The Extraction of the No. 1 Shaft Pillar at Blyvooruitzicht Gold Mine

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The planned extraction of the No. 1 shaft pillar at Blyvooruitzicht Gold Mine highlighted a number of potential problems. Despite the relative shallowness, careful thought had to be given to stope phasing owing to the detrimental effect of the anticipated geology and the fact that uninterrupted hoisting was necessary.

Extensive use was made of computer modelling to design the stope phasing, determine the influence of regional support and to predict the direction and magnitude of displacements in the shaft. The paper describes the success of this design approach, and measured displacements in the stoping horizon and shaft barrel are compared to those obtained from the original computer model.

Introduction

The Blyvooruitzicht Gold Mining Company, Limited was floated in early 1937, after encouraging values had been obtained in surface boreholes from reef intersections of the Carbon Leader Reef. The decision was made in the same year to establish the mine, and sinking of No. 1 shaft commenced in November 1937.

The outbreak of the Second World War curtailed operations, and the shaft was only completed to its final depth of 1536 m in June 1941.

The Carbon Leader Reef was intersected by the shaft, at a depth of 1426 m, just below the 5 level station, as shown in Figure 1.

As was common practice in those days, a shaft pillar, or a planned unmined portion of the ore reserves, was left surrounding the shaft to protect it from the effects of mining the remaining ore reserves.

By 1977 the emphasis of mining on Blyvooruitzicht had shifted to the western side of the lease area and the ore reserves served by this shaft had been exhausted, the decision was made to extract the shaft pillar.

It was appreciated that the displacements and strain induced at the reef intersection in the shaft barrel would impede high speed hoisting in the shaft. Alterations were necessary in the shaft guides and timbering, but it remained to be seen to what extent the shaft would be affected.

It was suggested that numerical modelling would assist in this decision. The correct stope phasing to minimize the Energy Release Rate (E.R.R.) and consequently the seismicity were also necessary considerations.

A general concept of a centrally mined block surrounded by satellite pillars was examined, using the numerical techniques available to the Rock Mechanics Department at the mine. This modelling allowed for:
FIGURE 1. East-west section through the shaft and ancillary excavations.
(a) Preliminary estimates of the stress-strain deformation for planning modifications to the shaft timbering.

(b) The correct scheduling and sequencing of the extraction of the ore reserves to minimize the above and E.R.R.

(c) Determination of the stability of the satellite pillars.

The role of modelling
 traditionally the design of underground excavations has been based on empirical and observational methods. More recently, numerical methods have become available which are able to model stresses and displacements around underground excavations.

Empirical design methods depend on relating experience gained elsewhere in similar situations. The observational method is that method which is changed to suit changing circumstances and has no scientific base.

Analytical design methods however rely on model studies in which the performance of an excavation can be analysed before it is created.

Analytical design methods allowed for model studies to be made of the shaft pillar extraction at the mine before any of the ore reserves contained in the pillar were mined. This numerical modelling was only a partial component in the investigation, and it was necessary to combine analytical with empirical and observational methods to ensure a satisfactory design. The cycle was:

(a) Site characterization : collecting geological data and rock strength data.

(b) Geotechnical model : the site was divided into areas of similar geotechnical properties.

(c) Modelling and design analysis : a modelling technique was selected.

Numerical methods used

The computer program FREEFS was extensively modified and adapted for use at Blyvooruitzicht by the Rock Mechanics Department staff. A number of pre- and post-processing programs have been developed in order to:

(a) allow easy modelling,

(b) generate grid patterns from digitized face position data,

(c) permit automatic 'windowing',

(d) provide visual output in the form of computer-generated line drawings.

A feature of the system is that any area of interest can be selected from disk files and analysed immediately. The entire mine (with portions of adjacent mines) is modelled using 512 x 333 blocks, each representing a 15 m x 15 m square area. Any area of 60 x 60 blocks (900 m x 900 m) can be selected for mining simulation, in each case the effect of mining outside this area being taken into account. See Figure 2.

The Rock Mechanics Department has available its own computer facilities which are also used for seismic research. The system is constructed around two Hewlett Packard (HP) 21 MX E-series mini-computers: See Figure 3.

The first unit (with 64 Kb) has a 12 bit A/D sub- system (H.P. 2313), a 20 Mb disk drive (H.P. 7906) with controller and switch- selectable use of graphic terminal (H.P. 2648). This unit and A/D is dedicated to continuous acquisition of data, detection of seismic events and storage of event data on disk.

The second unit (with 1 Mb) has a 20 Mb disk drive and the graphic terminal, together with a line printer (HP 2608), plotters (HP 9872 and HP 2648A) and magnetic tape drive (HP 7970E).

The graphics terminal has a fully refresh graphics display with 720 x 360 addressable
points and communicates at 9600 Baud. A 50 Mb disk drive (HP 7920) and a Summographics digitizer were also later added.

Modelling approach

Ten different stope phasing schemes were examined, each scheme having up to ten separate mining steps. The general layout of a centrally mined block, with two square stabilizing satellite pillars in the upper east and west corners of the pillar with a NE-SW striking pillar, downdip from the shaft, enclosing a dyke, was used in each case study. It was reasoned that this dyke would present a barrier both from a mining
and seismic viewpoint, hence the decision not to mine it but to bracket it with unmined ore reserves updip and downdip. The general layout is shown in Figure 4.

Intensive examination of old geological data showed no other major discontinuities which would slow down or impede mining, thus no alterations were necessary to the original mining concept.

Further details of the parameters used in the FREEFS model are:
- Young's Modulus: 70 Gpa
- Poisson's ratio: 0.2
- Grid size: 15 m
- Stoping width Sm: 1.0 m
- Reef dip: 20°
- Vertical stress gradient: 0.027 Mpa/m
- Horizontal to vertical virgin stress ratio: k = 0.5

In each of the models examined, extraction was simulated in an easterly and westerly direction symmetrically outwards from the shaft up to a maximum of nine mining steps.

Strains and displacements induced along the vertical axis of the shaft were considered in evaluating its long term stability. To calculate these parameters fourteen off-reef points were modelled along the vertical axis of the shaft at 50.0 m intervals, above and below the reef horizon.

A reef projection of the points was used thus the distance, above and below the reef horizon, is the normal distance. Above reef being regarded as positive and below as negative. Vertical displacements were computed at each of these off-reef points. Maximum principal stresses were also calculated at each of these off-reef points for each stage of mining in each of the phasing schemes. A limitation of the program was that off-reef points were required to be one grid square away from the reef horizon.

The on-reef solution was considered important and on-reef stresses and convergences were also calculated. E.R.R.'s...
FIGURE 4. Plan of stope phasing scheme accepted for the extraction of No. 1 shaft pillar
were also calculated as multiple mining steps were modelled.

The stope phasing scheme eventually chosen by the mine offered the optimum combination of these design parameters.

Reef level stresses

Two aspects of the stress field at the reef elevation were considered to be important to the stability of the shaft barrel:
(a) absolute value of the stress;
(b) change in stress experienced.

It was considered that should the stress level \( \sigma_e \) be above a certain fraction of the uniaxial compressive strength of the rock \( \sigma_c \), failure would occur. Rockmass strength in the range 180 - 200 Mpa was used. A proportion \( \sigma_e / \sigma_c < 0.5 \) was used as the limiting ratio.

A tabulation of maximum principal stress versus distance above and below reef shows that at no place, except in the immediate reef intersection area, was the ratio exceeded. The shaft barrel was bolted and meshed for a distance of 20 m above and below the reef intersection to counteract expected rock failure and subsequent dilation of the sidewalls of the shaft. Yielding stope support in the form of low load to compression packs was also installed for a radius of 10 m around the shaft barrel at the shaft/reef intersection. Crush pillars in the form of 1,5 m wide strips of reef were left in the immediate vicinity of the shaft to inhibit the initial convergence in the reef horizon.

In Table 1 the relationship of the maximum principal stress and the maximum change in the principal stress at each of the off-reef points modelled along the vertical axis of the shaft may be seen.

<table>
<thead>
<tr>
<th>Distance above reef (m)</th>
<th>Maximum ( \sigma_e ) (Mpa)</th>
<th>Maximum change (Mpa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>550</td>
<td>27.3</td>
<td>9.2</td>
</tr>
<tr>
<td>500</td>
<td>31.1</td>
<td>11.0</td>
</tr>
<tr>
<td>450</td>
<td>35.4</td>
<td>13.4</td>
</tr>
<tr>
<td>400</td>
<td>41.2</td>
<td>17.5</td>
</tr>
<tr>
<td>350</td>
<td>46.5</td>
<td>21.7</td>
</tr>
<tr>
<td>300</td>
<td>52.3</td>
<td>27.4</td>
</tr>
<tr>
<td>250</td>
<td>58.0</td>
<td>34.2</td>
</tr>
<tr>
<td>200</td>
<td>64.3</td>
<td>41.4</td>
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<tr>
<td>150</td>
<td>67.3</td>
<td>43.6</td>
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<td>100</td>
<td>66.3</td>
<td>43.0</td>
</tr>
<tr>
<td>50</td>
<td>55.4</td>
<td>36.7</td>
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<tr>
<td>- 50</td>
<td>51.8</td>
<td>16.1</td>
</tr>
<tr>
<td>- 100</td>
<td>65.2</td>
<td>35.0</td>
</tr>
<tr>
<td>- 150</td>
<td>72.5</td>
<td>38.4</td>
</tr>
</tbody>
</table>

Horizontal and vertical displacements and strains

The discontinuity in displacement which occurs at the shaft/reef intersection during mining is most conveniently resolved into horizontal and vertical components when dealing with modification to shaft timbering as opposed to normal convergence and ride components used in rock mechanics. Since the timbering in the shaft was braced against the sidewalls distortion of it would occur if this predicted movement was not catered for.

In dipping strata, the direction of the horizontal displacement is such that the hangingwall moves updip relative to the footwall. Modelling of the extraction was symmetrical in the strike direction so that the horizontal component of dislocation was in the dip direction. This was considered to be important because if the mining is not symmetrical an additional component of
DISTANCE ABOVE REEF IN METERS

--- ACTUAL SMOOTHED VALUES.
--- THEORETICAL VALUES.

FIGURE 5. Theoretical and actual strain plotted along shaft axis for slope phasing stages 1, 2, 3, 5, 6 and 9

horizontal dislocation in the strike direction will be induced.

The maximum values of the horizontal dislocation was calculated from:

\[ \Delta h = -\left( \frac{2k \sin \alpha}{1 + k + (1-k) \cos 2\alpha} \right) Sm \]

where: \( k \) = ratio of horizontal to vertical virgin stress
\( \alpha \) = dip of strata

A maximum of 182 mm horizontal movement was calculated. This was incorporated into the planned modifications. The vertical movement was calculated at 680 mm, and was
resolved into 270 mm upwards and 410 mm downwards. A limiting vertical strain of 0.001 was used. This figure was derived empirically from examination of previous shaft pillar extractions.

The vertical strains calculated for the accepted stope phasing are shown on Figure 5.

**Energy Release Rates (E.R.R.)**

Examination of some 10 000 located seismic events at Blyvooruitzicht had enabled the Rock Mechanics Department to derive a relationship between seismicity and E.R.R. Eighteen regions within Blyvooruitzicht were considered on the basis of distinguishable concentrations of seismicity. These regions were then grouped in four E.R.R. ranges, of namely, 0-20, 21-40, 41-80, and 81-150 MJ/m². In Figure 6 it can be seen that the rate of seismicity expressed as the number of seismic events with ML > 1.5 per 1 000 m² mined, is proportional to the E.R.R.

The stope phasing scheme accepted for implementation was the model which allowed for the lowest E.R.R. and consequently the lowest predicted seismicity. Table 2 shows the calculated E.R.R. for each mining step modelled.

![Figure 6](image_url)

**FIGURE 6. Relationship between number of seismic events of M > 1.5 per 1 000 m² mined and E.R.R. as found at Blyvooruitzicht Gold Mine**
Table 2. E.R.R. calculated for each mining step.

<table>
<thead>
<tr>
<th>Mining step</th>
<th>Average E.R.R. in MJ/m²</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>10</td>
</tr>
<tr>
<td>2</td>
<td>11</td>
</tr>
<tr>
<td>3</td>
<td>20</td>
</tr>
<tr>
<td>4</td>
<td>37.1</td>
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<tr>
<td>5</td>
<td>61.0</td>
</tr>
<tr>
<td>6</td>
<td>57.1</td>
</tr>
<tr>
<td>7</td>
<td>64.9</td>
</tr>
<tr>
<td>8</td>
<td>54.4</td>
</tr>
<tr>
<td>9</td>
<td>62.0</td>
</tr>
</tbody>
</table>

**Pillar stresses**

The dimensions of the two stabilizing satellite pillars and the NE-SW striking pillar, downdip from the shaft, were calculated using the on-reef stress option of the FREEFS program. An average pillar stress of 400 Mpa was the design parameter. This parameter being obtained from investigations conducted previously on the mine. The average pillar stress of a stable pillar on the mine was calculated using the well known analogue at the Chamber of Mines. The calculated stress of 600 Mpa it was reasoned would give a safety factor of 1.0. A 400 Mpa average pillar stress would consequently result in a safety factor of 1.5.

**Measurements and results**

**Measuring stations**

In order to keep check on the magnitude of the vertical dislocation in the shaft, measuring stations were installed between points 200 m above and 50 m below the reef horizon. Measuring stations were installed at 10 m intervals vertically below each other. Measurement was effected by means of a 200 m tape suspended from the stations. Measurements were initially taken monthly.

Ride and closure was calculated from measurements obtained from two three-dimensional ride closure meters installed in the reef horizon, close to the shaft. Three holes were drilled in the footwall of the stopped out area to form the corners of an approximate right angled triangle with one side along dip and one side along strike. Pegs were grouted into these holes and the distances between them were measured using a metallic tape. A hole was drilled in the hangingwall approximately vertically above the apex of the footwall triangle. The distances between the hangingwall and footwall pegs were also measured.

To monitor the seismicity induced by the mining in the pillar area three additional geophones (SENSOR M 6/9/B 14 Hz) were installed in excavations below the reef horizon. These geophones were linked to the mine wide seismic network operated by the Rock Mechanics Department.

**Evaluation**

The relative displacement between the measuring stations, in the shaft, was related to the off-reef points to enable a check to be kept of the theoretical and actual displacements. Figure 5 shows a comparison of the actual and theoretical displacement profiles for the stope phasing shown in Figure 4. Close agreement with the theoretically calculated values can be seen.

A total of 500 mm vertical shortening and 150 mm horizontal dislocation has been measured to date. A vertical shortening of 480 mm was computed for the same stage of mining of the stope phasing.

**Preparation of the shafts**

**Reinforcement of the shaft barrel**

The sidewalls of the shaft were supported...
by a 1.0 m square pattern of 2.1 m rock bolts and mesh for 20 m above and below the reef horizon. This was a precautionary measure to cater for the anticipated fracturing and subsequent dilation in the reef/shaft intersection area owing to the computed high stresses.

Suspension of timbering
Because of the predicted vertical and horizontal dislocation, at the reef/shaft intersection area, it was felt advisable to free the timber from the shaft sidewalls for at least 10.0 m above and below the reef/shaft intersection.

This was achieved by installing a bearer, on 5 level, resting on the station floor on the station side and on bearer plates bolted to the western sidewall. The bearer sets were suspended on hanging bolts between sets to create a framework independent of the shaft itself. Telescopic joints were installed in the guides, corner posts and studdles above the top bearer while below the bottom bearer compression type telescopic joints were installed.

The telescopic joints were designed to accept a total movement of 680 mm.

Conclusions
(a) Extraction of the ore reserves contained in the shaft pillar has progressed as per the stope phasing planned. The extraction is now in the planned stage 9 of the stope phasing. 93% of the ore reserves contained in the pillar have been extracted according to schedule.
(b) There has been no disruption to hoisting at any stage of the extraction.
(c) Displacements and vertical strains have been close to those computed indicating that the correct modulus was used for the numerical modelling.
(d) Measures taken to protect the shaft have been successful.
(e) No major seismic event has been located on any of the stabilizing pillars indicating that no pillar failure has occurred. The design criteria used were thus sound.
(f) Seismicity has been within the expected limits with an overall seismicity of 1.2 events / 1000 m² mined for \( M_L > 1.5 \).
(g) Numerical modelling has contributed to the safe and successful extraction of the ore reserves in the pillar.

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References

1. MCKINNON, S.O. AND OZBAY, M.U.