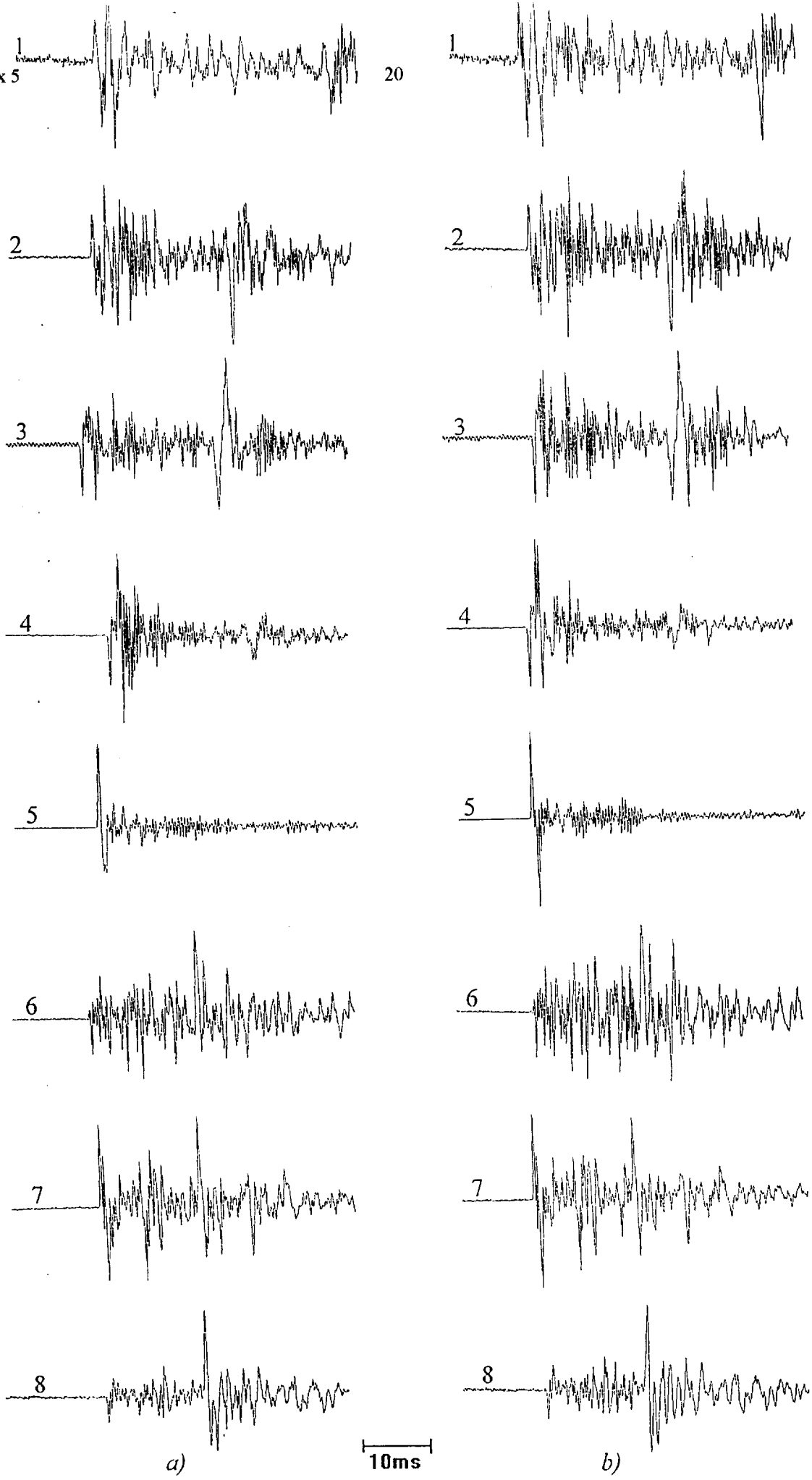
Fig. 4(*a, b*). The spatial distribution of six blasts recorded at Buffelsfontein gold mine:

a) before relocation; *b)* after relocation.

Fig 5. Diagram showing matching factor (M) as a function of inter-event separation (with a logarithmic regression line) for data recorded at Buffelsfontein gold mine.

Fig. 6. A commutative distribution of location error normalised by the average hypocentral distance. Location errors were obtained by the following methods: O - denotes the location error after applying the HRRL correction; Δ - the location error from the JHD method; \square - the location error from the P and S arrival-times location method.

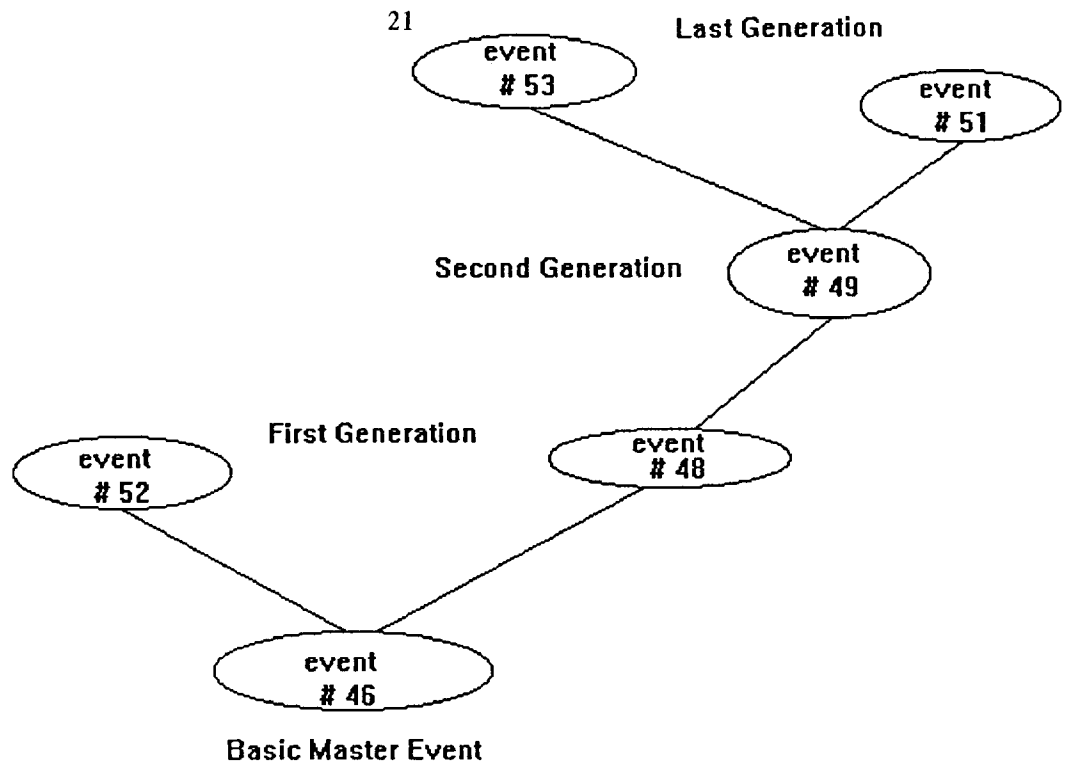
Fig. 7(*a, b*). Plan view of part of Blyvooruitzicht gold mine and the related seismic locations: *a)* before relocation; *b)* after relocation.

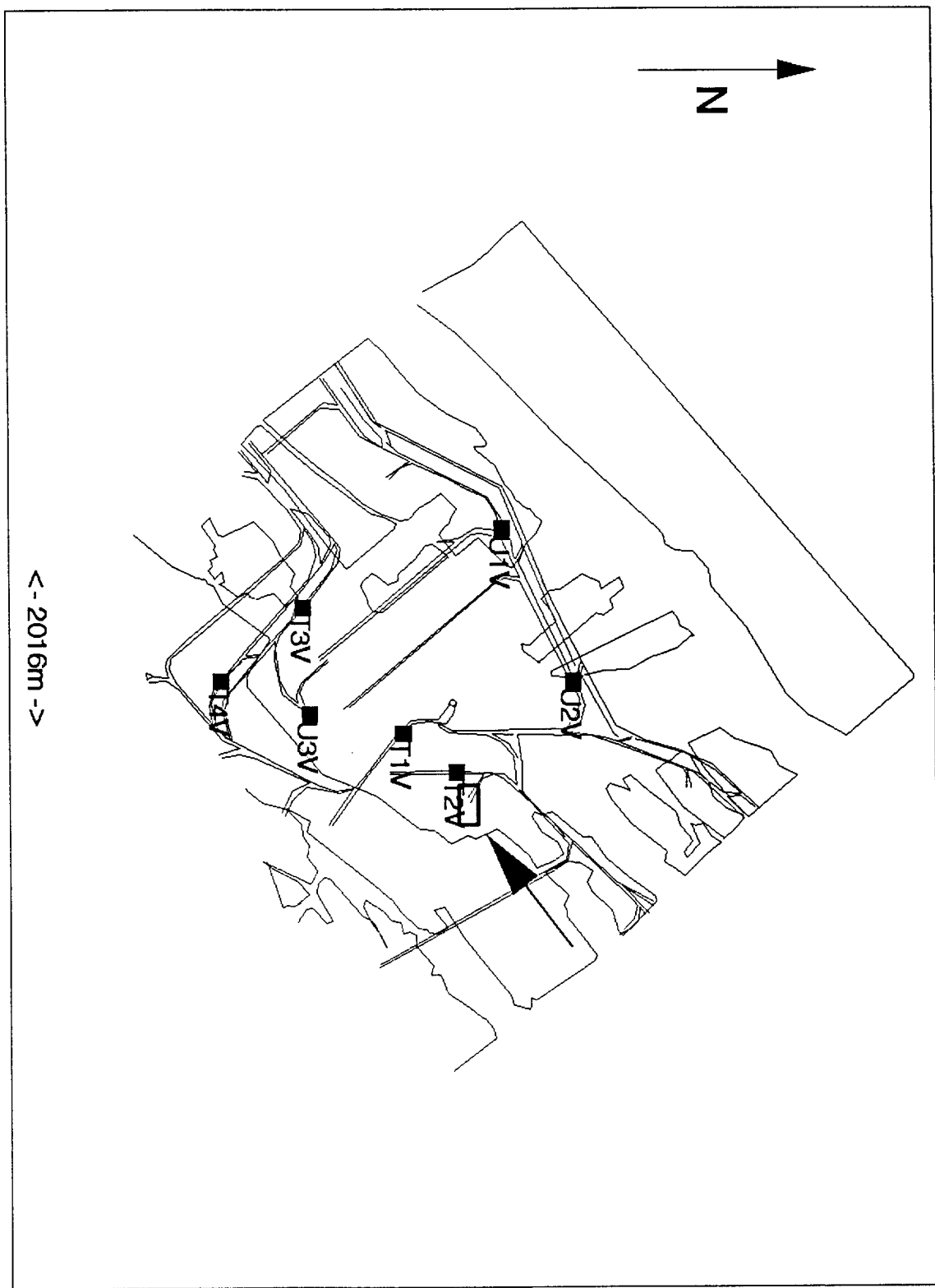


a)

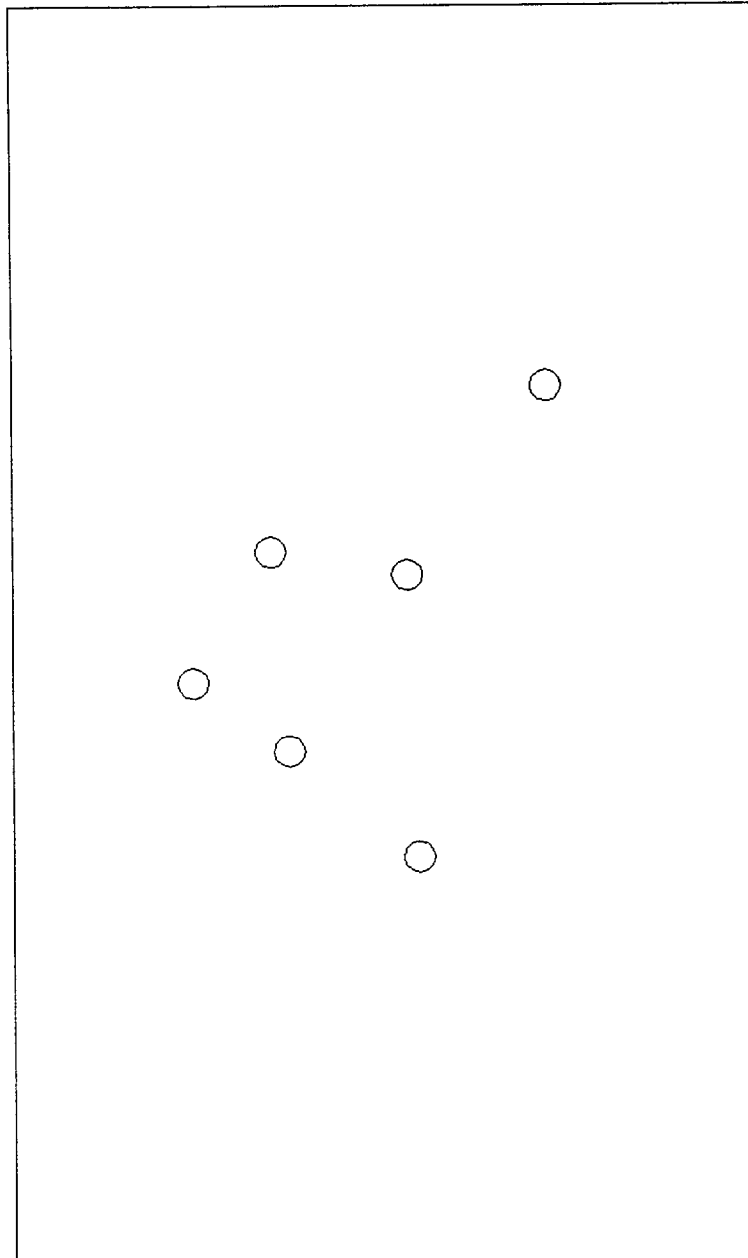
10ms

b)

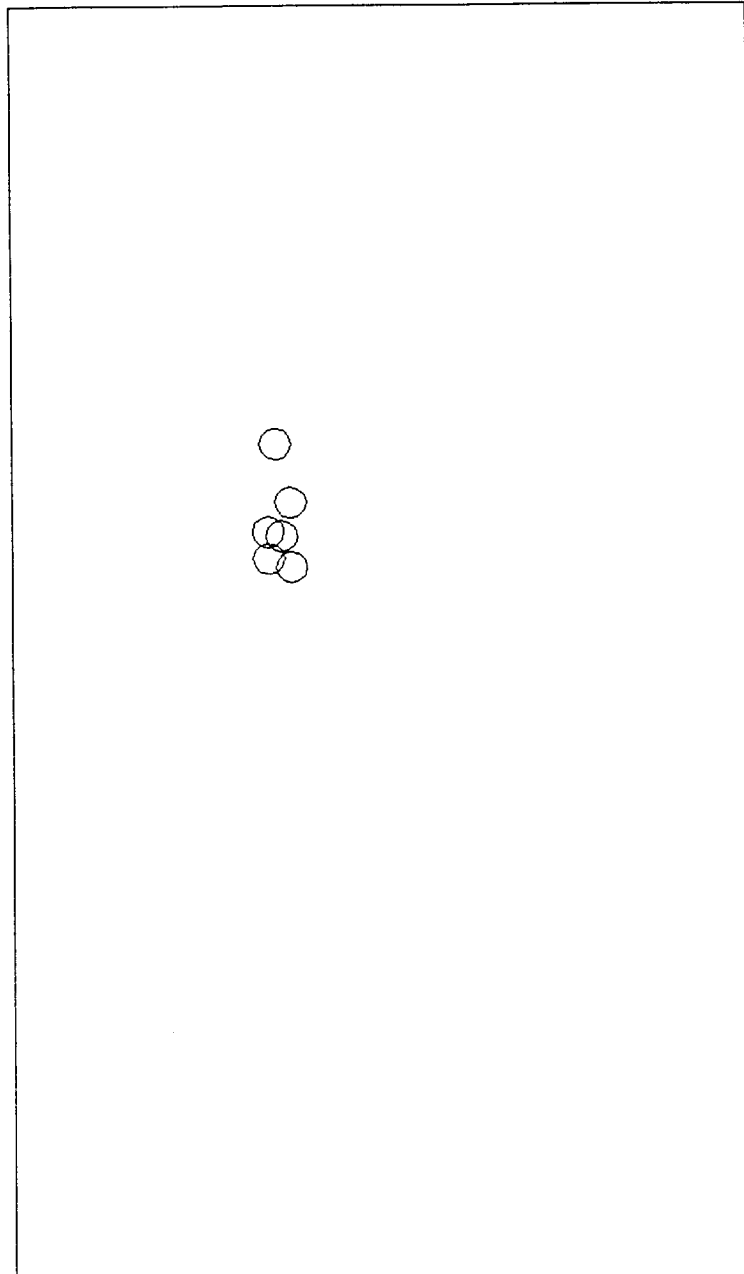


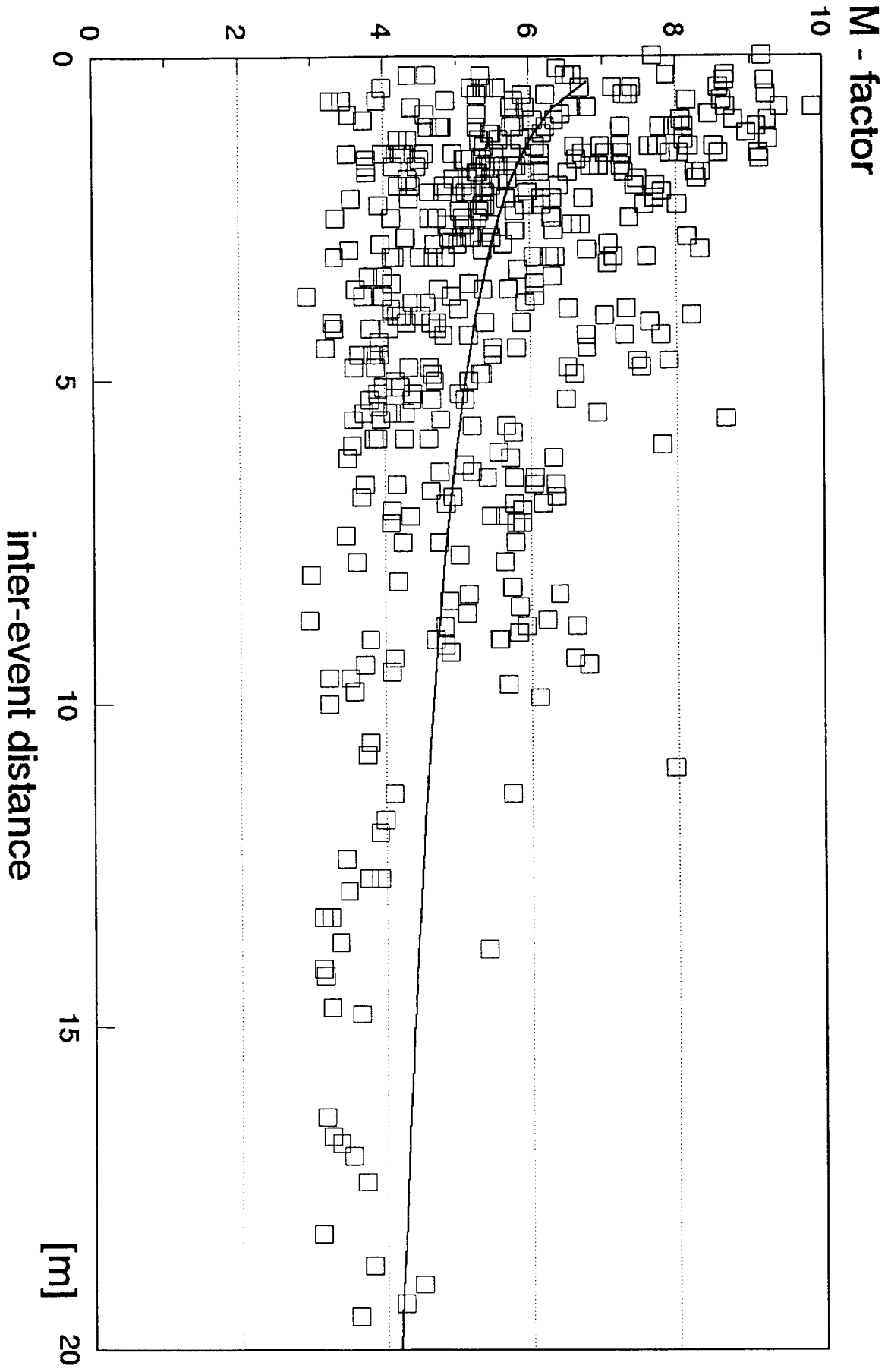


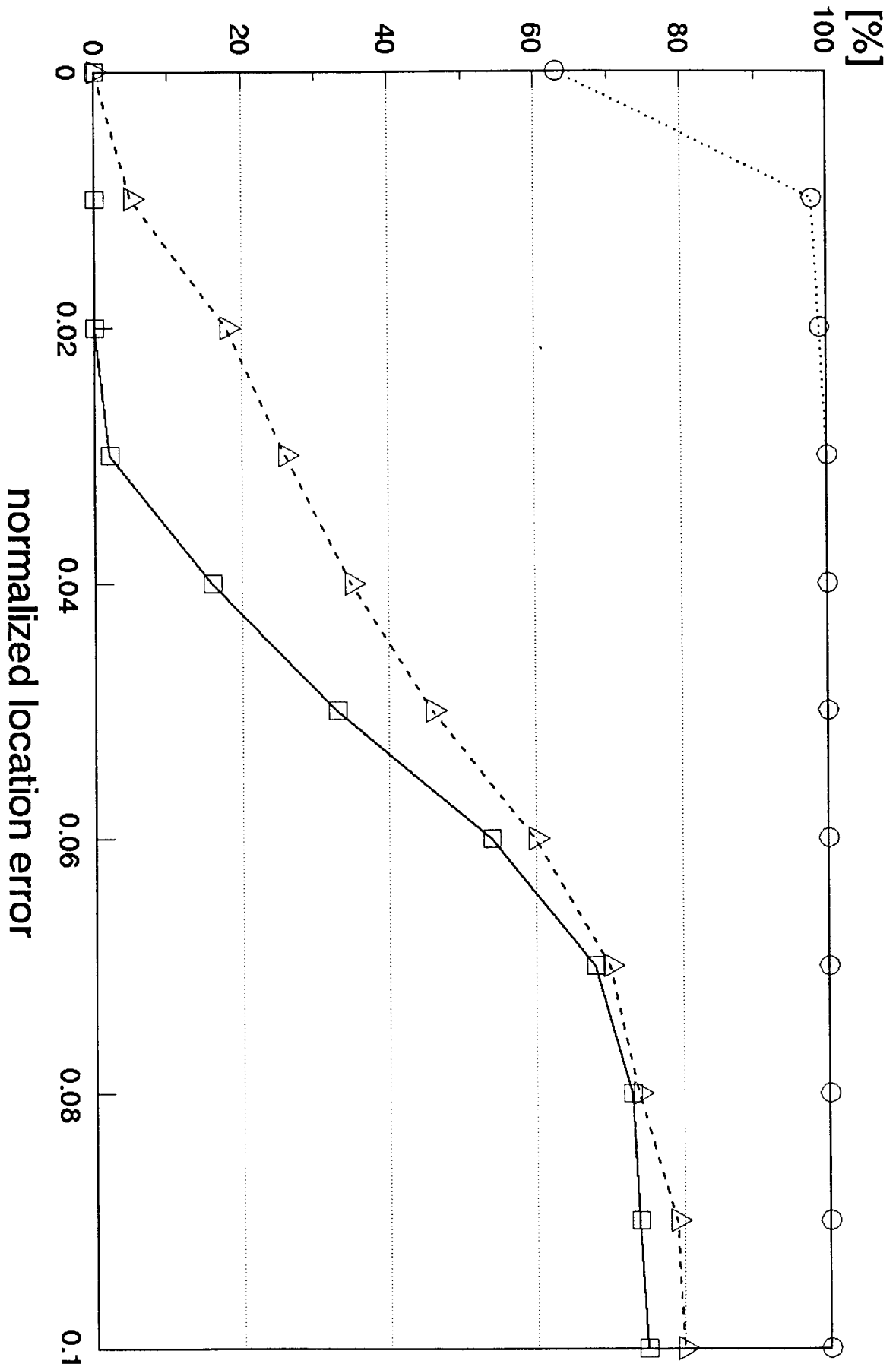
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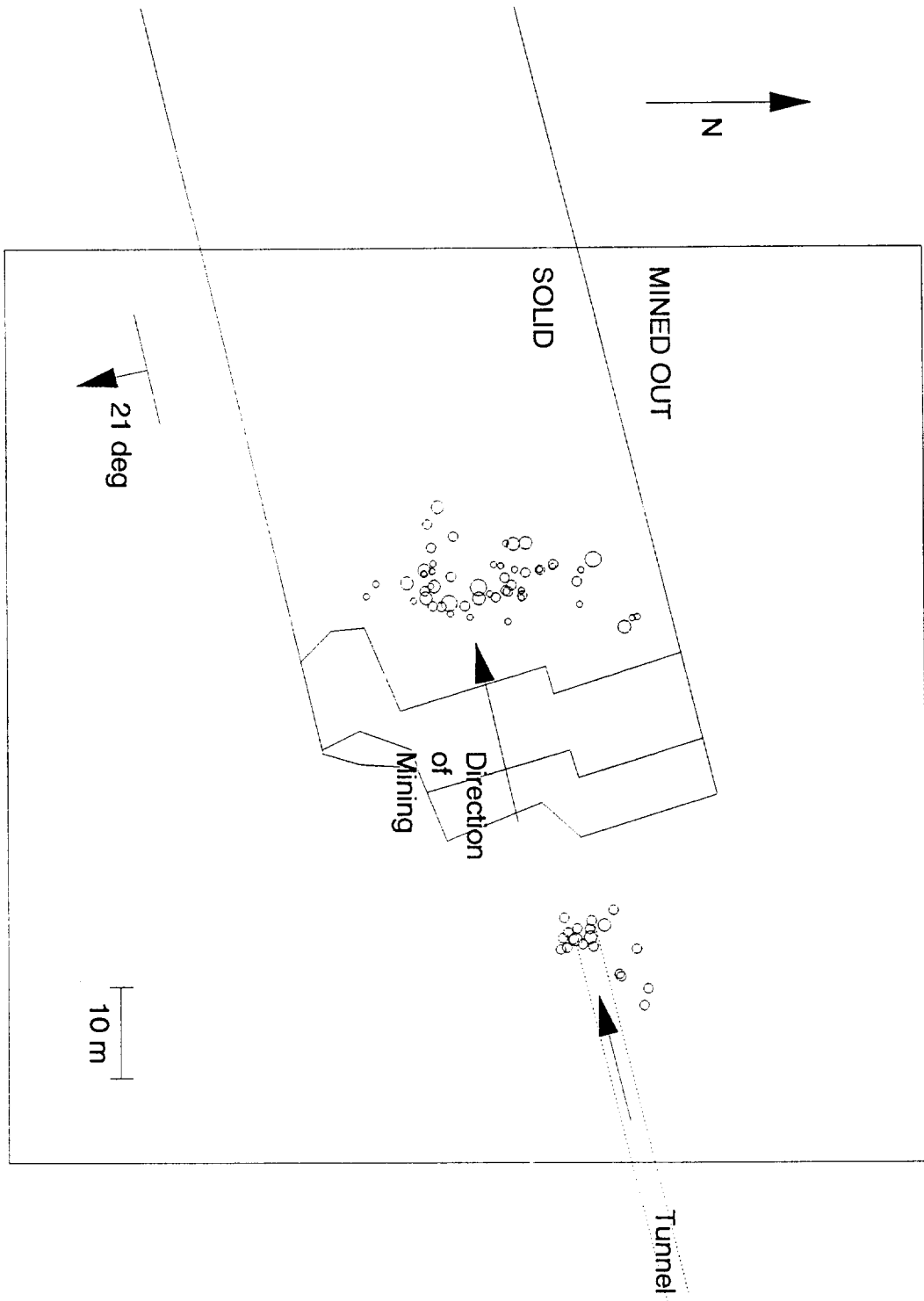


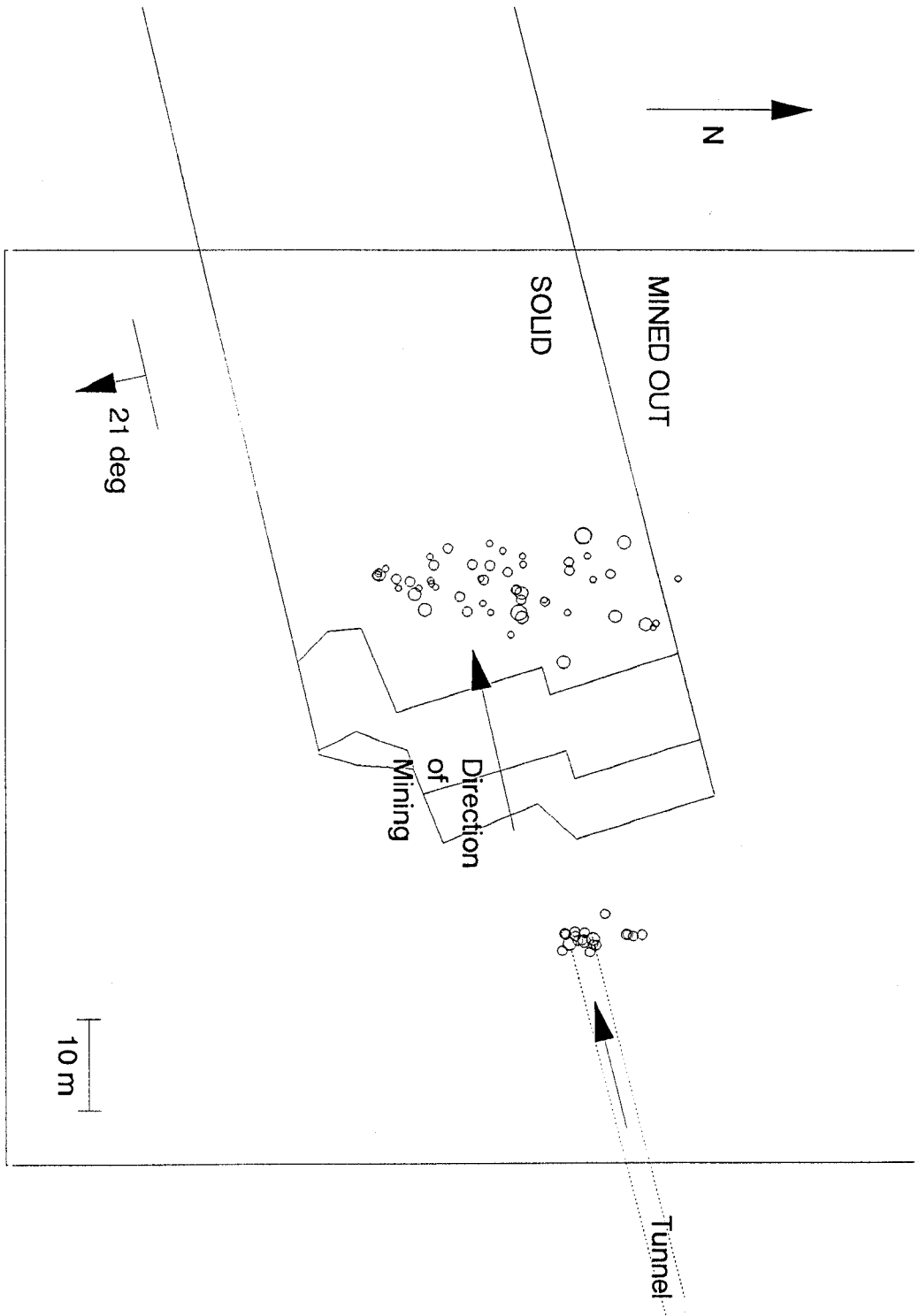
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PLANES OF LOCATION IN MINE-INDUCED SEISMIC CLUSTERS

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ABSTRACT

A large number of mining-induced seismic events show a high degree of waveform similarity. This similarity was analysed by a procedure with the following stages:

(i) Cross-correlation of the P- and S-seismograms and their envelope functions.

(ii) High Resolution Relative Location (HRRL) using P- and S-arrival times corrected with the time differences obtained from the correlation procedure.

(iii) Define clusters or families of events, with very similar waveforms.

(iv) Determine the planes of location within the cluster by means of the Principal Component Analyses (PCA).

(v) Plotting the poles to the planes of location using lower hemisphere stereographic projection and analysed their disposition and orientation.

This procedure was applied to data recorded underground at deep level South African gold mines. HRRL provide very robust and accurate relative locations. Large number of the events were relocated

reliably and formed clusters. Cluster positions and orientations were aligned with joint patterns and with face parallel structures.

METHOD

We used a procedure for study of mine-induced seismic clusters and the planes of events location. This was implemented by a two stage process:

1. AUTOLOC - procedure for automatic location of similar seismic events (Figure 1) on the basis of cross-correlation analysis. Consisted following steps:

- apply Hilbert transform to calculate the energy envelope function for each event, channel and phase and store envelope files on disk - ENVEL.

- loop through the 100 previous events to find the best envelope cross-correlation and correct with the envelope lag the P and S arrival times and the position of the correlation windows - CCM.

- loop through the 100 previous events to find the best P- and S-phase cross-correlation and write the correlation matrix - CCM.

- read the correlation matrix, recognise events doublets and perform high resolution relative location for the good fits - HRRL.

-using multiple master event for creation of clusters or families of related events - HRRL.

2. PCA - to obtain the spatial distribution of "planes of events location" within a cluster. As follows:

Each plane is defined by at least three locations, *A*, *B* and *C* (Figure 2), whose normal is plotted in a lower hemisphere projection.

The pole \vec{n}_k to the k^{th} plane is presented as function of dip angle ϕ and direction of dip θ

$$\vec{n}_k = \vec{n}_k(\theta, \phi)$$

Small or flat triangles were rejected when the perpendicular distance from any of the three events was less than three times the error of that event from the line joining the other events.

We consider the poles to the planes \vec{n}_k as a principal components of the cluster and studied their disposition and orientation using iterative stereographic technique.

DATA AND RESULTS

The data used here were recorded underground by short period Portable Seismic System (PSS). The selected data sets originated from two deep level gold mines in South Africa: Western Deep Levels South Mine and Blivooruitzicht Gold Mine. Each data set consisted about 600 consecutive events in temporal order and magnitudes in the range $-2.0 \leq M_L \leq 4.0$.

Figure 3 *a* is a plan of part of Western Deep Levels South Mine and related seismic locations. The clusters shown are very well separated by the HRRL procedure. They are oriented parallel to the neighbouring active mining faces - updip or North faces mined. The poles concentrations for both clusters given on Figure 3 *b* show similar orientations and dip close to the vertical.

Figure 4a illustrate seismicity associated with pillar removal at Blivooruitzicht Gold Mine. The seismic events were located using manual P & S arrival times (standard procedure). On Figure 4b we show the original locations of a cluster with 25 similar seismic events using CCM. Figure 4c illustrate the new locations obtained by HRRL and the PCA projection plane. The dips of the plane was 78° towards $N71^\circ E$.

CONCLUSIONS

The results described above illustrate the capabilities of HRRL method for the identification and precise automatic location of similar seismic event.

Using a multiple master event in the relocation procedure improves the clustering of related seismic events.

The PCA technique provides a robust method for the determination of the location planes.

The location planes, derived by PCA method, show a good agreement with the active mining features.

The results obtained are encouraging and motivate further PCA analyses on different data sets.

FIGURE CAPTIONS

Figure 1 Set of similar seismograms recorded at Blyvooruitzicht Gold Mine and the procedure description.

Figure 2 Lower hemisphere view of plane of locations defined by three events A , B and C ; \vec{n}_k is the pole to the plane, \vec{n}'_k stereographic projection of the pole, θ is dip direction angle and ϕ is a dip angle.

Figure 3a Area view and locations of two clusters at Western Deep Levels South Mine: "+" denotes the events of Cluster # 1 and " Δ " denotes the events of Cluster # 2

Figure 3b Stereonet projections of poles to the planes of locations at Western Deep Levels South Mine: correspond to Cluster # 1 and Cluster # 2.

Figure 4a Original location of cluster with 25 events

Figure 4b HRRL location & PCA plane of cluster with 25 events

FIG 1

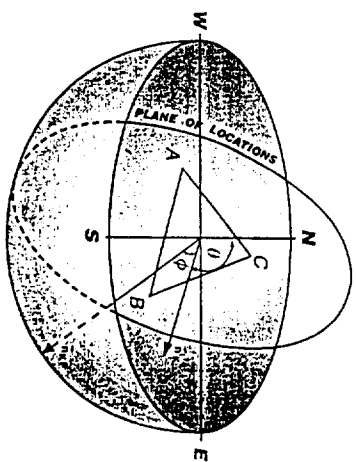
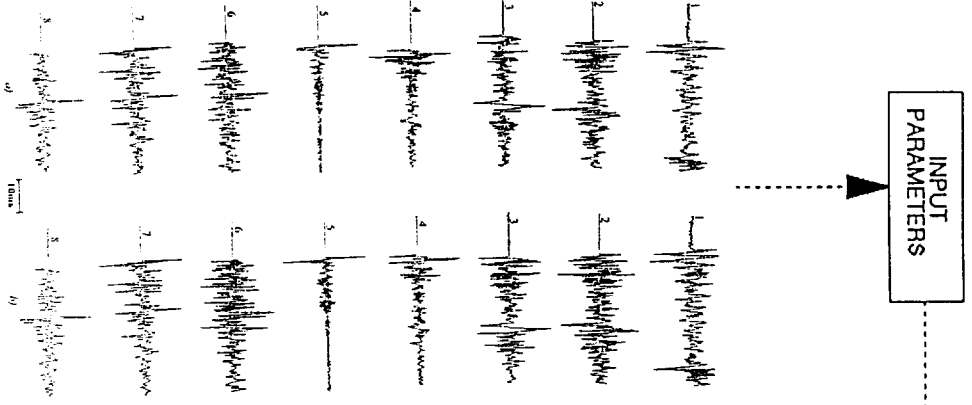
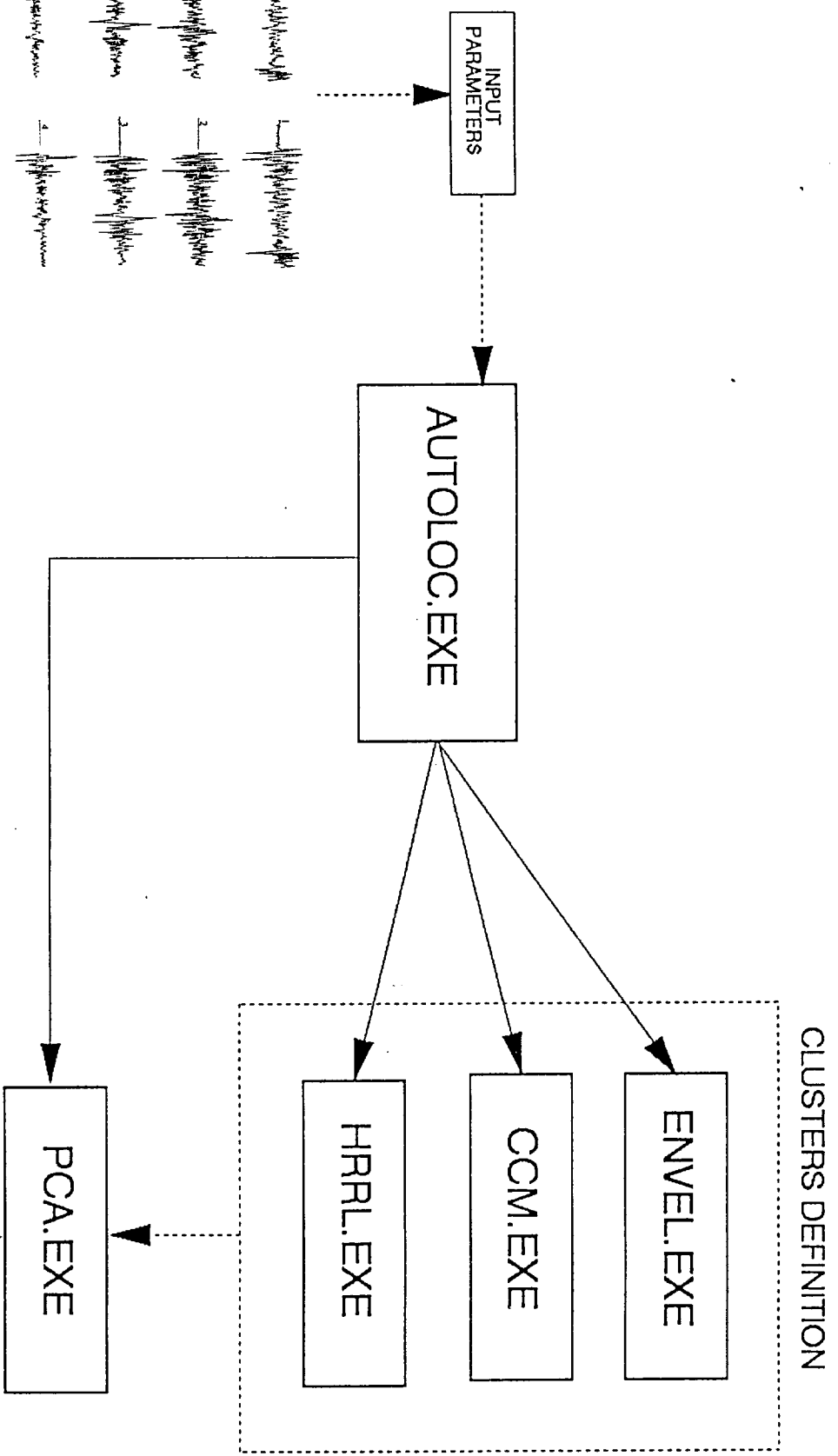
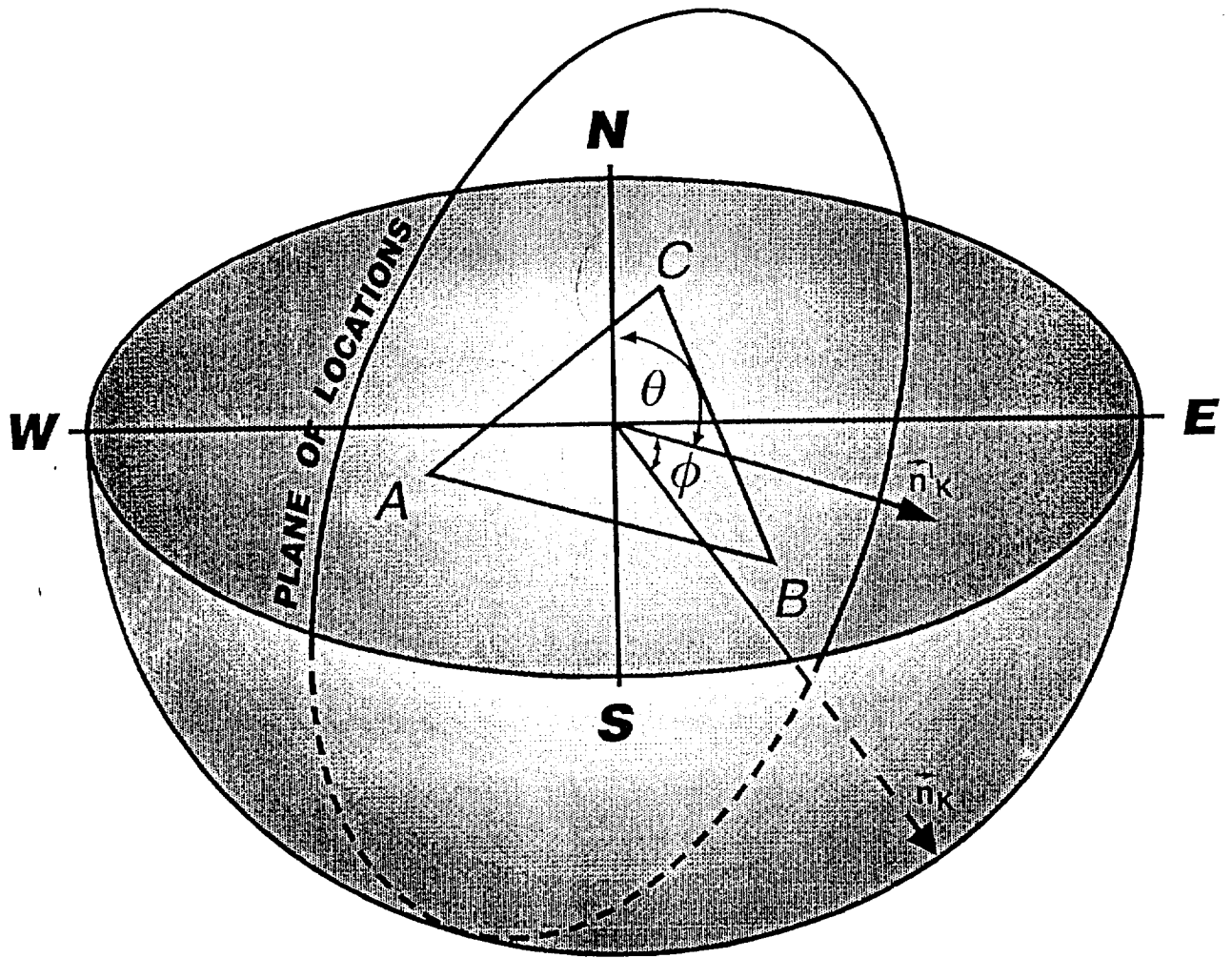


FIG 2





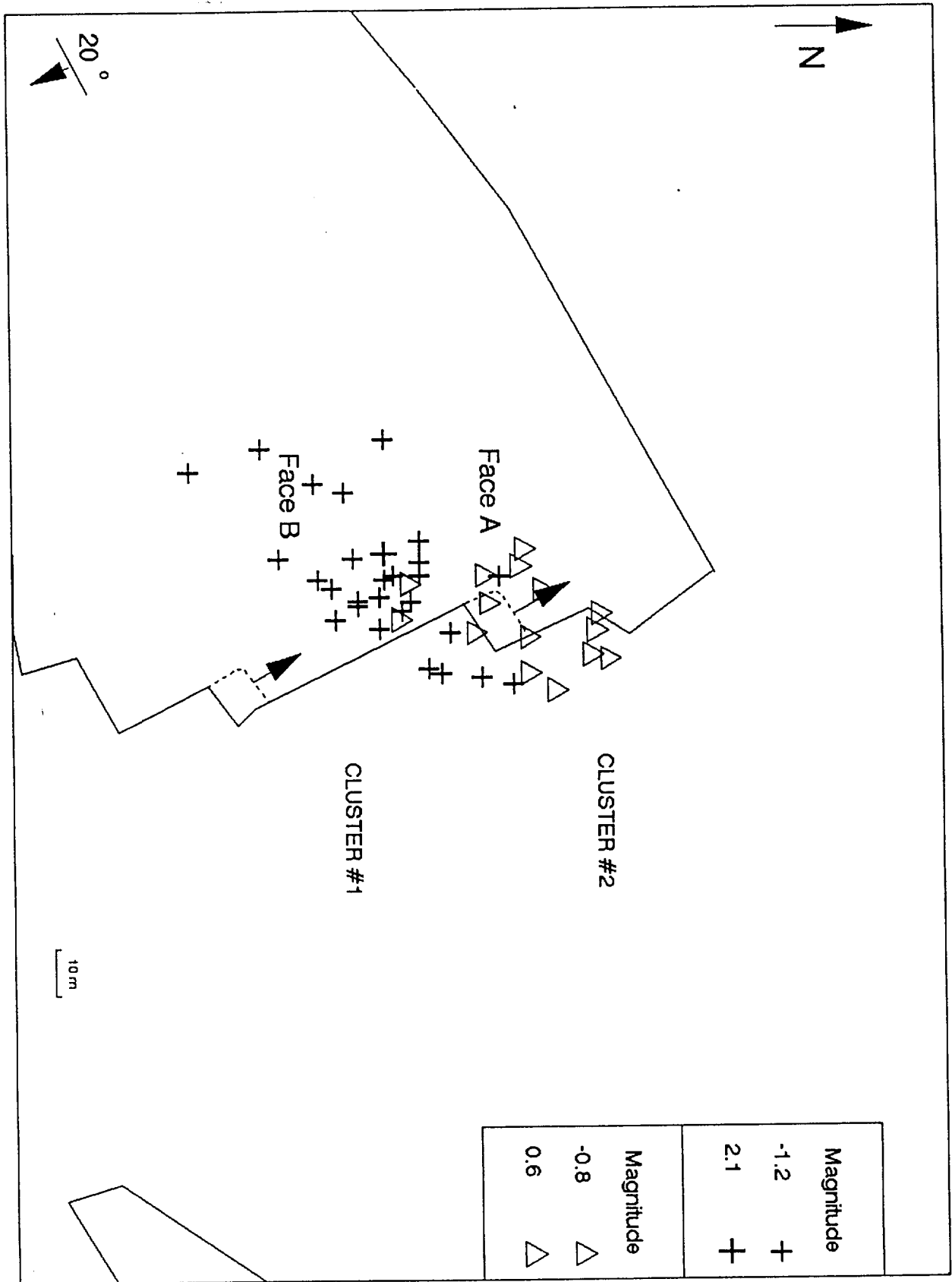
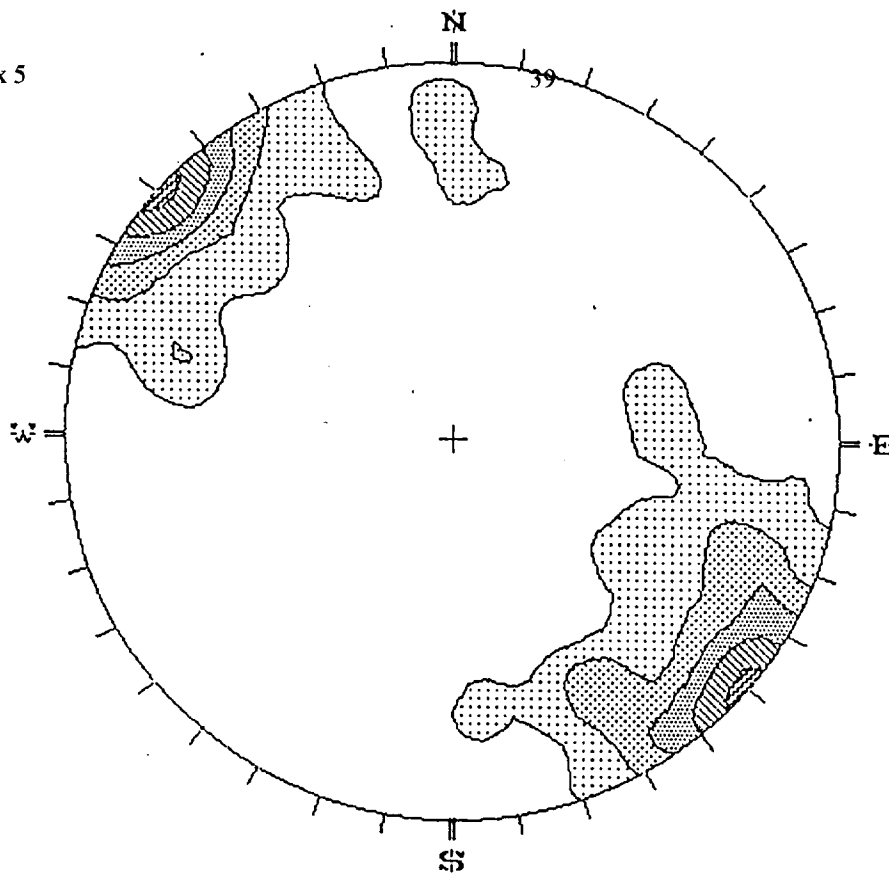
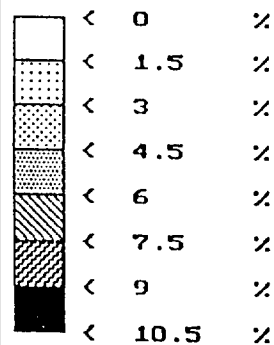


FIG 3A



Western Deep Levels - South Mine

FISHER POLE CONCENTRATIONS
% of total per 1.0 % area



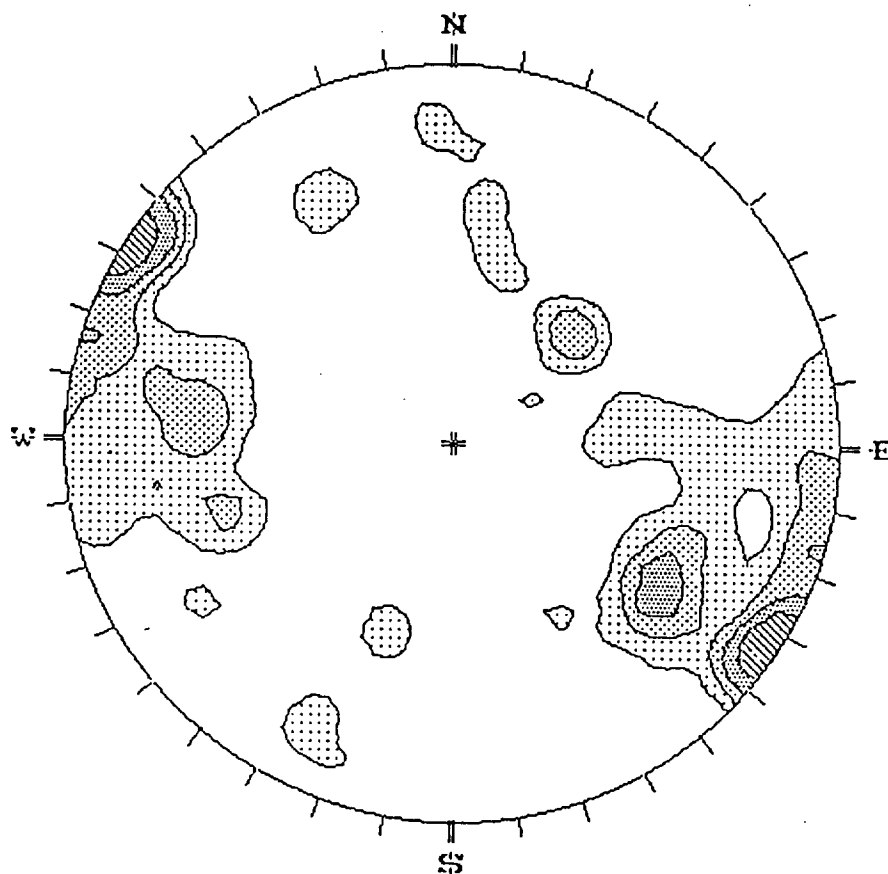
EQUAL ANGLE

LWR. HEMISPHERE

1161 POLES
1161 ENTRIES

NO BIAS CORRECTION

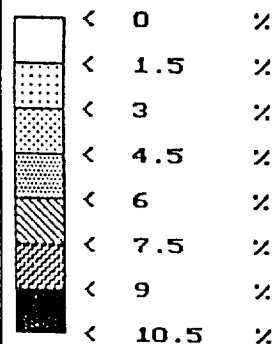
Principal Component Analysis Cluster # 2



Western Deep Levels - South Mine

CONTOUR PLOT

FISHER POLE CONCENTRATIONS
% of total per 1.0 % area



EQUAL ANGLE

LWR. HEMISPHERE

230 POLES
230 ENTRIES

NO BIAS CORRECTION

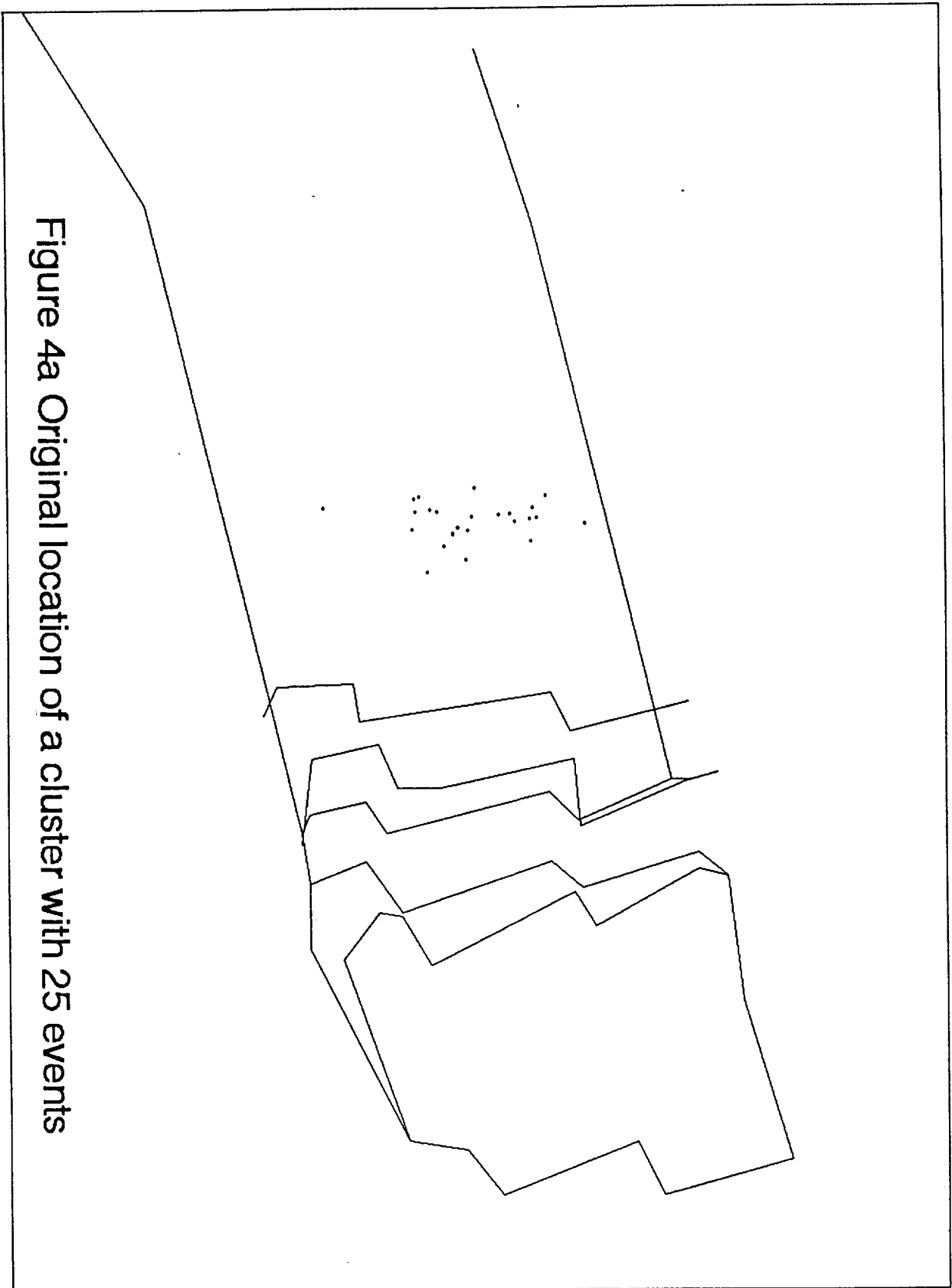
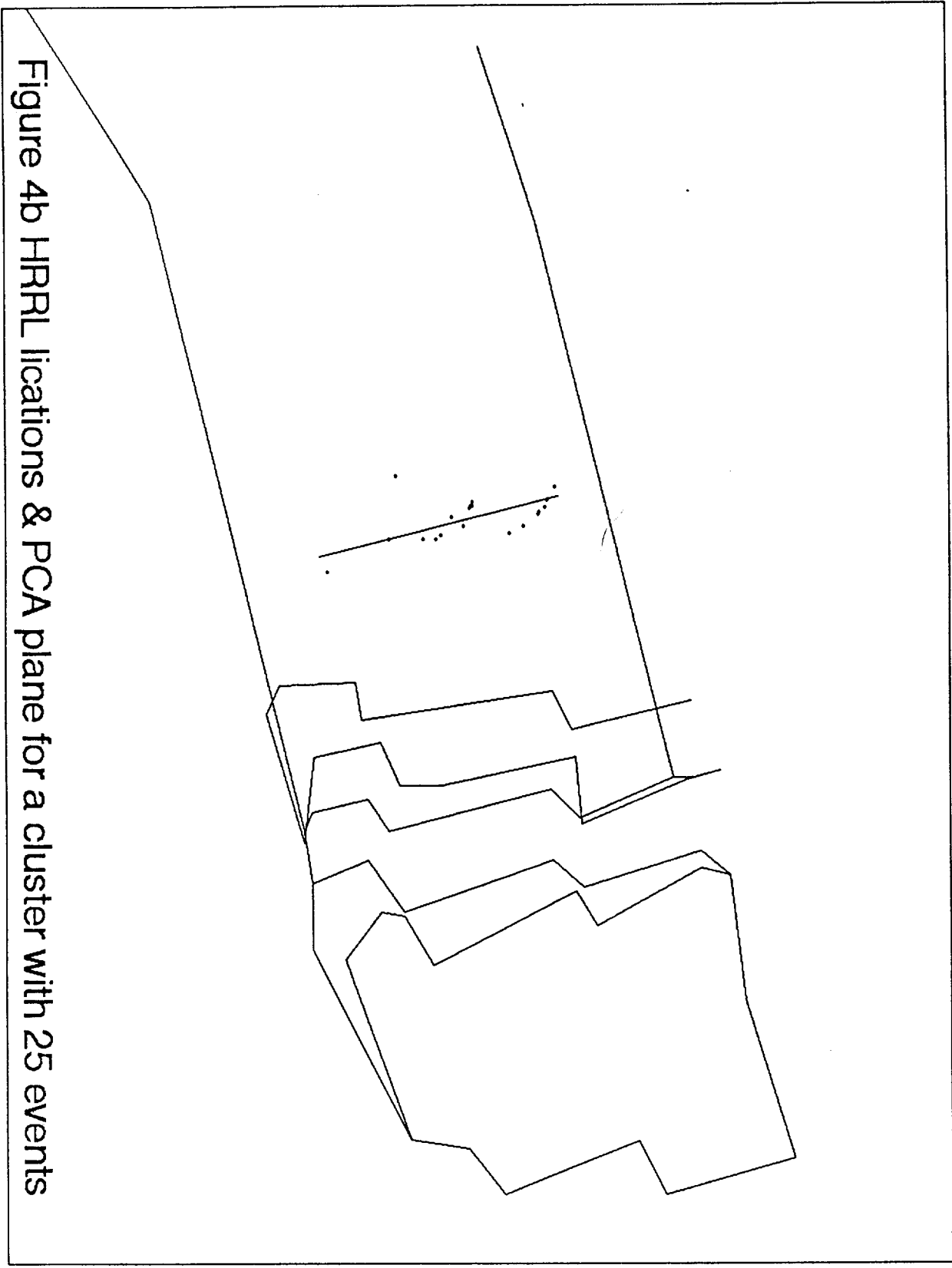


Figure 4a Original location of a cluster with 25 events

Figure 4b HRRL locations & PCA plane for a cluster with 25 events



Appendix 6.

GEOTECHNICAL AREAS CLASSIFICATION

GEOTECHNICAL CLASSIFICATION OF DEEP AND ULTRA-DEEP WITWATERSRAND MINING AREAS

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ABSTRACT

Depth of mining of the auriferous Witwatersrand deposits is continuously increasing. Some mines are already mining at depths exceeding 3500m. Production costs are similarly increasing, with health and safety aspects rightly gaining in importance. Novel approaches, considering mining strategies, layouts and support systems have to be adapted, in order to mine deep orebodies safely and efficiently.

Deformation mechanisms associated with Witwatersrand orebodies are complex. The importance of geological features is unquestionable, although their impact is obscured by mining parameters. The geological features to be considered are grouped into two categories, i.e. primary and secondary geological features; both are of rock engineering importance and contribute to the definition of geotechnical areas.

Primary geological features referring to the rock type, are largely defined by texture and composition, which is also reflected in varying rock engineering properties. These vary vertically and laterally, i.e. in time and space. These variations are observed in the orebody itself, and its hanging- and footwall rocks. Primary geological features also include the frequency and mineralogical characteristics of sedimentary structures, such as the various kinds of bedding planes and lithological boundaries. These are in most instances reef parallel. Reef geometry is also considered as a primary feature.

Secondary geological features are faults, dykes and joints. These have, to date, received most attention from the rock engineers. Compositional and textural characteristics of these discontinuities are also deemed important.

Primary and secondary geological features impact, especially at great depth, on the rockmass behavior, expressed as closure rate, seismicity, and mining induced extension and shear fracturing. They play an important role in identifying the appropriate mining strategy, layout and support. Geological features can be predicted into deep, unmined areas. The use of these predictions of the rockmass at ultra-depth aims at the identification and recommendation of the safest and most efficient extraction of the orebody.

The majority of the orebodies presently being mined will, potentially, be extracted from ultra-depth. The majority of these, in turn, are situated in complex geological environments.

INTRODUCTION

The Witwatersrand gold resources are continuously depleting. In 1970 some 1266t of gold (79% of the world production) were produced from this depository (Tainton, 1994). Production had decreased to 614t of gold in 1992 and some 230-300t are forecast for the year 2025 (Sanders, 1994; Tainton, 1994).

Mining inevitably has to consider resources at ultra-depth. Depending on rock strength, ultra-deep mining is defined as mining which will take place between 3500m and 5000m depth. It is possible to hoist economic tonnages from 4000m depth in a single shaft system (Stewart, 1994) and auriferous deposits are known to extend to at least 6000m below the surface (Tamlyn, 1994). Some mines are already mining at depths exceeding 3500m, such as Kloof and Western Deep Levels Gold Mines, East Rand Proprietary Mines (ERPM), and Driefontein Consolidated.

Rockburst related fatalities increase significantly with depth (e.g. Gurtunca and Gay, 1993; Roberts et al., 1994). Numerous researchers simultaneously stress the importance of, and requirement for geotechnical information, especially geological features, and of gaining an improved understanding of their behavior (e.g. Adams et al., 1981; Gay, 1986; Gay and Jager, 1986a and b; Roberts and Jager, 1991, 1993; Kullman et al., 1994; Gay et al., 1995).

Two quotes taken from Gay and Jager (1986) are illustrative of these requirements:

" A full understanding of the rock mechanics of an underground mining situation is not possible without considering the effect of local geology. Fundamentals of strata control, such as stress distribution, rock strength, the way rock fractures, the incidence of seismicity, and the stability of unsupported strata are all governed, to some extent, by basic geological features such as petrology, stratification, difference in competence between strata, and the incidence and orientation of structural features."

"... depending on their petrographic compositions rocks may respond in a brittle or ductile manner to forces. This range in rheologic behavior is likely to become more marked as mining extends to yet greater depths, and knowledge of the behavior of fractured rock and the viscoelastic properties of the Witwatersrand strata will be of paramount importance to the rock mechanic engineer. The role of the mine geologist will be crucial for obtaining this information."

The identification, delineation and quantification of geological characteristics at great depths is important. It is suggested that in many cases current mining methods can not simply be extended to greater depths without compromising safety, and thus significant modifications or completely new mine designs will be required which take the prevailing geological conditions into account.

We will first review the relevance of geotechnical information for Witwatersrand Gold Mines. Secondly, we will identify the orebodies and regions that are currently, and will potentially, be mined at ultra-depths. Some of these orebodies are then broadly categorised, according to rock assemblage and other geological

features, to enable the prediction of the rockmass behavior at ultra-depth. This classification further assists in evaluating potential mining layouts and methods for application in specific environments. Illustrative examples are given and discussed. We consider the total Witwatersrand Basin, straddling a distance exceeding 300 kilometers, including extensions to orebodies and potential new mining leases.

GENERAL OVERVIEW

The stability of excavations is a complex function of mining layout and method, local and regional support and various geological parameters. Mining parameters include the extent of unsupported spans, the rate of face advance, regional support strategy (such as pillar spacing/orientation and backfill, e.g. Arnold et al., 1994) mining method, layout and stoping width. Backfill, for example, may reduce the total closure by 70% (Gay et al., 1995), which would have a significant impact on any analysis of the rockmass behavior. The influence of geological features on seismicity was identified in both backfilled and unfilled stopes (Gurtunca and Gay, 1993a). The effects of geological features are likely to increase in importance with increasing spans along longwalls (i.e. higher ERR levels; Gay, 1993), a trend that is expected to be even more pronounced at great depth.

Our review led us to subdivide geological parameters into two broad categories, termed primary and secondary geological features:

Primary Geological Features of Rock Engineering Significance

Primary geological features are defined as those that formed during the deposition of the rocks of the Witwatersrand Basin. The relevant primary geological features fall into three broad groups:

a) Rock Type

Predominantly sedimentary, and minor volcanic rocks are associated with the orebodies of the Witwatersrand Basin. The various types of metamorphic rocks are neglected in this study, considering that the rocks under consideration have undergone a regionally comparable metamorphic grade (greenschist facies metamorphism), at temperatures of up to 350 °C (Phillips et al., 1990; Meyer et al., 1992).

The major sedimentary rock types are classified as shale/silt, quartzite (ranging from argillaceous to siliceous quartzite; pebbly varieties do exist), and conglomerate (ranging from matrix supported to clast supported conglomerate). The change of rock type along the same stratigraphic horizon, due to proximal/distal relationships (e.g. quartzite being transitional into shale) also has to be taken into account. The various textures and compositions of these lithologies determines, whether a rock is competent or incompetent, or transitional in between the two. However, the rock type and its associated properties should not be viewed in isolation. Marked differences in the competency of the hangingwall and footwall rocks, for example, may cause support problems such as punching or excessive damage to the hangingwall or footwall. A variety of studies considered the textural and compositional characteristics of Witwatersrand

strata (e.g. Henckel and Schweitzer, 1990, 1994; Henckel et al., 1990a and b; Hallbauer and Schweitzer, 1992). To date, no attempt has been made to investigate the potential rock engineering significance of these studies.

The importance of rock type has previously been discussed by Gay and Jager (1986): Tunnels, for example, should be positioned in competent strata, and require special support when positioned in shale. Rock type also influences total closure, and the degree of hangingwall blast damage. Ductile behavior of strata, such as shale, may also cause bulging of the footwall into the stope area. The distinct rockmass behavior in the Klerksdorp area, compared to the Carletonville Goldfield, has also been attributed to the generally weaker (about 200 MPa) quartzites associated with the excavations in the Klerksdorp area. "Weak" footwall rocks can degrade the pillar strength, which can be further reduced due to creep behavior of shale or other, laminated strata. Consequently, pillars behave differently under various geological and mining conditions, which clearly illustrates the requirement for geotechnical data for pillar design.

Volcanic rocks associated with the Witwatersrand orebodies are predominantly basalts and andesites, associated with pyroclastic rocks, of the Ventersdorp Supergroup. These overlie the Ventersdorp Contact Reef (VCR) and the different rockmass behavior of, especially the basalts when compared to the andesites, has recently been demonstrated (M. Roberts, pers. communication, 1994). Primary partings associated with these volcanic rocks are due to strongly amygdaloidal and brecciated, or auto brecciated horizons (mainly close to flow contacts), and thin pyroclastic and sedimentary intercalations. Horizontal mining induced movements on stope parallel hangingwall partings yield areas prone to fallout between face support units.

b) Sedimentary Structures

The second group covers the various kinds of bedding; planar bedding, graded bedding, trough- and planar cross bedding. The frequency of these bedding planes, and the lithological contacts are of importance in rock engineering (Atkins and Keen, 1984; Gay and Jager, 1986; Jager and Turner, 1986; Kullmann et al., 1994). These sedimentological partings are commonly parallel to the reef plane, may part with variable ease, depending on their mineral composition, and determine factors such as beam thickness. Partings may also cause spalling of pillar faces, resulting in the reduction of pillar width (Gay and Jager, 1986). The spacing of bedding planes also influences tunnel stability (Gay and Jager, op. cit.).

c) Reef Geometry

Reef geometry considers reef thickness, and the "rolling" of the orebody. As the effects of seismicity are substantially reduced with narrow widths (e.g. Arnold et al., 1994), reef thickness and hence stoping width are significant to future mine planning. Such information also indicates appropriate face area support, stoping method and the effectiveness of regional layouts and support.

"Rolls" have been identified as hazardous areas of mining (J. Hamman, pers. communication, 1993). "Rolling" of the orebody results from geomorphological variations of the old palaeosurface onto which the

reefs deposited and is, to some extent, predictable due to its close link to the under- and overlying strata (McWha, 1994; Germs and Schweitzer, 1994; Henning et al., 1994; Schweitzer et al., 1991, 1992, 1993, 1994).).

Secondary Geological Features of Rock Engineering Significance

Secondary geological features are predominantly encountered as joints, faults and dykes. These structures ahead of the face may result in an increased rockburst frequency towards these structures (peak rockburst frequency within approximately 20m of the structure) in both, longwall and scattered mining situations (Gay, 1986; Gay and Jager, 1986; Kullmann et al., 1994). Such features are often associated with relatively large falls of ground (Gurtunca and Gay, 1993). The importance of these structures for the weakening of off-reef excavations and pillars has also been highlighted. The orientation of a structure in relation to the excavation is important and not only for the placement of bracket pillars. Joughin and Jager (1983) suggest that the effects of secondary geological features are least when at right angles to the face, and greatest when parallel to the stope. Hangingwall faults dipping towards the unmined area and footwall faults that dip towards the mined out area were identified as being associated with large tremors on the fault planes (Hepworth and Diamond, 1983). Dykes which strike at high angle to the bedding plane have been identified as dangerous by Gay and Jager (1986).

Mineralogical characteristics of the fault and joint planes, and dyke/host rock contacts are also of rock engineering importance. A great variety of fault rock types are associated with the faults of the Witwatersrand Basin (Roering et al., 1991), and their parting potential ranges from good to nil (Jager and Turner, 1986). The rock type hosting the secondary geological features, may control fault and joint attitudes (e.g. Jager and Turner, 1986), geometry, secondary mineral assemblages and consequently friction angles and excess shear stresses.

In the case of dykes, dyke composition ranging from basaltic to rhyolitic, could be of importance in determining their burst-proneness. The siliceous dykes particularly should be capable of storing significant amounts of elastic energy, therefore being most likely to fail. However, this has to be confirmed through future studies.

Primary and Secondary Geological Features, Seismicity, Mining Induced Fracturing, and Computer Modeling

Seismicity:

A close link between seismicity, geology and mining methods is proposed. It is predicted that, as mining operates at greater depths, the number and magnitude of seismic events also increases (Gay et al., 1995), resulting in a relatively higher number of rockburst-related fatalities (Gurtunca and Gay, 1993b; Roberts et

al., 1993, 1994). Differences in seismicity have been documented for the Carletonville and Klerksdorp Goldfields, and these differences have been attributed to different mining methods, different degrees of geological disturbance, and different rock types and assemblages (Gay, 1986; Gay and Jager, 1986; Gay, 1993). The close interrelationship between mining geometry, seismicity and fault properties for the Klerksdorp has been highlighted by Spottiswoode (1986). In general, two distinct classes of seismic events are distinguished and these are related to fault slip and the unstable fracturing of intact rock (Joughin, 1965; Joughin and Jager, 1983; Brummer, 1988). Researchers agree that large magnitude seismic events are likely to be associated with geological structures, with wave velocities and burst-proneness differing within distinct rock types, affecting both stoping and development areas (Joughin and Jager, 1983; Rorke and Roering, 1984; Gay, 1986; Gay and Jager, 1986a; Gay et al., 1993; Kullmann et al., 1994; Gay et al., 1995)..

Mining Induced Fracturing

The importance of mining induced fracturing in deciding on the appropriate support strategy and mine layout, and its impact on the extent of damage caused by rockbursts, is generally recognized. In the absence of geological disturbances, "normal" fractures develop and ensure a stable state of mining (e.g. Brummer, 1988). Rock type has been shown to control fracture attitudes. For example, siliceous rocks tend to be more intensely fractured (e.g. Jager and Turner, 1986), with fractures being displaced along the fine bedding of shale horizons (Adams et al., 1981). The distance of the fracturing to the face is controlled by the rock strength, ERR, excavation dimensions, presence of bedding planes and other geological discontinuities, such as joints (e.g. Piper, 1984). These discontinuities may result in a reduction of fracture intensity, or even their complete absence (Roering, 1981; Joughin and Jaeger, 1983; Gay and Jager, 1986). Different fracture intensities have even been observed on opposite sides of a fault (Adams et al., 1981). Fracturing is equally important in and around pillars!(e.g. Yilmaz and Ozbay, 1993).

Computer modeling:

Various computer modeling approaches were undertaken, assisting in an improved understanding of the rockmass behavior under various mining conditions. The objective of this modeling is to identify the most suitable mining strategies, mine layouts and regional support designs for improving underground safety (e.g. Napier, 1991; Johnson, 1994). The selection of realistic rockmass strength parameters, the length and characteristics of parting planes and multiple discontinuities (such as bedding planes, faults and joints), and the impact of textural rock properties (e.g. grain size and shape) on fracture propagation, have been identified as important, or of potential importance (Napier and Stephansen, 1987; Napier and Hildyard, 1992; Snyman and Martin, 1992; Sweby and Smith, 1994; Napier and Kuipers, 1995; Napier and Peirce, 1995). Modeling also provides insight into the energy change in the vicinity of multiple discontinuities that intersect the reef plane at any angle, slippage on fault planes, and the effects of backfill (e.g. Napier and Stephansen, 1987; Napier, 1991). The need for an improved understanding of the formation of mining induced fracturing has been highlighted (e.g. Adams et al., 1981) and modeling has shown, for example, that parting planes can form a barrier to fractures (Napier and Hildyard, 1992).

The above examples clearly demonstrate the importance of geological features and their impact on some rock engineering disciplines. In the following we will delineate areas of ultra-deep mining and some of their geotechnical characteristics.

AREAS, OREBODIES AND MINES/PROJECT AREAS UNDER CONSIDERATION

A comprehensive database has been compiled which facilitates the identification of ultra deep regions of mining. It also includes relevant reef features, such as dip and stoping width. Tonnages and depth of mining for the various Goldfields have been projected into the future. When this information is linked to the corresponding mines and reefs, it is possible to identify major areas where mining at ultra-depth will take place.

Figures 1 a and b, prepared from this data-base, show the percentage production for the Witwatersrand Basin for the years 1990, 2000 and 2010 for various stoping widths and dips. It is implied that the majority of orebodies mined within the future will have dips ranging between 0 and 30° and will be extracted at stoping widths of 120-180 centimeters.

The trend of increasing depth with advancing time for the whole of the Witwatersrand Basin is depicted in Figure 2. A shift towards deeper mining is projected, with some 15% of the total tonnage being forecast to be derived from depths exceeding 3500m in the year 2010.

Information deduced from the database suggests that ultra-deep mining will not occur in the Evander and Free State Goldfields. Limited mining at depths of about 3000m, however, takes place at Freddie's Gold Mine, where the Basal Reef is extracted. Ultra-deep mining in several other areas has been proposed, or is being developed: Target, Sun, Strathmore, South Deep and Moab or East Vaal. In addition, exploration has identified other deep level ore reserves which could be developed in the longer term (e.g. Potchefstroom Gap). The above examples represent the interests of all major South African mining houses and cover all current mining districts.

The identified regions and reef types to be mined at ultra-depth are illustrated in Figure 3. Stratigraphically, these orebodies are positioned in the Central Rand Group (Fig. 4). Most of the presently mined orebodies will be mined at ultra-depth. One of these is the exceptional Ventersdorp Contact Reef (VCR, e.g. Germs and Schweitzer, 1994), which is overlain by the volcanic rocks of the Ventersdorp Supergroup. Inspection of Figure 4 also reveals that the majority of orebodies currently being, or will be mined at ultra-depth, are associated with shale horizons. These horizons are characterized by frequent, closely-spaced bedding planes. The reefs and mines/project areas under consideration are also summarized in Table 1.

associated with shale horizons. These horizons are characterized by frequent, closely-spaced bedding planes. The reefs and mines/project areas under consideration are also summarized in Table 1.

Our approach is to regionally correlate stratigraphically corresponding orebodies (employing the recently proposed stratigraphic correlation scheme of the Witwatersrand Task Group of the South African Committee of Stratigraphy, SACS, S. McCarthy, pers. communication, 1994), assuming that similar geological processes occurred throughout the Witwatersrand Basin at the same time. For example, the Basal Reef is taken as the Orange Free State equivalent of the Vaal Reef in the Klerksdorp area; the Carbon Leader (West Wits Line) is the time equivalent of the Main Reef in the Central Rand.

In the following we will detail some of the geotechnical information according to Goldfield/Project area, straddling the margin of the Witwatersrand Basin in an anti-clockwise manner, from east to west. We consider strata characteristics between 100m beneath and above the orebody. This facilitates the consideration of the rock engineering environment in development areas.

East Rand Goldfields - ERPM mines the Main Reef at a depth of about 3500m and mining is expected to exceed 4000m in depth. The Main Reef is characterized by a quartzitic footwall and hangingwall (Fig. 5a). The rock engineering properties of the footwall and hangingwall quartzites vary laterally, with the quartzites being less competent in the eastern portion of ERPM (Fig. 5a). Argillaceous partings and shale horizons are present in the hangingwall at various distances from the orebody.

The majority of seismic foci in ERPM have been located in the hangingwall strata (McGarr et al., 1975), because it is able to store more strain energy than the footwall. This was thought to be as a result of the hangingwall having a significantly higher Young's modulus (86 GPa) than the footwall (75 GPa), Gay and Jager (1986). Our information, however, implies that these rock engineering properties are comparable for the footwall and hangingwall quartzites (Fig. 5a). A possible explanation is that reef parallel sills, such as aplite sills (rhyolitic in composition) in the hangingwall at ERPM have sharp contacts with the sedimentary host rocks which tend to burst or collapse, resulting in roof damage (Gay and Jager, 1986).

Central Rand Goldfield - Future, deep mining could occur in the South Central Rand (Fig. 2a). This area is located towards the south of City Deep, with the Main (Fig. 5a) and South Reefs potentially mined. It has previously been noted that most Central Rand tunnels are developed in undisturbed quartzite with uniaxial compressive strengths of 250 MPa (Gay and Jager, 1986), contributing to tunnel stability in this area.

West Rand Goldfield - Reefs to be mined at depth are the VCR (Fig. 5b), the Upper Elsburg's and probably the South Reef (Fig. 3). The South Reef is the time equivalent of the Main Reef, as encountered at ERPM, and is under- and overlain by quartzite (Fig. 5a).

The VCR is currently being mined at ultra-depth at Kloof Gold Mining Co. Ltd. and it is expected that this mining will continue. In this area it is predominantly underlain by synformally folded, competent quartzites, and minor conglomerates, of the Elsburg Quartzite Formation (Fig. 5b). The hangingwall consists predominantly of Westonia Formation lavas (colloquially termed "soft" lava) with a distinct parting, which separates the soft from the hard lava, being situated some <1-40m into the hangingwall. It is believed that the presence of this "soft" lava in the hangingwall contributes significantly to the stability of deep level stabilizing pillars. The average stoping width is about 120m, and selected cut extraction is practiced in areas where the uppermost portion of the VCR consists of unpayable quartzite. The average thickness of the VCR at South Deep is 150 cm (highly variable), dips at 29° towards the east (Haslett, 1994), and is, similar to the VCR at Kloof Gold Mine, mined conventionally (Circular to Members of Western Areas and South Deep, 1995).

The Upper Elsburgs' at South Deep will be mined differently along their extent, due to lateral lithological variations. Trackless mining is used to mine the proximal, thick, conglomeratic portion of this orebody in the west, whereas conventional mining methods will be applied in the distal, deep eastern portion, highlighting the importance of orebody geometry. In the eastern portion individual gold bearing conglomerate horizons are separated by unpay quartzites (Circular to Members of Western Areas and South Deep, 1995). Up to 15 mineable orebodies are contained in the Upper Elsburgs' and the thicknesses of conglomerate horizons vary between 50-1200 centimeters.

At South Deep, pillars are strategically placed along the N/S-trending faults and dykes to limit spans and ERR. Dykes are ultrabasic and syenitic and trend N/S and E/W. Three phases of faulting are associated with the VCR, characterized respectively by cataclasis, pseudotachylite, and slickensides.

Carletonville Goldfield - Reserves of the Carletonville Goldfield are known to extend to at least 6000m below the surface (Tamlyn, 1994) and we will consider the Carbon Leader at Driefontein Consolidated Mines Ltd. and Western Deep Levels, and the VCR (Fig. 5b) at Western Deep Levels, Elandsrand Gold Mine, Deelkraal, and to a lesser extent Driefontein. In deep areas at Driefontein provision has been made for 18% geological loss and 12% loss due to stabilizing pillars (Tamlyn, 1994). Faults strike NE/SW and N/S and exhibit displacements of several hundred meters.

The Carbon Leader is generally characterized by a competent quartzite footwall, with a complex succession of rock types overlying this orebody (Fig. 5c). These are a bedded, siliceous quartzite (0-3m in thickness), the Rice Pebble Marker (0.5 to 1m in thickness) and the shaly Green Bar (1 to 5m in thickness). Pronounced partings are developed along the contacts of these lithologies. Partings are characterized by quartz veining and slickensides.

Pronounced lithological variations are observed beneath and above most of the reef horizons considered. The four major footwall and hangingwall lithologies considered are shales, quartzites, argillaceous quartzites, and conglomerates. Two additional hangingwall lithologies overlie the VCR. These are informally termed the soft and hard lava, which correspond with the official terms Westonaria Formation and Alberton Porphyry Formation respectively.

The majority of reefs considered are located in a complex geological environment, and are also characterized by variable associations of footwall/hangingwall lithologies. Secondary geological features are also prominent. Different geological environments will be expressed by different mining conditions at ultra-depth and will impact on the mining strategy. The compiled information will be used as input for numerical modeling.

DISCUSSION

Many of the current deep level mine-layouts and strategies will need to be modified or replaced for mining at ultra-depth. A better understanding of the geotechnical environment at ultra-depth is an essential input in assessing such mining methods.

Recently, it has been confirmed that geological characteristics, such as rock-type, influence the rock mass behavior around excavations, e.g. the closure rate or the attitude of the mining induced stress fracturing (M. Roberts, pers. communication, 1994). However, our review reveals that the quantification of primary and secondary geological features of rock engineering significance has only commenced. It is recommended that these features receive more attention in the future, and that the information is continuously evaluated in a multidisciplinary team consisting of rock engineers and geologists.

A detailed review of the various fault and dyke populations, and associated joint characteristics, as observed in the regions to be mined at ultra-depth is beyond the scope of this study. However, examination of the literature (Antrobus, 1986; Antrobus et al., 1986; Engelbrecht et al., 1986; Minter et al., 1986; Tucker and Viljoen, 1986; Tweedie, 1986; Killick and Roering, 1994; McCarthy, 1994; Vermaakt and Chunnnett, 1994; Berlenbach, 1995) reveals that fault and dyke characteristics and attitudes differ within the various Goldfields. Frequency and extent of faulting impact on mine layout. Scattered mining is practiced in heavily faulted terrains, with longwall mining being performed in structurally less disturbed areas. Jointing is closely linked to the various fault and dyke populations and it is anticipated that joint characteristics and attitudes will also differ from Goldfield to Goldfield. This needs to be confirmed through future investigations. The interpretation of the various structural settings is complicated due to the frequent reactivation of the various fault and dyke planes (e.g. Berlenbach, 1995), resulting in a complex structural setting. The rock engineering significance of different fault rock types (e.g. Roering et al., 1991) also needs further attention. Different fault rocks could possess distinct slip proneness.

The VCR depth contours and associated footwall lithologies at depth between 3000 and 4500m for Western Deep Levels are shown in Figure 6. A great variety in the features of footwall quartzites is noted, ranging from siliceous to argillaceous, with varying chlorite contents indicated by the variations in colour. Conglomeratic footwall horizons are less frequent and shaly footwall is encountered in the eastern portion of Western Deep Levels (Fig. 6). The hangingwall of the VCR towards the west of Driefontein consists of hard lava only.

Klerksdorp Goldfield - Future mining at depth will largely concentrate on the Vaal Reef (Fig. 5d) in the eastern portion of Vaal Reefs Gold Mine (Moab extension). The Vaal Reef is generally overlain by "weak" argillaceous, horizontally extensive quartzites. Argillaceous partings and shale horizons are present in the footwall and hangingwall.

Production at Moab, where the sub-vertical shaft goes to depths of about 3700m, is expected to commence in the shallow areas towards the end of 1996 (Gilroy et al., 1994). Channel widths are about 83 cms with the dips varying in between 22 and 30°. Faulting is pronounced with three major stages of faulting being observed. Displacements are generally in the region of several hundreds of meters, some even exceeding 1000m.

Analyses of quartzites the footwall and hangingwall of the Vaal Reef reveal that the compressive strengths and Young's Moduli are significantly higher in siliceous quartzites than in the argillaceous quartzite (Gay et al., 1984). The Ventersdorp Supergroup dykes are known for their burst-proneness (Gay and Jager, 1986).

Bothasville Goldfield - Several, economically important reef horizons have been identified at Sun and Target (e.g. Gray et al., 1994; Tucker et al., 1994), which occur at depths exceeding 3000m. The reef horizons are the Basal Reef, B Reef (or Mariasdal Reef), A Reef, Elsburg Reefs, Big Pebble Reef, and the Kruidfontein Reef (VCR equivalent). Borehole MA1, for example, intersected the Basal Reef, showing encouraging values, at a depth of 3975m. The Khaki shale and the argillaceous quartzite in the hangingwall and footwall of the B-reef respectively, and shale horizons are especially common within the Kimberley succession, or Aandenk Formation which are associated with several conglomeratic bands of the Big Pebble Reefs and the Mariasdal Reef are known to impact on mining conditions.

SUMMARY

About 20 mines and project areas that are mining, or will potentially mine, at great depth have been identified. The VCR, Carbon Leader, Main Reef, reefs of the Elsburg and Kimberley successions, Vaal Reef, and the Basal Reef are the major targets. Some of the important rock engineering characteristics of these reefs are summarized in Table 2.

Evaluation of the data reveals that a variety of rock types, having distinct rock engineering properties, are located in close proximity to the excavations. Partings and dykes are also prominent.

The Witwatersrand rocks were, on a number of occasions, intruded by dykes with varying textures and compositions. Compositions vary from basaltic to rhyolitic, which is also reflected in varying competencies of the various intrusive rocks. Future work could consider the potential link between the texture/composition of the dykes and their rock engineering properties. Establishment of such a link would enhance the prediction of dyke behavior at ultra depth.

It has previously been stated that rock engineering problems are affected by unpredictable discontinuities and mining proceeding into unfamiliar geological terrain (e.g. Atkins and Keen, 1984; Piper, 1985). We would like to argue this statement. A combination of the various geological disciplines, in conjunction with rock engineering, enables the projection of the various rock types, and their associated features (such as rolls; rock engineering properties) into the unknown. Proximal/distal relationships are similarly predictable into the unmined (van Niekerk et al., 1990, 1994). Distinct footwall, but also hangingwall, strata, and the transitions in between these, can be related to distinct reef geometries, to depths exceeding 5 km. Seismic surveys and other geophysical techniques, in addition, enable the prediction of geological discontinuities on a local and regional scale (Fenner et al., 1994; Circular to Members of Western Areas and South Deep, 1995).

CLASSIFICATION

Data and information gathered as part of this work has been represented in a classification by reef and depth as shown in Figure 6. Mining condition is made up of a number of weighted factors for each reef and for different depths to obtain a single classification. This is almost certain to change for different districts and mining methods. Weightings may be changed and additional information included. It is hoped that this will initiate discussion and improvements towards a useful geotechnical/rock mechanics classification.

Acknowledgements

This work formed part of SIMRAC GAP033. The rock engineers and geological staff of the various Mining Houses are thanked for their contributions and cooperation during the course of this study. Mr P Willis is acknowledged for providing most of the information that was incorporated into the database. Mr Dave Arnold and the staff at ERPM have contributed significantly to this work.

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DEEP LEVEL PRODUCTION TRENDS

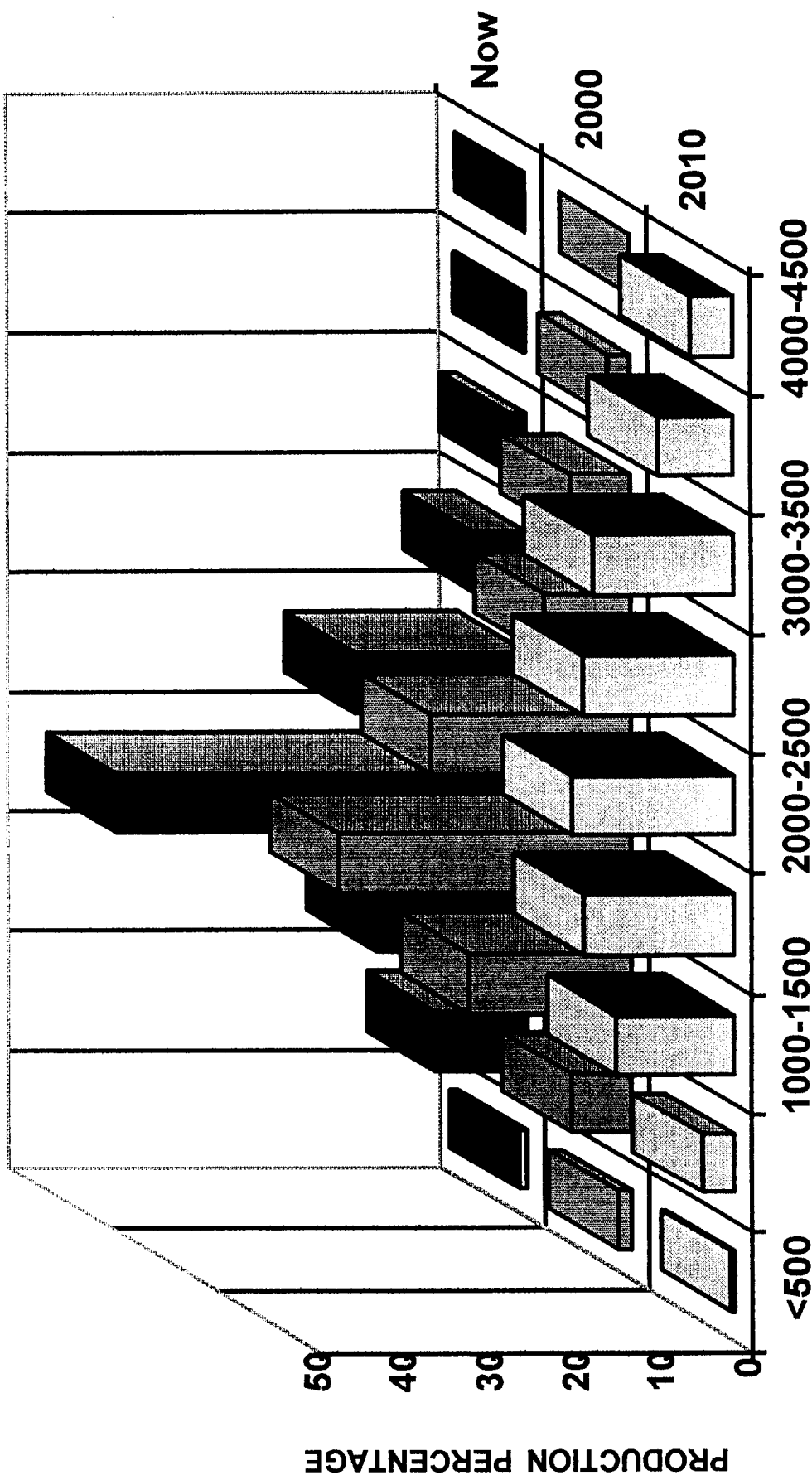


Figure 2. Percentage production versus time and depth below surface as determined and projected for the gold production from the Witwatersrand basin. Data courtesy of P. Willis (Pers. communication, 1995)

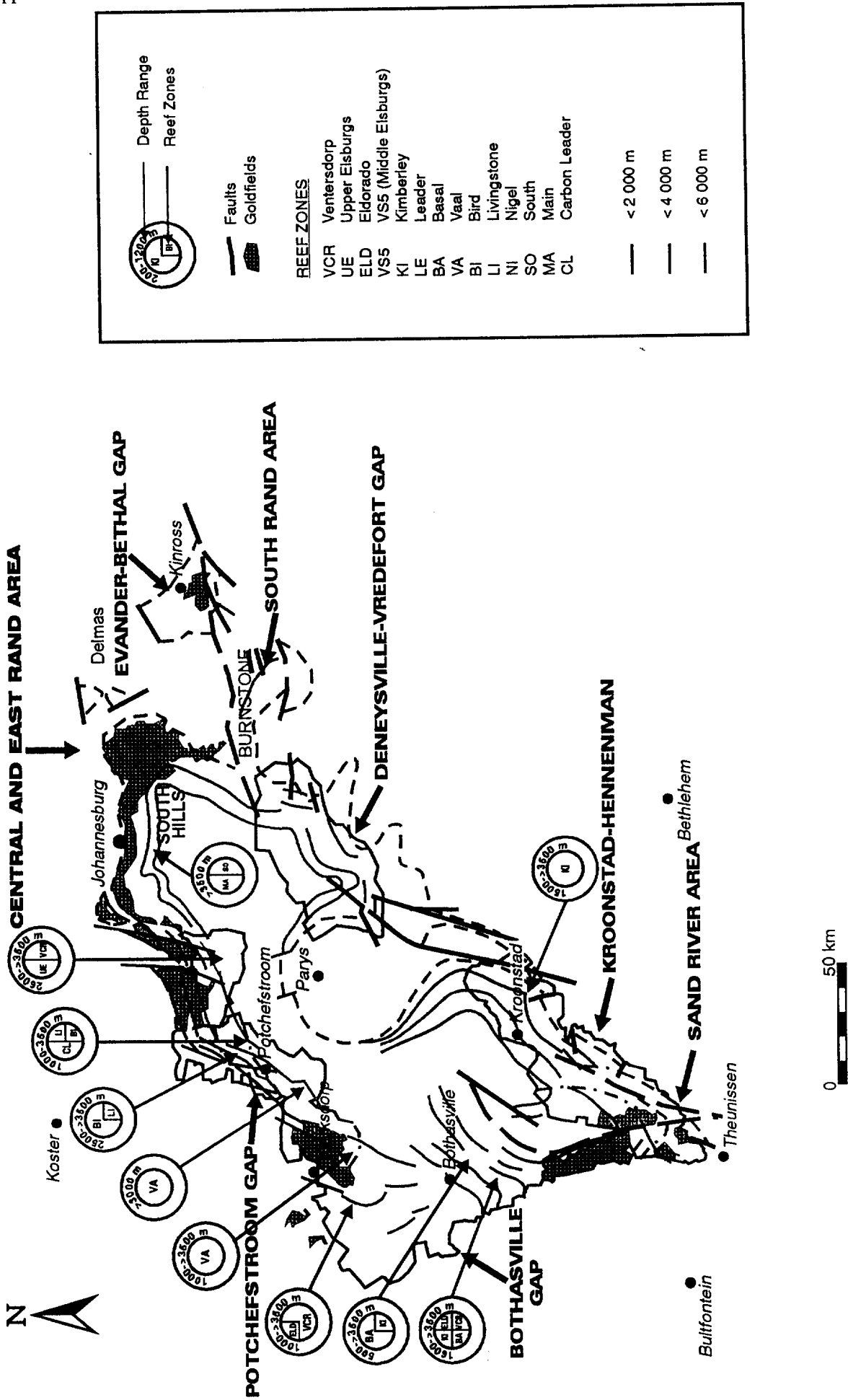


Figure 3: Outline of the Witwatersrand Basin, the Goldfields and associated mines and depth contours for the Central Rand Group. Also shown are reef-type and depth-range for potential mines in exploration areas. Information was also deduced from Tainton (1995) and The Witwatersrand Basin Map (1986).

OREBODIES CURRENTLY AND POTENTIALLY MINED AT DEPTHS EXCEEDING 3 500 m

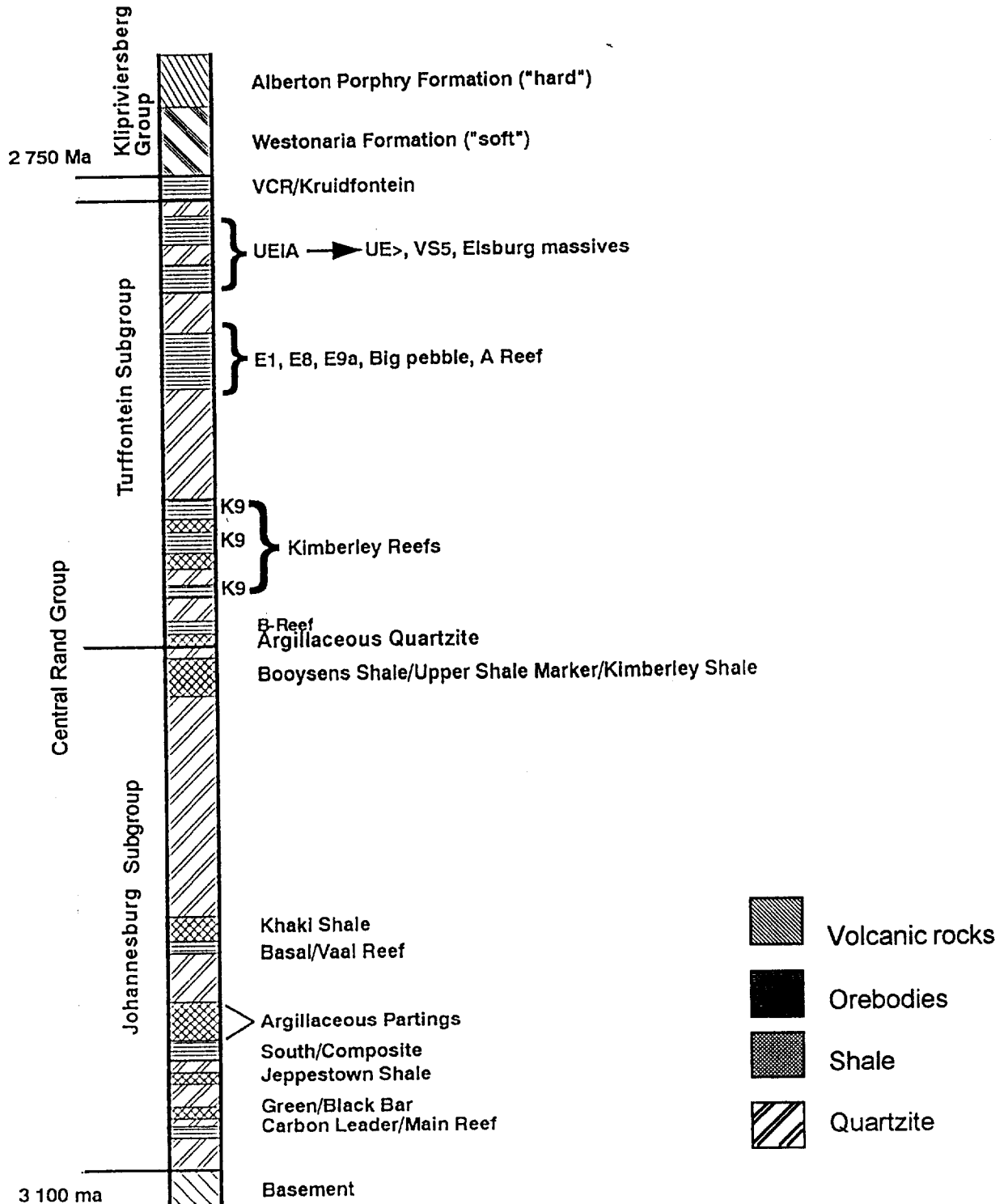
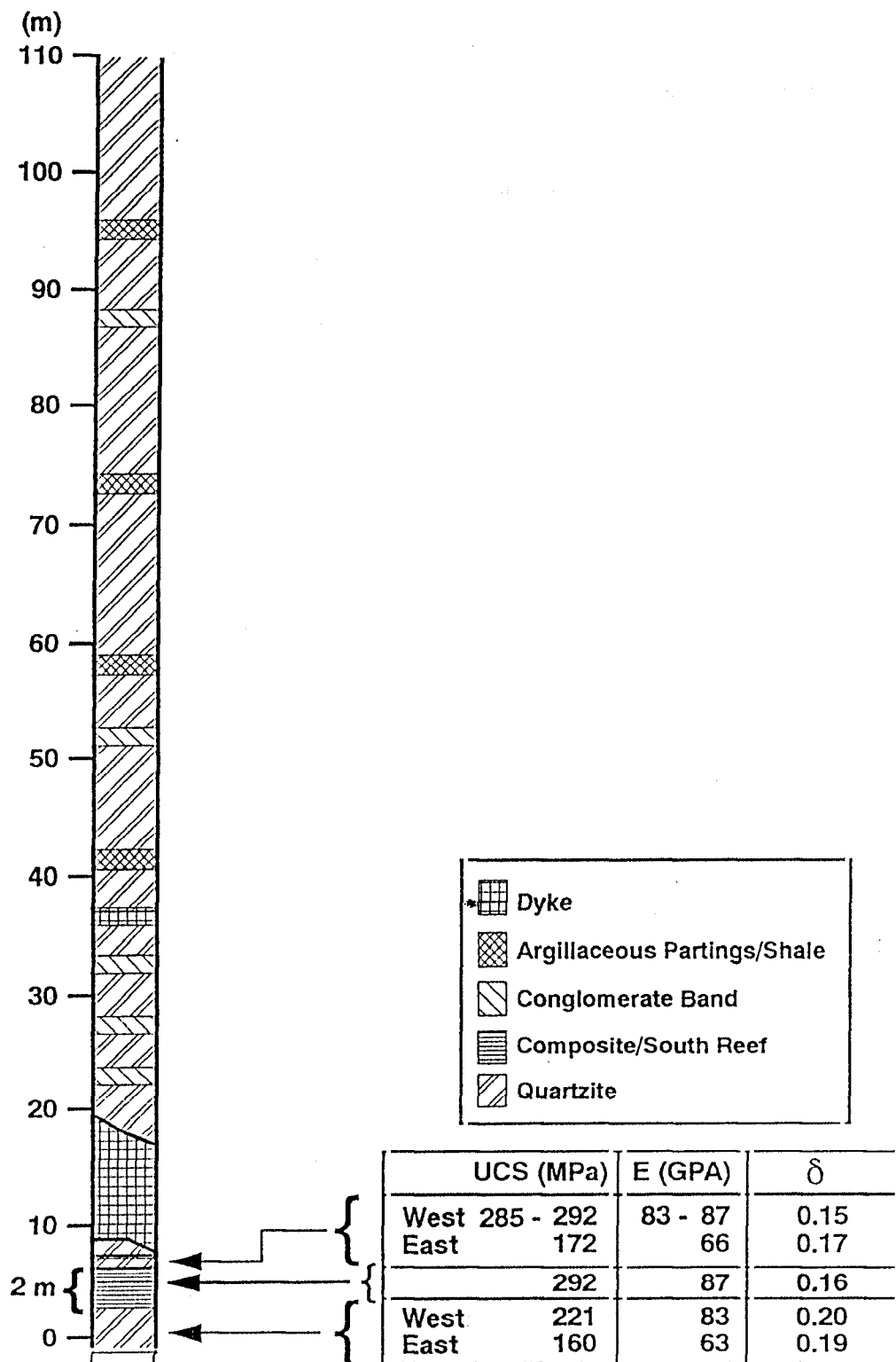


Figure. 4: Schematic stratigraphic column of the Central Rand Group delineating the orebodies that are currently and will be potentially mined at depths exceeding 3500m.

SCHEMATIC PROFILE OF THE ROCK TYPES ASSOCIATED WITH THE COMPOSITE/SOUTH REEF



Figures 5 a - d: Stratigraphic profiles delineating the various rock types associated with the Main Reef (Fig. 5a), Ventersdorp Contact Reef (Fig. 5b), Carbon Leader (Fig. 5c), and the Vaal Reef (Fig. 5d). Also provided are some rock engineering property analyses for the reefs and footwall and hangingwall strata.

SCHEMATIC PROFILE OF THE ROCK TYPES ASSOCIATED WITH THE VCR

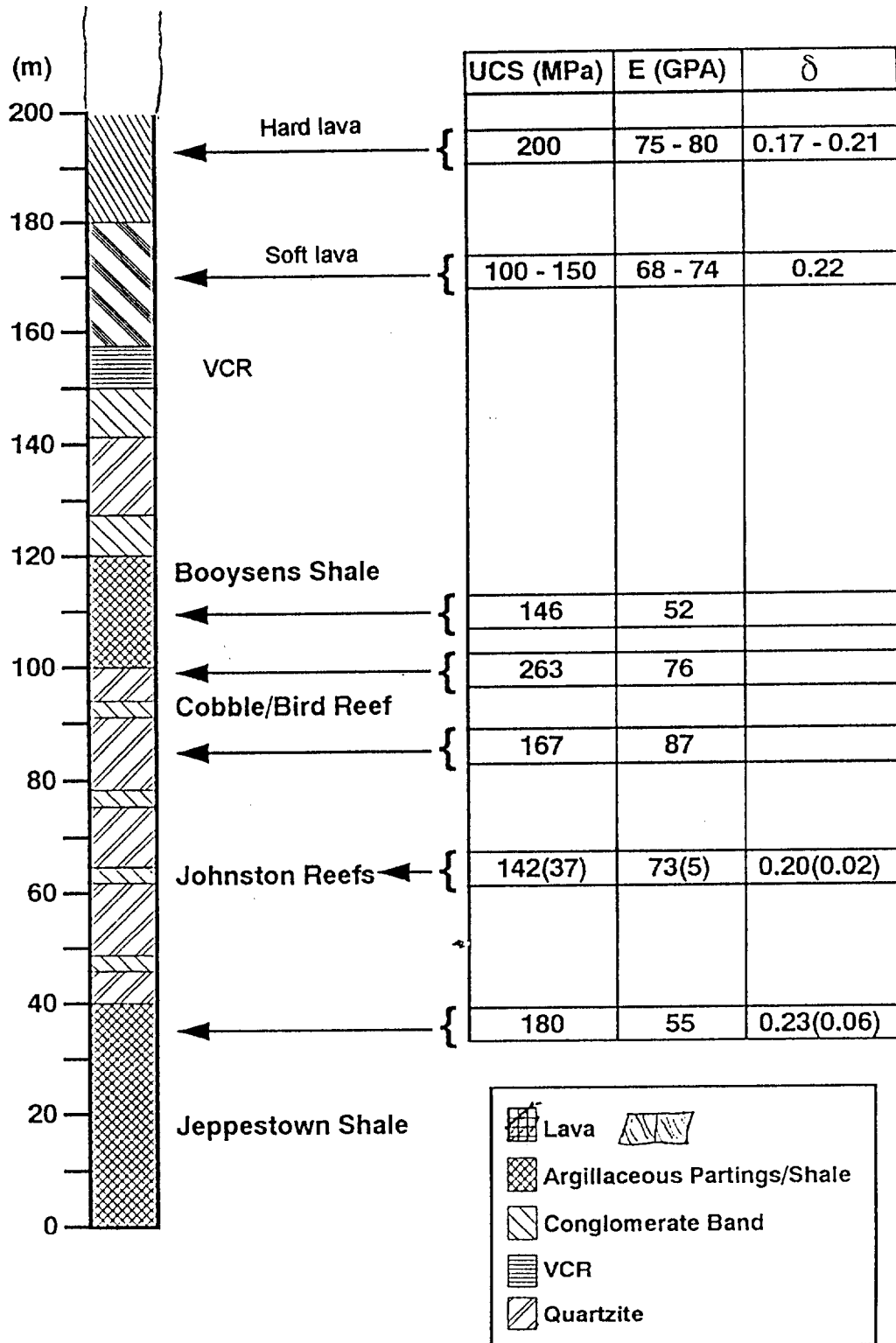


Figure 5b

SCHEMATIC PROFILE OF THE ROCK TYPES ASSOCIATED WITH THE CARBON LEADER

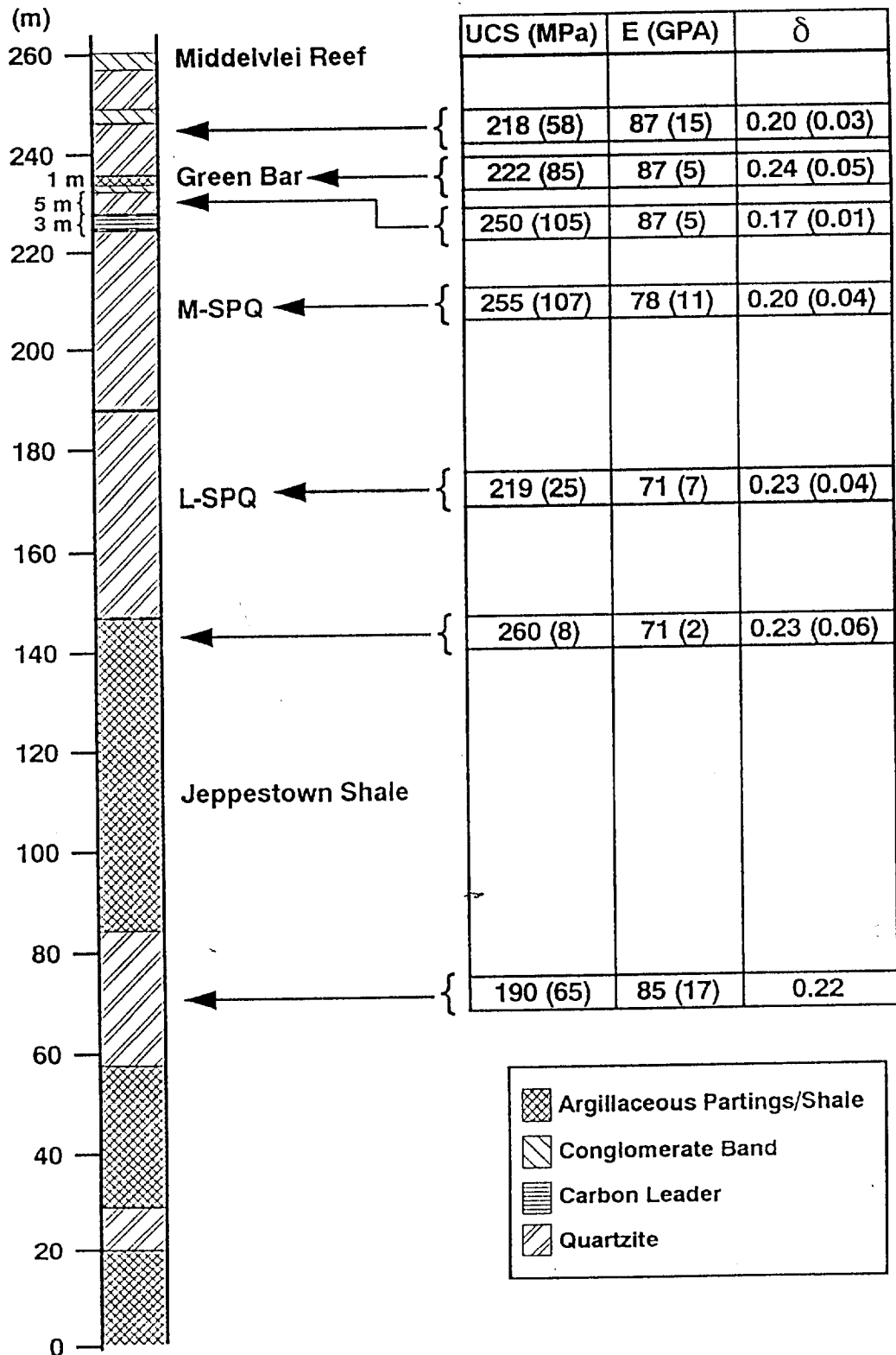


Figure 5c

SCHEMATIC PROFILE OF THE ROCK TYPES ASSOCIATED WITH THE VAAL REEF

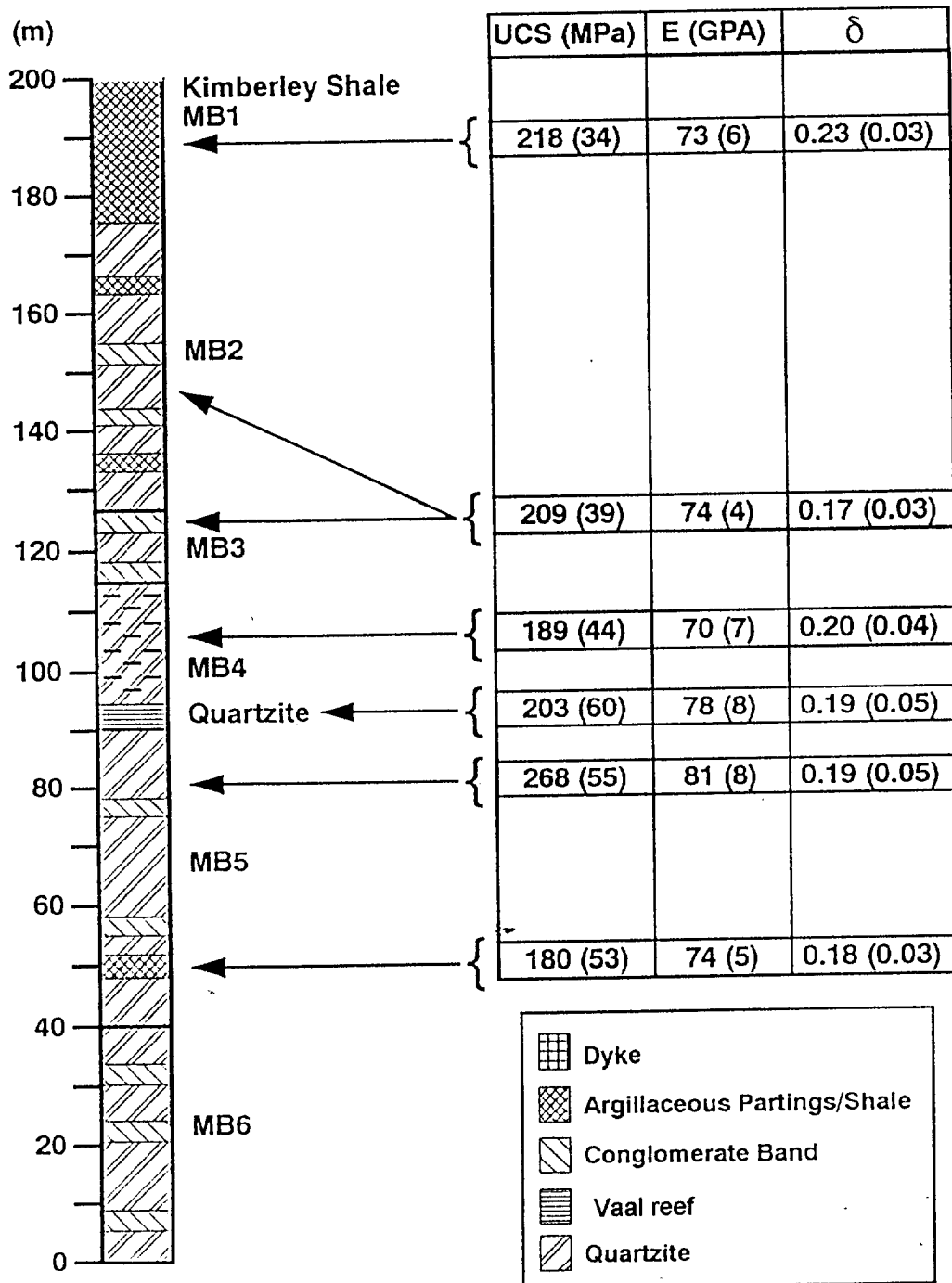
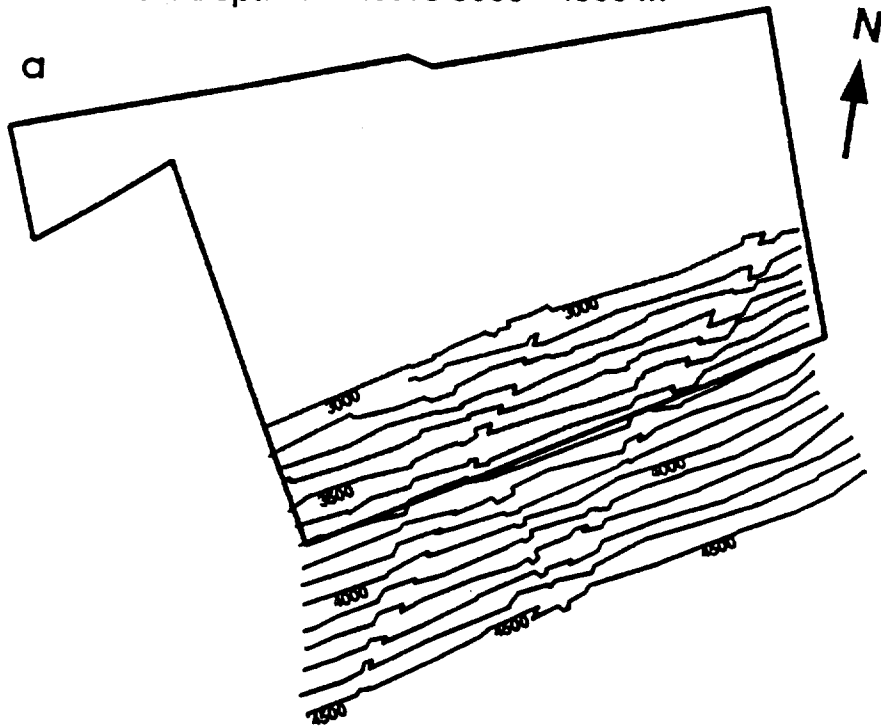
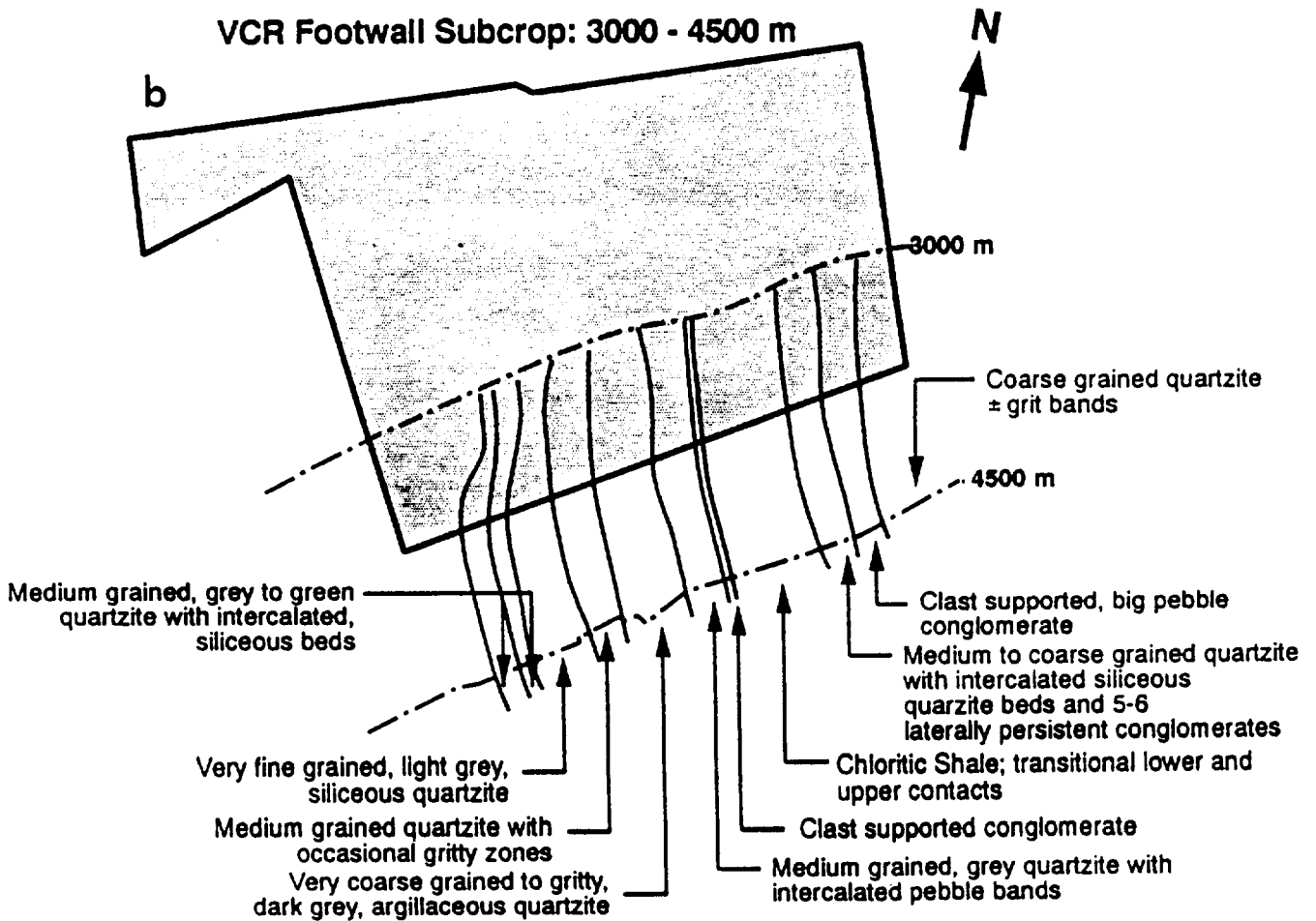


Figure 5d

WESTERN DEEP LEVELS
VCR Depth Contours 3000 - 4500 m



VCR Footwall Subcrop: 3000 - 4500 m



Figs. 6a and b: 3000m to 4500m depth contours for the VCR at Western Deep Levels - South Mine (Fig. 5a) and the corresponding variations in footwall lithologies. The hangingwall is comprised of hard lava in this area.

GEOTECHNICAL AREAS CLASSIFICATION

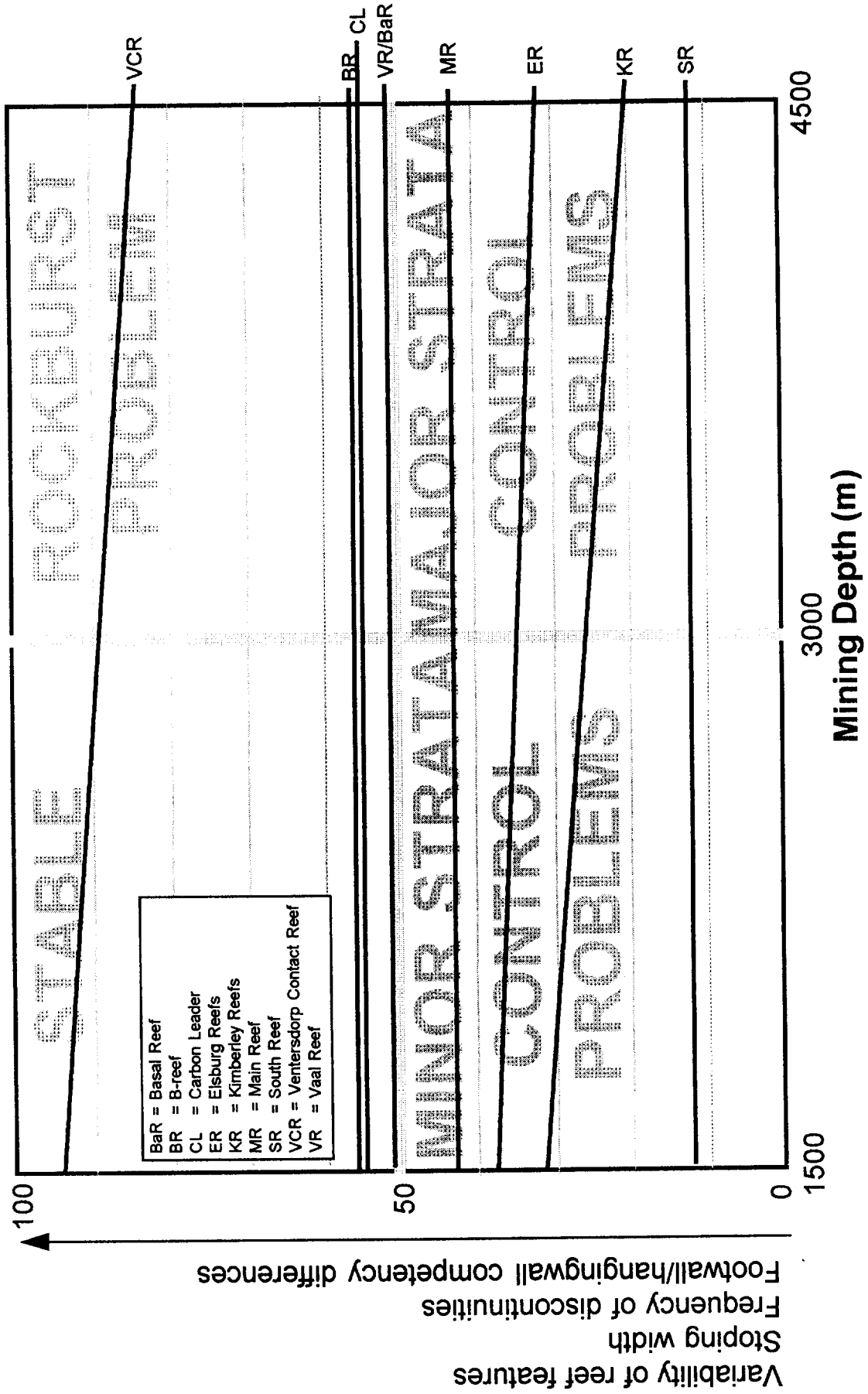


Figure 7. Proposed geotechnical classification - a weighted mining risk for differing reefs and depths.

GOLDFIELD	MINE/PROJECT AREA	REEF	MINING HOUSE
Evander	Rosespruit	Kimberley	GENCOR
Central Rand	ERPM South of City Deep Durban Roodepoort Deep	Composite Reef Main Reef Main Reef, Kimberley	Randgold Randgold Randgold
West Rand	Randfontein Estates South Deep Kloof Libanon/Venterspost Leeudoorn	South Reef VCR, Elsburgs VCR VCR, Main Reef VCR	JCI JCI GFSA GFSA GFSA
Carletonville	East Driefontein West Driefontein Western Deep Levels-North Western Deep Levels-South Elandsrand Deelkraal	VCR, Carbon Leader VCR, Carbon Leader Carbon Leader VCR VCR VCR	GFSA GFSA AAC AAC AAC GFSA
Klerksdorp	Buffelsfontein East Vaal (Moab)	Vaal Reef Vaal Reef	GENCOR AAC
Bothasville Gap	Sun/Target	Basal Reef, B-Reef A-Reef, Elsburgs, Kimberley, VCR	AV
Orange Free State	Freegold Sand River	Basal Reef Kimberley, Leader Reef	AAC GFSA

Table 1: Summary of mines/project areas and reef types that are currently and will potentially mine at ultra-depth.

OREBODY	REGION	FOOTWALL	HANGINGWALL	CHANNEL WIDTH	GRADE	DIP (deg.)	COMMENTS
VCR/ (Kruidfontein)	Carletonville Klerksdorp Bothaville	Highly variable Shale-Conglomerate	Westonaria Formation (0-40 m) Alberton Porphyry Formation (< 500 m)	0 - 2.00 m	Erratic	15 - 35	Rolls Pilloids Bedding parallel faulting
Upper Eisburgs/ Massives	West Rand	Quartzite	Quartzite	1.20 - 1.80 m > 1.80 m	Erratic	0 - 15	Pillar spacing
Eisburgs-UEIA BP, A	West Rand	Quartzite	Shales above grey hangingwall quartzite	1.20 - 1.80 m > 1.80 m	Erratic	0 - 15	Pillar spacing
Kimberley	Bothaville West Rand Evander	*K ₈ * Shale Quartzite	\pm Argillaceous partings	1.20 - 1.80 m	Erratic	15 - 45	Channels support on shales
B	Bothaville	Strongly Argillaceous Quartzite Shale	Quartzite	0.10 - 2.00 m	Even-erratic	15 - 45	Channels < 200 m wide
Steyn/ Basal/Vaal	Klerksdorp Orange Free State	Quartzite	Argillaceous Quartzite Shale	(< 1.20) 1.20 - 1.80 m	\pm Even	15 - 30 (OFS) 15 (Klerks.)	Flat channels Closure
South/ Composite	Central Rand	Quartzite (\pm Shale)	Quartzite	1.20 - 1.80 m	\pm Even	30 - 45	
Carbon Leader/ Main	Carletonville	Quartzite	Quartzite (Bedding planes) Shale (1 - 2 m thick)	1.20 m	\pm Even	15 - 30	Green Bar

Table 2: General characteristics of the orebodies and associated strata under consideration.

Appendix 7.
CRUSH PILLARS

The Influence of Limited Backfill Confinement on Quartzite Material in the Post-failure Regime

R. A. Johnson

Abstract

The strengthening effects of limited uncemented backfill confinement on the post-failure behaviour of quartzite samples of several aspect ratios has been measured in laboratory studies and replicated with the use of numerical modelling. These results may be extrapolated to the use of backfill reinforced crush pillars as part of a mining method applicable in high stress environments and having both local and regional support benefits. The results are detailed and applications are discussed in this paper.

Introduction

Experience to date with the use of squat pillars for regional support in deep level gold mining gives rise to several concerns at increasing depth. In certain geotechnical environments stabilizing pillars are prone to foundation failure and pillar induced seismicity with associated damage to haulages and working areas causing concern related to worker safety and productivity. In addition there are economic considerations with extraction percentages as low as 60 - 80 %, a figure likely to decrease with increasing depth. These pillar reserves are at best sterilised for many years and are inherently difficult to extract.

The need for effective regional support to limit closure and hence ERR and seismicity is well established (Anon, 1988). Recent work indicates that more than extensively placed backfill is required to achieve acceptable levels of ERR and face stress at depths exceeding 3000m (Johnson, 1995). Adams et al (1989) have shown that concrete pillars have potential, however this requires significant changes to the mining method, is expensive and difficult to place. Several other methods including crush pillars are being investigated at present for application in high stress conditions anticipated at ultra-depth.

The crush pillar alternative is being considered because they are proven effective at shallower depths. At ultra-depth such pillars will be crushed ahead of cutting eliminating the risk of bursting. Yield pillars can accommodate convergence without bursting (Roberts and Jaeger, 1993). It has been found that crush pillars alone at >90% extraction cannot satisfy currently accepted criteria, and increasing the pillar percentage is unattractive because of the complexity of mining and economic considerations. Recent findings on the effectiveness of backfill in reducing accidents (Gurtunca, 1995) coupled to the need to stiffen such fill for application at ultra-depth motivates the investigation of crush pillars in conjunction with extensive backfilling. This can give in excess of 90% extraction and promises to provide acceptable mining conditions. It is felt that the backfill will increase the strength of the crush pillars (borne out in this study) and that the crush pillars will in turn provide early stiffness to the face area improving local support. This is similar to the finding of Ryder (1992) who describes how flyash surrounding pillars and not necessarily placed to the roof, can stabilise coal pillars. The method is used to some extent in Canadian mines, as reported by Swan & Board, 1989.

Little information is available on the post-failure behaviour of brittle rock. Interesting early work by Jaeger & Gay (1974) showed that fractured rock fragments and powder compressed between plattens is able to sustain very high stresses. At lower strains (5 - 40%) a residual of around 12 MPa was obtained for unconfined 2:1 cylindrical samples. This is similar to the load transferred in blocky assemblages such as cave material (Arnold, 1994 & Johnson, 1994). Similarly Kicker & Park (????), also in a laboratory experiment found that yielded material (pillars) recompacted and was able to carry very high stresses, concluding that such yield pillars have potential for application in controlling high stress areas.

Several papers describe & quantify the use of yield pillars in hardrock stope support at shallower depths, where these principally perform a strata stabilizing function. Ozbay & Ryder (1988) examine such pillar application using an analogue between UCS laboratory testing and loading of a single pillar, using width to height ratio of greater than one. In outlining design considerations they distinguish between intact and fractured pillars at the time of cutting. They cite an empirical residual strength of a pillar as 13 MPa, which is similar to the Jaeger & Gay result.

This paper reports on a laboratory study that has been carried out to investigate the confining effects of backfill on quartzite samples varying the extent of fill and the pillar aspect ratios. The significant findings of the laboratory work are that the confining effects of limited backfill dramatically strengthen the pillar and that a residual strength as low as 2 - 10 % of the UCS was obtained. The latter result being significantly lower than anticipated.

Method

Preliminary modelling using FLAC confirmed the potential of the crush pillar/backfill combination as a support technique. An experimental program was carried out at the University of the Witwatersrand using their MTS servo-controlled testing facility (see Figures 1a & 1b), to quantify the post-failure behaviour of brittle quartzite and the influence of backfill on this behaviour.

Standard uni- and tri-axial tests (see Figure 2a) were carried out to give numerical modelling parameters. Backfill likewise was tested yielding results shown in Figure 2b.

52 mm cylindrical samples of footwall quartzite were prepared at different lengths to make up 1:1, 1½:1, 2:1 and 3:1 width to height ratio samples. These were prepared to a close tolerance of better than 10 microns parallelity.

Initially circumferential control was used on 1:1 samples, however, due to the brittle rock behaviour and servo-response as shown in Figure 2c. This method proved unattractive as the test duration was excessive and results difficult to interpret.

It was possible to fail samples with a width/height ratio > 1:1 in a stable manner using only the coarse LVDT control on the main servo, which simplified the testing process. All samples were tested to initial failure after which samples were alternately tested with and without backfill confinement to around 40%

strain (see Figure 3a). Those without fill providing the unconfined large strain behaviour of the material, or the residual strength. Others were surrounded by a 52mm annulus of backfill (1 sample diameter) - see Figure 3b at either 100%, 90% or 50% fill height (Figure 3c) placed immediately after failure. The backfill was essentially unconfined, a paper sleeve being used only to facilitate its placement.

Most of the testing was carried out using 1½:1 and 2:1 width to height samples as these seem most suited to underground application in high stress conditions. Photographs in Figure 4 illustrate different stages of the experiment: a) Sample between plattens around the time of failure, with damage and spalling of the sample evident. b) Typical sample damage immediately after failure. c) Backfill placed, confined by a paper sleeve. d) Sample and fill after testing to 40% strain. It can be seen that the sample, though damaged & crushed is relatively intact. It is usually possible to remove a single re-compacted sample from the fill after testing.

Summary of Results

Unconfined post-failure results for 2:1 sample are shown in Figure 5. Initial failure occurred consistently at between 200 and 230 Mpa and at around 2½% strain. In six of the seven cases a residual of between 5 and 12 MPa was obtained for strains from 5% to 35%, after which an increasing trend is evident. One sample was significantly higher at 15 - 20 Mpa. On average the residual stress lies between 2 and 6 % of the UCS.

Confined (filled to 100% of the sample height) results again for the 2:1 case are shown in Figure 6. This indicates that such fill confined samples are on average 40 times stronger than the unfilled cases at 35% strain. Even at 10% strain the filled case is 3 - 4 times stronger.

Similar results were obtained for the 1½:1 samples (Figure 7). In this case the filled samples were on average 5 times stronger at 35% strain, again with an increasing trend. One sample at 3:1 was tested to 75 times as strong with fill at 35% strain - see Figure 8. The platten failed at around 1000 MPa, apparently in tension caused by friction and sample dilation.

The above results are summarised in Figure 9, with Figure 9b showing the influence of fill height on the confined sample strength. Even filled to only half the sample height gives a threefold increase in the post-failure strength. The average for 90% filling gives an 22 fold increase in strength, and while only 55% of the fully filled case is surprisingly high.

Discussion and Application Potential

It is felt that these results encourage the application of crush pillars to some specific mining conditions both at high stress and in deep levels. A methodology for designing crush pillars, albeit for shallower conditions, is given in Ryder & Ozbay (1990) which includes the influence of size, shape, strength and aspect ratio. Pillar residual strengths of 13 - 15 MPa have been found by Jaeger & Gay (1974) and by Ozbay & Roberts (1988). The former show substantial increase in these values at strains as high as 70%. Roberts and Jager (1993) empirical results similarly show an increasing trend at 40% strain. They further emphasise the need for geotechnical data as an input to such pillar design. Parameters such as

joint spacing & characteristics, intact and post-failure characteristics of the pillars and surrounding strata, stress levels and k-ratio.

Ozbay & Roberts (1988) describe several layouts using crush pillars for shallow and intermediate depths - see Figure 10 a & b. With some modification as in Figure 10c crush pillars with backfill may provide a suitable method for mining in high stress conditions anticipated at ultra-depth. This promises improved local support, by stiffening the backfill, which in the back areas at higher closures providing an effective regional support function.

Conclusions

Residual strengths were found to be consistently between 5 & 12 MPa corresponding to 2 - 6 % of the UCS. These values are considerably lower than anticipated thought. These results vary little with changes in width to height between 1½ and 3:1. The larger width/height samples show a more marked tendency to increase in strength with increasing deformation.

The strengthening effects of even very limited backfill are significant. The 1½:1 pillar being three times as strong with fill, with the 2:1 and 3:1 cases some 40 and 75 times as strong respectively. In cases where the fill is 90% of the sample height, the above figures are reduced by about 40%. Fill to only half the sample height provided a three fold strength increase.

While further work is indicated, the laboratory work to date confirms the potential of using backfill & crush pillars in which the fill strengthens the crush pillars at higher strains and the pillars provide early stiffness to the fill in the face area. There may well be other application areas for backfill to strengthen or stabilise underground pillars.

Further Work Indicated

This work has led to several significant conclusions, however further work is necessary to apply them in practise. Underground trials of pillar/backfill combinations are essential to translate the lab results to in situ conditions.

No results available on the stress distribution in the crushed samples. Examination of the samples would indicate highest stresses at the confined core. This could be verified with additional laboratory or numerical modelling.

Acknowledgements

The assistance of Dr Ryder with the modelling and planning of the experiment is acknowledged. The practical assistance and advise of Dr Ozbay and Mr Dede at Wits university are appreciated, as well as the work of Mr Henderson who carried out much of the testing.

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Figure 8. a) 3:1 results including platten failure.

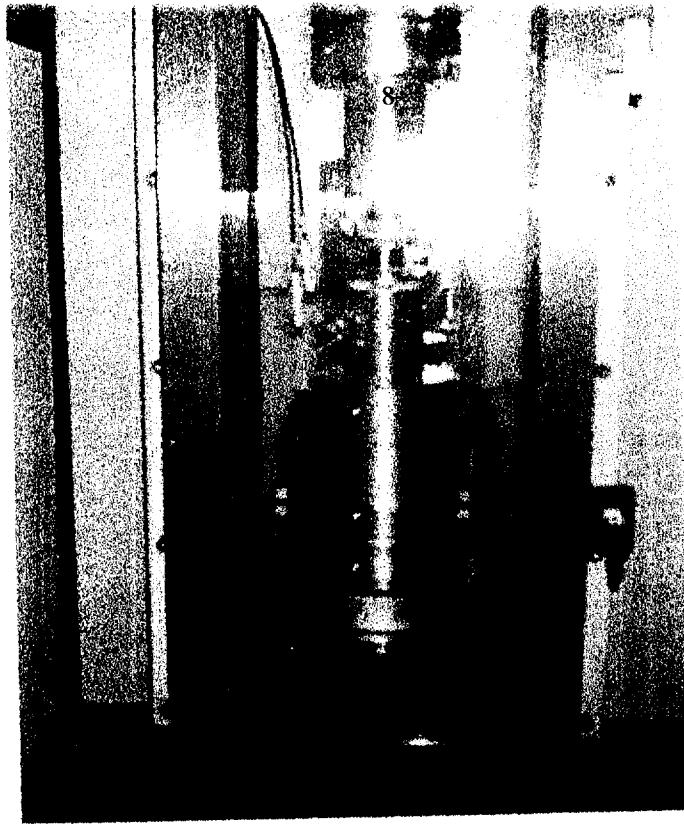
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Figure 10. Possible crush pillar layouts:

- a & b) Examples of yield pillar layouts used in shallow mines. (After Ozbay & Roberts, 1988)
- c) Proposed deep level crush pillar and backfill layout.



b

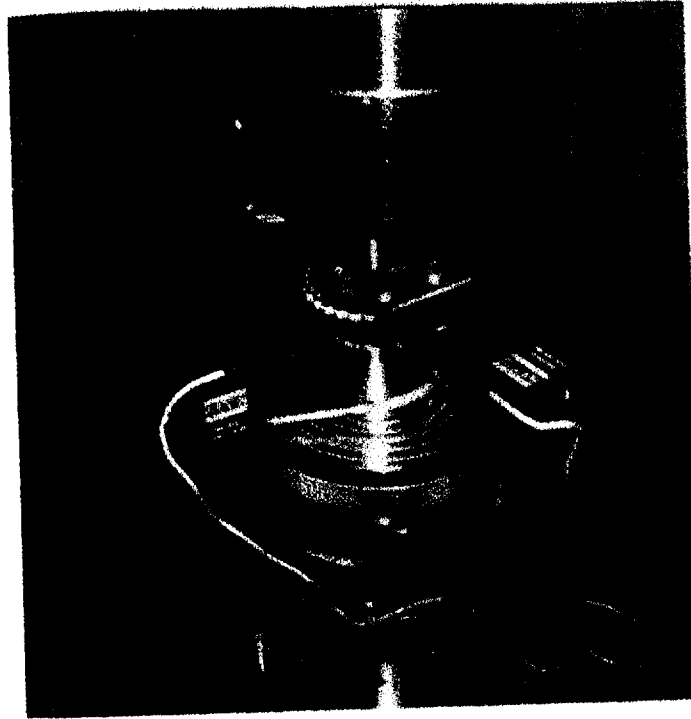


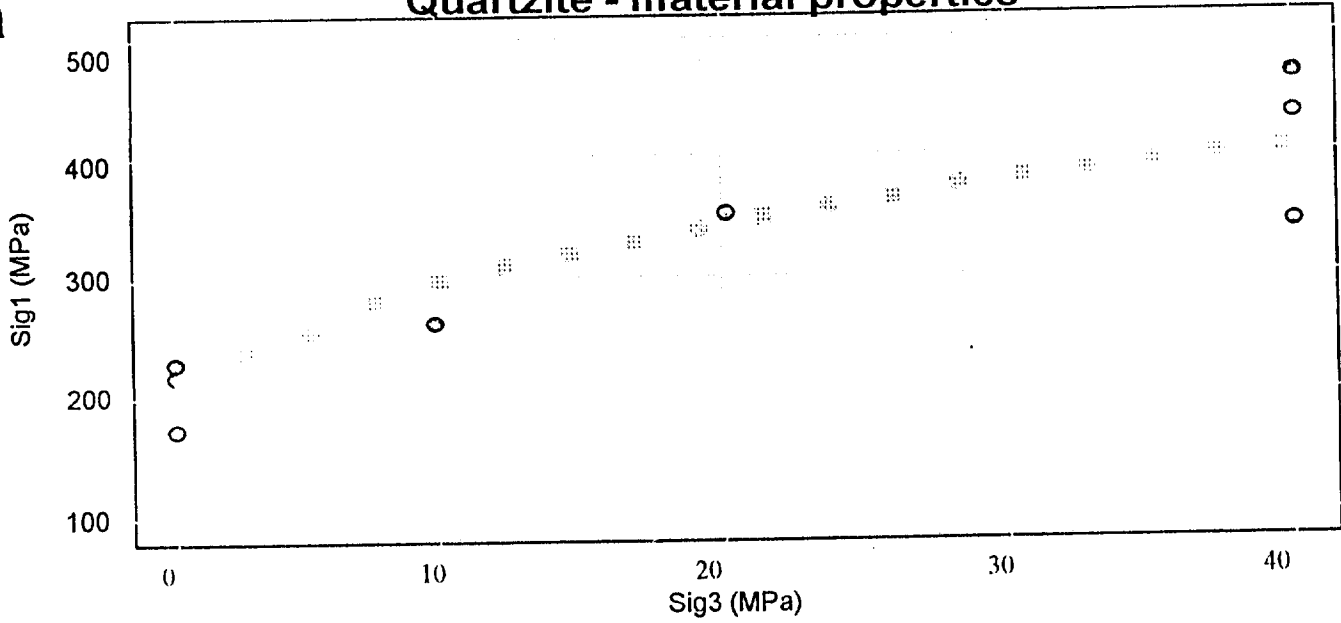
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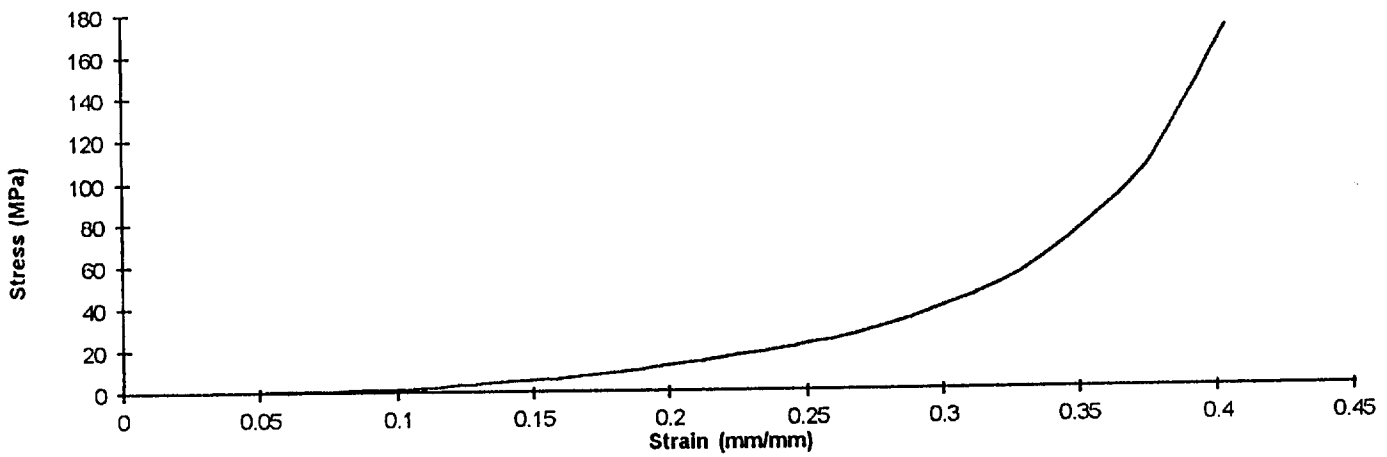
Quartzite - material properties

a



b

Backfill Stress Strain Curve



c

Quartzite - 1

diameter = 51.8 mm; height = 51.9 mm

Quartzite - 2

diameter=51.9mm; height=51.9mm

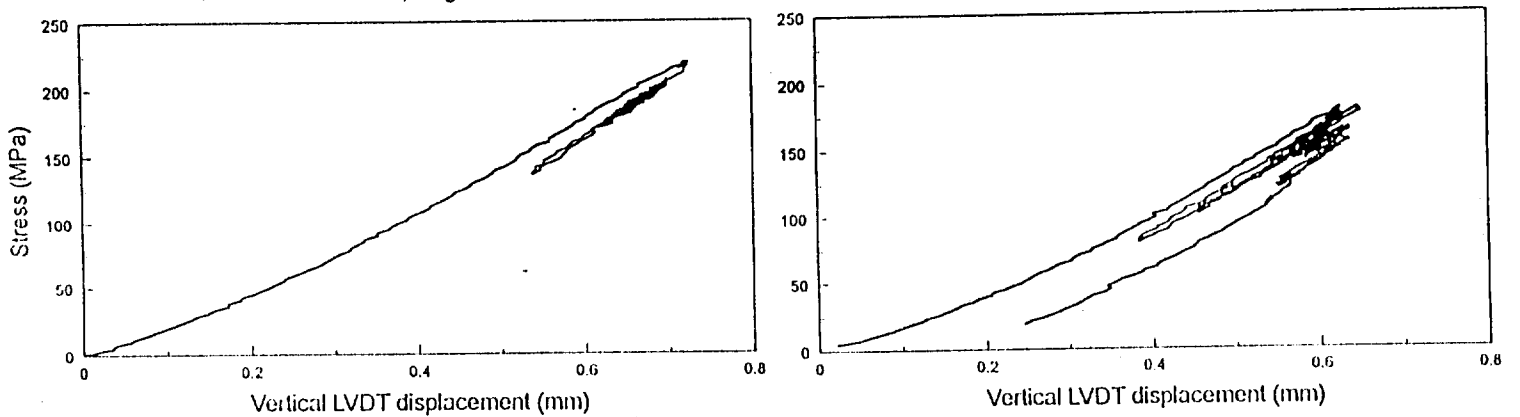
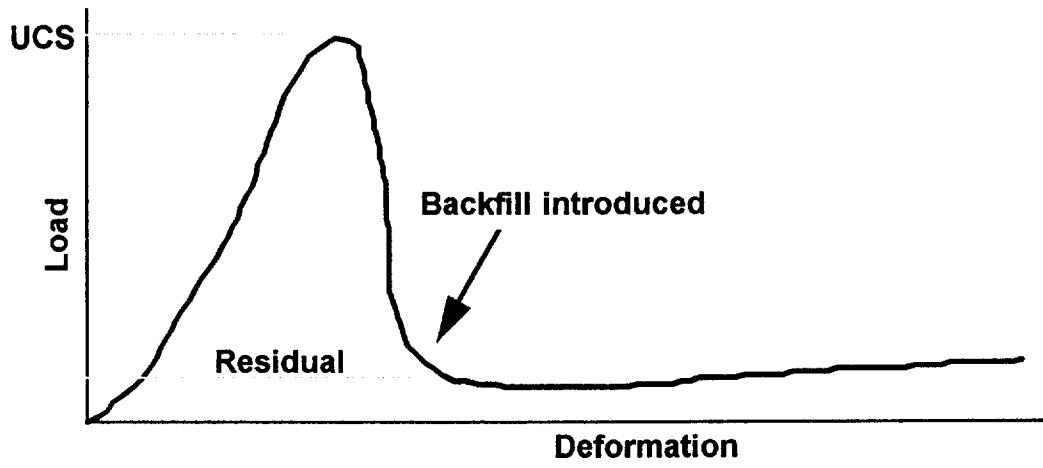
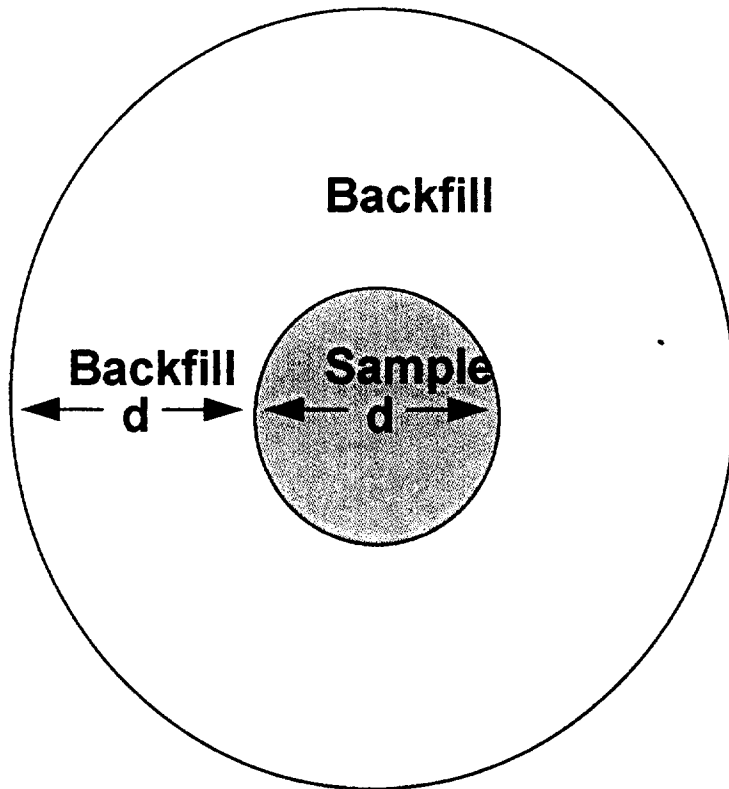


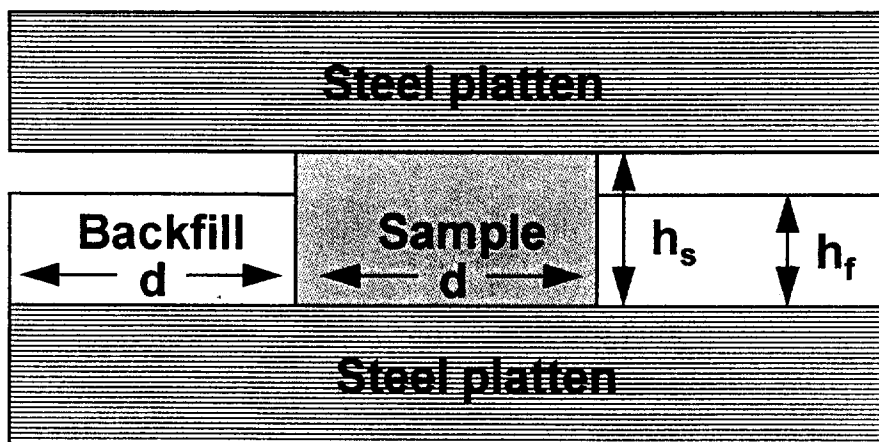
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b



c



h_s - Sample height

h_f - Fill height

d - Sample width

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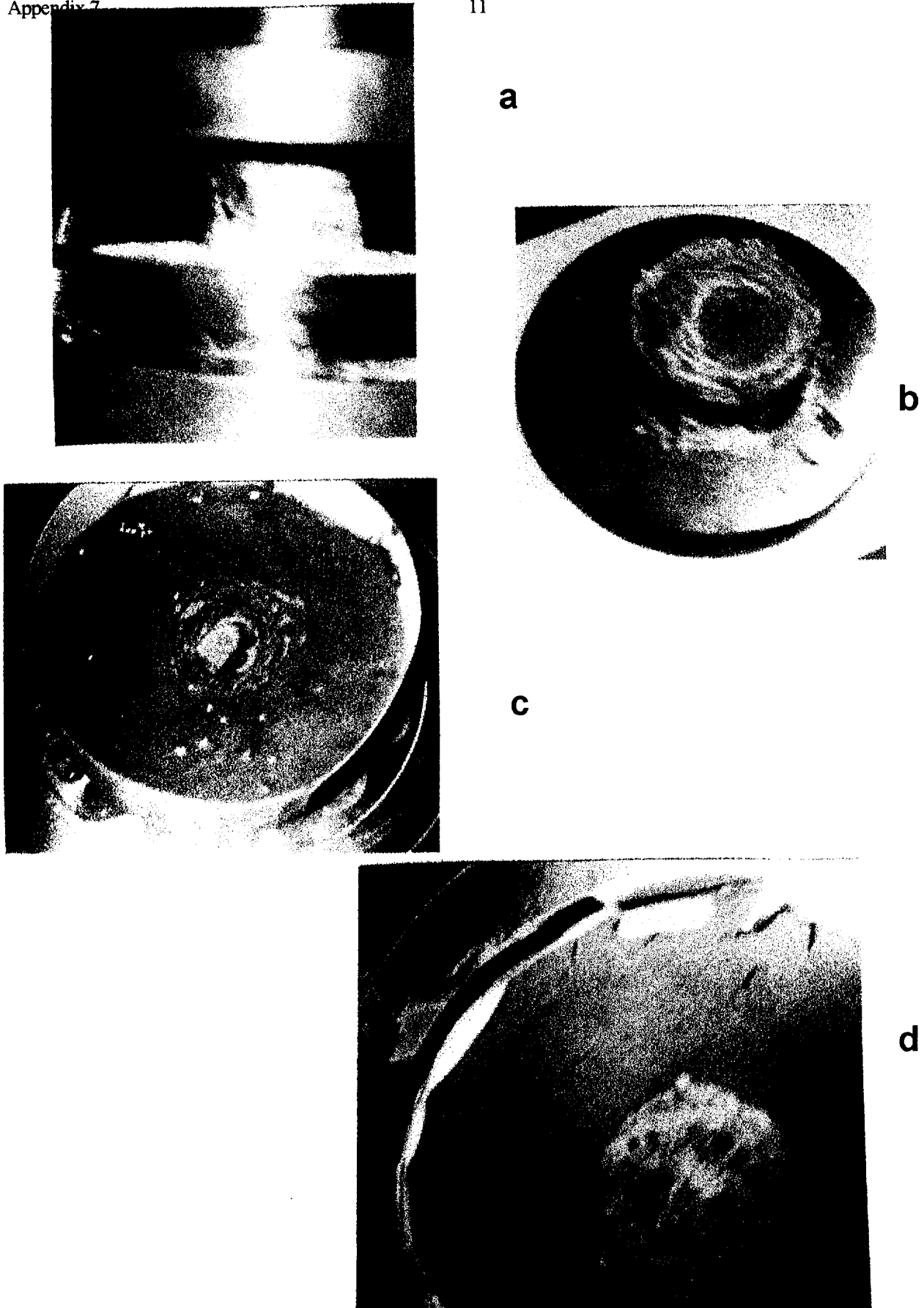
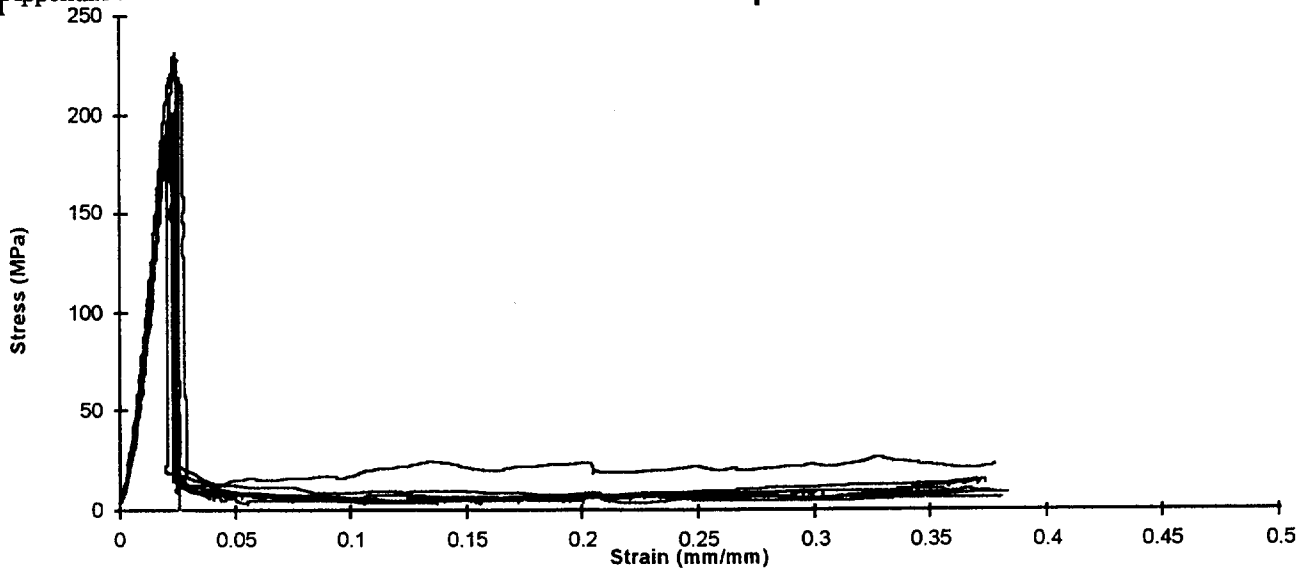


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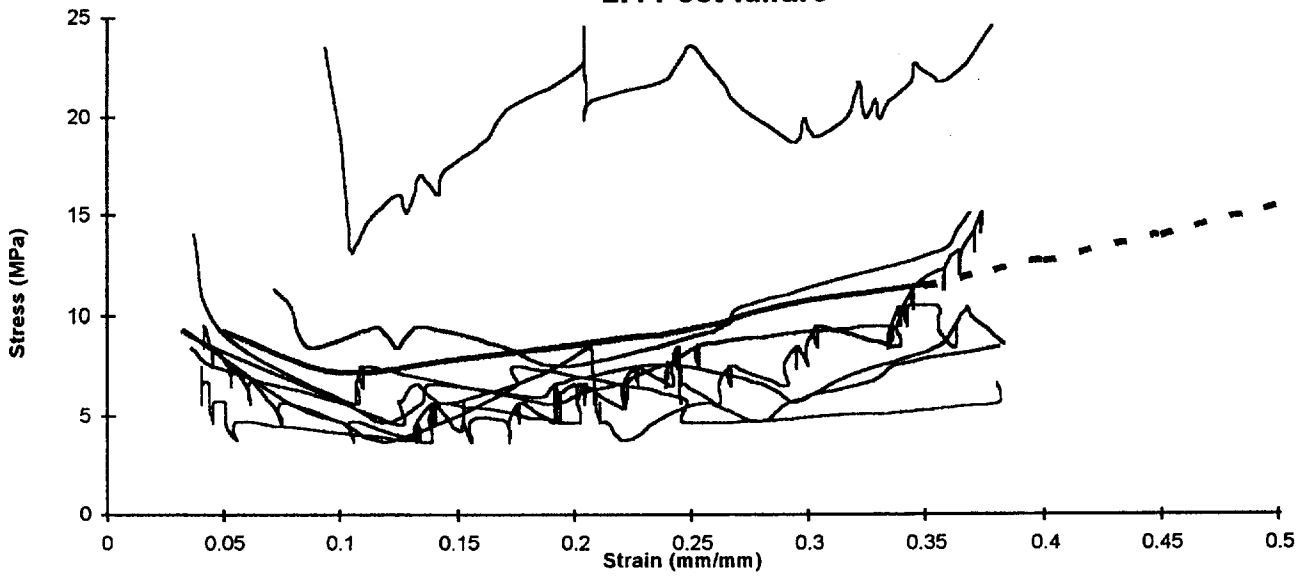
Appendix 7

2:1 Crush pillar



b

2:1 Post-failure



c

2:1 Post-failure (line removed)

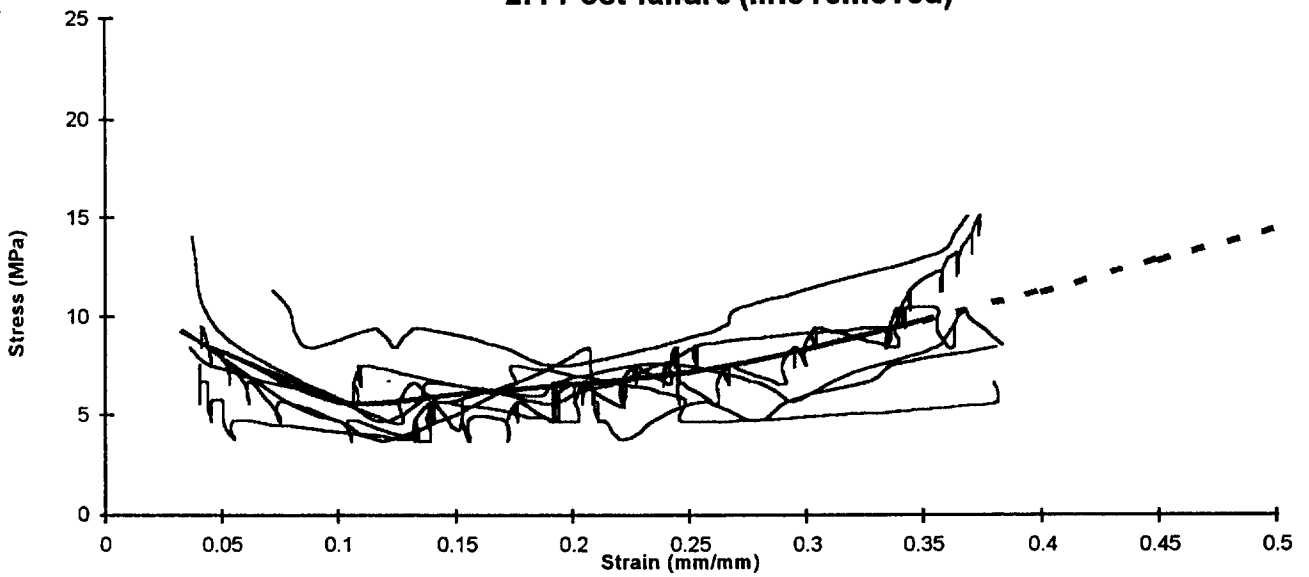
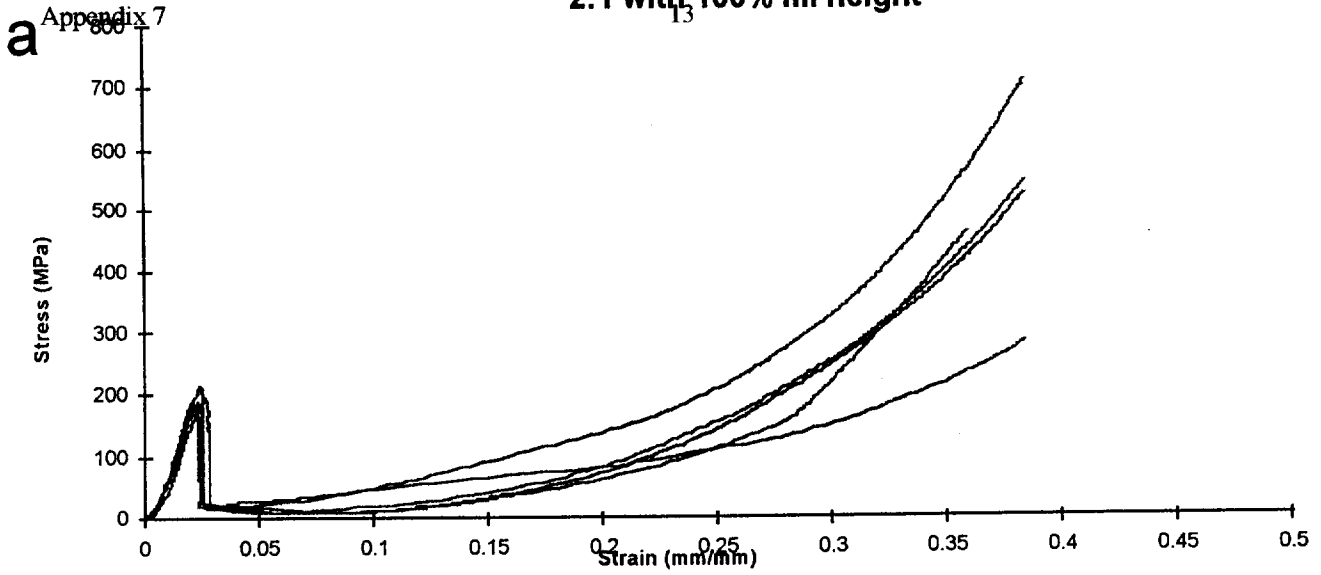
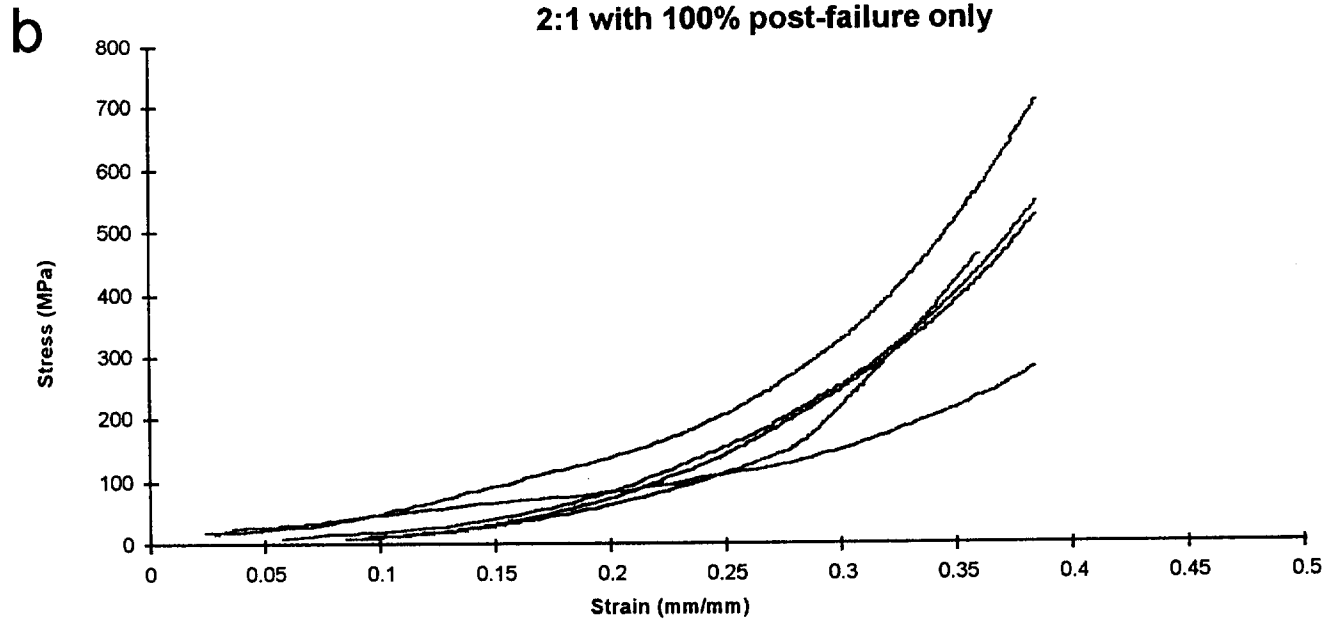


Figure 5. a) 2:1 unconfined results including initial failure.
b) 2:1 unconfined results - post-failure with average.
c) 2:1 unconfined results (outlier removed) - post-failure with average line.

2:1 with 100% fill height



2:1 with 100% post-failure only



2:1 with 100% post-failure & averages

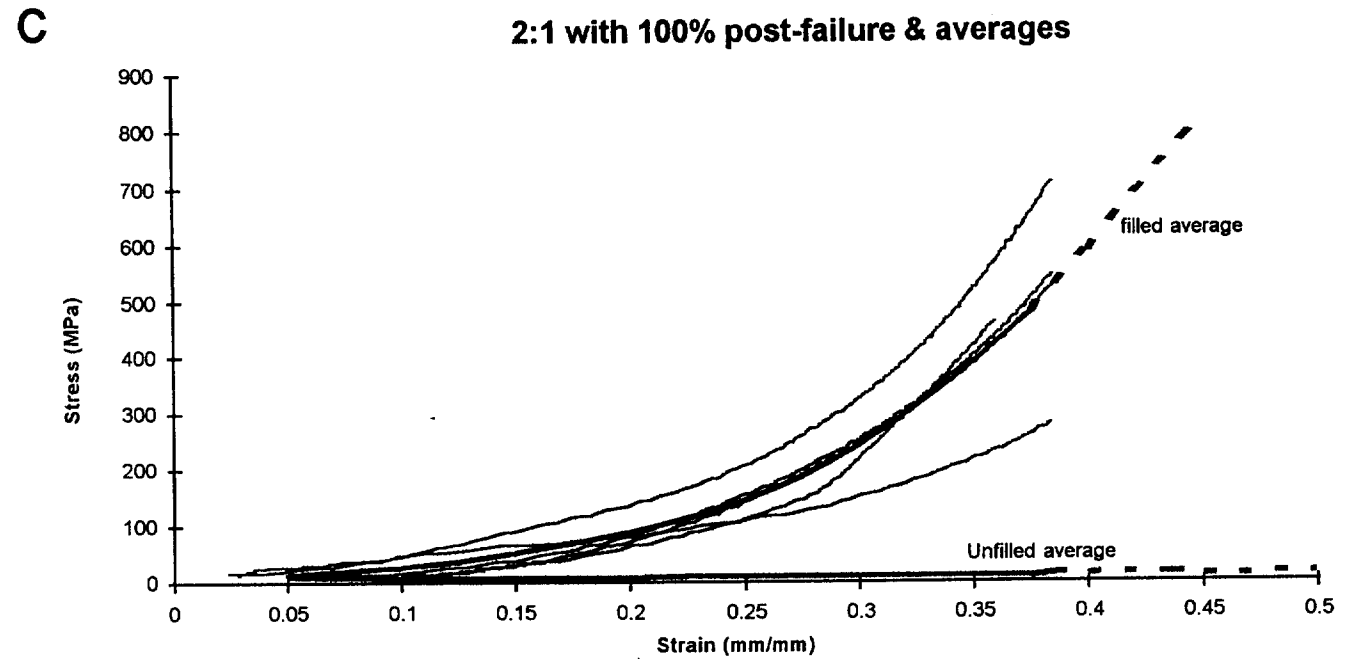
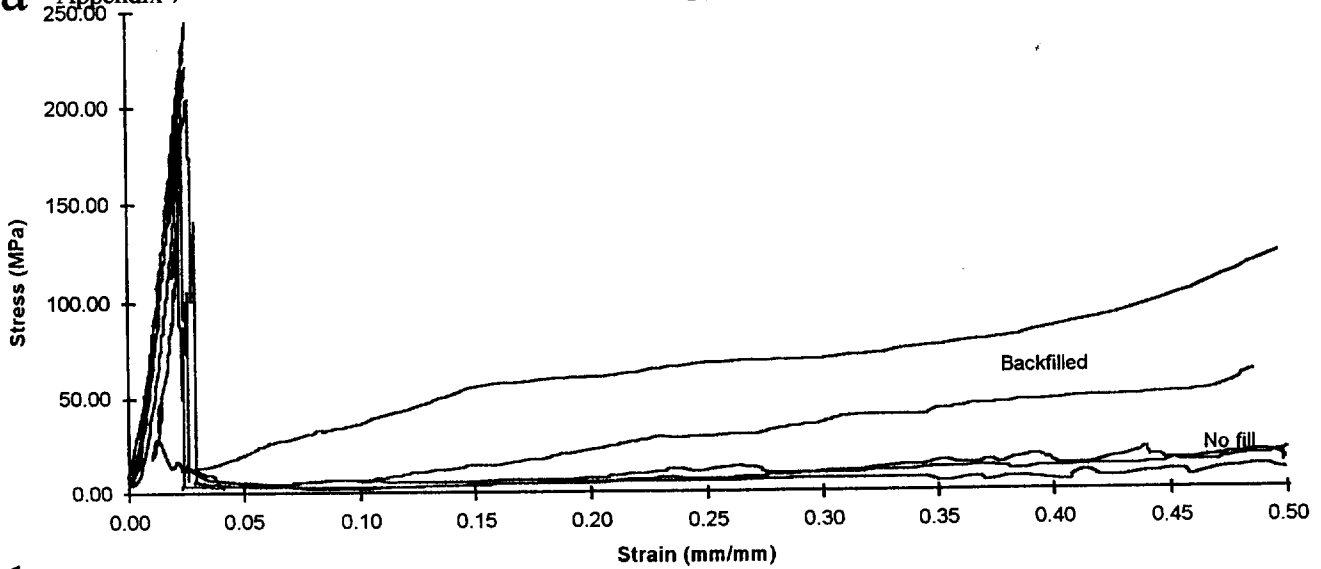


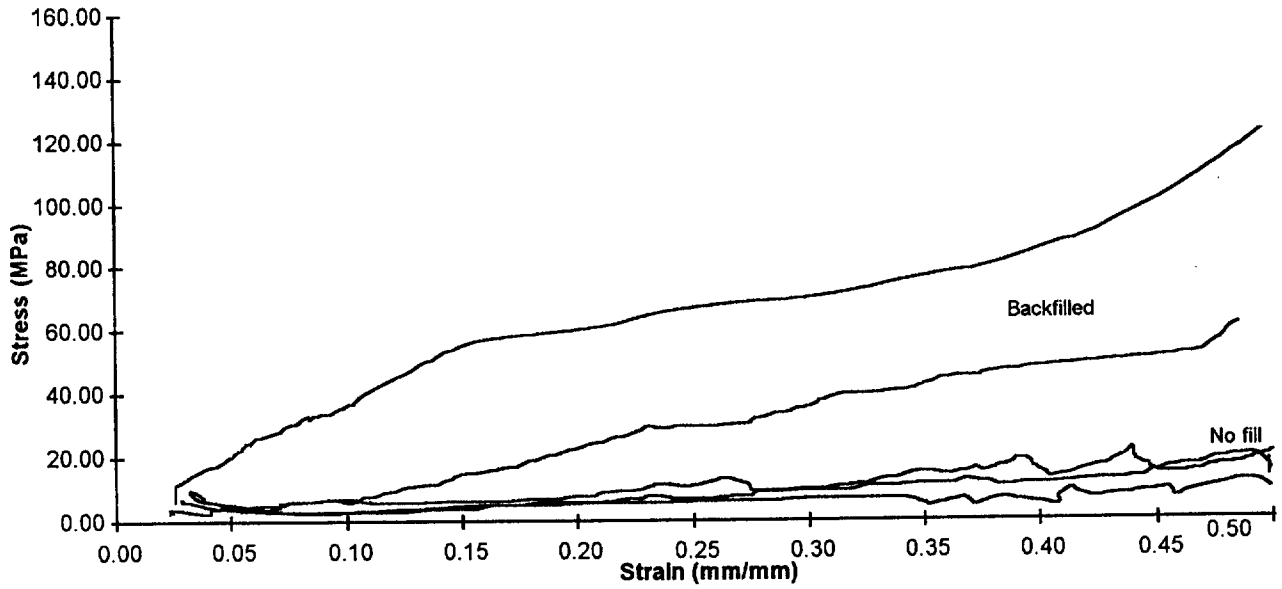
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1.5:1 Crush Pillar

Appendix 7
14



b 1.5:1 Post-failure only



c 1.5:1 Post-failure & average

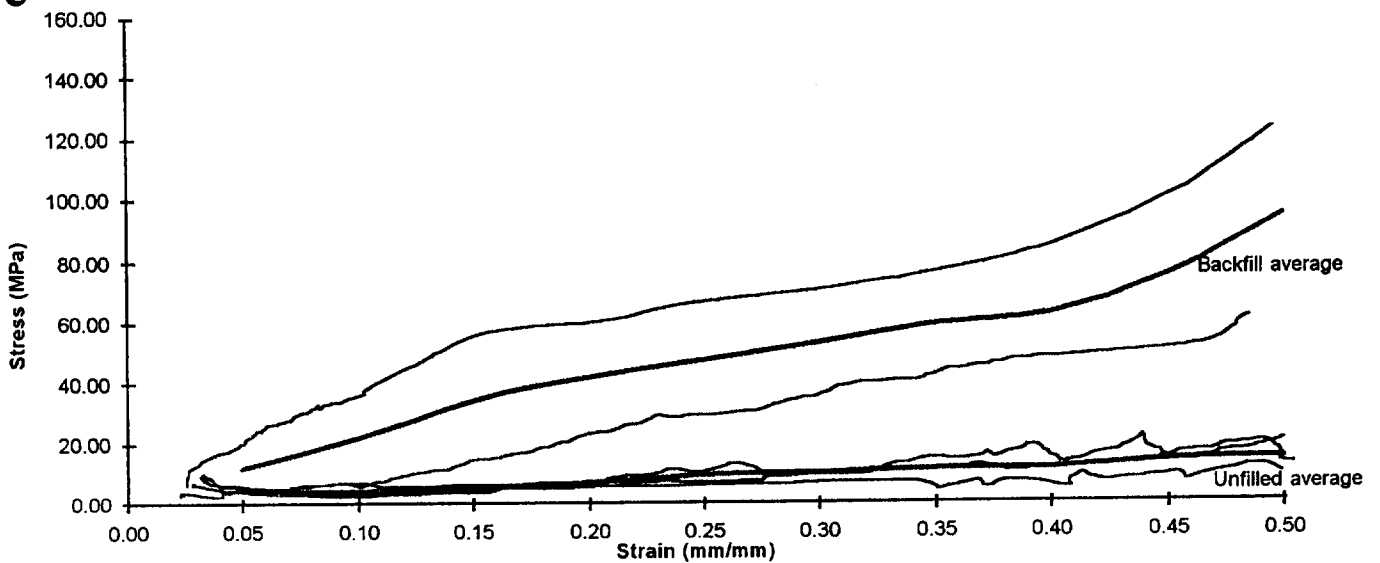


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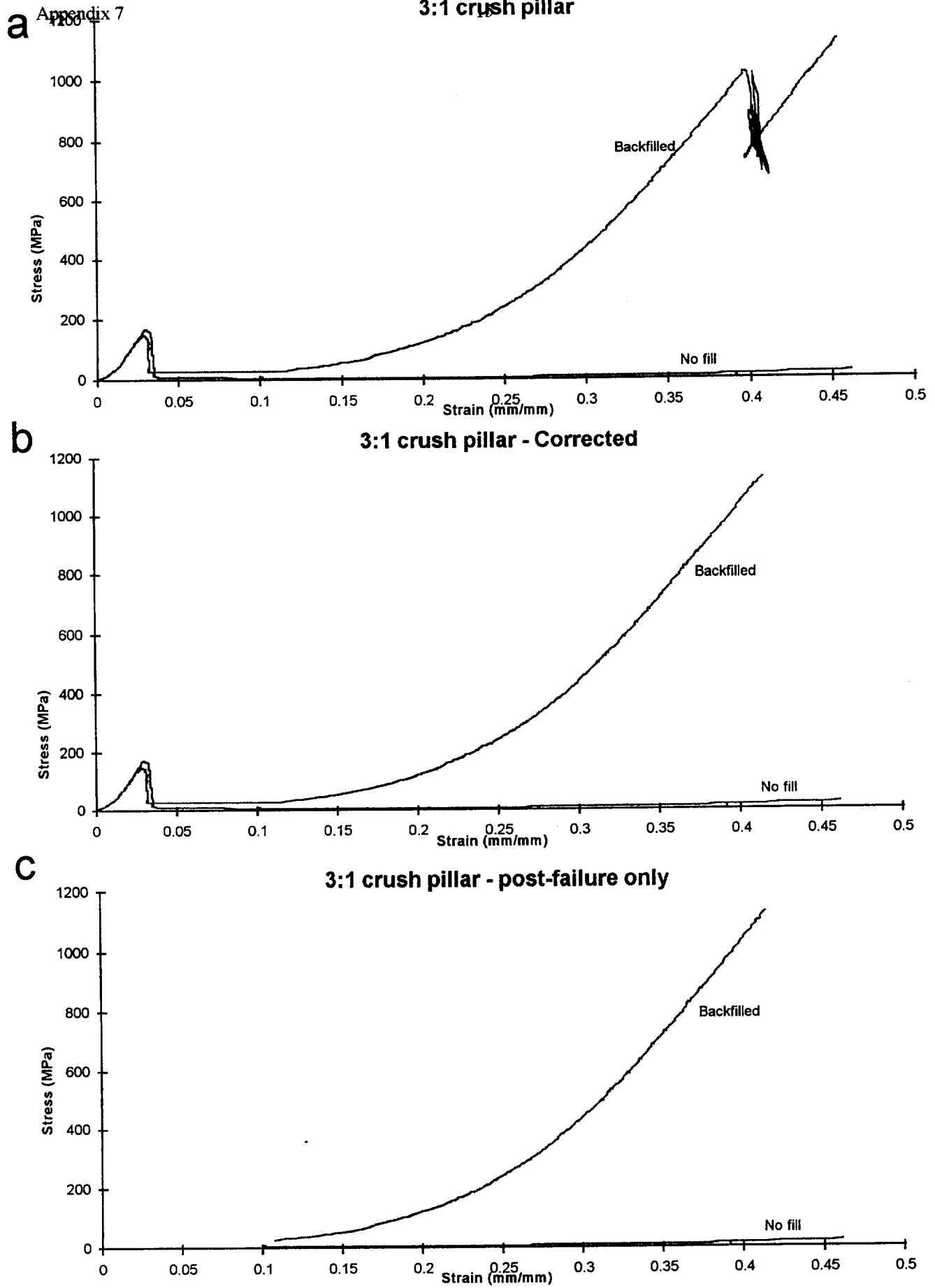


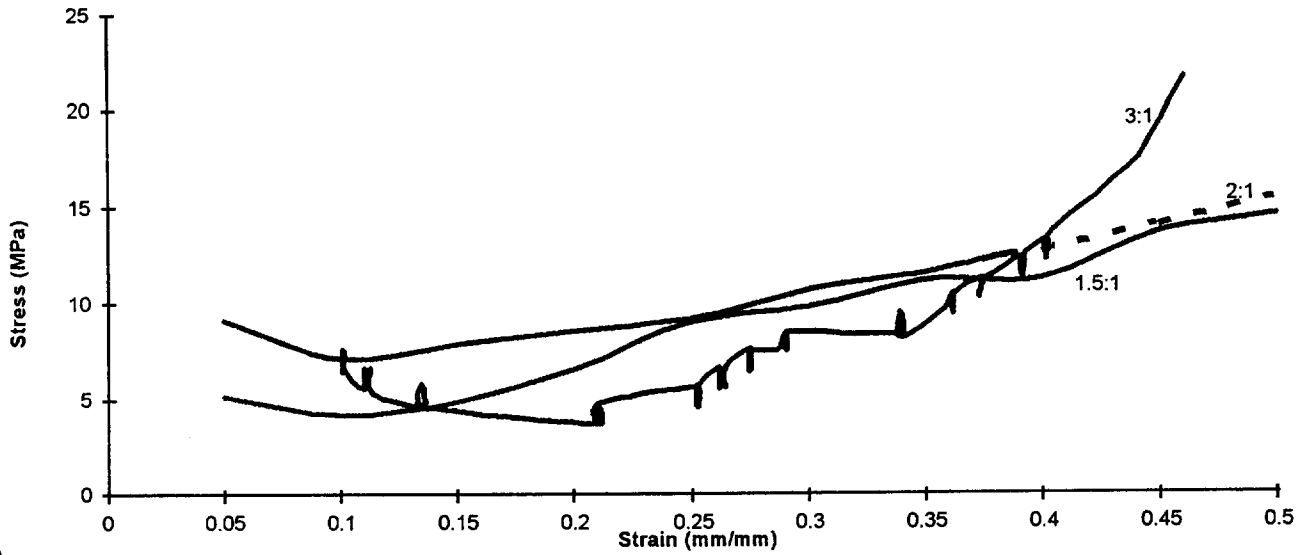
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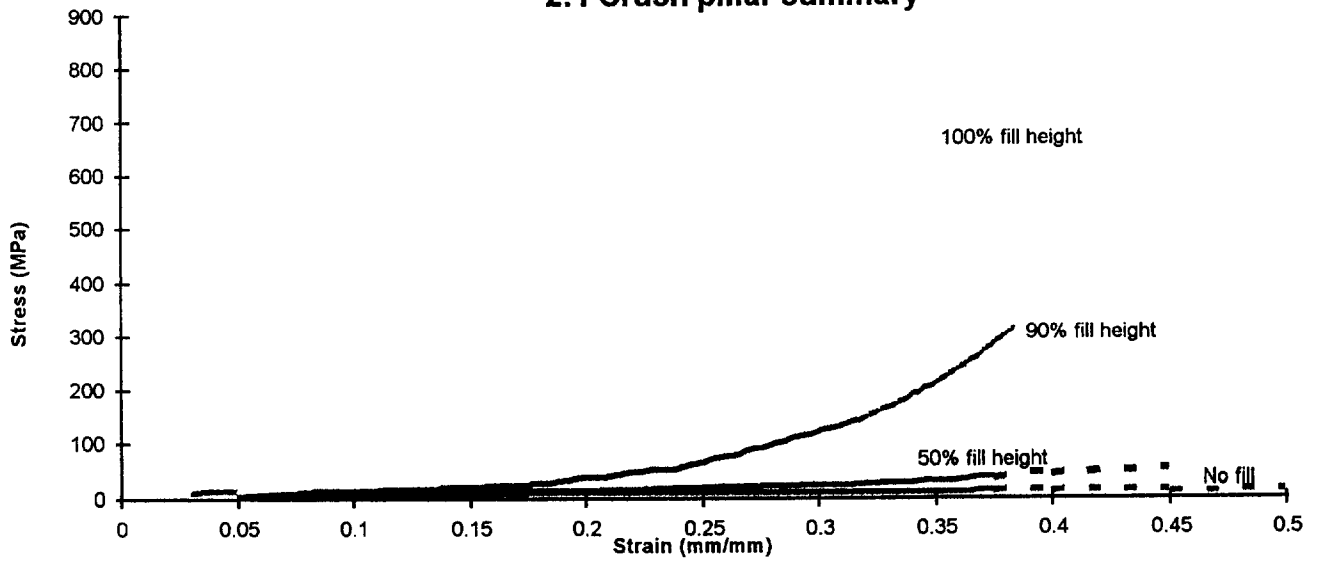
a Appendix 7

w/h summary - unfilled



b

2:1 Crush pillar summary



c

w/h summary - filled

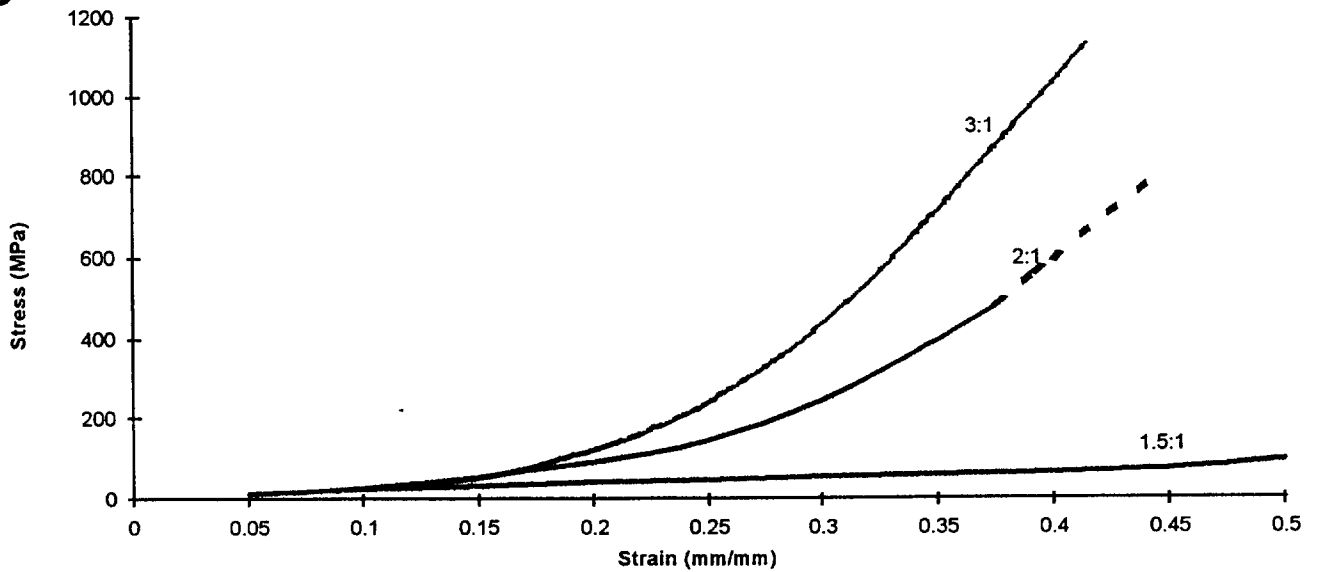


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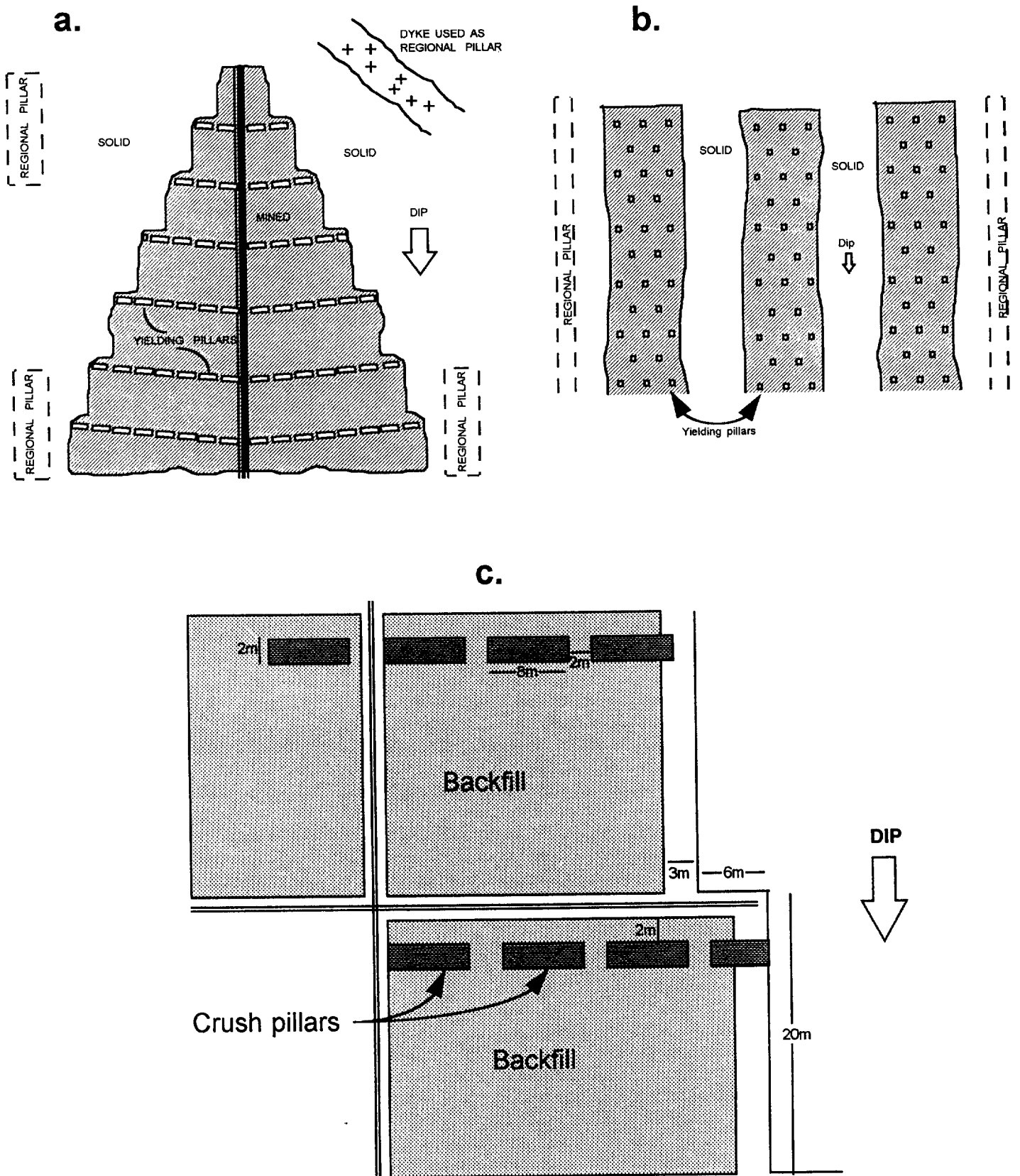


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