# Risk Evaluation and Mitigation Measures for Slope Management at Chingola Open Pit D&F, Zambia

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Konkola Copper Mines' open pits started in 1955. The main stay of open-pit copper sulphide ore production in 2014 was Chingola Open Pit (COP) D&F. Mining was of low-grade copper ore. The treatment plants had moved from ball to semi-autogenous mill technology designed to treat bulk volumes of low-grade ore. This has caused a problem in that the increased demand for low-grade ore places a lot of pressure on mining and designing of slopes to maximise safe resource utilisation at a cost-effective stripping ratio. Part of the problem is the current slope design practice at COP D&F which uses a deterministic approach, based on the factor of safety and probability of failure. The main objective of this paper is comparative analysis of the evaluation of the risk and consequences of slope failure of traditional bench-and-ramp design compared with a risk-optimised approach. The methodology adopted is to carry out computerised bench and ramp designs with geotechnical data acquired in accordance with the mine's traditional practice. In order to test lowering of risk of slope failure, selected geotechnical inputs, with the largest degree of uncertainty, were revisited afresh by additional fieldwork and laboratory testing, and then used for risk-optimised pit redesign. Results obtained from the fatality accident rate from UK Health and Safety Executive guidelines and a Zambian benchmark showed that risk-optimised slope design at COP D&F mine maximised resources with reduced stripping ratio, removing 4.5 million tonnes of waste from the current pit design. The major risk, evaluated using the 'rock fall risk assessment for quarries', was the increase of rock falls. Recommendations of an array of risk mitigation measures were made, which included slope water management, fall-back satellite slope beacon monitoring, greater emphasis on in-pit slope stability radar, as well as remote control and robotic technologies for loading and hauling equipment.

## INTRODUCTION

As the COP D&F Open Pit approaches closure, reconsideration of design parameters is a key strategy to extend the life of mine. The pit wall designs will not have to remain stable a long time. Therefore, a decision to steepen the pit slopes is a very attractive strategy in order to either excavate additional ore or to mine the same tonnage of ore but at reduced stripping ratio before transition to underground mining. For both the deterministic approach as well as the risk-optimised approaches, there are risks associated with the methodologies. The fact that there is always an uncertainty in the model input parameters means that risk is an inherent part of any slope design. Literature surveys conducted have shown that there has been a lot of study on the risk-optimised approaches, with particular focus on the probability of failure of slopes (Terbrugge et al., 2006; Steffen et al., 2008; Brown & Booth, 2009).

The concept of risk in mining is addressed in terms of both probability of failure and consequences of the same failure, that is:

$$Risk = Probability\ of\ failure \times Consequences \quad [1]$$

In this paper, focus is placed on the consequences of slope failures on personnel, resulting in loss of quality of life, injury and/or fatality, as well as equipment damage.

**Objectives**

The primary objectives of this research are to:

- Assess the optimisation of the slopes of the COP D&F using a risk-optimised approach to extend the life of the mine at a cost-effective stripping ratio before the COP D&F/COP C-E-F underground transition;
- Define risk and the need for risk treatment in slope design at the mine site;
- Propose strategic and tactical management options of risk mitigation measures to reduce the risks to *As Low as Reasonably Possible (ALARP)*.

## METHODOLOGY

The approach used to optimise the slopes of COP D&F to either achieve a cost-effective stripping ratio or extend life of mine (LOM) was done by redesigning the pit, in sections of the mine with competent rock formations only, using the bottom-up principle. The primary data used in the generation of parametric inputs for the risk-optimised design in this paper were established from new discontinuity surveys using scanline mapping on the proposed sections of the pit. The shear strength of the rock, together with the uniaxial comprehensive strength, were obtained using laboratory tests of 20 samples per rock formation in the mine stratigraphy.

The risk-optimised approach provides site-specific acceptance criteria, which are used in conjunction with traditional factor-of-safety (FOS) and probability-of-failure (POF) approaches. With this, uncertainties or risks were grouped into six broad categories, namely geological model, structural model, rock mass, laboratory rock testing, hydrogeological and geotechnical model uncertainties. A review of COP F&D data identified structural and rock mass models, together with laboratory rock testing, as sources of the largest uncertainties in the slope design protocol at the mine site. One of the authors personally undertook scanlines and also collected the samples and tested them. In summary, pit design was done in accordance with the standard computerised design at the mine. The design input data with greatest degree of uncertainty were identified. New data were collected and processed to replace this. The new data were then used in the redesign of the pit.

## DATA COLLECTION

Data collection focused on laboratory testing and field work data. Discontinuity survey using scanline mapping was carried out at the COP F&D hanging wall, covering a total of 300 m of scan lines from 0 to 180 meter bench (mb). This was supplemented by 58 point load tests and 150 triaxial tests across the stratigraphy of the hanging-wall rock formations, as shown in Tables I, II and III. This information was used to characterise the rock mass. The blocky nature of the rock mass was established, together with the rock mass strength, for pit slope optimisation.

## DATA ANALYSIS

A redesign of the pit slopes in the region (165 mb to 210 mb), where more confidence in geological and discontinuity data had been acquired, and therefore fewer uncertainties, was done, as shown in Figure 1. A bottom-up design approach was adopted for this methodology, by accepting the same width at the bottom and therefore same ore tonnage but steepening face angles and reducing berm width in areas with less model parametric input uncertainties.

*Table I. Typical scanline mapping data for Upper Roan Dolomite (URD) at COP D&F.*

**SCANLINE SURVEY FORM**

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| DATE | 15 | AUGUST | 2014 | SITE | COP FD H/W | LOGGED BY | C Silwamba |
| LOCATION | | X | Y | Z | DIP | $75^{O}$ | |
| | START | 5906.49 | 15553.31 | 1262.58 | DIP DIRECTION | $46^{O}$ | |
| | END | 5897.51 | 15561.34 | 1262.29 | LENGTH OF SCANLINE | 13 m | Rock type: Upper Roan Dolomite (URD) |

| Serial Number | ORIENTATION OF JOINT | | CONDITION OF JOINT | | | GROUND WATER CONDITION | JOINT SPACING ($S_J$), m | JOINT SETS ($D_n$) | TRACE LENGTH, m | APERTURE, mm | WEATHERING | COMMENTS |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Dip Direction (°) | Dip (°) | Roughness | | Smoothness | | | | | | | |
| | | | Waviness % | Smoothness | Infilling | | | | | | | |
| 1 | 89 | 75 | 61 | R | CJ | DRY | 5 | S1&S2 | 1/4.0 | 1 | N/W | URD |
| 2 | 78 | 75 | | R | CJ | DRY | | | 3.5/4 | 1 | N/W | URD |
| 3 | 355 | 90 | 23 | R | CJ | DRY | 1.4 | S1&S2 | 3.5/1.5 | 5 | N/W | URD |
| 4 | 346 | 80 | 10 | R | CJ | DRY | | | 3.5/4 | 3 | N/W | URD |
| 5 | 343 | 75 | 1 | R | CJ | DRY | 1.2 | S1 | 3.5/4 | 1 | N/W | URD |
| 6 | 54 | 74 | 3 | S | THF | DRY | | S1 | 3.4/4 | 1 | N/W | URD |
| 7 | 178 | 85 | 8 | R | CJ | DRY | | S1 | 5.0/5.0 | 1 | N/W | URD |
| 8 | 80 | 73 | | S | CJ | DRY | 1.1 | S1&S2 | 2/1.2 | 1 | N/W | URD |
| 9 | 85 | 60 | 10 | VR | THF | DRY | | | 2/1.7 | 6 | S/W | URD |
| 10 | 89 | 80 | 1 | S | CJ | DRY | 0.6 | S1 | 15.0/15.0 | 1 | N/W | URD |
| 11 | 53 | 76 | 10 | S | CJ | DRY | | S2 | 2.0/1.2 | 1 | N/W | URD |
| 12 | 16 | 70 | 2 | S | CJ | DRY | | S3 | 1.7/0.5 | 1 | N/W | URD |
| 13 | 75 | 90 | 1 | S | CJ | DRY | | S3 | 3.0/1.2 | 1 | N/W | URD |
| | | | | | | | | | 97 | | | |

Note: RQD= 115 - 3.3$J_v$; Joint frequency = 1/$S_J$ where $S_J$ is average size of block (rockmasses) which represents average spacing of discontinuities and $J_V = D_N(1/S_J)$ $D_N$ is estimated number of joints.

| WAVINESS = measured maximum amplitude (a)/(measured length along joint (L) | SMOOTHNESS | WAVINESS | INFILLING | WEATHERING |
|---|---|---|---|---|
| | Very Rough = VR | Large undulation = > 3 % | CJ = Clean Joint | N/W = No weathering |
| | Rough = R | Small undulation= 0.3 - 3 % | TF = Thin Filling | H/W = Highly weathered |
| | Smooth = S | Planar =< 0.3 % | THF = Thick Filling | W = Weathered |
| | Slickensided = P | | | S/W = Slightly weathered |

*Table II. Summary of rock mass classification of COP D&F, 0 mb to 90 mb*

**SUMMARY OF SCANLINE SURVEY- 0 to 90 metre benches**

| DISCONTINUITY | DIP $^{O}$ | DIP Direction $^{O}$ |
|---|---|---|
| Plane 1 | 65 | 27 |
| Plane 2 | 75 | 112 |
| Overall strike of face -3 | 65 | 107 |

**ROCK MASS CLASSIFICATION USING GSI SYSTEM - 0 to 90 metre bench**

| GSI Value | 76-95 | 56-75 | 36-55 | 21-35 | < 20 |
|---|---|---|---|---|---|
| Rock Mass Quality | Very Good | Good | Fair | Poor | Very Poor |

*Table III. Rock shear strength and uniaxial compressive strength.*

| | Traditional Approach | | | | Risk Optimised Approach | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | Cohesion C (KCM) kPa | | Friction Angle ϕ (KCM) degrees | | Cohesion C (KCM) kPa | | Friction Angle ϕ (KCM) degrees | | UCS Mpa |
| | Upper Bound | Lower Bound | Upper Bound | Lower Bound | Mean | Std Dev | Mean | Std Dev | |
| Shale with Grit (SWG) | 150 - 300 | | 25 - 27 | | 101 | 17 | 15 | 3 | 14 |
| Chingola Dolomite (CDOL) | 20 - 20 | | 11 to 11 | | 33 | 10 | 18 | 2 | 40 |
| Dolomitic Schist (Dolschist) | 200 - 300 | | 25 - 30 | | 163 | 30 | 18 | 2 | 16 |
| Upper Banded Shale (UBS) | 150 -250 | | 25 - 25 | | 118 | 23 | 18 | 4 | 13 |
| The Felspathic Quartzite (TFQ) | 250 - 350 | | 30 - 35 | | 143 | 62 | 29 | 8 | 284 |
| Banded Sandstone Upper (BSSU) | 120 - 200 | | 25 - 25 | | 97 | 57 | 29 | 6 | 27 |
| Pink Quarzite (PQ) | 200 - 300 | | 25 - 30 | | 185 | 56 | 21 | 5 | 73 |
| Lower Banded Shale (LBS) | 150 - 250 | | 25 -25 | | 86 | 59 | 24 | 9 | 91 |
| Arkose (Ark) | 400 - 620 | | 30 - 40 | | 463 | 38 | 20 | 2 | 149 |
| Granite (Gr) | 400 - 620 | | 30 - 40 | | 560 | 62 | 19 | 4 | 332 |

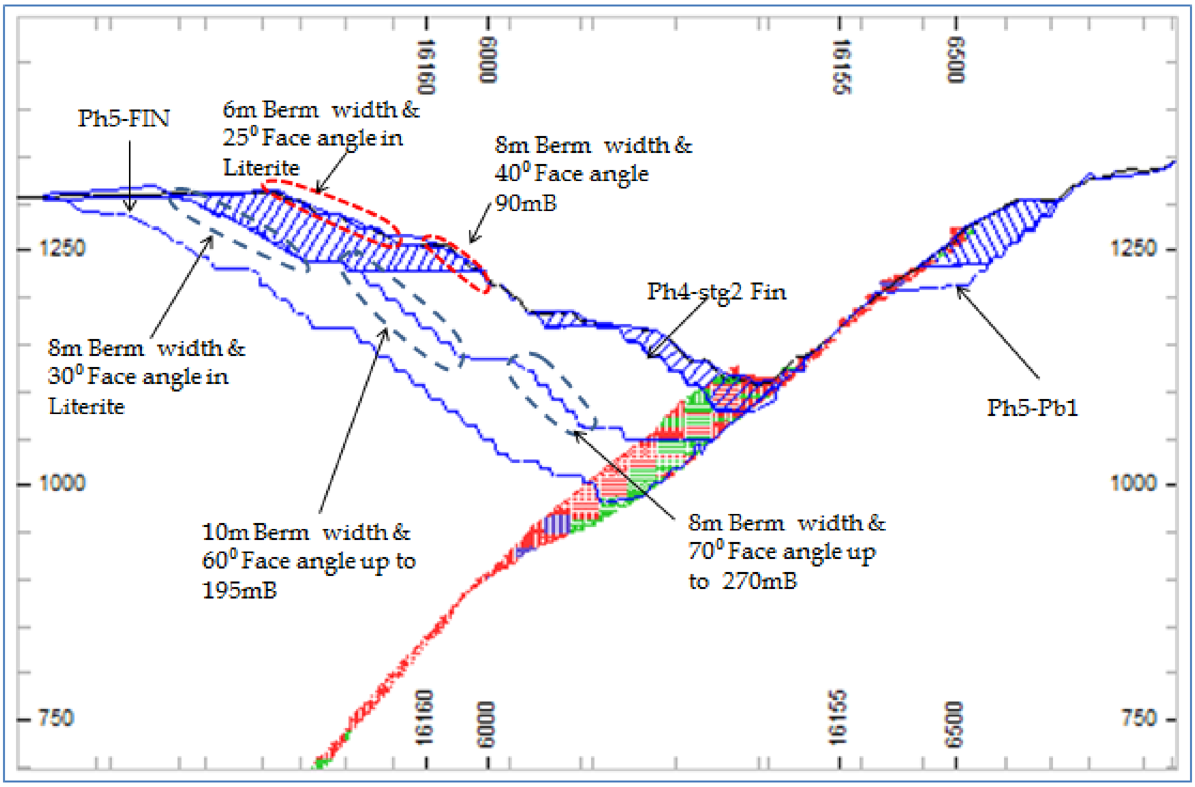



*Figure 1a. Mine designs on dip sections - design based on traditional approach.*

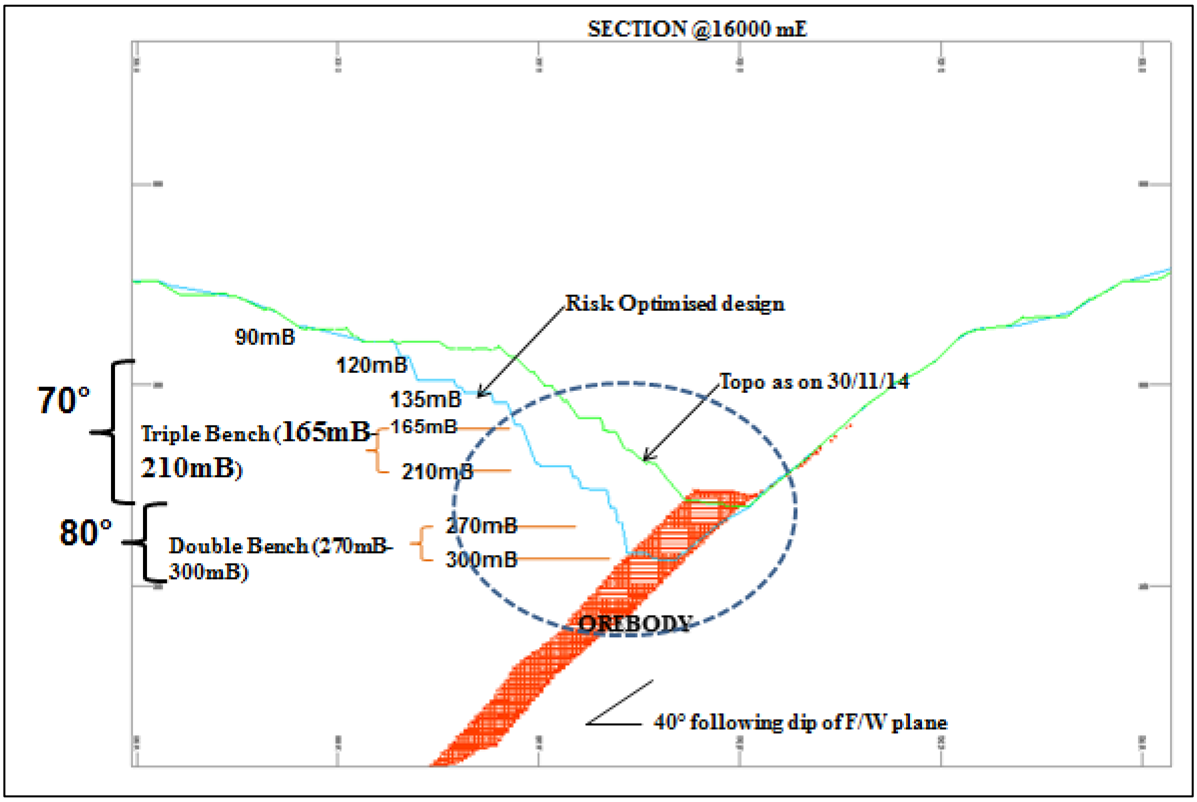



*Figure 1b. Mine designs on dip sections - design based on risk-optimised approach.*

Evaluation of the two designs (traditional and the risk-optimised design) indicated a reduction of approximately 4.5 million tonnes of waste before ore intersection, as shown in Table IV.

*Table IV. Waste evaluation comparisons - traditional approach to design vs. risk-optimised design.*

| Waste evaluations -Traditional Approach | | | | | Waste evaluations - Risk Optimised Approach | | | |
|---|---|---|---|---|---|---|---|---|
| Elevation | BENCH | TONNES | | | ELEVATION | BENCH | TONNES | |
| *1,243* | *90* | *1,552,855* | Traditional approach | Pre-strip Waste | *1243* | *90* | *552* | Risk Optimized Approach |
| *1,228* | *105* | *3,766,423* | | | *1228* | *105* | *1,434,977* | |
| *1,213* | *120* | *3,710,184* | | | *1213* | *120* | *2,982,899* | |
| *1,198* | *135* | *3,545,112* | | | *1198* | *135* | *2,791,012* | |
| *1,183* | *150* | *2,285,423* | | | *1183* | *150* | *2,828,341* | |
| *1,168* | *165* | *2,337,149* | | | *1168* | *165* | *2,398,124* | |
| *1,153* | *180* | *2,462,359* | | Copper Ore Mining (4.68x$10^6$t) | *1153* | *180* | *2,714,736* | |
| 1,138 | 195 | 2,417,391 | | | 1138 | 195 | 2,417,391 | |
| 1,123 | 210 | 2,393,124 | | | 1123 | 210 | 2,393,124 | |
| 1,108 | 225 | 2,599,569 | | | 1108 | 225 | 2,599,569 | |
| 1,093 | 240 | 2,792,093 | | | 1093 | 240 | 2,792,093 | |
| 1,078 | 255 | 2,153,731 | | | 1078 | 255 | 2,153,731 | |
| 1,063 | 270 | 1,161,581 | | | 1063 | 270 | 1,161,581 | |
| 1,048 | 285 | 438,556 | | | 1048 | 285 | 438,556 | |
| 1,033 | 300 | 151,960 | | | 1033 | 300 | 151,960 | |
| | | 33,767,510 | | | | | 29,258,094 | |

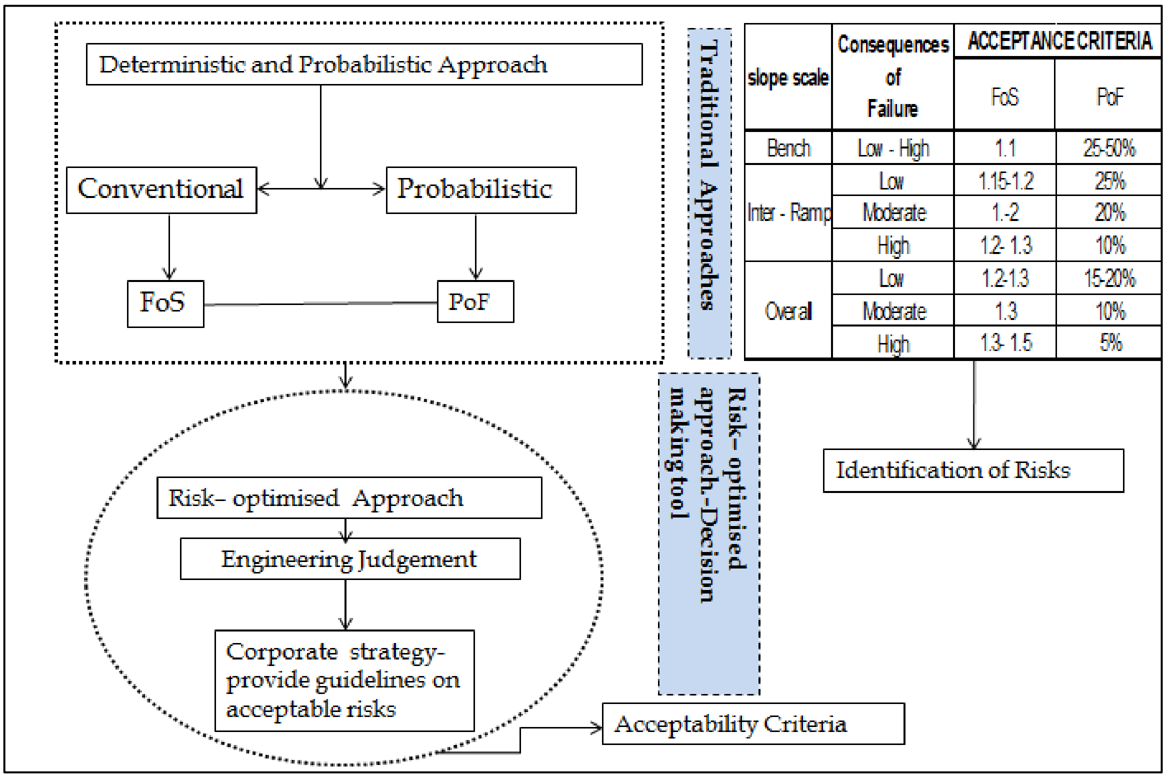
## RISK DEFINITIONS AND EVALUATION

The definition of acceptable risk levels is a very complex issue. It is very difficult to determine acceptable risk levels that individuals and society may accept. From a technical view point, however, tolerable and acceptable risks have been defined differently. 'Tolerable risk' is defined as the risk that society is willing to live with as long as that risk is monitored and management options taken to reduce it. In contrast, 'acceptable risk' is the risk society is prepared to accept without any specific risk-management options (Lee & Jones, 2004).

### Risk Evaluation and International Standards in Risk Management

Gaps in knowledge of the behaviour of rock mass, and hence uncertainties and risk, demands a structured approach to risk management. ISO 31000:2009, based on the AS/NZS 4360:2004 standard, provides a generic approach to risk management which can be adapted to any part of the mine operation, describing the principles and guidelines for managing any form of risk in a systematic and credible manner.

A local simplified conceptual framework of slope design for risk-optimised approaches was developed for COP, as shown in Figure 2. The organisation's corporate strategy allows for review and monitoring of the risk-management strategies outlined. This will, in turn, ensure that the acceptable criterion being used by the technical team is backed by management. In addition, identified risks will have mitigation measures that will reduce risks to *as low as practically possible*.




| slope scale | Consequences of Failure | ACCEPTANCE CRITERIA | |
|---|---|---|---|
| | | FoS | PoF |
| Bench | Low - High | 1.1 | 25-50% |
| Inter - Ramp | Low | 1.15-1.2 | 25% |
| | Moderate | 1.-2 | 20% |
| | High | 1.2- 1.3 | 10% |
| Overall | Low | 1.2-1.3 | 15-20% |
| | Moderate | 1.3 | 10% |
| | High | 1.3- 1.5 | 5% |

*Figure 2. Local simplified conceptual framework for risk-optimised design of COP slopes.*

**Risks to Individuals**

In this paper, the acceptability criterion is viewed from the stand point of tolerable risk. Regarding individual risks to life, acceptable risk levels are determined by comparison with other risks. Acceptable collective risks to life are treated by the cumulative **F**requency of accidents and the **N**umber of fatalities (F-N curves). The F-N curves in Figure 3 indicate the frequency–magnitude relationship of adverse consequences. Of particular interest is the single fatality of $10^{-4}$ identified as the 'Proposed BC Hydro individual risk'. This number represents the risk exposure to 'natural death' by the safest population group in North America. Accepting this as a criteria for slope design implies, therefore, that a person subjected to the open-pit working environment is not exposed to greater death risk resulting from a slope failure than they are of dying due to natural causes.

With due consideration for the data of the various identified risks and existing government guidelines, it is recommended that open-pit mines be designed to a fatality risk level of between 1:1000 and 1:10 000 persons per year, which falls within the *ALARP* region (Contreras & Steffen, 2006).

An effort at demonstrating the concept of risk of fatality in open-pit mining at COP is made below. Statistics on fatal accidents at Konkola Copper Mines (Figure 4) were converted into a format that can be compared with the risk of fatality.

***Fatal Accident Rate (FAR) Calculations for KCM Open Pits***

FAR= (Expected no. of fatalities x $10^8$)/ (No. of hours exposed to risk) [2]

1 worker year = 50 work weeks/year x 40 h/week = 2000 h/year. Based on 1000 employees working for 50 years during their life time (total workforce at the open pits with varying exposure levels visit the open pit at one point or the other, therefore have potential of exposure to risk of rock falls): So, 1000 x 50 x 2000 = $10^8$ working hours or exposed hours.

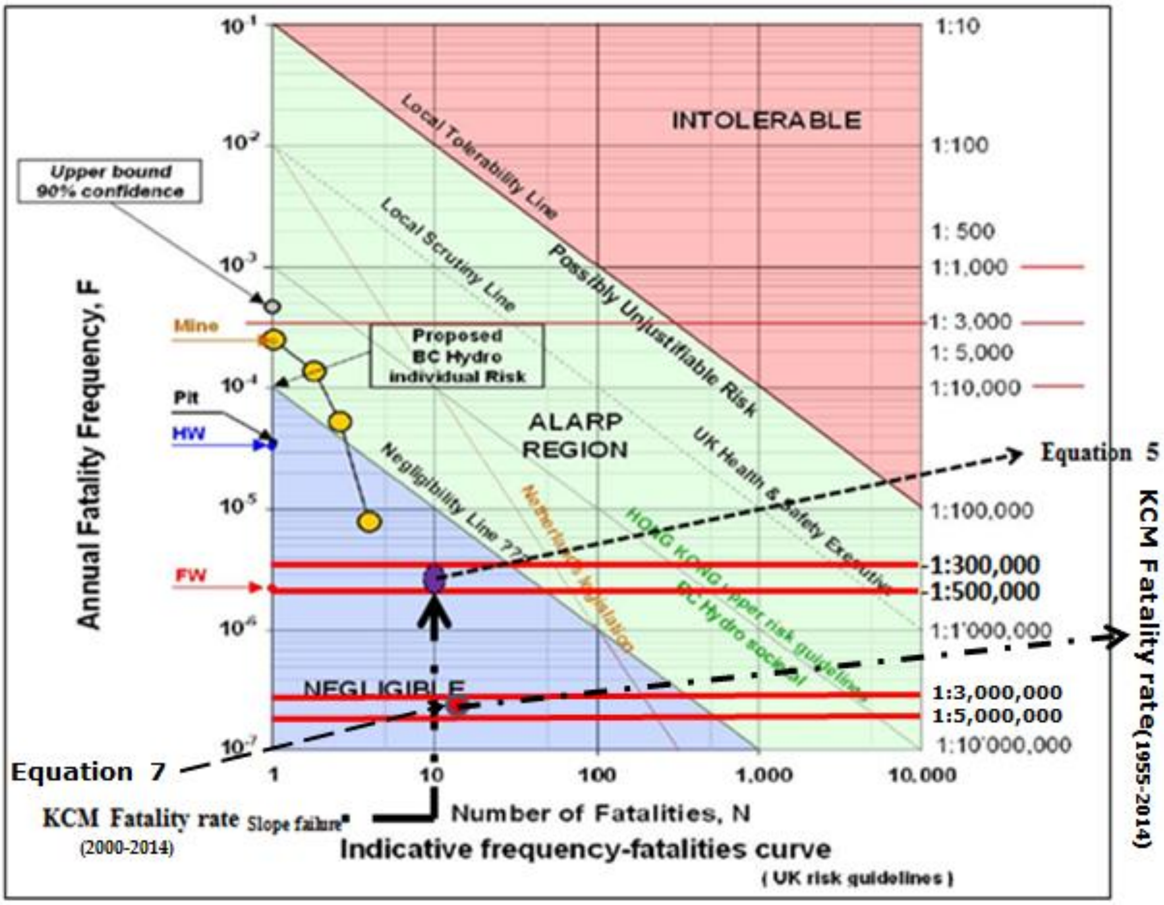



*Figure 3. Benchmark criteria for safety impact of slope failure with KCM slope fatal accident rate from 2000 to 2014 and 1955 to 2014 included by authors (modified from UK Health and Safety Executive, 1992).*

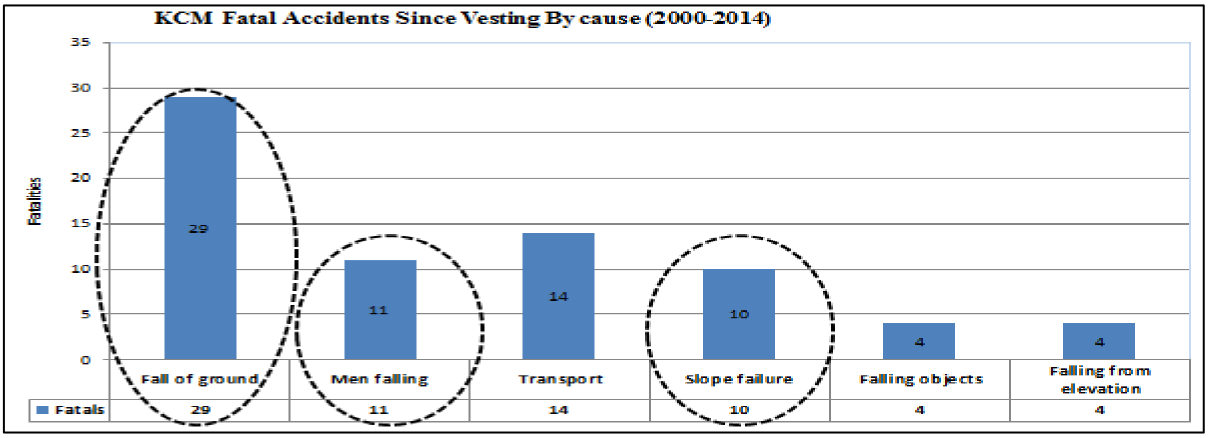



*Figure 4. Konkola Copper Mines fatal accident statistics 2000–2014 (KCM internal reports, June 2014).*

$$FAR = \text{Number of fatalities} \times 10^8 / (\text{total working hrs by all employees during period}) \quad [3]$$

$$FAR_{\text{SLOPE FAILURE (2000-2014)}} = 10 \times 10^8 / (10^8 \times 14) = 0.71 \quad [4]$$

Fatality Rate = (exposed hrs/person per year) x FAR/$10^8$)
= (50 weeks/year x 40 h/week) x 0.71/$10^8$
= 1.42 x $10^{-5}$ deaths/person.year [5]

***Fatality Rate Calculations since Start of Nchanga Open Pit (Slope Failures form 1955 to 2014)***

$FAR_{(1955\ TO\ 2014)}$ = 11 x $10^8$/ ( $10^8$x 59) = 0.19 [6]

Fatality Rate = 50 weeks/year x 40hrs/week) x 0.19/$10^8$
= 3.8 x $10^{-6}$ deaths/person.year [7]

The calculated fatal accident rate for Nchanga Open Pit due to slope failures falls below the negligible line (Figure 3) using the benchmark criteria for safety impact of slope failures (UK Health and Safety Executive, 2001).

It can be seen from Table V that a person working at COP is not exposed to greater risk of death from slope failure than he/she is of travelling along the Kitwe–Chingola Road (the nearest international highway) in the period under consideration.

*Table V. Chingola–Kitwe Road (nearest international highway to Democratic Republic of Congo) accidents statistics (Government Republic of Zambia, 2011).*

| Chingola - Kitwe Road Accident Statistics for General Public | | | | | |
|---|---|---|---|---|---|
| Year | Fatal | Serious Injuries | Slight Injuries | Damage of property | Total |
| 2001 | 36 | 57 | 71 | 224 | 388 |
| 2002 | 39 | 38 | 79 | 219 | 375 |
| 2003 | 32 | 50 | 101 | 230 | 413 |
| 2004 | 36 | 80 | 91 | 273 | 480 |
| 2005 | 71 | 95 | 118 | 283 | 567 |
| 2006 | 50 | 105 | 137 | 390 | 682 |
| Total | 264 | 425 | 597 | 1619 | 2905 |
| | | | | | |
| | Fatal/ year | Serious Injuries | Slight Injuries | Damage of property | |
| FAR | 44.0 | 70.8 | 99.5 | 269.8 | |
| Fatality Rate | 8.8 x $10^{-3}$ | 1.42 x $10^{-2}$ | 1.99 x $10^{-2}$ | 5.4 x $10^{-2}$ | |
| Assuming 50 KCM employees travel this road every morning and evening to go for work | | | | | |

**Rock Fall Analysis for Chingola Open Pit Using Empirical Approaches - ROFRAQ**

The major risks related to the risk-optimised approach at COP D&F site is the increasing risk of rock falls posing danger to both the quality of human life and equipment. The general understanding in mining geotechnics is that people are at low risk from medium- to large-scale failures in open-pit mines: '….*slopes seldom fail without giving adequate warning*' (Hoek & Bray, 1972). However, the rise in incidents of fatalities and serious disabilities is due to small-scale failures and rock falls.

One empirical tool which assesses the risk associated with rock falls is the Rock Fall Risk Assessment for Quarries and Open-Pit Mines (ROFRAQ), as shown in ROFRAQ Table VI and its associated assessment classification as shown in Table VII. To measure the likelihood of a rock fall occurring on a slope, irrespective of the final outcome (accident/no accident), a primary version of the ROFRAQ index was used, called $ROFRAQ_{BASIC}$, which is based on four ratings (A to D):

$ROFRAQ_{BASIC}$ = (A x B x C x D) [8]

Rating E is based on the COP D&F mine plans while F is based on site accident statistics due to rock falls.

The preliminary assessment of rock fall hazards, as described by the ROFRAQ datasheet (Table VI), was used to examine rock fall hazards at COP D&F in Table VII. The chance of a coincidence between a rock falling down and hitting a person on the foot is very low in the area which was designed using a risk-optimised approach. For machines working close to the toe of the slopes, like drill machines, require some mitigation measures to avoid rock falls hitting the machine. Nevertheless, a proactive approach is adopted by enhancing risk mitigation measures.

ROFRAQ also provides a differentiated E rating for machines ($E_{machines}$) and for workers ($E_{persons}$). The reason for this is that there are many manual workers for whom the risk of a fatality is higher than for workers protected by machines. ROFRAQ GENERAL accounts for risk of rock falls irrespective of any danger to either persons or machines, hence the equations below:

$$\text{ROFRAQ}_{\text{GENERAL}} = (A \times B \times C \times D \times E \times F) = 11.29 \quad [9]$$

$$\text{ROFRAQ}_{\text{PERSON}} = (A \times B \times C \times D \times E_{persons} \times F) = 7.59 \quad [10]$$

$$\text{ROFRAQ}_{\text{MACHINE}} = (A \times B \times C \times D \times E_{machine} \times F) = 19.01 \quad [11]$$

An estimated probability of an accident due to rock fall occurring on the slope = ROFRAQ/100,000 [12]

## RISK MITIGATION MEASURES

### Proposed Risk-Mitigation Tactical Options for COP D&F

The adoption of a risk-optimised approach to slope designing at COP D&F would underpin management involvement and commitment to reducing risks associated with slope failures to *ALARP* while maximising resource utilisation. The management tactical options to mitigate physical risks from slope failures in the operational areas would have to limit the usage of manually biased options, which have not been successful in the past, and instead to deploy the following:

- Automated slope stability radar: The foundation for personnel safety before, during and after major ground movement would remain, as now, the real-time data capturing and analysis provided by automated slope stability radar (SSR™) technology or variants.
- Fallback satellite and survey slope monitoring beacon system: This would ensure continuity of data capturing and analysis of the behaviour of slopes for decision making in the event of loss of the real-time capability of the slope stability radar.
- Pit bottom remotely controlled loading and hauling equipment: For certain critically marked sections of the pit, this would be a cross function between robotically operated haul trucks and loading equipment and human-operated equipment remotely controlled from a specially designed view point at a safe location within the pit.

### Slope Water Management

Water in the slopes at Chingola and Nchanga open pits has been at the root of slope failure numerous times in the past. In the case of COP D&F, water was historically the root cause of slope instability, especially from 0 to 90 mb, and so an aggressive strata dewatering programme was initiated, including surface well dewatering systems using borehole pumping and sub-horizontal geo-drains, as well as surface capping and cambering to prevent rain-water infiltration into the slopes.

*Table VI. ROFRAQ for rock fall risk in quarry & open pit slopes (Alejano & Stockhausen, 2008).*

ROCKFALL RISK ASSESSMENT FOR QUARRIES (ROFRAQ)

QUARRY______________________ SLOPE:______________________ DATE:______________

| ARE THESE POTENTIAL FALLING BLOCKS? | Joint sets (a) | Joint continuity (b) | | Faults (c) | | Observed overblasting damage (d) | Bench & berm cleanup (e) | | Blocks on slopes (f) | Slope height (g) | A = [[(a*b)+c+d+e+f]/2]*g |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | 0-1 | 0.5 <1m | 0.6 | NO | 0 | Pre-spit | -2 Very regular | -3 Many | 7 - 10 <25m | 0.5 | |
| | 2 | 1 1-3m | 0.8 | Small x 1 | 0.5 | Half barreis | -0.5 Regular | -1 Several | 3-7 50m | 0.8 | 0<A<10 |
| | 3 | 3 3-10m | 1 | Large x 1 | 1 | None | 0 Occassional | 0 Some | 1-3 100m | 1 | |
| a | 4 | 5 10-2Cm | 1.2 | 2 | 1.5 | Slight | 1 Infrequent | 1 Few | 0.1-1 150m | 1.3 | |
| | >5 | 7 >20m | 1.4 | >2 | 2 | Excess overbreak | 2.5 Little/none | 3 None | 0.0-0.1 >250m | 1.5 | |

| ARE THE BLOCKS PONTENTIALLY UNSTABLE? | GEOTECHINICAL STUDY + JOINT INTERPRETATION + IN SITU OBSERVATION OF SLOPE INSTABILITY MECHANISMS | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Mechanism type (observed or estimated) | | | | | Affected slope area (%) | | | | |
| | SIMPLE | B+ (*0.8) | COMPLEX | B* (*1.1) | EVOLUTIVE B* (*1.4) | | IN-SITU OBS | GEOTECH STUDY | FALLEN BLOCKS AT SLOPE TOE | AVERAGE (correct B*) P |
| | Plane failure | | Ravelling | | Block toppling | A | | | | |
| | Wedge failure | | Many blocks | | Flexural toppling | B | | | | |
| b | Circular failure | | Free Blocks | | Footwall slopes | C | | | | |
| | B* is a correction factor according to failure mechanism complexity | | | | | D | | | | |
| | When Pi is the corrected average for the affected slope ares % Rating (B) is calculated as B=10 x {1 -[22/7 (1-Pi/100)]} | | | | | | | | 0<B<10 | |

| IS A TRIGGERING EFFECT LIKELY TO PRODUCE INSTABILITY? | Max 24h rainfall for a 50 year return period (a) | | Average $0^0$ frost-free period (days) (b) | | Slope water (C) | Weathering/erosion (D) | | Blasting vibration (specific load) (e) | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | <50mm | 0.1 | >300 | 0.1 | Dry 0 | Unweathered | 0 | <250 g/m³ | 0.1 | |
| | 50 - 80mm | 0.5 | 250-300 | 0.4 | Damp 0.1 | Slghtly weathered | 0.1 | 250-400 g/m³ | 0.5 | C=a+b+c+d+e |
| | 80 - 110mm | 1.5 | 200-250 | 1 | Wet 0.3 | Weathered | 0.3 | 400-550 g/m³ | 1 | 0<C<10 |
| c | 110-150mm | 4 | 150-200 | 2 | Dripping 0.7 | Very weathered | 0.7 | 550-700g/m³ | 1.5 | |
| | >150mm | 8 | <150 | 2.5 | Flowing 1 | Extremely weathered | 1 | >700g/m³ | 2 | |

| ARE THE BLOCKS LIKELY TO REACH THE MINE BOTTOM | Bench heihgt (m) & Berm/catch-bench width (BW) (m) | | | | | General Slope dip (a) | Berm conditons & cleanup (b) | Face irregularity (c) | D=(Dъ + a +b +c)*d |
|---|---|---|---|---|---|---|---|---|---|
| | 5 | 10 | 12 | 15 | 20 | | | | |
| | BW Dъ | BW Dъ | BW Dъ | BW Dъ | BW Dъ | | | High +2 | |
| | <1 10 | <1 10 | <2 10 | <3 10 | <4 10 | 40* +2 Optimal | 0 | Medium +0.5 | |
| | 1.5 7 | 2 9 | 3 8 | 4 9 | 5 9.5 | 48* +1 Good | +0.5 | Low 0 | 0<D<10 |
| | 2 4 | 3 6 | 4 5 | 5 6 | 6 7.5 | 56* +0.5 Limited | +1 | Mesh/ditches (d) | |
| | 2.5 1 | 4 2 | 5 2.5 | 6 3 | 7.5 2.25 | 60* +0.5 Poor | 1.5 | | |
| d | <3 0 | 5 0.2 | 6 1 | 7 1 | 9 0.5 | 72* 0.5 Very poor | 3 | Yes 0 | |
| | | >5.5 | 0 7> | <8.5 0 | 10 0 | 80* -1 | | No 1 | |

| ARE BLOCKS LIKELY TO IMPACT ON WORKERS OR MACHINERY | Block size (f falling block) (a) | Rock volume (>1 falling blockes (a) | Time spent at slope toe by machinery workers (%) (b) | Space occupied by machinery Vs slope length ( ) | Position of machinery/workers in relation to slope toe (d) | $E_{mech}$ = (b/100)*(c-mach/100* (a or a)*d*10 |
|---|---|---|---|---|---|---|
| | <0.001m³ 0.9 | <0 m³ 1 | Days/year | Space occupied by machinery Vs slope length ( ) | Very close (x/H <10%) 1 | Epers = [(b/100) *(c-Epers)/100)*(a1 or a2)*d*10 |
| | 0.001-01m³ 1 | 0.1-5m³ 1.5 | Shift/day | | Close (x/H <25%) 0.5 | |
| | 0.1-1m³ 1.2 | 5-50m³ 2 | hours/shift | space occupied by persons (1m Vs slope length ( ) | Average (x/H <50% 0.1 | E = 10 -[(10-Emach)*(10-Epers)/10] |
| e | >1m³ 1.5 | >5C m³ 2.5 | % time under slope | | Far (x/H <100%) 0.01 | 0.00025<E<10 |
| | | | | | Ver far (x/H >100% 0.001 | |

| f | SLOPE ROCKFALL HISTORY | | | | | |
|---|---|---|---|---|---|---|
| No recorded rockfalls | Very few rocksfalls | No data Unreliable observations | Occasional rockfalls | Several rockfalls. No accidents | some rockfalls. 1 reported accident | Frequent rockfalls >1 reported accident |
| 0.75 | 0.9 | 1 | 1.1 | 1.2 | 1.4 | 1.5 |

*Table VII. Assessment of the slope face hazard according to ROFRAQ datasheet (summary equations 9 to 12).*

| Very low risk | Low Risk | Low to Average Risk | Average Risk | High Risk | Very High Risk |
|---|---|---|---|---|---|
| No special measures required | Simple measures required including regular observations | Some simple safety precautions required. E.g avoid traffic during rainy period. | Important safety measuresrequired. Improve berms and catch benches, clean ups, blasting control techniques | Highly Hazardous, redesign mine plan, enlarge catch benches | Redesign pit slopes, install meshes, build ditches |

**Cost Factor of Risk Optimisation and Mitigation Measures**
The cost of mitigating measures compared with the benefits of risk optimisation in avoiding 4.5 million tonnes of waste mining needed to be evaluated. At the simplest, by Zambian Copperbelt standards, the cost of mining 4.5 million tonnes of waste at up to 180 m in depth is in the order of tens of millions of US dollars. On the other hand, the cost of the mitigation measures suggested would be just a few million dollars.

## CONCLUSIONS

Firstly, the risk-optimised approach of slope designs applied between 90 mb and 180 mb of the COP D&F hanging wall, using a bottom-up design principle, resulted in face angle optimisation from 60° to 70° and berm width reduction from 10 m to 5 m. With this approach, approximately 4.5 million tonnes of waste rock material would be reduced compared with the traditional approach, while maintaining the same ore volume compared with that obtained from the traditional methodology.

Secondly, the concepts of tolerable risk and acceptable risk are identified as applicable in a scenario similar to that of the COP D&F. International risk evaluation systems were considered and measures of risk at global level were identified and benchmarked to KCM slope FAR from years 1955 to 2014, which indicated that KCM statistics were within negligible range compared to global values. A local international highway was used to domestically benchmark the KCM open-pit slope FAR from years 2000 to 2006, Chingola D&F included, which indicated that FAR for a risk-optimised approach was below that of the Chingola–Kitwe highway. An additional analysis, using the ROFRAQ approach yielded values between 7.59 and 19.01 for 'machines', 'personnel' and 'general', which put the risk of any rock fall hitting people or machinery as 'Low Risk', according to the ROFRAQ scale.

Thirdly, strategic and tactical management options of risk mitigation measures to reduce the risks of fatality or injury to *ALARP* are proposed. Even if calculations of risk indicate, as in this paper, 'neglible' or 'low risk', dependent on the benchmark used, risk optimisations need to develop real-life engagement with the real-world fears that the workforce, boards, management and other stakeholders may have of any slope-management option which includes the word 'risk': it is both an emotional and technical term. In recognition of this, an array of technological options were proposed for cost-effective in-pit deployment to enhance personnel and equipment safety, including slope water management, automated slope stability radar, satellite based and survey beacon monitoring, as well as remote and/or robotic loading and hauling in critical areas.

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**REFERENCES**


Alejano, L.R. and Stockhausen, H. (2003). An empirical method to estimate the risk of accidents due to rock falls in quarries (II): a case study. *Proceedings of International Conference on Slope Engineering,* Hong Kong. pp. 140–145.

AS/NZS4360 (2004). Risk management, principles and guidelines standards Australia. Available from https://www.saiglobal.com/ [accessed 18 November, 2013].

Brown, E. T. and Booth, A. (2009). *Risk Management and Guidelines for Open Pit Design*. CSIRO, Melbourne.

Contreras, L.F. and Steffen, O.K.H. (2006). Risk based pit slope design. SRK Consulting, Johannesburg, South Africa.

Government of the Republic of Zambia (2011). National Assembly of Zambia: Kitwe–Chingola–DRC accident statistics. https://www.parliamnet.gov.zm/index.php? Option=com_content & task. [Accessed 13 August, 2013].

Hoek, E. and Bray, J.W. (1972). *Rock Slope Engineering*. Institution of Mining and Metallurgy, London.

Lee, E.M. & Jones, D.K.C. (2004). *Landslide Risk Assessment*. Thomas Telford, London.

Steffen, O.K.H., Contreras, L.F., Terbrugge, P.J. and Venter, J. (2008). A risk evaluation approach for pit slope design. *Paper ARMA 08-231 presentation at the 42nd US Rock Mechanics Symposium and 2nd U.S.-Canada Rock Mechanics Symposium*, San Francisco, June 29-July 2, 2008.

Terbrugge, P.J., Wesseloo, J., Venter, J. and Steffen, O.K.H. (2006). A risk consequence approach to open pit slope design. *Journal of the South African Institute of Mining and Metallurgy*. pp 82, 86-87.

UK Health and Safety Executive (1992). *The Tolerability of Risk from Nuclear Power Stations*. London, UK.