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Decision-making criteria in rare earths exploration projects: an interview study

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Abstract: When the rare earth prices skyrocketed in 2011, over 400 exploration projects appeared in the rest of the world (ROW). Before an exploration project comes into production, it has to pass through various stages of project development and face multiple challenges at each of these stages. According to Cooper's stage-gate system, a decision about whether to move on to the next project stage should be based on the evaluation of certain criteria. The case of rare earth elements (REEs), however, differs from other metals in terms of mineralogy, market, technology, environmental issues and strategic importance. Therefore, the decision criteria might also partly differ. To find these criteria for the case of rare earths, interviews with decision-makers from several rare earths projects at different stages of development were conducted. In this paper, obtained criteria are listed, explained and analysed for each stage/gate. Suggestions about their application to project management are made.

Keywords: exploration; mining; project development process; project management; project evaluation; decision-making; stage-gate system; decision-making criteria; rare earth elements; interview study.

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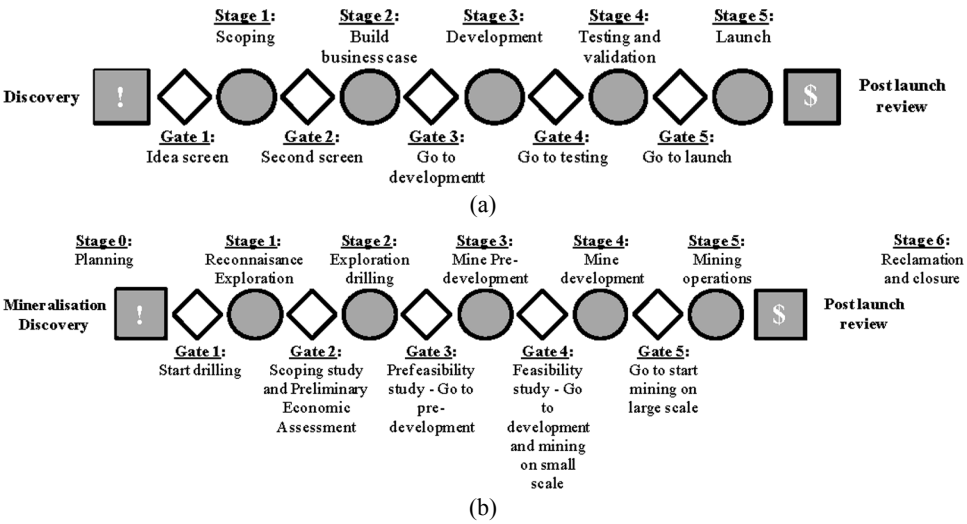
1 Introduction

Exploration project development is a complex process, which consists of various stages and can easily take 10–15 years from planning exploration to mining and market entrance (e.g., Hartman and Mutmansky, 2002; Moon and Whateley, 2006). It can be represented and explained by a concept of Cooper (1990) to manage the development of new products, the so-called stage-gate system. According to this approach, the process is divided into stages, which represent actions and gates – decision points (see Figure 1).

A decision, whether to proceed to the next stage, wait with, abandon or revise the project, is made based on decision criteria, which are set for every gate and evaluated by gatekeepers, i.e., a multidisciplinary and multifunctional group of members with senior status. On the basis of these criteria and inputs from the project management, gatekeepers evaluate the project and decide on its further development. The criteria vary along the project development process (Cooper, 1990, 2008).

Figure 1 Illustration of stage-gate system and an exploration project development process



Source: (a) adopted from Cooper (1990) and (b) authors based on Moon and Whateley (2006)

Similarly to Cooper's concept, Noort and Adams (2006) in their approach to the effective management of mining projects state the necessity of project audit or peer-review where project fundamentals covering technical, environment-social, legal and economic aspects are evaluated by company seniors to study whether the project is

able to advance to the next stage or to identify directions of further work to ensure this. These project fundamentals in this case play to some extent a role of decision criteria as in Cooper's stage-gate approach.

Mining project development typically follows a path similar to Cooper's (Figure 1), often taking formalised steps like preliminary economic assessment (PEA), prefeasibility study (PFS), feasibility study (FS) and the final investment decision. In huge mining companies, the decisions are made by the senior project management, therefore it decides on the needed information and criteria. In turn, in junior exploration companies, unlike Cooper's concept, decisions about further investment in a project are made by investors. Thus, they define required information. Reliability of information is promoted by reporting standards such as NI43-101 or JORC, or stock exchange rules and mining regulations. Viable mining regulation is hence a key issue to the decision process and investment.

For the exploration projects, the decision criteria are generally not clearly defined and listed individually for each of the stages/gates. Various authors/scholars having published papers with relevance to our paper do not directly refer to decision-making criteria. In their works, they either speak about factors influencing economic characteristics of mining projects (e.g., Fettweis et al., 1990; Slaby and Wilke, 2005), or generally and randomly mention factors influencing mining project success (e.g., Evans and Moon, 2006; Golev et al., 2014; Laird, 1997; Moon and Whateley, 2006; Rudenno, 2009; Scott and Whateley, 2006; Wellmer and Neumann, 1999; Wellmer, 2008), or focus just on one specific issue (e.g., Evans, 2006; Pike and Thibodeau, 1981).

Furthermore, there can be additional criteria depending on, e.g., the metal to be mined. For example, critical metals, e.g., REEs, have several special characteristics, which make their exploration or mining projects different from other projects. In the case of REE, these characteristics are, for example, connected to the rare earths market, processing technology, missing comparative project data, joint occurrence in minerals, end-products, by-products and strategic considerations (Klossek and Boogaart, 2013).

The aim of our paper is, thus, to clearly define and assign decision-making criteria to each of the stages/gates of REE projects and highlight those criteria that only or mostly hold specifically for an REE project.

The paper is organised as follows: Section 2 describes the methodology we used in the paper. Section 3 presents results of the interviews. Sections 4 and 5, respectively, describe results of the analysis and discuss their implications. Finally, in Section 6, conclusions and possible avenues for further research are provided.

2 Methodology

Our methodological approach consists of two major steps: First, we used interviews to collect the necessary data; second, we structured the obtained data according to different categories.

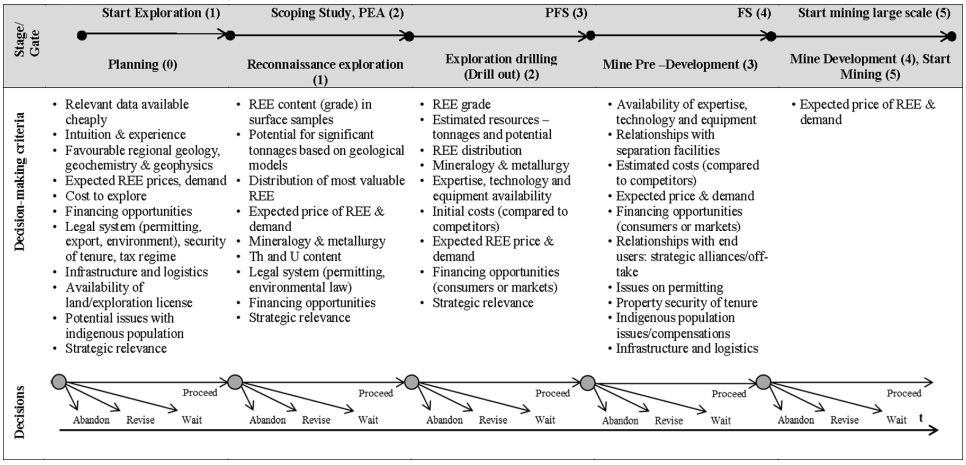
Step 1: Interviews

To define the criteria used for decision-making in REE projects, we interviewed decision-makers (Chief Executive Officers (CEOs), Chief Operating Officers (COOs), Managing Directors, Chief Geologists, or similar) of some of the most developed REE projects, based on their availability and openness to talk (for the list of interviewees see

Appendix 1). In some cases, we had a chance to talk to several people from the same company (e.g., CEO, Project Development Manager and Senior Geologist). In total, we interviewed 16 persons from 11 companies. Additionally, we spoke to an REE market expert Professor Dudley Kingsnorth from IMCOA, Australia, who is advising some of the leading REE companies concerning their market strategies. The majority of interviews were performed personally; few were made per e-mail or Skype. They took place in the period from November 2011 to May 2012.

First, we created a form to be used as a basis for discussion with interviewees (see Appendix 2). The form graphically represented the exploration project development process (stages and milestones, i.e., gates) with possible decision-making criteria for each stage. This information was extracted from the selective review of relevant literature (Evans, 2006; Evans and Moon, 2006; Golev et al., 2014; Fettweis et al., 1990; Laird, 1997; Moon and Whateley, 2006; Pike and Thibodeau, 1981; Rudenno, 2009; Scott and Whateley, 2006; Slaby and Wilke, 2005; Wellmer and Neumann, 1999; Wellmer, 2008). Interviewees were asked to challenge and complete a given list of criteria. Also, some of the interviewees added information on the project development process. At the time of the interviews, the projects were at different stages of development (from early exploration to small-scale mining). Hence, we collected views from different standpoints on the REE project development and related decision criteria. Having combined the results of the interviews, we obtained one cumulative list of decision criteria and one collective view on the REE project development process. To be able to show all obtained information, we changed the way of graphical representation and more clearly defined milestones,
i.e., gates, in the project development process (see Figure 2).

Figure 2 Results of the interviews: REE decision-making criteria



Step 2: Categorisation

First, we analysed the information obtained to discover which criteria are specific for the REE project development process.

Second, we characterised the criteria according to their influence on (see Figure 3):

- *Viability of the project*. The criteria with this characteristic determine the viability of the project and are, therefore, knock-out criteria. Qualitative evaluation of them suffices. An example could be a regulation prohibiting mining radioactive REE ores.
- *Economics of the project*. We assigned this characteristic to those criteria that determine costs or earnings of the project and for which quantitative evaluation is required. A typical example is the expected basket value of separated REE oxides. Economic factors can become viability factors if the costs (revenues) exceed (go below) a certain threshold and make the project uneconomical, leading to abandoning the project.
- *Uncertainty of the project*. These criteria represent uncertainty of the project economics and viability. A typical example could be uncertainty in developing an economical processing method for a particular type of ore.

Figure 3 Analysis of obtained decision-making criteria

Stage/Gate	Stage 0, Gate 1	Stage 1, Gate 2	Stage 2, Gate 3	Stage 3, Gate 4	Stage 4, Gate 5
Prevailing	V/E/NM	V/I/NM	Ec/I/NM	V/I/M	Ec/E/NM
Decision-making criteria	**Geological:** • Favourable regional geology, geochemistry & geophysics (V/U, I, NM) **Technological:** - **Market: …** • *Expected REE prices&demand (Ec/U, E, NM)…* • *Strategic relevance (Ec, E, NM)* **Project economics:** • Relevant data available cheaply (Ec, E, NM) • Intuition & experience (Ec, I, M) • Cost to explore (Ec, I, M) **Financing:** • Opportunities for project financing (V, E, NM)… • Reconnaissance exploration financing available (V, I, M) **Stakeholder related: …** • Legal system (permitting, export, environment), security of tenure (V, E, NM) • Tax regime (Ec, E, NM) • Availability of land/exploration licence (V, E, NM) • Potential issues with indigenous population (V, E, M) **Other :** • Infrastructure and logistics (Ec, E, M) • Accessibility of potential deposit (V, E, NM)	**Geological:** • REE content (grade) in surface samples (V, I, NM) • *REE basket (V, I, NM)* • Potential for significant tonnages based on geological models (V, I, NM) • *High Th and U content in combination with legislation and corporate policy (V, I, NM)* • *Mineralogy (V, I, NM)* **Technological:** • *(Potential) availability and acceptability of metallurgical options (V/U, I, NM)* **Market: …** • *Strategic relevance (Ec, E, NM)* **Project economics:** - **Financing: …** • Exploration and PFS financing available (V, I, M) **Stakeholder related: …** **Other :** -	**Geological:** • Estimated REE resources and remaining potential to discover more (Ec, I, NM) • *Mineralogy (Ec, I, NM)* **Technological:** • *Availability and acceptability of metallurgy – positive results in laboratory (V/U, I, NM)* • *Expertise, technology, equipment availability on the market (V/U, E, NM)* **Market: …** • *Strategic relevance (Ec, E, NM)* **Project economics:** • Initial mining & process costs (Ec, I, NM) • Initial costs compared to competitors (V, I, NM) **Financing: …** • FS financing available (V, I, M) **Stakeholder related: …** **Other :** -	**Geological:** - **Technological:** • *Relationships with separation facilities (technical partnerships) (V, I, M)* • *Availability of expertise, technology (positive results in Pilot Plant) and equipment in project (V, I, M)* **Market: …** • *Relationships with end users: strategic alliances/off-take (V, I, M)* **Project economics:** • Estimated Opex & Capex (Ec, I, M) • Costs compared to competitors (V, I, M) **Financing: …** • Design and construction financing available (V, I, M) **Stakeholder related: …** • Mining permit (V, I, M) • Property security of tenure (V, I, NM) • Social license (V, I, M) **Other:** • Infrastructure and logistics (Ec, I, M)	**Geological:** - **Technological:** - **Market: …** **Project economics:** - **Financing:** - **Stake holder related:**- **Other:** -
Groups	Present	Focus	Focus	Not present	**Geological**
	Not present	Present	Focus	Focus	**Technological**
	Present/Monitored	Present/Monitored	Present/Monitored	Present/Monitored	**Market**
	Focus	Not present	Focus	Focus	**Project economics**
	Present/Monitored	Present/Monitored	Present/Monitored	Present/Monitored	**Financing**
	Focus	Monitored	Monitored	Focus	**Stakeholders**
	Present	Not present	Not present	Present	**Other**

LEGEND
V – influences project viability; **Ec** – influences project economics; **V, Ec/U** – represents uncertainty of project viability or economics; **I** – internal origin; **E** – external origin; **M** – modifiable
NM – non-modifiable
… to be permanently monitored criterion; *Criterion* – critical for the case of REE.

Third, we differentiated the criteria according to their origin, i.e., internal or external. Internal criteria are project-related, e.g., the grade in the deposit. External criteria are related to the project surroundings. They represent an opportunity or challenge equal to all the projects in the same area. Internal is about how the project management uses this opportunity or deals with this challenge. Mining legislation is such an external criterion. Having obtained mining licence is the corresponding internal criterion.

Fourth, depending on if a criterion can be changed by the project management, we characterised it as modifiable or non-modifiable. E.g., project acceptance by a local community is a classic modifiable criterion. A good project management can build the necessary trust with local communities over time.

Fifth, we allocated the criteria into the groups: geological, technological, market, project economics, financing, stakeholder related and other criteria.

Finally, we analysed the results horizontally (along the project development) to identify tendencies and discussed the application of the criteria to the project management.

3 What is different for the case of REE?

Most of the criteria obtained in the interviews hold for all exploration projects. Some of them, however, are specific for the REE project development process (criteria marked in italics in the following text and in Figure 3). Such criteria are REE expected price and demand, REE basket, mineralogy, technological availability, uranium and thorium contents in combination with the environmental legislation and corporate policy, strategic relevance, strategic relationships with separation facilities and REE consumers. In the following, the REE project development process and the REE decision criteria (see Figure 2) will be discussed stage/gate wise.

Stage 0: *Planning stage*

This stage is mostly about collecting and reviewing available data, the exploration history, and based on this, selecting a region and an area for reconnaissance (e.g., Moon and Whateley, 2006). For the case of REE, studying exploration history for uranium is considered by the interviewees to be helpful in finding an REE mineralisation as they often occur together in deposits. Once a region and an area are selected, regional mineral trends are identified and deposit and exploration models are built (e.g., Evans and Moon, 2006). Additionally, stakeholders are identified and country risk is evaluated (e.g., Scott and Whateley, 2006).

Gate 1: Start exploration

On the basis of the information collected and analysed, a decision about starting exploration is made.

Criteria:

- Relevant data available cheaply

At the planning stage, availability of relevant data at the lowest possible cost is an important criterion. At such an early stage, project management cannot afford high costs as financing is still to be found.

- Intuition and experience

In the process of region and area selection as well as creating exploration models, the experience and often intuition of a geologist play an important role.

- Favourable regional geology, geochemistry and geophysics

Naturally, geology, geochemistry and geophysics have to be estimated as favourable. Otherwise, it does not make sense to continue with the project.

- Expected prices of REE, expected demand and costs to explore

It is important to track REE market developments and be acquainted with demand and price forecasts, still keeping in mind that they are highly unsure (e.g., Scott and Whateley, 2006). REE prices are strategically influenced by China, which makes it difficult to predict their development. *This makes the 'expected price' criterion especially critical for the case of REE.*

- Financing opportunities

Financing opportunities should be estimated as financing is a critical issue for the project development.

- Legal system, infrastructure and logistics, availability of land and exploration licences and relationship with the indigenous population

These external factors need to be studied in detail as they may cause project failure, delays or other complications as well as additional costs. It is important to establish a trustful relationship with local communities, learn their needs and involve them in project development from the very beginning of the project. Good infrastructure (e.g., access to roads, energy, water and work force, etc.) in place is essential and brings project cost advantages (Rudenno, 2009). In general, country risk or political risk is important to be estimated. However, high country risk can be balanced out by minimising payback period of a project (Wellmer and Neumann, 1999) or if geological characteristics are favourable (Wellmer, 2008).

- Strategic relevance

Strategic relevance of REE for the country of operation and its industry is an additional factor, which can foster the REE project development. *This is another criterion, which is REE case-specific.*

Stage 1: Reconnaissance exploration

The aim of this stage is to study in more detail the area of interest identified at the previous stage. Exploration rights are acquired. At this stage, geoscientific surveys and studies are conducted, REE surface samples are analysed, targets and their size are identified, and initial drilling is conducted (e.g., Moon and Whateley, 2006). Additionally, mineralogy is determined and initial metallurgical tests are conducted (e.g., Evans, 2006). It is especially critical for the case of REE to perform the tests as early as possible. This is due to the complex, or in some cases, unknown mineralogy, for which ecologically or economically acceptable processing technologies do not exist in the ROW. Therefore, additional extensive metallurgical tests are highly relevant at the following exploration drilling stage.

Gate 2: Scoping study

On the basis of the information obtained in Stage 1, scoping study is prepared within which the first economic assessment is made. Scoping study aims at defining project potential and eliminating non-viable options (i.e., operating scenarios). PEA serves as a basis for the decision about financing of the further exploration activities (e.g., Noort and Adams, 2006).

Criteria:

- REE content (grade) and tonnages

There is already first geological information available, which can be evaluated, e.g., REE content in surface samples and distribution of the most valuable REE. Resource potential is estimated based on the defined cut-off grade (e.g., Whateley and Scott, 2006). Also, the potential for significant tonnages can be estimated based on geological models. If the REE content in surface samples shows potential for high grades of total rare earth oxides (TREO) in the deposit, it is a positive sign for the project. High grade means higher efficiency, meaning that fewer amounts should be mined and processed to obtain the same amount of product, and, therefore, lower costs. However, even if the potential grade is low, it can still be balanced out by high tonnages and favourable shape and location of ore body, i.e., close to surface, etc. Additionally, high tonnages mean longer life of mine (at the constant production rate), which means longer period to amortise mining and processing capital expenditures (CAPEX), but, on the other hand, more uncertainty. High tonnages can give the project a possibility to benefit from economies of scale (e.g., Scott and Whateley, 2006). In this regard, an optimal life of mine is calculated (e.g., Wellmer and Neumann, 1999).

- REE basket (distribution of the most valuable REE), expected price and demand

As well, it is highly important for an REE project to have potential for high grades of the most critical REE. It is well known that the demand for most of the light REE (LREE) will be soon satisfied, while shortages of such REE as dysprosium (Dy), neodymium (Nd), yttrium (Y), terbium (Tb), erbium (Er) and europium (Eu) are expected for 2015 (Citigroup – Global Commodities Research, 2013). It means that there are higher chances to enter the REE market and survive competition for those REE projects that can deliver these critical REE to the market. Such projects have higher basket price, as such elements as Tb, Eu and Dy have the highest price from all tradable REE (Metal-Pages, 2014). Higher basket price means higher earnings. *REE basket (distribution), expected price and demand are REE case-specific decision-making criteria.*

- Mineralogy and metallurgy

The deposits that are especially rich in heavy REE (HREE; some of them are critical as discussed before), generally, have more complex mineralogy. For example, at least nine mineral species in the Strange Lake deposit of Quest Rare Minerals are considered as REE-bearing (Gysi and Williams-Jones, 2013). Moreover, these REE minerals belong to different mineralogical groups, i.e., silicates, carbonates and phosphates (Gysi and Williams-Jones, 2013). This requires more complex technological approaches. Apart from that, some minerals can appear to be unknown. Processing technologies are usually deposit and mineralogy specific, which means that new project-specific processing approaches have to be developed, which is an uncertainty factor. These issues result in higher processing costs.

Furthermore, there is no market for REE mixed oxides outside China, although it might emerge when and if mixed oxides and separation facilities become available on the market. Thus, unlike conventional mining projects, REE projects are forced to deal with further stages of the value chain and face additional technological challenges connected with, above all, separation. As a trade-off, the projects, however, get more value, most of

which is naturally being created at the end of the value chain. *These facts make mineralogy and metallurgy especially critical criteria for the case of REE.*

- Uranium and thorium content and legal system

An additional issue for the REE project development can be potential for high uranium and thorium content in the deposit. Uranium and thorium often occur together with REE in the same deposit as they are chemically very similar to each other (REITA, 2012). Not only content of these radioactive elements in the deposit has to be analysed, but also their content in the mineral concentrate, which can become an issue, e.g., for the transportation. Uranium and thorium content has to be evaluated in combination with the environmental legislation of the country of operation and further processing as there can be an additional challenge in the permitting process. For example, a Greenlandic REE project called Kvanefjeld has been facing challenges in permitting process owing to the high content of uranium and thorium until Greenland's parliament decided on removing zero-tolerance policy concerning radioactive elements in October 2013 (Greenland Minerals and Energy, 2013). *Uranium and thorium content is an REE case-specific criterion.*

Even if the legislation tolerates issues related to radioactivity, uranium and thorium presence signifies an additional technological challenge, which means higher investment and operating costs, e.g., for securing waste water in the mine, tailings management, separation of uranium and thorium, etc. In some cases, radioactivity issue can cause some problems in relationship with indigenous, i.e., local communities. For example, there were numerous protests organised by local communities of Malaysia and supported by the Australian community against the REE processing plant of Lynas, REE company based in Australia (BBC News Business, 2012; Stop Lynas, 2014). In this case, it is important to invest early in education and informing the locals about acceptable levels of radioactivity, etc., to ensure social acceptance. All these issues have also to be taken into account in making decisions about producing uranium and thorium as by-products.

- Financing opportunities

As at the first stage of the REE project, financing opportunities need to be estimated (Scott and Whateley, 2006). At this stage, junior exploration companies often go public to get an opportunity to finance their exploration activities via the stock market.

- Strategic relevance

The company estimates the strategic relevance of REE for customers in a certain country and accordingly makes a decision to get listed. *This is an REE-specific criterion.*

Stage 2: Exploration drilling

At this stage, more drilling is performed to make a well-grounded resource estimate. When there is more information on the ore body, its shape and location, a suitable mining method (open pit vs. underground mining) and mining costs are estimated (e.g., Evans and Moon, 2006). More extensive mineralogical and metallurgical tests are conducted to identify potential technological options and make an estimate of processing costs.

Gate 3: Prefeasibility study (PFS)

The results of the activities at Stage 2 are assessed in a PFS. The aim is to evaluate operating scenarios preferred in the scoping study and select just one case (rarely two) out of them (e.g., Logan et al., 2006; Noort and Adams, 2006).

Criteria:

- REE grade, tonnage, distribution, mineralogy and metallurgy

The purpose of Stage 2 is to obtain more information with higher accuracy, precision and confidence about geology of the deposit (i.e., REE grade, tonnages and REE distribution), *mineralogy as well as technological feasibility* based on the laboratory results *(REE-specific criteria).*

- Initial mining and processing costs (compared with competitors), expected price and demand

On the basis of the previously described information, initial mining and processing costs can be estimated, evaluated and compared with competitors. For this purpose, industry cost curve logic is very helpful (Bernstein Research, 2012). Already at this stage it is important to know which place and share the project would potentially have on the market to evaluate chances for the project to survive price falls. Thus, *REE expected price and demand (REE-specific criterion)* are additionally analysed.

- Expertise, technology and equipment availability

Technologies and expertise for REE processing in economically and ecologically acceptable ways are hardly available in the ROW. *Thus, this criterion is especially critical for the case of REE.*

- Financing opportunities and strategic relevance

At this project stage, financing is especially important as the following activities, i.e., pilot plant, preparation of a definitive feasibility study, etc., require a significant amount of investment (e.g., Scott and Whateley, 2006). As REEs are subject to strategic considerations, apart from financial investors there are also REE consumers, who can invest in REE projects to secure their supply of required REE. An example from the industry is a joint venture founded in 2012 of Matamec Explorations and Toyota Tsusho Corp. with Toyota owning 49% of shares to secure access to HREE (Matamec Explorations Press Release, 2012). Thus, *strategic relevance of REE (which is REE-specific criterion)* and, therefore, of the project needs to be evaluated together with financing opportunities.

Stage 3: *Mine pre-development stage*

This stage is especially important as multiple aspects have to be clarified. For this purpose, more drilling is performed, if necessary. Consequently, annual production capacity and production rate is decided on (e.g., Scott and Whateley, 2006). At this stage, processing methods are tested in a pilot plant and the process flow sheet is finalised for chosen production capacity. On the basis of this, capital (CAPEX) and operating (OPEX) expenditures are estimated with higher confidence and precision, sufficient for building a financing strategy. Finding financing of further activities is the most critical aspect of Stage 3. Without sufficient financing, the project cannot proceed to the most capital

intensive stage – mine development (e.g., Rudenno, 2009). For this purpose, a bankable feasibility study is usually prepared, based on which banks can make a decision about financing the project. Market studies are conducted to estimate product price development. Having defined costs, production rate and product price, REE reserves are estimated and economic feasibility of (the) future mine is assessed (Laird, 1997). Another important part of this stage is environmental assessment to evaluate potential ecological impact of the future mine and develop measures on how to avoid or reduce it. Additionally, on-site infrastructure and logistics are analysed and planned and socio-economic factors are assessed (Laird, 1997). Moreover, it is important to ensure property security of tenure. Another critical issue is mine permitting procedure, which is initialised at this stage and finalised during the next stage.

Gate 4: Feasibility study (FS)

The aspects described before are contained in the definitive FS (e.g., Scott and Whateley, 2006). The aim is to refine the optimal case selected in the PFS to make a decision about proceeding to the mine design and construction (e.g., Logan et al., 2006; Noort and Adams, 2006). Final technological, economic and ecological assessment is covered by the definitive FS.

Criteria:

- Expertise, technology and equipment availability

Positive results in a pilot plant ensure the technological viability of the project. Additionally, access to required expertise and equipment has to be evaluated within the project. *The criterion is REE specific.*

- Relationships with separation facilities

The market for REE mixed oxides might emerge in the ROW. In this case, establishing strong relationships with separation facilities might be relevant, as well as in the form of a technological partnership, depending on the project production strategy. For example, Frontier Rare Earths and Korea Resources Corp. formed a joint venture with the purpose of building a separation plant in South Africa where the REE deposit Zandkopsdrift is located (Humphries, 2013). *Strategic relationship with separation plants and technological partnerships are a criterion, which is especially critical in the case of REE.*

- Estimated OPEX and CAPEX (compared with competitors), expected price and demand

Similar to the previous stage, estimated CAPEX and OPEX of processing should be compared with the costs of competitors. *REE expected price and demand (REE-specific criterion)* should be monitored.

- Financing opportunities from consumers or markets and relationships with end-users: strategic alliances/off-take

As REEs are not traded on the open market, it is important for the project to establish relationships with end-users in the form of off-take agreements. So, for example, in 2011 Lynas Corporation signed a long-term off-take agreement with BASF for delivery of lanthanum (BASF, 2011). Moreover, signed off-take agreements serve as securing future earnings and, therefore, help REE projects to obtain bank loans.

Additionally, as discussed before, project management may find possibilities of finding strategic joint ventures with potential customers so that the project is financed by the partner. *Relationship with REE end-users is an REE case-specific criterion.*

- Issues on permitting, property security of tenure, indigenous issues/compensations, infrastructure and logistics

These issues have to be clarified as in most cases they delay the project development so that contract obligations cannot be fulfilled in time. Obtaining a mining permit might become an issue for an REE project owing to radioactivity issues. Project infrastructure and logistics have to be evaluated to know if additional investment is required. Also, logistics planning needs to be done.

Stage 4: Mine development

After all the issues described before have been clarified and financing found, it comes to the last project stage, i.e., mine development. At this stage, engineering, procurement and construction manager and team are selected to start mine construction. The activities at this stage aim at implementing the optimal case studied in definitive FS.

Gate 5: Start mining operations

Once the mine has been constructed, mining operations can be started and production capacities achieved.

Stage 5: Mining operations

At this stage, necessary adjustments are made to optimise the project value (e.g., Noort and Adams, 2006). When the production on a large scale has been started, and products have been brought to the market, the project management needs to track market developments to eventually plan an expansion. In this case, more REE off-take agreements need to be signed.

4 Analysing decision-making criteria

First, having characterised the criteria we identified several tendencies (see Figure 3). Generally, at the first two stages and gates of the project development (Stage 0, Gate 1 and Stage 1, Gate 2), one looks at criteria that influence the viability of the project. The criteria are judged qualitatively, as there is no quantitative information available, or it has low level of confidence. At Stage 0, the criteria mostly have external origin, i.e., one looks at the project surroundings. In turn, at Stage 1, criteria with internal origin prevail because on this stage the first geological information becomes available and external criteria had been checked on the previous stage. On both stages, the criteria are mostly non-modifiable as on these stages one primarily has to do with what is given by nature, e.g., mineralisation, and with external factors, such as legislation or tax regime, which a project is not in power to change.

At Stage 2, the majority of criteria influence project economics. After conducting drilling and tests, there is more quantitative information available with higher level of confidence. The criterion 'mineralogy' changes its character. At Stage 1, it was described as influencing project viability, while at Stage 2 it becomes project economics related. At Stage 1, one could just qualitatively evaluate this criterion (e.g., complex or not

complex mineralogy). And at Stage 2, having more information, one is able to quantitatively estimate its influence on costs. Similarly to Stage 1, at Stage 2 the internal origin of criteria is prevailing because on this stage mostly project-related geological or technological parameters are analysed. As on the previous two stages, the criteria here are mostly non-modifiable for the same reason.

At Stage 3, the criteria mainly influence the viability of the project. This is because on this stage such critical issues have to be dealt with as mining permit and establishing strategic relationships. On this stage, the criteria also mainly have internal origin, i.e., project related. External factors are also checked and change only slowly. As discussed before, one can see that some previously external criteria are internal on this stage as they become a part of the project (e.g., infrastructure and logistics). Differing from all previous stages, the criteria here are mostly modifiable by the project management. This is due to the content of the stage, which is mainly about organisational issues and also because non-modifiable criteria have been checked in the prior stages.

At Stage 4, one has to track the REE expected price, so the character of the criterion is economic, external and non-modifiable. At the end of this stage, modifiable criteria should be in favour of the project.

4.1 Uncertainty

Generally, the main sources of uncertainty for the project economics and viability are the REE market, availability of processing technology and geology. Market represents uncertainty for the project economics, whereas technology and geology exhibit uncertainty for the viability of the project. The criteria that represent uncertainty are obviously non-modifiable; they can have internal (geological and technological uncertainty) or external (market uncertainty) origins.

Geological and technological uncertainty decreases with the project development, as more information with higher confidence becomes available. However, market uncertainty is always present. It is highly relevant to consider uncertainty factors in the economic evaluation as a part of feasibility studies. A method that allows accounting for uncertainty in economic evaluation is the real option method (for numerical examples, see, e.g., Dehghani and Ataee-pour (2013)).

4.2 Grouping criteria

We analysed the obtained groups of criteria horizontally to find tendencies throughout the project development. While analysing the groups of criteria, we differentiated between the groups that are present on a particular project stage and that are in main focus on this stage (see the lower part of Figure 3).

4.2.1 Geological

These criteria are analysed on the first three project stages (Stages 0–2), i.e., from planning to exploration drilling. However, they play an important role especially at Stages 1 and 2. On these stages, the geological information is obtained, without which technological and economic feasibility of the project cannot be estimated.

4.2.2 Technological

These criteria are present at Stages 1–4, being especially critical at Stages 2 and 3. On these stages, technological options are studied more in detail and further tested, first in a laboratory and then in a pilot plant. The results of these tests indicate the technological feasibility of the project, which is prerequisite to the economic evaluation of the project.

4.2.3 Project economics

The criteria related to project economics are evaluated at Stages 0, 2 and 3. They are also important on all these stages. At Stage 0, there are some costs connected to information collection, etc. As well as this, the initial exploration costs are estimated based on studied data. At Stages 2 and 3, estimates of mining and processing costs are done and compared with the competitor projects.

4.2.4 Market and financing

When grouping the decision-making criteria, we realised that some of them need to be permanently monitored no matter the project stage (see Figure 3, marked with '…'). Opportunities for project financing should be continuously tracked. Also, availability of financing of the activities at the following stage should be analysed. Additionally, it is important to monitor the REE expected price and demand at every project stage and, based on the changes, update forecasts and re-estimate own position on the market.

4.2.5 Stakeholder-related criteria

Similarly, the stakeholder-related criteria are evaluated at every project stage, in case some conditions have changed. For instance, the legislation related to radioactivity or taxes can change over time, which might positively or negatively influence the project. However, being present at every stage, at Stages 0 and 3 these criteria are more extensively analysed. At Stage 0, this is due to the preliminary evaluation of potential project environment and risks connected to it. At Stage 3, it is connected to passing certain official procedures, e.g., obtaining mining permit and social licence.

4.2.6 Other criteria

Finally, other criteria, among them infrastructure and logistics, are analysed at Stages 0 and 3. At Stage 0, accessibility of a potential deposit is evaluated in addition to the infrastructure of the region.

5 Discussion

In the previous section, we clustered and grouped the decision criteria with the purpose of finding tendencies. In this section, we will discuss the applicability of the results to the REE project management.

The obtained decision-making criteria can be incorporated into the stage-gate approach to the REE project management, for example, to internally evaluate the project at different stages and sort out different options time efficiently.

The type of decision, whether to proceed with the project, wait, revise or abandon it, depends on the characteristics of the criteria. For example, if a viability non-modifiable (V/NM) criterion such as accessibility of a potential deposit is unfavourable, it would be logical to abandon the project (or option) as it is unsure that the situation changes. On the other hand, if a non-modifiable criterion, which influences project economics such as REE expected prices (Ec/NM), is estimated to be unfavourable, a decision to wait or revise can be made as there is a probability that the situation will improve.

Additionally, to better manage the project, the management needs to be aware of the important issues, at each stage of the project development. Therefore, the better you know the decision-making criteria, the better you can manage them. It is critical to know those issues that influence the viability of the project and to take care of those that are modifiable. The viability criteria that are not modifiable should be perceived as knock-out factors. Even if one of such criteria is unfavourable, the project cannot be realised. The economic criteria that cannot be modified should be analysed in detail and be monitored as they can become viability factors if changing extremely. This especially holds for REE expected price and demand, REE resources and mineralogy. If, e.g., REE price is expected to fall markedly, it can render the project uneconomical.

Regarding the criteria that influence the economics of the project, which are modifiable, the management should keep in mind that they can turn into viability criteria if not treated properly or in time. For example, if potential problems with infrastructure and logistics are neglected at the planning stage, it can become too expensive or even legally not possible to solve them later on, which can, in the end, influence the viability of the project. Therefore, these criteria also require special attention.

It is additionally advisable to be aware of criteria of external origin, i.e., project surroundings, which represent external risks. The external factors can speed up or hinder the development of the project. Also, it is important to know those non-modifiable criteria that represent uncertainty for the project economics and viability. Uncertainty can generally be managed, i.e., reduced through acquiring more information on the issue (i.e., exploration, tests, etc.). However, market uncertainty is permanently present.

To summarise the suggestions to the REE project management:

1 know the V, NM criteria of your project, as they are the knock-out criteria, i.e., the most critical criteria for the project at a particular stage

2 know early the viability criteria, which you can modify (V, M) and work on their improvement

3 know the criteria that influence project economics, which you cannot modify (Ec, NM) and keep an eye on them for changes.

4 know modifiable criteria that influence economics of your project (Ec, M) and work on their improvement as otherwise they can turn into viability ones

5 evaluate criteria of external origin (E) early and monitor them throughout the project development

6 be aware of criteria representing uncertainty for the project economics and viability (Ec/U, V/U). And, consider uncertainty in your decision-making.

6 Conclusion

As the first of its kind, this paper defines and assigns decision-making criteria to each of the stages/gates of an REE project development process. Using existing frameworks for exploration project development and the stage-gate approach, the paper also highlights those criteria that only or mostly hold specifically for an REE project.

An important conclusion of this paper is that the REE project development process needs more attention than the development of a conventional exploration project owing to the geological, technological, environmental, market and other challenges. The paper shows that REE project development process and decision-making criteria are partly different from conventional exploration projects. More specifically, such criteria as long-term REE expected price and demand in a strategically influenced market, REE basket, availability of processing technology for given mineralogy, uranium and thorium contents in combination with the environmental legislation and corporate policy, strategic relevance, strategic relationships with separation facilities and REE consumers are specific to the REE project development process.

Having characterised and grouped the criteria, we analysed the tendencies.

The character of criteria is different at different project stages. Also, different sets of groups of criteria prevail at various stages.

Obtained decision-making criteria can be incorporated into the stage-gate system as an approach to the REE project management and internal evaluation. Especially critical are those criteria that influence the viability of the project and cannot be modified. They are the knock-out criteria for the project. Knowing these criteria is essential for the project management.

The most significant limitation of the paper is the small number of interviewees, connected with the fact that the REE industry outside China is small and young at the moment. Additionally, none of the REE projects, which management was interviewed, started mining on a large scale at the moment of interviews. Therefore, it would be relevant for future research to repeat the interviews, once more projects are producing (e.g., in 5–10 years), to challenge the list of obtained criteria and study correlation between the criteria and project success. Additionally, weighing the importance of the criteria would be another aim for further research.

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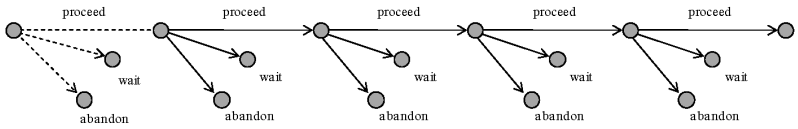
Appendix

Appendix 1 List of interviewees

No.	*Company*	*Name of interviewee*	*Position at the moment of interview*
1	Avalon rare metals	Ian London	Avalon's Energy and Markets Advisor
2	Tasman	Mark Saxon/Henning Holmström/Glenn Patriksson	CEO/Project Development Manager/Senior Geologist
3	Lynas	Eric Noyrez	COO
4	Northern minerals	Robin Wilson/Robert Sills	Exploration Manager/Commercial Manager
5	Alkane	Ian Chalmers	Managing Director
6	GWMG	Jim Engdhal/C.Reid McDonald	CEO/Manager, Operational Research
7	Molycorp	Mark Smith	CEO
8	Commerce resources	Jenna Hardy	Director
9	Montero mining and exploration	Tony Harwood/Michael Spratley	CEO/Mining Engineer
10	Rare element resources	Donald E. Ranta	President, CEO & Director
11	Nuna minerals	Peter van Maastrigt	Chief Geologist
12	*IMCOA*	*Dudley Kingsnorth*	*Executive Director and REE Expert*

Appendix 2 Initial form for the interviews

Milestones and decision options	Region and area selection	Application for permits and licenses, scoping study and preliminary economic evaluation	Pre-feasibility study	Feasibility study, mining rights, permits and licenses, environmental impact studies	Construct facilities, excavate deposit	Large-scale production of ore
						
Stages	**Planning**	**Reconnaissance exploration**	**Exploration drilling**	**Mine Pre – Development**	**Mine Development**	**Start Mining**
Decision-making criteria	1. Relevant data available 2. Intuition (Gut feeling) 3. ...	1. Relevant data available 2. REE content Tonnages 3. Current price of REE 4. Expected price of REE 5. Strategic relevance 6. Financing 7. Legal system in the country of deposit 8. …	1. REE content 2. HREE content 3. Tonnages 4. Current price of REE 5. Expected price of REE 6. Strategic relevance 7. Financing 8. Estimated costs 9. Processing methods 10. …	1. Current price of REE 2. Expected price of REE 3. Strategic relevance 4. Financing 5. Legal system in the country of deposit 6. ...	1. Current price of REE 2. Expected price of REE 3. Strategic relevance 4. Financing 5. ...	

Experimental study for the assessment of suitability for vegetation growth on coal mine overburden

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Abstract: Owing to increased production and productivity of opencast coal mines, large amount of waste rock is removed and stacked in the form of waste dumps. Positive utilisation of such waste rock not only saves considerable dumping land but also reduces problem of maintaining stable and environmentally friendly dumps. One of the major utilisation of waste rock is to use it for vegetation. Therefore a systematic investigation to study the fertile characteristics of overburden waste rock for vegetation was conducted. Waste rock samples collected from an opencast coal mine dump in South India were used in the laboratory experiments. Observations showed the suitability of mine wastes for vegetation when supplemented with additives/nutrients – bottom ash, fly ash, lime stone powder and secondary sludge from sewage treatment plants (STPs) were considered for this purpose as additives. Results suggest that the mine-overburden supplemented with sewage waste (atleast 25% by volume) to be suitable for effective vegetation.

Keywords: overburden; restoration; reclamation; additives; fertility of waste rock.

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1 Introduction

Mining of minerals is essential for the growth of any country. But, mining activity results in various environmental aspects like dust pollution, noise pollution and land degradation (Ram Chandar and Singh, 2001a, 2001b). Owing to the increased opencast mines, land degradation has become a major problem (Murthy et al., 2004). Damage owing to mining activity has also reflected on decrease in green cover or water resource or both. The measures commonly employed in the mined out areas are: compensatory afforestation, reclamation and rehabilitation based on mine closure guidelines. The highwall formed from opencast excavation as well as the waste dumps formed on the surface with waste rock are found to be stable (Sastry and Ram Chandar, 2010, 2013). Successful and long-term mine soil reclamation of overburden dumps requires the establishment of stable nutrient cycles from plant growth and microbial processes (Singh et al., 1996; Kavamura and Esposito, 2010; Sheoran et al., 2010) and the selection of appropriate species for vegetation. For reclamation, studies addressing the soil fertility, structure, microbial populations, top soil management and nutrient cycling have to be done to retain the land to be useful and as self-sustaining ecosystem. Fertility of mine waste rock can be increased by adding various natural materials like saw dust, wood residues, sewage sludge and animal manures, as these stimulate the microbial activity, which provides the nutrients (N, P) and organic carbon to the waste material (Sheoran et al., 2010). Compost provides organic matter, decreases bulk density and erosion and increases aggregate stability, aeration, water infiltration and retention (Tester, 1990; Serra-Wittling et al., 1996; Ros et al., 2001). It also increases concentrations and availability of micro-

and macronutrients (Guerrero et al., 2001; Martínez et al., 2003), providing a wide range of nutrients than inorganic fertilisers, with less nitrate leaching and water contamination (Gagnon et al., 1997; Mamo et al., 1999). Biosolids also serve many purposes, including pH control, metal control and fertilisation. Establishment of revegetation on mine overburden dumps renders biologically productive, and contributes towards physical stabilisation, pollution control, visual improvement and removal of threats to surrounding population.

2 Materials and methods

Initially, samples were collected from four different places of an opencast coal mine dump site at different horizons after rainy season using spade and trowel. From each site, 30 cm × 30 cm dimension pits were dug with 30 cm deep using spade and trowel and about 5 kg of sample was scraped from each of the four pits randomly chosen. As the samples are of similar in nature in terms of physico-mechanical properties, they were mixed thoroughly. The following analysis was done for samples collected and also after addition of additives to the sample.

2.1 Texture analysis

Texture (or particle size) analysis in this study was used to determine the relative proportions of the different particle (or grain) sizes, which make up a given soil/material mass. Two techniques were used in this exercise to separate the soil particles into particle size ranges namely clay (<0.002 mm), silt (0.002–0.05 mm) and sand (0.05–2 mm). Coarse particles (sands and gravels) were analysed with mechanical sieves. The distribution of fine particle sizes (silts and clays) is determined by uniformly dispersing the sample in water measuring how quickly the particles fall in the mixture, the sedimentation analysis.

2.1.1 Sieve analysis

1000 gm of oven-dried sample was weighed in a tray and soaked in water. The test was conducted as per IS; 2720(Part IV)-1985. The soaked sample was puddled thoroughly and transferred to 4.75 mm sieve and passing through it was collected separately and oven-dried. The dried material was sieved through 2.36 mm, 1.18 mm, 600 µm, 300 µm, 150 µm and 75 µm size sieves, respectively. A minimum of 10-minute sieving was done and the fraction retained on each sieve was collected in a separate container and the mass was determined. The percentage of fraction retained on each sieve was calculated on the basis of total mass of sample taken (1000 gm).

2.1.2 Sedimentation analysis using hydrometer

The particle size distribution was determined using a density hydrometer.

Dispersion of soil: 50 gm of oven-dried sample is taken and the soil solution is prepared using distilled water with dispersing agent (sodium silicate) of 4 drops. The soils solution is shaken vigorously in mechanical shaker for few minutes.

Sedimentation test with hydrometer:

- The well-shaken solution (prepared up to 1 l) was taken in to a measuring cylinder. The hydrometer is gently immersed to a depth below its floating point and then allowed it to float freely. Using stop watch, the hydrometer readings are noted down after periods of 15 s, 30 s, 1 min, 2 min, 4 min and 8 min, respectively. After the 8th minute reading, the hydrometer was taken out and rinsed in distilled water and allowed to stand in a jar containing distilled water.
- The hydrometer is re-inserted in the suspension and the readings are noted down after periods 15th min, 30th min, 1 h, 2 h, 4 h, 8 h and 24 h, respectively.
- Composite correction (C) was determined by taking the readings of top meniscus of the hydrometer immersed in the dispersing agent (sodium silicate, 0.5 mL) along with distilled water in another 1000 mL measuring cylinder at the same temperature as that of test solution. Since the temperature 27°C was constant during the experiment, it was found the composite correction (C) to be equal to meniscus correction (Cm), i.e., (C = Cm = 0.5) found during the experiment.
- The specific gravity (G) of the dried soil sample obtained from passing the 75 μm IS sieve is determined by density bottle using expression:

$$= \frac{(M2 - M1)}{(9M2 - M1) - (M3 - M4)}$$

where $M1$ is the mass of density bottle, $M2$ is the mass of bottle and dry soil, $M3$ is the mass of bottle with soil and water and $M4$ is the mass of bottle with water only.

- Factor F is calculated using expression:

$$F = \frac{\sqrt{3000}\eta}{((G-1)\gamma - w)}$$

is a constant factor for given values of η (viscosity of the liquid/water) and G. Here, the viscosity η of the water at 27°C is 0.00855×10^{-4} KN-s/m^2.

- The diameter of the particles in suspension at sampling time 't' is found from the following expression:

$$D = 10^{-5} F \frac{\sqrt{He}}{t},$$

where He is the effective depth.

- The percentage of finer N is found based on the mass (Md) of the dry sample taken passing through 75 μm IS sieve (here, sample taken for test is 50 gm), using expression

$$= R \frac{100\,G}{(Md(G-1))}.$$

- The percentage of finer N based on total mass (m) of dry soil sample taken (here, 1000 gm) is found from the relation, where M' is the cumulative mass passing 75 µm IS sieve.

2.2 Soil pH and electrical conductivity (EC)

pH is the measure of the intensity of the alkalinity or acidity of the soil suspension. The electrical conductivity of the sample is the measure of the soluble salts present. EC of the samples was measured using conductivity meter according to BIS: 2720 part XIX (1977). The soil suspension of soil to water ratio 1 : 2.5 was prepared and was used as sample.

2.3 Determination of moisture content by oven drying method

Empty weight of container ($M1$) and 500 gm of moist sample along with the container was found as ($M2$). Container with moist sample was kept in the hot air oven and the temperature of the oven was maintained between 105°C and 110°C for about 16 h. The container was taken out and cooled in a desiccator and the mass ($M3$) of dried sample was found. Moisture content was calculated as follows:

$$W = \frac{M2 - M3}{M3 - M1} \times 100 \text{ (percent)}.$$

2.4 Soil organic carbon (SOC) by modified Walkley-Black method

Soil organic matter increases water holding capacity, soil porosity, infiltration and cation exchange capacity (CEC) (Sheoran et al., 2010). Mining and reclamation activities cause drastic loss of the antecedent SOC and nitrogen (N) in mined and newly reclaimed mine soils (Akala and Lal, 2001).

Procedure:

- Suitable quantity (2 gms) of sample is weighed and transferred into a 500 mL conical flask.
- 10 mL 1 N potassium dichromate and 20 mL of concentrated sulphuric acid were added slowly and was shaken vigorously after adding. The mixture was kept for 30 min idle.
- 200 mL of distilled water and 10 mL phosphoric acid were added to the conical flask (Figure 1).
- Ferrous ammonium sulphate (FAS) (0.4 N) was taken in a burette and filled it up to the mark.
- Three drops of diphenylamine indicator was added to the conical flask, the solution changed to blue and titrated immediately against FAS till the end point is green in colour. The readings were noted down as mL of FAS consumed against sample solution (T).

- A blank was prepared using the same procedure as mentioned earlier without soil sample in the conical flask and titrated against 0.4 N FAS till the end point is green in colour. The reading was noted down as mL of FAS consumed against blank solution (S).

Calculation: Percentage organic carbon is calculated using the following expression:

$$\%c = \frac{3.951}{g}\left(1 - \frac{T}{s}\right)$$

where

T: Volume of FAS consumed against sample extract solution

S: Volume of FAS consumed against blank solution

g: Weight of the soil sample taken (2 gms).

Percentage of organic matter is calculated using the following expression:

$$\%\text{OM} = \%c \times 1.724.$$

Figure 1 Samples ready to be analysed for % organic carbon (see online version for colours)

2.5 *Cation exchange capacity (IS:2720(Part XXIV)-1976)*

Cation exchange is the physico-chemical process whereby one type of ions (cation) adsorbed on soil particles is replaced by another type. The CEC signifies the capacity of soil to retain cations up to its highest limit or it can also be defined as the soil to combine with cation in such a manner that they cannot be easily removed by leaching with water, but can be exchanged by an equivalent amount of other cations. It is the sum of exchangeable metallic ions plus hydrogen ions in the given soil.

Procedure:

- 100 mL of 0.05 N hydrochloric acid is pipetted into a test-tube (175×32 mm^2), a weighed amount of soil (1–5 gm) was added, stoppered and stirred well and allowed to stand overnight

- filtered through a dry 11-cm Whatman No. 30 or equivalent filter paper, collecting the filtrate in a dry 125-mL Erlenmeyer flask, rejecting first portion
- 25-mL aliquot is titrated against standard lime water using bromothymol blue as indicator. As the end-point is approached, 2 or 3 more drops of indicator is added to overcome absorption by any sesquioxide precipitate
- a blank titrated similarly.

Calculation: The approximate value for total exchangeable metallic ions in milliequivalents percent is given by the following expression:

$$\text{Meq} = (B - T) \times N \times \frac{100}{25} \times \frac{100}{W}$$

where,

meq: Total exchangeable metallic ions

B: Blank titration in mL of lime water of known normality

T: Actual titration in mL of lime water of known normality

N: Normality of the lime water

W: Mass of soil taken.

2.6 Heavy metal contamination

Metal concentrations in soil range from less than 1 mg/kg to high as 100,000 mg/kg, whether owing to the geological origin of the soil or as a result of human activity (Blaylock and Huang, 2000). Excess concentrations of some heavy metals in soils such as Cd+2, Cr+6, Cu+2, Ni+2 and Zn+2 have caused the disruption of natural aquatic and terrestrial ecosystems (Gardea-Torresdey et al., 1996; Meagher, 2000). Cadmium and lead, which have no known beneficial effects, may become toxic to plants and animals if their concentrations exceed certain values (Adriano, 1986; Gough et al., 1979). Scanning electron microscope (SEM) was used for energy-dispersive X-ray spectroscopy (EDS, EDX or XEDS), which is an analytical technique used for the elemental analysis or chemical characterisation of a sample. It relies on the investigation of an interaction of some source of X-ray excitation and a sample.

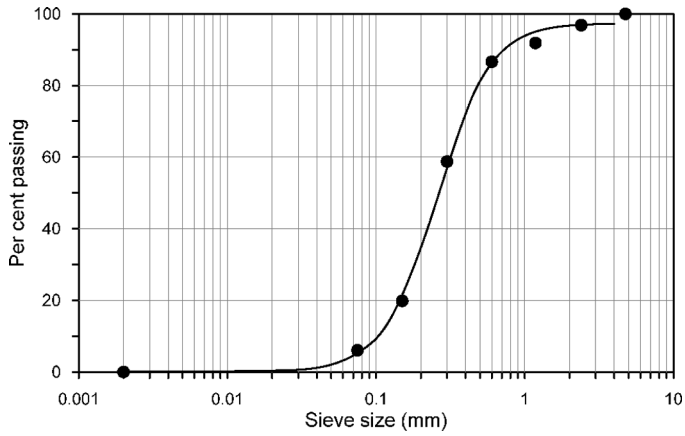
3 Results and discussion

3.1 Texture analysis

After sieve analysis and sedimentation by hydrometer method, the particle size distribution was determined and then the particle or grain size distributing curve was plotted (Figure 2). From the graph and from hydrometer results, the texture of the sample is found to be poorly graded loamy and silty. The samples collected are loamy sand to sandy loam to sand in nature as per soil classification, where the sand fraction was found to increase but has lower silt and clay fractions. Generally, this kind of sample is not

suitable for vegetation. So, some additives are added in 25% and 50% by volume to the sample and structural and textural analysis was done and observed that there is considerable increase in clay content with addition of fly ash and sewage from STP.

Figure 2 Textural analysis of soil sample



Additives used are:

- sludge from secondary clarifier from STP
- bottom ash collected from project site area, which is being used for stowing
- fly ash from thermal power plant
- lime stone powder.

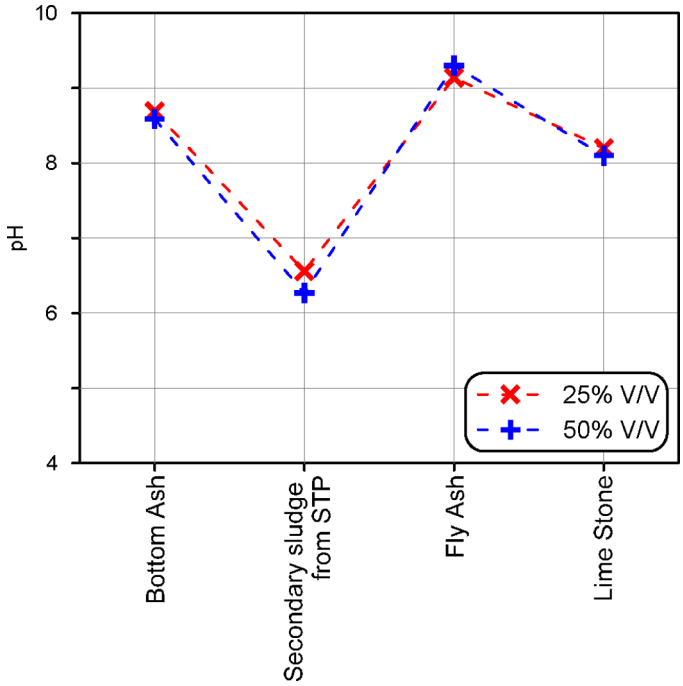
3.2 Determination of pH

pH influences the soil structure formation, root growth and nutrient availability for the vegetation on the mine soil. Low pH or acidic soils have several adverse effects including Al and Mn toxicity and nutrient deficiencies. pH was determined after adding the additives and details are given in Table 1. Initially, the pH of sample was 7.72 but on addition of additives it varied from 6.27 to 9.14. A large variation was seen on addition of fly ash in 50% v/v combination as shown in Figure 3, whereas pH was decreased on addition of STP sludge in 25%v/v and 50% v/v combination.

Table 1 pH values after adding additives

Additives/Mix %	*25% V/V*	*50% V/V*
Bottom ash	8.69	8.59
Secondary sludge from STP	6.56	6.27
Fly ash	9.14	9.30
Lime stone	8.20	8.10

Figure 3 Variation of pH values after adding additives (see online version for colours)



3.3 *Electrical conductivity (in ms)*

Initial electrical conductivity of the soil sample was 0.36 ms. As shown in Table 2, on addition of additives, the EC of mixture varied from 0.28 ms to 2.8 ms. Bottom ash additive mixed with sample showed maximum conductivity in 50%v/v combination. On addition of lime stone powder, the EC decreased from 0.36 ms to 0.28 ms. Figure 4 shows the decrease in EC values from bottom ash to fly ash. Maximum value is obtained for bottom ash whereas minimum for limestone powder.

3.4 *Bulk density (in g/cm^3)*

Initial bulk density of the sample was 1.76 g/cm^3. As shown in Table 3, on addition of additives, bulk density was decreased. On addition of bottom ash to sample, the bulk density decreased to 1.19 g/cm^3, which was the lowest among other additives. Figure 5 indicates that the lesser the volume added, the higher the bulk density value except in the case of lime stone powder, which shows a little increase in bulk density value.

Table 2 EC values after adding different additives

Additives/Mix %	*25% V/V*	*50% V/V*
Bottom ash	2.30	2.80
Secondary sludge from STP	2.03	2.18
Fly ash	1.07	1.57
Lime stone	0.34	0.28

Figure 4 Variation of electrical conductivity (EC) with different additives (see online version for colours)

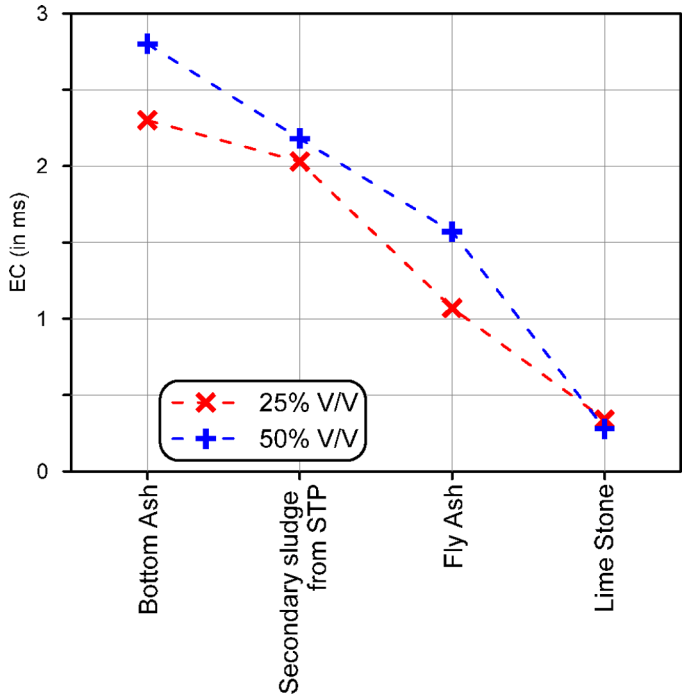


Figure 5 Variation in bulk density with additives (see online version for colours)

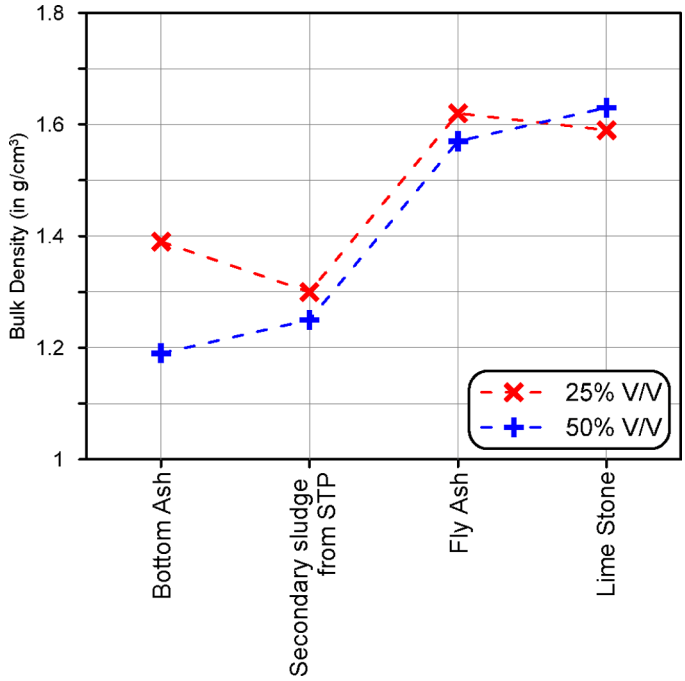


Table 3 Bulk density values after adding additives

Additives/Mix %	*25% V/V*	*50% V/V*
Bottom ash	1.39	1.19
Secondary sludge from STP	1.30	1.25
Fly ash	1.62	1.57
Lime stone	1.59	1.63

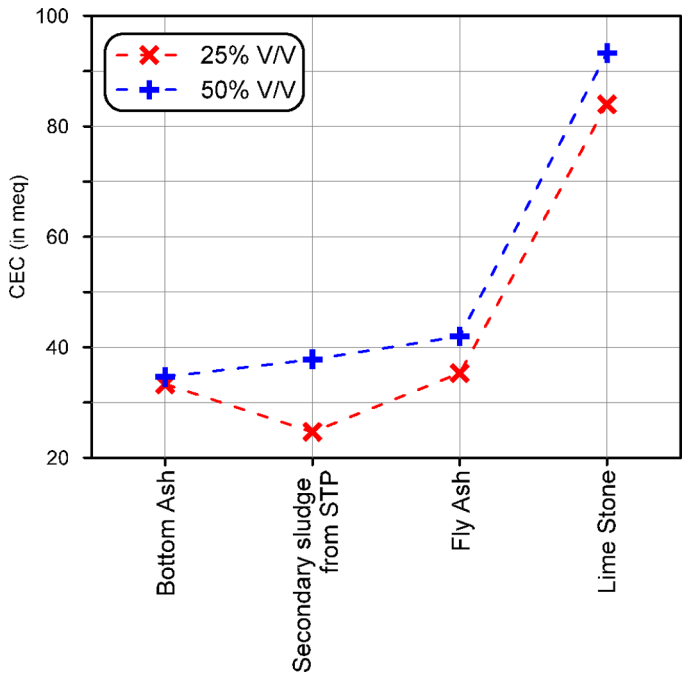
3.5 Cation exchange capacity (in meq) (IS:2720(Part XXIV)-1976)

Initial CEC of the soil sample is 31.99. As shown in Table 4 and Figure 6, it can be clearly observed that on addition of lime stone powder there is a considerable increase in the CEC, whereas it remained almost same on addition of bottom ash and decreased with the addition of STP sludge in 25% v/v.

Table 4 CEC (meq) values after adding additives

Additives/Mix %	*25% V/V*	*50% V/V*
Bottom ash	33.33	34.66
Secondary sludge from STP	24.66	37.8
Fly ash	35.32	41.99
Lime stone	83.99	93.29

Figure 6 Variation of CEC with additives (see online version for colours)



3.6 Soil organic carbon (SOC) (in %) by modified Walkley-Black method

SOC of sample is 0.08%. As shown in Table 5, the SOC varied between 0.30% and 3.58% on addition of additives. Maximum increase was found when STP sludge was added to sample (Figure 7). The higher-value SOC is better for vegetation.

3.7 Soil organic matter (in %)

Soil organic matter of sample is 0.15%. SOM values with different additives are given in Table 6. From Figure 8, it can be found that on addition of STP there is a prominent increase in the % of organic matter, whereas the results of bottom ash, fly ash and lime stone remained below 0.5% SOC. Organic matter depends on the SOC. Relative increase in organic matter can be observed with increase in organic carbon.

Table 5 SOC (%) values after adding additives

Additives/Mix %	*25% V/V*	*50% V/V*
Bottom ash	0.49	0.30
Secondary sludge from STP	2.75	3.58
Fly ash	0.32	0.35
Lime stone	0.43	0.30

Figure 7 Variation of SOC (%) with different additives (see online version for colours)

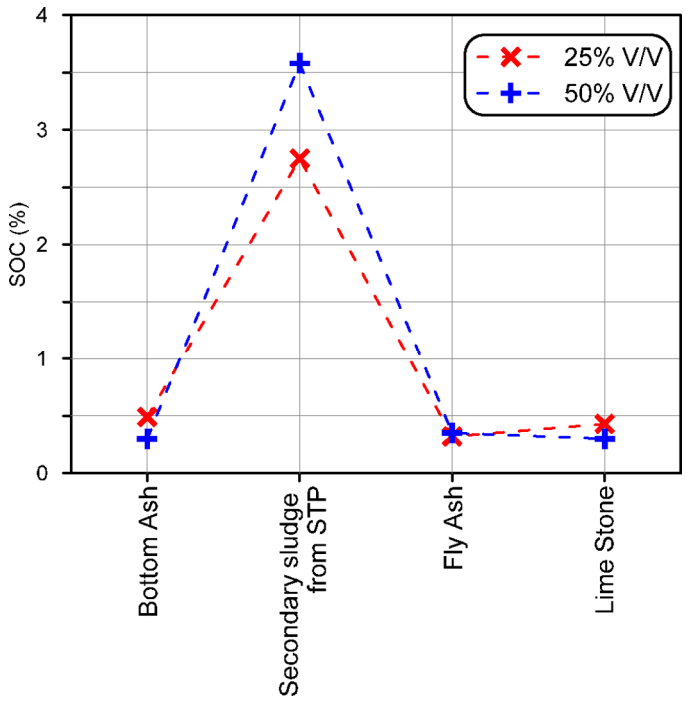
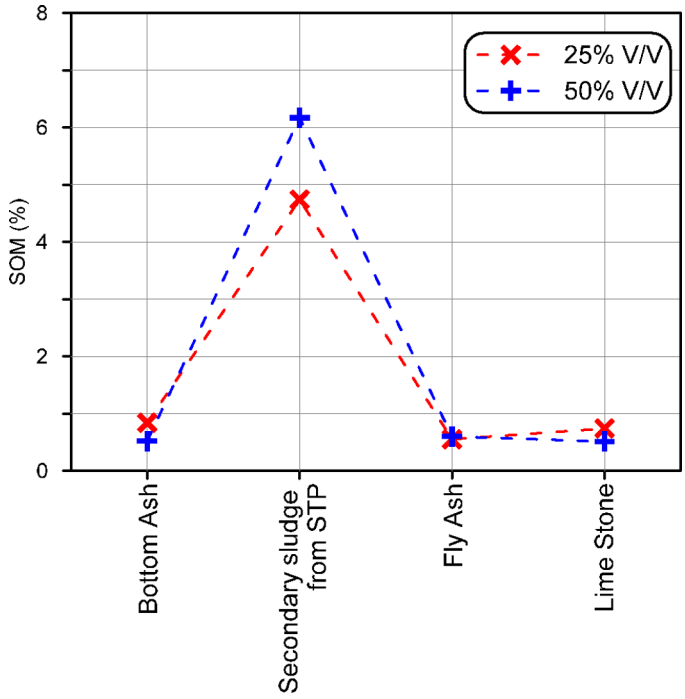


Table 6 SOM (%) values after adding additives

Additives/Mix %	*25% V/V*	*50% V/V*
Bottom ash	0.84	0.52
Secondary sludge from STP	4.74	6.17
Fly ash	0.55	0.60
Lime stone	0.74	0.51

Figure 8 Variation of organic matter with different additives (see online version for colours)



3.8 Elemental analysis

A SEM was used to determine the elements present in the samples by using electron dispersive X-ray spectroscopy (EDX) technique. The main purpose of this analysis is to find the existence of heavy metal contamination in the sample. A typical output is shown in Figure 9. Compositions of different elements are shown in Figure 10.

It was found that there is no heavy metal contamination. The same procedure was carried to find whether any additive is contaminated and found that no additive is contaminated with heavy metals.

Figure 9 Microscopic view of the sample (see online version for colours)

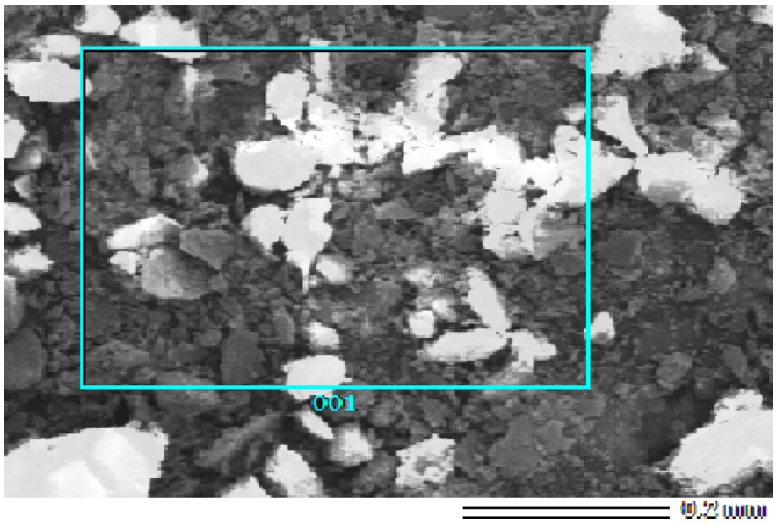
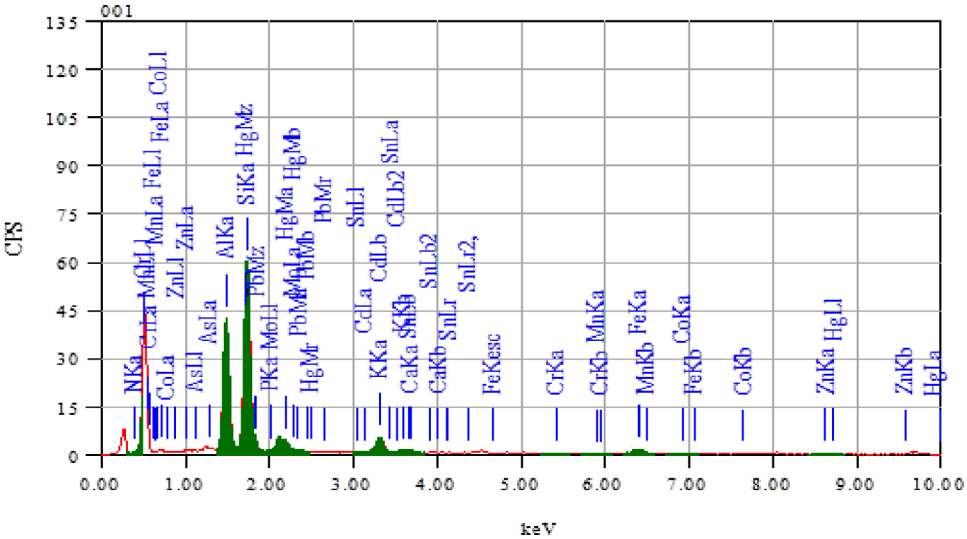


Figure 10 Composition of different elements of sample (see online version for colours)



4 Conclusions

On the basis of the experimental studies, the following conclusions are drawn:

- The major part of the samples collected are sandy loam to sandy in nature where the sand fraction was found to increase but has lower silt and clay fractions.
- The pH of samples ranged from 7.71 to 8.06 (neutral to moderately alkaline). On addition of additives, pH varied from 6.27 to 9.14, which was initially 7.72.

- From the experiments, it is clear that the soil is free from heavy metal contamination and it is almost neutral soil and no liming is required to increase the pH.
- Electric conductivity (EC) varied from 0.28 ms to 2.8 ms. EC mostly increased on addition of additives.
- Bulk density decreased from 1.72 to a range of 1.19–1.63 on addition of additives.
- The SOC varied between 0.30% and 3.58% on addition of additives; initially, it was found to be 0.08%. SOM also increased on addition of different additives.
- With these results, the best suited additive is sewage sludge from STP as it is the only additive that is decreasing the pH. The favourable pH for the growth of plant species is 5.5–7.0. The clay percentage also increased considerably and shown better results than fly ash and lime stone powder in EC. The bulk density was decreased to 1.30 which is favourable condition for plant growth. It also contains organic carbon and organic matter which plays a major role in the plant growth. So can be concluded that the mine over burden can be used for effective vegetation by adding sewage waste by at least 25% by volume.

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Mathematical programming applications in block-caving scheduling: a review of models and algorithms

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Abstract: Production scheduling is one of the most important problems in a mining operation, as it has a significant impact on a project's profitability. As open-pit mines go deeper, because of the massive amount of waste removal, which is required to extract the ore, as well as high operational costs per tonne, underground mining has become more attractive. Among underground mining methods, block caving could be a good alternative because its high rate of production is similar to that of open-pit mining, and it has the added advantage of low operational costs. Relying only on manual planning methods or computer software based on heuristic algorithms will lead to mine schedules that are not the optimal global solution. Block-caving scheduling has been the subject of a lot of research. Most studies have applied mathematical programming, simulation and stochastic approaches. This paper reviews mathematical programming applications in block-caving scheduling, highlights and suggestions for future works.

Keywords: block caving; mathematical programming; production scheduling.

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1 Introduction

These days, most surface mines work in a higher stripping ratio than in the past. In the following conditions, a surface mine can be less attractive to operate and underground mining is used instead. These conditions are

- too much waste has to be removed to access the ore (high stripping ratio)
- waste storage space is limited
- pit walls fail or
- environmental considerations could be more important than exploitation profits (Newman et al., 2010).

Among underground methods, block-cave mining, because of its high production rate and low operation cost, could be considered an appropriate alternative. Projections show that 25% of global copper production will come from underground mines by 2020. Mining companies are looking for an underground method with a high rate of production, similar to that of open-pit mining. Therefore, there is an increased interest in using block-cave mining to access deep and low-grade ore-bodies.

Production scheduling is one of the most important steps in the block-caving design process. Optimum production scheduling could add significant value to a mining project. The goal of long-term mine production scheduling is to determine the mining sequence, which optimises the company's strategic objectives while honouring the operational limitations over the mine life. The production schedule defines the management investment strategy. An optimal plan in mining projects will reduce costs, increase equipment use and lead to optimum recovery of marginal ores, steady production rates and consistent product quality (Chanda, 1990; Chanda and Dagdelen, 1995; Dagdelen and Johnson, 1986; Winkler, 1996; Wooller, 1992). Although manual planning methods or computer software based on heuristic algorithms are generally used to generate a good solution in a reasonable time, they cannot guarantee mine schedules that are the optimal global solution.

Mathematical programming with exact solution methods is considered a practical tool to model block-caving production scheduling problems; this tool makes it possible to search for the optimum values while considering all of the constraints involved in the operation. Solving these models with exact solution methodologies results in solutions within known limits of optimality.

In this paper, block caving and production scheduling in cave mining have been introduced, and mathematical programming methods, which can be used as a tool for production scheduling, have been discussed. Finally, the related research in this area is presented, as are some conclusions and new ideas for future studies.

2 Block caving

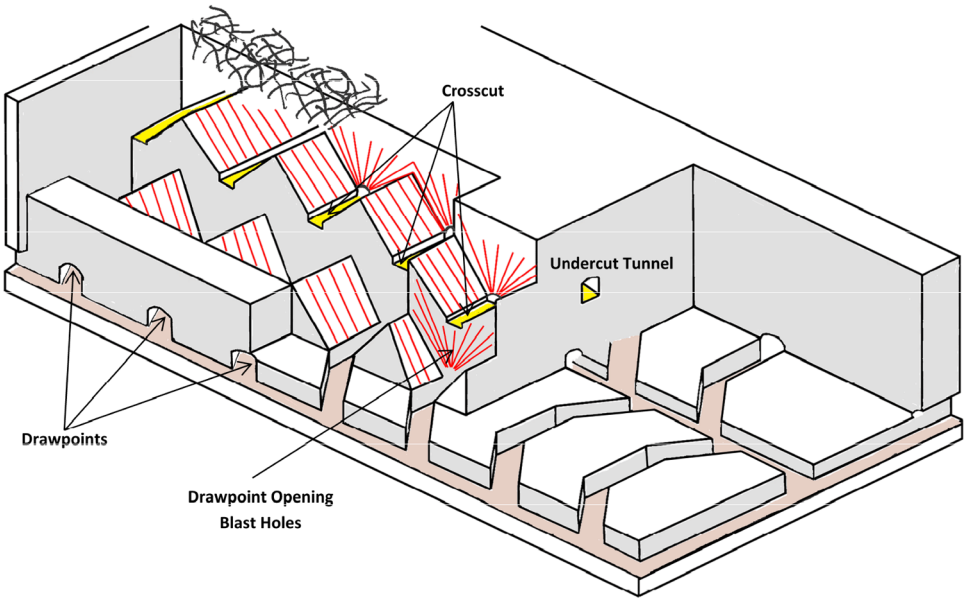
Generally speaking, underground mining methods can be classified in three categories:

- caving methods such as block caving, sublevel caving and longwall mining
- stoping methods such as room-and-pillar, sublevel stoping and shrinkage
- other methods such as postpillar cut-and-fill and Avoca (Carter, 2011).

Block caving is usually appropriate for low-grade and massive ore-bodies in which natural caving could occur after an undercut layer is made under the ore-body. Laubscher (1994) refers block caving "to all mining operations in which the ore-body caves naturally after undercutting its base. The caved material is recovered using drawpoints".

Depending on the ore-body dimensions, inclination and rock characteristics, block caving could be implemented as block caving, panel caving, inclined drawpoint caving and front caving. The low-cost operation could be understood from the natural caving. In other words, during the extraction period, there is no cost for caving unless some small blasting is needed to deal with hang-ups. In block caving (Figure 1), the pre-development period can last for more than 5 years. This is a significant period of time with no cash back. But when the production starts, the extraction network can be used for the life of the drawpoint, so the operating cost is low and production rate can be remarkable. To sum up, block caving has the lowest operating cost of all underground mining methods. In some cases, its cost is comparable with that of open-pit mining.

Figure 1 Block-caving mining (see online version for colours)



There are three methods of block caving. In the grizzly or gravity system, the ore from the drawpoints flows directly to the transfer raises after sizing at the grizzly, and then is gravity-loaded into ore cars. In the slusher system, slusher scrapers are used for the main production unit. In the load-haul-dump (LHD) system, rubber tyred equipment

are used for ore handling in production level (Hustrulid, 2001). Table 1 shows some examples for each method. Caterpillar jointly with the Chilean mining company Codelco has developed a continuous haulage technology for block-caving operation. In this method, the LHD at the drawpoint is replaced by a rock feeder. This device pushes the rock into the haulage access, where it drops onto a hard rock production conveyor.

Table 1 Some real cases for different block-caving methods

Method	*Mine*	*Ore Type*	*Location*
Gravity (Grizzly)	San Manuel	Copper	Arizona
	Andina	Copper	Chile
Slusher	Climax	Molybdenum	Colorado
	Tong Kuang Yu	Copper	China
LHD	Henderson	Molybdenum	Colorado
	Ertsberg	Copper	Indonesia
	El Teniente	Copper	Chile
	New Afton	Copper-Gold	Canada

Source: Bergen et al. (2009), Julin (1992), Song (1989)

The size of the caved material, the mine site location, availability of labour and economics are some aspects that determine the block-caving system (Julin, 1992). Factors that have to be considered in block caving include caveability, fragmentation, draw patterns for different types of ore, drawpoint or drawzone spacing, layout design, undercutting sequence and support design (Laubscher, 2011). Some large-scale open-pit mines will be transferred to underground mining as they go deeper; they need to produce in a similar rate to open-pit mines to provide their processing plants with feed, so block caving with a high production rate could be an attractive alternative. Around the world, more than 60 mines have been closed, are operating or are planned to be mined by block caving (Woo et al., 2009).

Laubscher (2000) identified 10 different horizontal LHD layouts as having been used in block-caving mines. Figures 2 and 3 illustrate two of them. Figure 2 shows offset Herringbone. In this layout, the drawpoints on opposite sides of a production drift are offset. This helps to improve both the stability conditions and the operational efficiency. This layout was used initially at the Henderson Mine, USA, and Bell Mine, Canada. Figure 3 shows the layout developed at the El Teniente Mine, Chile. In this layout, the drawpoint drifts are developed in straight lines oriented at 60° to the production drift (Brown, 2003).

One of the most critical processes in block-cave mining is undercutting. The undercutting strategy can have a significant influence on cave propagation and on the stresses induced in, and the performance of the extraction-level installations (Brown, 2003, 2007). The three mostly used undercutting strategies are post-, pre- and advanced undercutting. In the post-undercutting strategy, undercut drilling and blasting takes place after the production level has been developed. In the pre-undercutting method, no development or construction takes place on the production level before the undercut has been blasted. In the advanced undercutting strategy, the production level is developed in advance of the blasting of the undercut. This method was introduced to reduce the

drawpoints' exposure to the abutment stress zones, which were induced as a result of the undercutting process. The next section presents production scheduling in mining operations, and particularly in block-cave mining.

Figure 2 Typical offset Herringbone layout (after Brown, 2007) (see online version for colours)

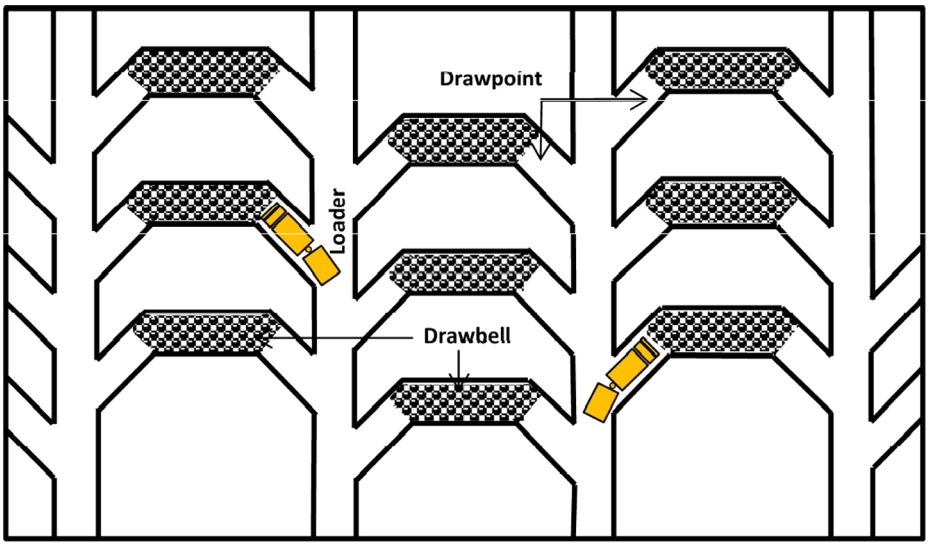
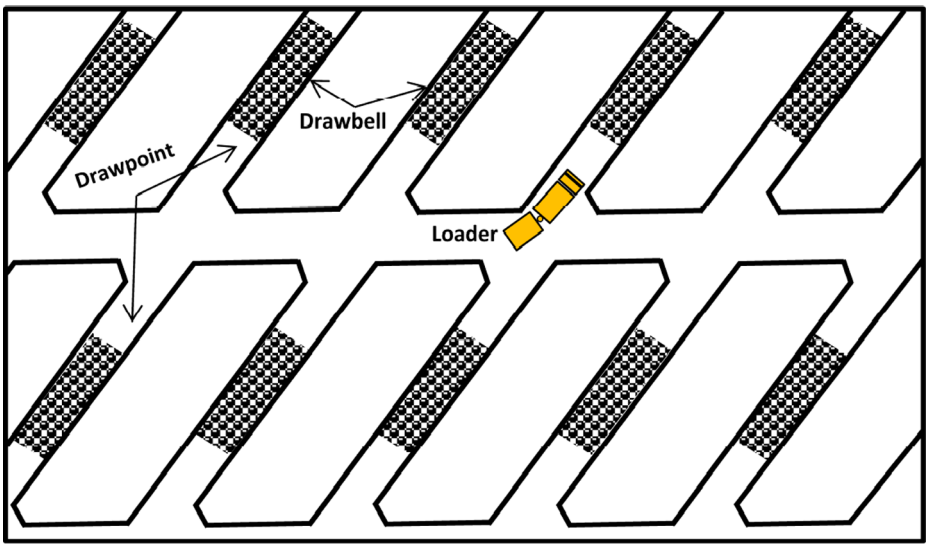


Figure 3 Typical El Teniente layout (after Brown 2007) (see online version for colours)



3 Underground mines production scheduling

The production scheduling in mining operations requires determining which blocks should be extracted and the time of their extraction during the life of mine, while

considering geomechanical, operational, economic and environmental constraints. Production scheduling for any mining system has an enormous effect on the operation's economics. Some of the benefits expected from better production schedules include increased equipment use, optimum recovery of marginal ores, reduced costs, steady production rates and consistent product quality (Chanda, 1990; Chanda and Dagdelen, 1995; Dagdelen and Johnson, 1986; Winkler, 1996; Wooller, 1992).

There are three time horizons for production scheduling: long, medium and short term. Long-term mine-production scheduling provides a strategic plan for mining operations, whereas medium-term scheduling provides a monthly operational scheme for mining while tracking the strategic plan. Medium-term schedules include more detailed information that allows for a more accurate design of ore extraction from a special area of the mine, or information that allows for necessary equipment substitution or the purchase of necessary equipment and machinery. The medium-term schedule is also divided into short-term periods (Osanloo et al., 2008).

The majority of scheduling publications to date have been concerned with open-pit mining applications. As a result, the software development for underground operations has been delayed and many of the scheduling concepts and algorithms developed for surface mining have found their way into underground mining. Underground mining methods are characterised by complex decision combinations, conflicting goals and interaction between production constraints.

Current practice in underground mine scheduling has tended towards using simulation and heuristic software to determine feasible, rather than optimal, schedules. A compromise between schedule quality and problem size has forced the use of mine design and planning models, which incorporate the essential characteristics of the mining system while remaining mathematically tractable. Different types of methods have been applied to underground mine scheduling. Similar to open-pit mines, production scheduling algorithms and formulations in literature can be divided into two main research areas:

- heuristic methods
- exact solution methods for optimisation.

Heuristic methods are generally used to generate a good solution in a reasonable amount of time. These methods are used when there is no known method to find an optimal solution under the given constraints. Despite shortcomings such as frequently required intervention and the lack of a way to prove optimality, simulation and heuristics are able to handle non-linear relationships as part of the scheduling procedure.

In addition to these categories, other methods such as queuing theory, network analysis and dynamic programming have been used to schedule production or material transport. This paper reviews mathematical programming applications in block-caving production scheduling. In block-cave mining, production scheduling determines the amount of material that should be mined from each drawpoint in each period of production, the number of new drawpoints that need to be constructed and their sequence during the life of mine (Pourrahimian, 2013). The same concerns in deep open-pit mining can be applied to block-cave mining; possibility of value changes of the project through scheduling is remarkable.

4 Mathematical programming methods

Mathematical programming (MP) is the use of mathematical models, particularly optimising models, to assist in making decisions. The MP model comprises an objective function that should be maximised or minimised while meeting some constraints that determine the solution space and a set of decision variables whose values are to be determined. Objectives and constraints are functions of the variables and problem data. Mathematically, an MP problem can be stated as,

$$\begin{aligned} &\text{Maximise} && f(X) \\ &\text{Subject to} && g_i(X) \le 0 \quad i = 1, 2, ..., m. \\ & && X \ge 0 \end{aligned}$$

where $X = (x_1, x_2 ... x_n)^T \in R^n, f(X), g_i(X), i = 1, 2, ... m$ are real-valued functions of X. If the functions $f(X)$ and $g_i(X)$ are all linear, the problem is known as a linear programming problem (LP). Otherwise, it is said to be a non-linear programming (NLP) problem (Sinha, 2006).

The modelling process in mathematical programming has eight steps (Eiselt and Sandblom, 2010): problem recognition, authorisation to model, model building and data collection, model solution, model validation, model presentation, implementation and monitoring and control. The mathematical programming models, which are considered for production scheduling, are linear programming (LP), mixed-integer linear programming (MILP), NLP, dynamic programming (DP), multi-criteria optimisation, network optimisation and stochastic programming (Shapiro, 1993). In an LP problem, when some or all of the variables are integers, the problem is called pure integer (IP) and mixed-integer programming (MIP, MILP), respectively. A linearly constrained optimisation problem with a quadratic objective function is called a quadratic program (QP).

The tractability of the mathematical models depends on the size of the problem, in terms of the number of variables and constraints, and the structure of the constraint sets. In the integer programming, as the size of an integer program grows, the time required for solving the problem increases exponentially. The most common problem in the MILP formulation is size of the branch and cut tree. The tree becomes so large that insufficient memory remains to solve the LP sub-problems. The size of the branch and cut tree can actually be affected by the specific approach one takes in performing the branching and by the structure of each problem. So, there is no way to determine the size of the tree before solving the problem (Pourrahimian et al., 2012). For instance, Pourrahimian et al. (2013) presented three MILP formulations at three different levels of resolution:

- aggregated drawpoints level
- drawpoint level
- drawpoint-and-slice level.

Table 2 summarises the number of decision variables and constraints if the proposed formulations are applied on a mine with 3000 drawpoints over 25 years.

Table 2 Number of decision variables and constraints for the models presented by Pourrahimian et al. (2013)

Level of resolution	*Description*	*Number of continuous variables*	*Number of binary variables*	*Number of constraints*
Aggregated DPs	3000 DPs aggregated into 200 clusters	5000	10,000	20,675
DPs	3000 DPs	75,000	150,000	309,075
DP-and-slice	3000 DPs and 60530 slices	1,513,250	1,663,250	640,185

5 Production scheduling optimisation in block-cave mining

Using mathematical programming optimisation with exact solution methods to solve the long-term production planning problem has proved to be robust, and results in answers within known limits of optimality (Pourrahimian, 2013). Lerchs and Grossmann (1965) applied mathematical programming in mine planning (open-pit mining) for the first time. Since the 1960s, considerable research has been done in mine planning using mathematical programming, both in open-pit and in underground mining. Newman et al. (2010) and Osanloo et al. (2008) have mentioned many of the studies related to open-pit mining. Alford (1995) listed problems, which have the potential of being considered optimisation problems in underground mining. These problems are:

- primary development (shaft and decline location)
- selection from alternative mining methods
- mine layout (i.e., sublevel location and spacing, stope envelope)
- production sequencing
- product quality control (material blending)
- mine ventilation
- production scheduling (ore transportation and activity scheduling).

Among these problems, product quality control and production scheduling have received the greatest consideration for optimisation (Rahal, 2008). Production scheduling optimisation is so important because its impact on a project's net present value (NPV) is critical. Therefore, it should be updated periodically. Scheduling underground mining operations is primarily characterised by discrete decisions regarding mine blocks of ore, along with complex sequencing relationships between blocks. To optimise block-caving scheduling, most researchers have used mathematical programming, LP (Guest et al., 2000; Hannweg and Van Hout, 2001; Winkler, 1996), MILP (Alonso-Ayuso et al., 2014; Chanda, 1990; Epstein et al., 2012; Guest et al., 2000; Parkinson, 2012; Pourrahimian, 2013; Rahal, 2008; Rahal et al., 2003, 2008; Rubio, 2002; Rubio and Diering, 2004; Smoljanovic et al., 2011; Song, 1989; Weintraub et al., 2008; Winkler, 1996) and QP (Diering, 2012; Rubio and Diering, 2004). LP is the simplest method for modelling and solving. Since LP models cannot capture the discrete decisions required for scheduling, MIP is generally the appropriate MP approach to scheduling (Pourrahimian, 2013).

Solving an MILP problem can be difficult when production system is large, but MILP is a useful methodology for underground scheduling (Rahal, 2008). This section includes reviews of MP applications in block-caving scheduling and some features for each methodology.

Song (1989) used simulation and an MILP model to find the optimal mining sequence in the block-cave operations at the Tong Kuang Yu mine in China. To obtain an optimal mining sequence, Song first simulated the caving process dependent on undercut parameters. Then, he determined ore-draw spacing and pressure distribution during ore-draw. Finally, he used caving simulation and analysis results to obtain the optimal mining sequence. He optimised the production schedule using total mining cost minimisation while considering the geometrical and operational limitations, which guarantee caveability and stability demands. Defining linear functions was an advantage of his methodology. The disadvantage, especially in long-term scheduling, was the solution time.

Chanda (1990) combined a simulation with MIP to model the problem of scheduling drawpoints for production at the Chingola Mine in Zambia. He computerised a model for short-term production scheduling in a block-caving mine. The model used MIP to determine the production rate in finger raises in each production drift considering some quality and quantity constraints. The objective was to minimise the deviation in the average production grade between operating shifts.

Guest et al. (2000) developed LP and MILP models to maximise the NPV of block-caving scheduling (long-term scheduling) over the mine life of a diamond mine in South Africa. This model tried to consider, as constraints, related aspects of mining: mining capacity, metallurgical issues, economic parameters, grades and geotechnical limitations. Applying this wide range of constraints is a remarkable advantage of this model. However, there were two problems with this approach; maximising tonnage or mining reserves will not necessarily lead to maximum NPV and draw control is a planning constraint and not an objective function. The objective function in this case would be to maximise tonnage, minimise dilution or maximise mine life (Rubio, 2002).

Rubio (2002) formulated two strategic goals; maximisation of NPV and optimisation of the mine life in block caving. As constraints, he considered geomechanical aspects, resource management, the mining system and metallurgical parameters involved in the mining operation. One of the main advantages of his model was that it integrated estimates of mine reserves and the development rate that resulted from the production scheduling. Traditionally, these parameters were computed independent of production scheduling. He also formulated a relationship between the draw control factor and the angle of draw. This relationship was built into the actual draw function to compute schedules with high performance in draw control. Opportunity cost in block caving was defined as the financial cost of delaying production from newer drawpoints; a drawpoint will stay active at any given period of the schedule, if it has enough remaining value to pay the financial cost of delaying production from newer drawpoints that may have a higher remaining value.

Rahal et al. (2003) described an MILP goal program with dual objectives of minimising deviation from the ideal draw profile while achieving a production target. They performed a schedule optimisation using a-life-of-mine approach in which all production periods were optimised simultaneously. They assumed that material mixing in the short term has a minimal effect on the panel's long-term state. The model's constraints were deviation from ideal practice, panel state, material flow conservation,

production quality, material flow capacity and production control. They applied the model to De Beers kimberlite mine. The results showed how different production control constraints regulate production from individual drawpoints, as well as recovery of the ideal panel profile by implementing an optimised draw schedule.

Diering (2004) described the basic problem in block-caving scheduling as trying to determine the best tonnages to extract from a number of drawpoints for various periods of time. Those periods could range a day to the life of the mine. Diering (2004) singled out NPV as the overall objective to maximise, subject to some constraints: minimum tonnage per period, maximum tonnage per period, maximum total tonnage per drawpoint, maximum total tonnage per period, ratio of tonnage from current drawpoint compared with neighbours, height of draw of current drawpoint with respect to neighbours, percentage drawn for current drawpoint with respect to neighbours and maximum tonnage from selected groups of drawpoints in a period (usually, the groups of drawpoints are referred to as production blocks or panels). He emphasised that it would be better to formulate the problem as an LP instead of an NLP because of solution time and the size of problems. He applied a multi-step non-linear optimisation model to minimise the deviation between a current draw profile and a defined target. It was shown that this algorithm could also be used to link the short-term plan with the long-term plan.

Rubio and Diering (2004) applied MP to maximise the NPV, optimise the draw profile and minimise the gap between long- and short-term planning. They integrated the opportunity cost into PCBC (GEOVIA) for computing the best height of draw in a dynamic manner. To solve their problem, they used different mathematical techniques such as direct iterative methods, LP, a golden section search technique and integer programming. In their formulation, mining reserves were not part of the set of constraints; the mining reserves were computed as a result of the optimal production schedule. They also used QP to minimise the differences between actual heights of draw vs. a desired target.

Rahal (2008) presented a draw control model that indirectly increases resource value by controlling production based on geotechnical constraints. He used MILP to formulate a goal programming model with two strategic targets: total monthly production tonnage and cave shape. This approach increased value by ensuring that reserves are not lost owing to poor draw practice. The model's advantage was that it allows any number of processing plants to feed from multiple sources (caves, stockpiles and dumps). There were three main production control constraints in the MILP: the draw maturity rules, minimum draw rate and relative draw rate (RDR). Rahal (2008) used MILP to quantify production changes caused by varying geotechnical constraints, limiting haulage capacity and reversing mining direction. He showed that tightening the RDR constraint decreases total cave production. He applied his model for three case studies and illustrated how the MILP can be used by a draw control engineer to analyse production data and develop long-term production targets both before and after a cave is brought into full production. Rahal et al. (2008) used MILP to develop an optimised production schedule for Northparkes E48 mine. They described the system constraints as minimum and maximum draw tonnage, the permissible RDR difference between adjacent drawpoints, drawpoint availability and the capacity of the materials handling system. The impact of different production constraints on total cave capacity was examined. It was shown that the strength of using MILP lies in its ability to generate realistic production schedules that require little manual manipulation.

Weintraub et al. (2008) developed an approach to aggregate the reduced models (which have been derived from a global model) using the original data for an MIP mine planning model in a large block-caving mine. The aggregation was based on clustering analysis. The MIP model was developed to support decisions for planning extraction of blocks and the decisions of exact timing for each block in the extraction columns. The final model was developed to integrate all mines for corporate decisions and to determine extraction from each sector, in each mine, for each period (for a five-year horizon). Weintraub et al. (2008) used two types of aggregation: *Priori* aggregation and *Posteriori* approach. Comparing the original model with the disaggregation, the first approach reduced execution time by 74% and the model dimension by 90%. The second approach reduced solve time by 88% and the model dimension by 15%.

Smoljanovic et al. (2011) presented a model to optimise the sequence of the drawpoint opening over a given time horizon. They incorporated sequencing and capacity constraints. Their model was based on an open-pit model (BOS2) adapted to underground mining. Binary variables were used to indicate whether a specific drawpoint had been opened. The real numbers represent the percentage of the column that was extracted. The model was applied in a panel caving mine in which the studied layout included 332 drawpoints. It was shown that the sequencing can change the value of objective function by as much as 50%. Smoljanovic (2012) applied MILP to optimise NPV and the mining material handling system in a panel caving mine. The model output selected the best sequence after considering different mining systems. Results showed that the out-coming NPV of the objective function for different systems could vary by up to 18%. The importance of the mining system and capacity constraints in the sequencing was shown in comparison with different scenarios.

Parkinson (2012) developed three integer programming models for sequence optimisation in block-cave mining: Basic, Malkin and 2Cone. The research was carried out to help provide a required input to a PC-BC program to find an optimised sequence in which the drawpoints are opened in an automated manner. The models were applied on the two datasets. A simple answer was not found. A combination of the presented models was proposed to help the planner to optimise the sequence. Parkinson demonstrated that integer programming models can generate opening sequences but that the process can be complicated.

Epstein et al. (2012) presented a methodology for long-term mine planning based on a general capacitated multi-commodity network flow formulation. They considered underground and open-pit ore deposits sharing multiple downstream processing plants over a long horizon. The model's target was the optimisation of several mines as an integrated problem. LP and IP with a customised procedure were applied to solve the combined model. For the production phase in underground mine, which it was block caving, constraints were production per sector, product and period, production cost, extraction times for each block (at most once), block and period priority, minimum blocks for each column, order of drawpoints, maximum duration of a drawpoint, extraction rate of each column, the column in each period, similarity of heights in neighbouring columns, bounds on the area, extracted rock per period and each sector extraction within its time window. The model developed by Epstein et al. (2012) has been implemented at Codelco. Production plans for a single mine and integrating multiple mines increased the NPV.

Diering (2012) used QP techniques for block-caving production scheduling. He focused on single-period formulations. He explained that the block-caving process is

non-linear (the tons that you mine in later periods will depend on the tons mined in earlier periods), so it would not be appropriate to use LP for production scheduling in block caving. The objective function was the shape of the cave. Three sets of constraints were applied in the model: mandatory, modifying and grade-related. This formulation omitted the sequence of drawpoint development (interaction between neighbouring drawpoints) as a constraint.

Pourrahimian et al. (2012) presented two MILP formulations at two different levels of resolution:

- drawpoint level
- aggregated drawpoints (cluster level).

The objective function was to maximise the NPV. Pourrahimian et al. (2012) used PCBC's slice file as an input into their model, but their models treat the problem in the drawpoint or cluster level as a strategic long-term plan, and the slices are not used in the presented formulations. To reduce the number of binary integer variables, they used fuzzy c-means clustering to aggregate the drawpoints into clusters based on similarities between draw columns and the physical location of the drawpoint and its tonnage. They used same data for both models and solved the problem for four different advancement directions. The execution time for aggregated drawpoints was reduced by more than 99%.

Pourrahimian et al. (2013) developed a theoretical optimisation framework based on an MILP model for block-cave long-term production scheduling. The objective function was to maximise the NPV. Pourrahimian et al. (2013) formulated three MILP models for three levels of problem resolution: cluster level, drawpoint level and drawpoint-and-slice level. They showed that the formulations can be used in both the single-step method, in which each of the formulations is used independently, and as a multi-step method, in which the solution of each step is used to reduce the number of variables in the next level and, consequently, to generate a practical block-cave schedule in a reasonable amount of CPU runtime for large-scale problems. They considered mining capacity, grade blending, the maximum number of active clusters or drawpoints, the number of new clusters or drawpoints, continuous mining, mining precedence, reserves and the draw rate as constraints, which were involved in the all three levels of resolutions. Using such a flexible formulation is very helpful because depending on the level of studies – prefeasibility studies (PFSs), feasibility studies (FSs) or detailed feasibility studies (DFSs) – a mine planner can use the appropriate level of solution and the related runtime. Pourrahimian et al. (2013) developed and tested their methodology in a prototype open-source software application with the graphical user interface drawpoint scheduling in block-caving (DSBC).

Alonso-Ayuso et al. (2014) considered uncertainty in copper prices along a given time horizon (five years) using a multistage scenario tree to maximise the NPV of a block-cave mine in Chile. The stochastic model then was converted into an MIP model. Alonso-Ayuso et al. (2014) applied the stochastic model in both risk-neutral and risk-averse environments. Results showed the advantage of using the risk-neutral strategy over the traditional deterministic approach, as well as the advantage of using any risk-averse strategy over the risk-neutral one.

Table 3 Summary of applied MP models in block-caving production scheduling

Author	*Model*	*Model objective(s)*	*Constraint*
Song (1989)	Simulation and MILP	Minimisation of total mining cost	• Geometrical limitations • Operational limitations
Chanda (1990)	Simulation and MIP	Minimisation of the deviation in the average production grade between operating shifts	• Maximum allowable output per shift • Maximum allowable number of working drawpoints per shift • Declaration of exhaustion for exhausted drawpoints • Required grade for each shift (equality) • Tonnage of blended ore in each shift

Guest et al. (2000)	LP and MILP	Maximisation of NPV	• Geotechnical constraints
			  • Column draw rates
			  • Precedence of accumulated tons drawn
			  • Limits in differences of accumulated tons drawn between columns within time horizons
			  • Limits in ratios of tons drawn between columns (neighbours) within time horizons
			• Mining constraints
			  • Ore flow constraints (tunnels, ore passes, haulage, underground accumulation areas, shaft systems)
			• Metallurgical constraints
			  • Treatment plant (capacities per period)
			• Economic constraints (revenue, costs)
			• Geological constraints (grade, size)

Table 3 Summary of applied MP models in block-caving production scheduling (continued)

Author	*Model*	*Model objective(s)*	*Constraint*
Rubio (2002)	MILP and NLP	Two models (a) maximisation of NPV and (b) optimisation of the mine life	• Development rate • Undercut sequence • Drawpoint status • Maximum opened production area • Draw rate • Period constraints • Mining reserves
Rubio and Diering (2004)	MILP and QP	Maximisation of NPV, optimisation of draw profile, and minimisation of the gap between long- and short-term planning	• Development rate • Undercut sequence • Maximum opened production area • Draw rate • Draw ratio • Period constraints
Diering (2004)	NLP	Maximising NPV for M periods and minimisation of the deviation between a current draw profile and a defined target	• Minimum and maximum tonnage per period • Maximum total tonnage per drawpoint and per period • Ratio of tonnage from current drawpoint compared with neighbours. • Height of draw of current drawpoint with respect to neighbours • Percentage drawn for current drawpoint with respect to neighbours • Maximum tonnage from selected groups of drawpoints in a period

Table 3 Summary of applied MP models in block-caving production scheduling (continued)

Author	*Model*	*Model objective(s)*	*Constraint*

Rahal (2008)	MILGP	Minimising deviation from the ideal draw profile while achieving a production target	• Deviation From Ideal Plan
			  • Ideal depletion
			  • Panel production rate
			  • Production from external sources
			• Contents of material sources
			• Material flow conservation (blocks, tunnels, ore pass, haulage, accumulation, shaft, plant)
			• Material flow and capacity (source flow, block, externals, ore pass, haulage, accumulation, shaft, plant)
			• Production Control
			  • Block available for draw
			  • Relative draw rate
			  • Block flow bounds
			  • Draw maturity rules (Lower Depletion Bound/Upper Production Bound)
			• Product quality and quantity
			• Economics

Weintraub et al. (2008)	MIP	Maximisation of profit	• Each cluster can be extracted only once • Sequence of extractions • The allowable speed • Capacity of extraction • Conservation of flows and logical relationships between variables

Table 3 Summary of applied MP models in block-caving production scheduling (continued)

Author	*Model*	*Model objective(s)*	*Constraint*

Smoljanovic et al. (2011)	MILP	Optimisation of NPV and mining material handling system	• Production Constraints
			  • Max and min amount of tonnage to be extracted per time period
			  • The overall mine capacity
			  • Total number of drawpoints to be opened at each time period
			  • Capacity per drawpoint
			  • Min percent of extraction for each drawpoint
			  • Lifetime of a drawpoint
			  • Capacity of haulage system
			• Geometric constraints
			  • Connectivity and shape constraints

Epstein et al. (2012)	MIP	Maximisation of NPV	• Production per sector (product and period) • Production cost • Extraction times for each block • Block and period priority • Minimum blocks for each column • Order of drawpoints • Maximum duration of a drawpoint • Extraction rate of each column • The column in each period • Neighbouring columns heights similarity • Bounds on the area • Extracted rock per period • Each sector extraction within its time window

Table 3 Summary of applied MP models in block-caving production scheduling (continued)

Author	*Model*	*Model objective(s)*	*Constraint*

Diering (2012)	QP	Objective tonnage
		(to optimise the shape of the cave)
			• Mandatory constraints
			  • Production capacity
			  • A maximum tonnage for each drawpoint based on the drawpoint maturity curve
			  • A minimum tonnage for each drawpoint.
			• Modifying constraints
			  • Maximum tonnage from production tunnels.
			  • Maximum tonnage from an orepass or crusher.
			  • Maximum tonnage from an entire sector
			• Grade-related constraints

Parkinson (2012)	IP	Finding an optimal opening sequence in an automated manner	• Each drawpoint starts once • Global capacity (processing plant capacity) • Tunnel development • Additional constraints: • Within-tunnel contiguity • Across-tunnel contiguity
Pourrahimian et al. (2013)	MILP	Maximisation of NPV	• Mining capacity • Grade blending • Maximum number of active clusters or drawpoints (according to the model resolution) • Number of new clusters or drawpoints (according to the model resolution) • Continuous mining

Table 3 Summary of applied MP models in block-caving production scheduling (continued)

Author	*Model*	*Model objective(s)*	*Constraint*

Pourrahimian et al. (2013)	MILP	Maximisation of NPV	• Mining precedence
			  • Slice
			  • Drawpoint
			  • Cluster
			• Reserves
			• Draw rate
			  • Draw column
			  • Cluster

Alonso-Ayuso et al. (2014)	MILP	Maximisation of NPV while considering uncertainty in copper price	• Each cluster is processed at most once • If a cluster is processed at a given period then all predecessor clusters are also processed by that period • The clusters in each set would be extracted simultaneously in each sector • Number of tons processed in each sector at each period • Flow conservation constraints for the processing stream • Number of tons processed in each period • Upper and lower bounds for the total area processed in each sector • Number of tons processed in each period • Upper bound due to the capacity of processing stream • The maximum increase and decrease of tons in each sector in each period

Table 4 Advantages and disadvantages of applied mathematical methodologies in block-caving production scheduling

Methodology		*Features*
LP	Advantage	• LP method has been used most extensively (Rahal, 2008)
		• It can provide a mathematically provable optimum schedule (Rahal, 2008)
	Disadvantage	• Straight LP lacks the flexibility to directly model complex underground operations which require integer decision variables (Winkler, 1996)
		• Mine scheduling is too complex to model using LP and the only possible approach is to use some combination of theoretical and heuristic methods to ensure a good, if not optimal schedule (Scheck et al., 1988)
MILP	Advantage	• Computational ease in solving a MIP problem (and MILP) is dependent upon the formulation structure (Williams, 1974)
		• MILP could be used to provide a series of schedules which are marginally inferior to a provable optimum (Hajdasinski, 2001)
		• MILP is superior to simulation when used to generate sub-optimal schedules, because the gap between the MILP feasible solution and the relaxed LP solution provides a measure of solution quality (Rahal, 2008)
		• MILP can provide a mathematically provable optimum schedule (Rahal, 2008)
	Disadvantage	• It is often difficult to optimise large production systems using the branch-and-bound search method (Rahal, 2008)
		• The block-caving process is non-linear (the tons which you mine in later periods will depend on the tonnes mined in earlier periods), so it would not be appropriate to use LP for production scheduling in block caving (Diering, 2012)
QP	Advantage	• Since the block-caving process is non-linear, QP could be an appropriate option to model it
		• It can find solutions in the interior of the solution space, which results in an even height of drawpoints as well as lower horizontal mixing between drawpoints (Diering, 2012)
	Disadvantage	• Solving this kind of problem could be a challenge. It must be changed to LP and then be solved, to ensure conversion errors

Rubio (2014) introduced the concept of portfolio optimisation for block caving. In this method, every decision related to mine design and mine planning could be a component of a set that defines a feasible portfolio. This set is optimised for different production targets to maximise return subject to a given level of reliability. Using this approach, a frontier efficient is proposed as a boundary to display different strategic designs and planning options for the set of variables under study. By this method, the decision-makers can define a point along the frontier efficient where they want to place a given project.

Table 3 shows the summary of the aforementioned MP applications in block-caving scheduling.

The available tools for block-cave production scheduling can be divided into two categories:

- commercial
- in-house tools.

One of the commercial software is GEOVIA PCBC. The program is integrated into a general-purpose geological modelling and mine-planning system so that it can be used for studies ranging from pre-feasibility to daily draw control. The simulation of mixing is an important part of the program. PCBC simulates the extraction from each active drawpoint period-by-period subject to a range of constraints and inputs (Diering, 2000).

In mathematical programming, we look for values of variables, which are allowed and which do not violate the constraints. This defines what is called a solution space, in which the edges of this space are the constraints. In case of an LP formulation, the solution must be on a boundary of this space. In the case of block-cave scheduling, an LP formulation will always seek to take the maximum tons from the highest value drawpoints and the least tons from the lower-valued drawpoints (Diering, 2012). As a result, this kind of scheduling may result in high levels of horizontal mixing between drawpoints because the draw columns have different heights. This is a potential disadvantage of LP application in block-caving scheduling. Table 4 summarises the advantages and disadvantages of methodologies examined in previous studies.

6 Conclusion

Increasing the use of block-cave mining in new-world mining environments has led many researchers to focus on this area to make mining operations as optimal as possible. Production scheduling in block caving, because of its significant impact on the project's value, has been considered a key issue to be improved. The problem is complex, unique for each case, large-scale and non-linear. Researchers have applied different methods to model production scheduling in block caving, for short-term and long-term periods of mining, some for real case studies as industry projects and others as academic research projects.

Generally speaking, confronting future challenges in block-cave mining can be divided into two categories:

- operational
- economical.

Block caving is known as a low-cost mining method, which makes it possible to mine the low-grade ore-bodies, therefore, optimal production schedule with lower cost is required. Block-cave mining is one of the best solutions for continuing the operation after shutting down the mine in deep open-pit mines. The new operation (block caving) has to feed the processing plant, which used to be fed by the open-pit mine. Therefore, the production rate in the block-cave operation has to be as high as the open-pit mining. Although some semi-auto mining equipment has been introduced for block caving, but it is just the starting point to reach the full-automated operations. Also, making decisions about the geometry of drawpoints, best height of draw, undercut level and the production level are critical and challenging. Block-cave mining usually requires much more development compared with other mining methods, which needs a long period of time before starting the production, so the high capital cost is needed to run the project. High capital cost increases the risk of project. The operational costs of block-cave mining is low but if the rock mass caveability is not achieved as it expected, the costs for additional drilling and blasting can be definitely challenging.

Most of the researchers have applied MILP to model production scheduling; it can be useful because both the integer variables (whether a block, slice or draw column should be extracted) and the continues variables (the constraints and mining operation details) can be modelled so that the optimal values can be achieved while considering the system's constraints. Basically, the block-caving operation is non-linear. Therefore, linear programming could be an inappropriate method to model production scheduling. Quadratic programming as a non-linear methodology has been applied by a few researchers for block-cave mining scheduling.

As computer-based algorithms are improved, we expect to see the development of more detailed models with more complexity, models that try to be more practical and include all aspects of mining systems, with new algorithms for faster solutions. Using non-linear methodologies for multi-period scheduling with a reasonable solution time would improve block-caving scheduling. In block-caving operations, decisions about current actions are often based on how those actions affect future actions. For this reason, a real options technique can be properly used in production scheduling optimisation.

There are some uncertainties in block-cave mining that should be involved in production scheduling. Grade uncertainty is one of the most common, because of the nature of ore-body, but in block-caving operations grade uncertainty is more critical and complicated, owing to the vertical and horizontal mixing, which occurs during the caving and production processes. Once the rock is fragmented, the particles of the rock flow towards the production level in different ways depending on the fragmentation profile and distribution. Price uncertainty is another variable that should be considered when attempting to achieve realistic optimal production scheduling.

Blasting operations in block caving has a critical impact on fragmentation and, as a result, on material flow. The gravity flow of fragmented rock plays an important role in the production rate and grade in block-caving operations. When considering the material flow in the presence of blasting parameters, production scheduling could result in more realistic plans. Since rock mass is broken by caving, the actual fragmentation expected at the drawpoints is uncertain. Therefore, drawpoint productivity is uncertain and the amount of area that needs to be developed and undercutted also becomes uncertain.

Geotechnical aspects of ore-body and its surrounding rocks determine caveability and the efficiency of block-caving operations. Using more geotechnical constraints in production scheduling modelling helps the mine planners to have more confidence about

scheduling results. If the rock mass is of high stress and competent, the cave propagation could be uncertain and triggered erratic dilution and non-uniform grade extraction.

Developing accurate clustering methods, with more flexible levels of problem resolution, could lead to better options for mine planners during the different stages of planning.

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Influence of rock mass rating and *in situ* stress on stability of roof rock in bord and pillar development panels

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Abstract: The re-orientation of bord and pillar layout is dictated by the direction of the horizontal stress. However, the competence of strata expressed in terms of rock mass rating (RMR) and the magnitude of in situ stress have a persuasive influence in deciding the gain out of re-orientation of galleries. However, a few queries like: a) "Is it essential to re-orient all bord and pillar workings vis-à-vis the direction of in situ stress?"; b) "What if the strata are competent and the workings are subjected to low magnitude of horizontal stress?"; c) "What happens to a bord and pillar workings affected by high horizontal stress and having incompetent strata?" have remained unaddressed. This paper, therefore, addresses such queries through a new approach of classification of the bord and pillar development workings using numerical modelling. This classification system will aid field engineers in deciding the need for re-orienting galleries vis-à-vis the RMR and in situ stress.

Keywords: bord and pillar; in situ stress; perturbed stress; rock mass rating; RMR; Sheorey's failure criterion; horizontal stress; stability.

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1 Introduction

In underground coal mines, knowledge of magnitude and orientation of horizontal stress field play a major role in deciding the mining method, size and orientation of roadways and support density. It is by now a well-known fact that one of the principal horizontal stresses is always greater than the vertical stress for at least up to a depth of 500 m (Moura coalmine, Australia, Enever and Wooltorton, 1983). Measurements of in situ stress in British coal measures of North Selby colliery also corroborate the above fact (Bigby et al., 1992)

Re-orienting the development layout in terms of changing the direction of dip and level galleries in bord and pillar panel can mitigate the adverse strata problems in development galleries (Sengupta et al., 2004; Sengupta and Sinha, 2011). The competence of roof rock strata expressed in terms of *RMR* and the magnitude of in situ stress together have a strong bearing in deciding the gain out of re-orientation of galleries. This particular issue is addressed in this paper using a new approach of classification of bord and pillar development workings making use of numerical modelling. This paper also discusses the strata control problems faced by two mines, namely Tandsi and Thesgora, which are characterised by weak roof rock and high horizontal stress.

The directional influence of high horizontal stress has been observed in a number of bord and pillar mining layouts in India (Anireddy and Ghose, 1994). High horizontal

stress and unfavourable orientation of roadways/galleries with respect to the direction of maximum horizontal stress adversely affect roadways' condition (Aggson and Curran, 1978). Investigations conducted in Australian coal mines (Gale and Fabjanczyk, 1991) have established a relation of roof failure in the roadways with the angle between the roadway axis and direction of maximum horizontal stress.

In a typical bord and pillar mining layout practised in India, the dip galleries and level galleries are driven perpendicular to each other. Now in a particular situation orienting one set of the galleries, say the level galleries, along the major principal stress will improve the roof condition in level galleries only but will severely affect the roof condition in dip galleries. The best approach would be re-orientation of the galleries in such a way that the level galleries make an angle of 45° with the direction of horizontal stress (Sinha et al., 2013). In such a situation the strata problems in the dip galleries will improve but the strata problems in level gallery will deteriorate to a manageable extent.

Competence of roof rock in conjunction with horizontal stress has a strong bearing on strata problem as stated below:

- re-orientation of the galleries having highly competent roof rock and subjected to low horizontal stress does not bring in much gain in terms of stability of roof rock. Such development workings are grouped as Class-A
- re-orientation of galleries having moderately competent roof rock and subjected to moderate horizontal stress helps in bringing about some improvement in strata problem (Class-B)
- re-orientation of galleries having incompetent roof rock and subjected to high horizontal stress does not yield much improvement in strata problem (Class-C).

Bord and pillar mines having low to high horizontal stress and weak to strong roof rock can be classified in to above three classes based on quantification of competence of roof rock expressed in terms of *RMR* and the value of stress ratio (*K*). The quantification of these two parameters namely, rock mass rating (*RMR*) and ratio of horizontal to vertical stress (*K*) for the purpose of the classification is done through parametric study using numerical modelling as discussed in the following section.

2 Influence of in situ stress on stability of development galleries

To investigate into the influence of in situ stress on stability of development galleries in bord and pillar mines, a panel consisting of 3×3 pillars was formed in *Examine*^{3D} program. *Examine*^{3D} is a CAD-based program utilising boundary element method to solve problems related to three-dimensional excavations in rock mass with sufficient accuracy (Sinha et al., 2007). Square pillars of size 45 m centres and gallery size of 4.5 m wide and 3 m height were selected. Gallery excavations were created by use of polylines, nodelines and skin commands of the program. The skin of the excavation was created between two nodelines for one set of galleries parallel to each other. The skin of excavation was discretised using the default discretisation scheme of the program with a mesh density factor of unity.

2.1 Initialisation of in situ stress in Examine^{3D}

The principal stresses with their respective dip values and dip directions are fed as inputs in *Examine^{3D}*. For parametric study the value of *K* was varied from 1.5 to 3.0 in steps of 0.5. The major horizontal principal stress was set to act in the east-west direction.

2.2 Rock mass property and failure criterion for evaluating the factor of safety

The elastic modulus of rock mass was taken as 5.0 GPa and Poisson's ratio as 0.25 in the model. For Indian coal measure rocks the Sheorey's (1997) failure criterion has been found suitable (Singh et al., 2002; Murali Mohan et al., 2001; Banerjee and Srivastava, 2007; Banerjee et al., 2007). This criterion uses the 1976 version of Bieniawski's *RMR*. The basic CMRS-*RMR* value was directly used here instead of Bieniawski's *RMR*, since failure criterion thus obtained has worked well in Indian coalmine (Kushwaha et al., 2010). Sheorey's failure criterion for rock mass is defined as:

$$\sigma_1 = \sigma_{cm}\left(1+\frac{\sigma_3}{\sigma_{tm}}\right)^{b_m} \tag{1}$$

where

σ_1: triaxial strength of rock mass

σ_3: minor principal stress or the confining stress

b_m: exponent of the failure criterion that controls the curvature of the nonlinear Sheorey's failure criterion;

σ_{cm} and σ_{tm}: compressive strength and tensile strength of the rock mass, respectively.

σ_{cm}, σ_{tm} and b_m are defined as follows:

$$\sigma_{cm} = \sigma_c \exp\left(\frac{RMR-100}{20}\right) \tag{2}$$

$$\sigma_{tm} = \sigma_t \exp\left(\frac{RMR-100}{27}\right) \tag{3}$$

$$b_m = b^{\frac{RMR}{100}}, \quad b_m < 0.95 \tag{4}$$

The symbols without the subscript *m* are the laboratory-determined values for intact rock.

As the program *Examine^{3D}* does not support the use of failure criterion other than the Mohr–Coulomb or the Hoek and Brown. The Mohr–Coulomb equivalent, which is the best fit of Sheorey's failure criterion was utilised. The rock mass shear strength (cohesion), τ_{sm}, the coefficient of internal friction, μ_{0m}, and the angle of internal friction, φ_{0m}, were obtained from Sheorey's criterion given by equations (5) through (7).

$$\tau_{sm} = \sqrt{\sigma_{cm}\sigma_{tm}\frac{b_m^{\,b_m}}{(1+b_m)^{(1+b_m)}}} \tag{5}$$

$$\mu_{0m} = \frac{\tau_{sm}^2 (1+b_m)^2 - \sigma_{tm}^2}{2\tau_{sm}\sigma_{tm}(1+b_m)} \tag{6}$$

$$\varphi_{0m} = \tan^{-1} \mu_{0m} \tag{7}$$

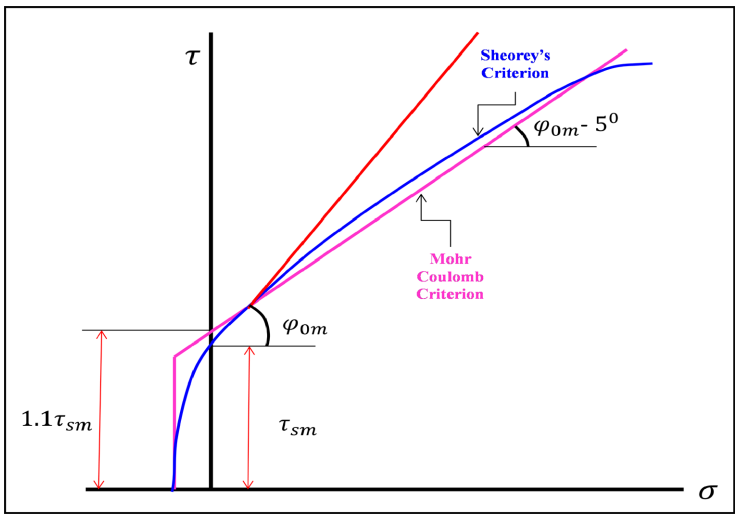
The stability of underground structures thus obtained by use of Sheorey's failure criterion based on factor of safety (FoS) is interpreted as summarised in Table 1.

Table 1 Stability of underground structures vis-à-vis factor of safety

Factor of safety (*FOS*)	*State of stability*
FOS < 0.6	Immediate failure within few hours. Applicable to ribs and stooks also
0.6 < FOS < 1.0	Stable for few days
1.0 < FOS < 2.0	Short-term stability. The pillar/rib can fail within few years
FOS > 2.0	Long-term stability, i.e., the pillars/ribs may not fail at all. Such pillars can be treated as indestructible

It is, however, found that the values of rock mass cohesion, τ_{sm}, and friction angle, φ_{0m}, have to be changed slightly to take into account the fact that the Sheorey's failure criterion is nonlinear (Figure 1) of the form $\tau = \tau_{sm}\left(1+\frac{\sigma}{\sigma_{tm}}\right)^{c_m}$, where, $c_m = \mu_{0m}^{0.9}\frac{\sigma_{tm}}{\tau_{sm}}$, whereas the Mohr–Coulomb failure criterion is linear. The value of τ_{sm} obtained from Sheorey's failure criterion was increased by 10% and that of φ_{0m} reduced by 5° for using Sheorey's failure criterion as replacement of Mohr–Coulomb failure criterion. Considering large number of Indian case studies, Murali Mohan et al. (2001) prescribed that the values determined by above mentioned method can be safely taken for modelling.

Figure 1 Schematic diagram for using Mohr-Coulomb equivalent of the Sheorey's failure criterion (see online version for colours)



Using the laboratory-determined compressive strength of coal specimen as 40 MPa, tensile strength of 4 MPa, the coefficient b_m in Sheorey's equation as 0.5 and varying the rock mass rating (RMR) value from 30 to 70 in steps of 10, the Mohr–Coulomb parameters (tensile strength, cohesion and angle of friction for rock mass) equivalent to Sheorey's failure criterion were calculated and summarised in Table 2.

Table 2 Mohr-Coulomb parameters for rock mass equivalent to Sheorey's failure criterion for different *RMR* values

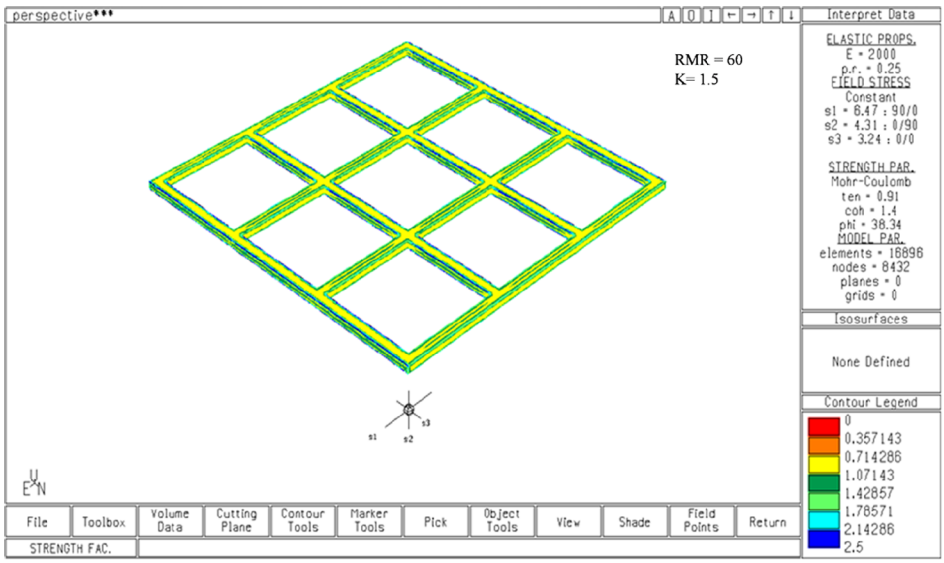
	Corresponding Mohr-coulomb parameters for rock mass		
RMR values	*Tensile strength, MPa*	*Cohesion, MPa*	*Friction angle, degrees*
30	0.3	0.35	30.75
40	0.43	0.56	33.34
50	0.63	0.89	35.88
60	0.91	1.4	38.34
70	1.32	2.2	40.74

2.3 Numerical modelling assisted classification system of development galleries in terms of severity of strata problems

Based on the inputs fed to the program *Examine*^{3D}, as discussed in the preceding Sections 2.1 and 2.2, a total of 20 numerical models were run. The FoS plots of the development workings were analysed in terms of roof problems in all the models and a classification system was developed using the criterion discussed in Section 1.0.

A representative view of the FoS plot of the roof rocks in development galleries of Class-A category is shown in Figure 2.

Figure 2 FoS plot of roof rock in development galleries of Class-A where on set of galleries is oriented across the direction of in situ stress and the compass coordinate (see online version for colours)



It is observed from Figure 2 that the roof rock does not experience severe strata problems in either set of the galleries. The galleries along the major horizontal principal stress having FoS of roof rock more than 0.7 reaching beyond 1.07 can be considered most stable. Such kind of galleries will remain stable for a few days even without support (Table 1).

Galleries oriented across the major horizontal principal stress and showing FoS in the range of 0.7 to 1.0 indicate that they will also remain stable for a few days without support (Table 1).

Under such circumstance, where the FoS of roof rock in galleries oriented along and across the major horizontal principal stress is almost same, re-orientation of direction of galleries does not make any sense. Galleries characterised with competent roof rock with high RMR value in conjunction with low horizontal stress may be classified as Class-A.

A close observation of the representative view of the FoS plot of the development galleries belonging to Class-B, as shown in Figure 3 reveals the following facts.

- One set of galleries aligned perpendicular to the major horizontal principal stress experience severe strata problems. This is evident from the fact that the FoS of roof rock in these galleries varies between 0.3 and 0.7. Any excavation surface having FoS < 0.6 is expected to fall immediately within a few hours of its creation if it remains unsupported (Table 1).
- Another set of galleries aligned parallel to the major horizontal principal stress, does not experience severe strata problems as the FoS of the roof rock in these galleries vary between 0.7 and 1.0. This means, the roof will remain stable for few days even without support (Table 1).
- Re-orientation of the galleries at an angle of 45 in this kind of development workings will assist in mitigating the problems of adverse roof condition. Galleries having moderately competent roof rock characterised by moderate RMR subjected to moderate horizontal stress may be classified as Class-B.

Figure 4 shows the FoS plot post re-orientation of Class-B category of workings. Here, the FoS in both the galleries have reached a manageable levels of 0.7 to >1.0 indicating better roof condition.

Similarly, Figure 5 is a representative view of the FoS plot of the development workings of Class-C.

It is observed from Figure 5 that in both sets of galleries the roof rock experiences severe strata problems as the FoS is less than 0.7 indicating that the roof fall will occur within a few hours of driving if remain unsupported. It also indicates that there will be spalling of pillars standing on either sides of galleries which are aligned perpendicular to the major principal stress direction.

The plot of FoS contours for the galleries, post re-orientation, as shown in Figure 6 is a representative case covered under Class-C category. It is interesting to note that even by re-orienting the galleries there is no relief from strata problems. Roof rock in both set of galleries, post re-orientation show a FoS < 0.7 indicating immediate failure within a few hours, if left unsupported.

Figure 3 FoS plot of roof rock in development galleries of Class-B where on set of galleries is oriented across the direction of in situ stress and the compass coordinate (see online version for colours)

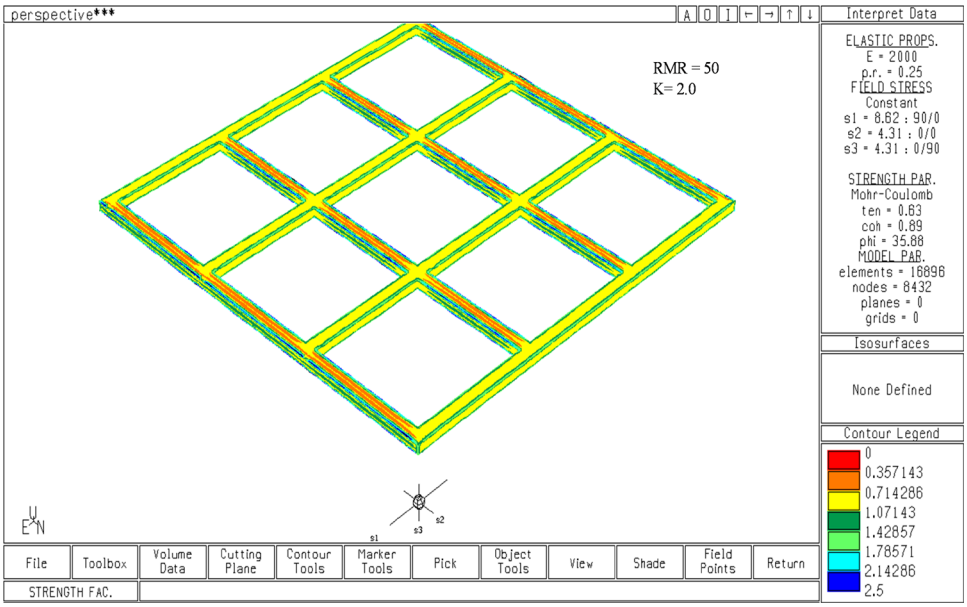


Figure 4 FoS plot of roof rock in development galleries of Class-B where on set of galleries is oriented at 45°to the direction of in situ stress and compass coordinate (see online version for colours)

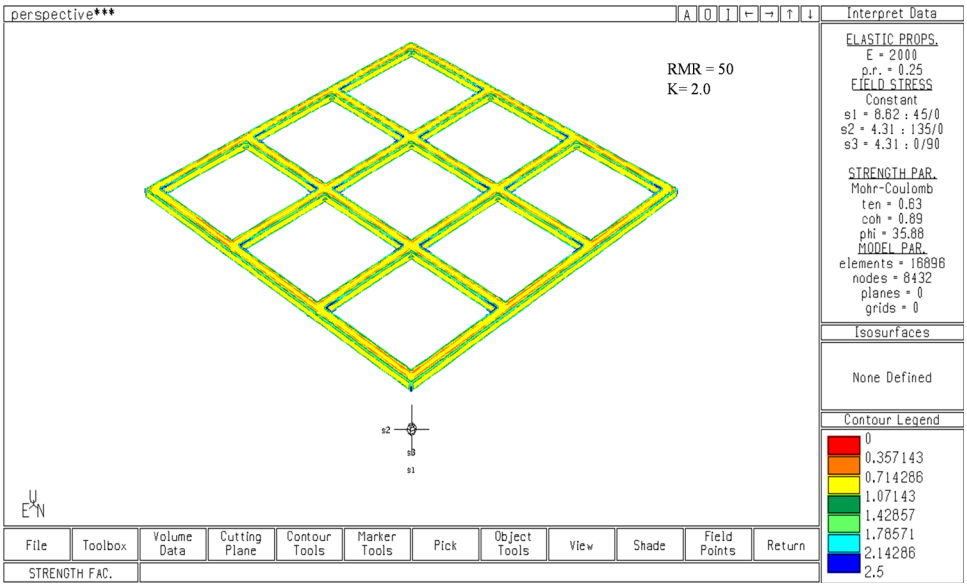


Figure 5 FoS plot for development workings of Class-C with one set of galleries across the stress direction and the compass coordinate (see online version for colours)

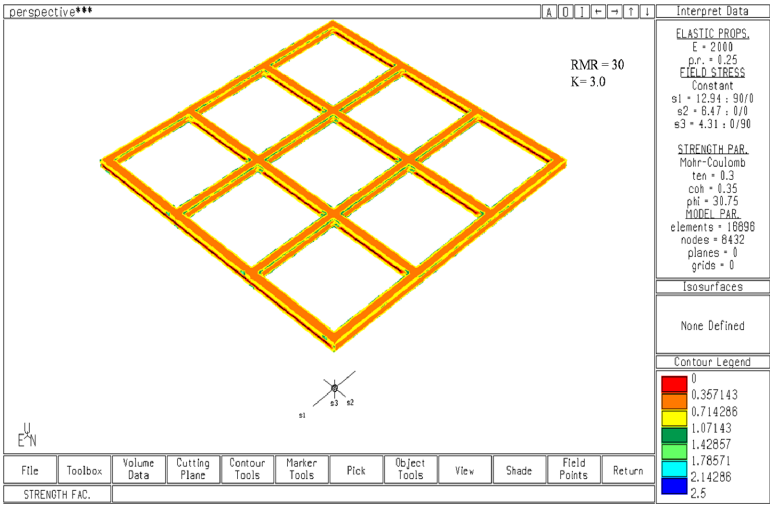
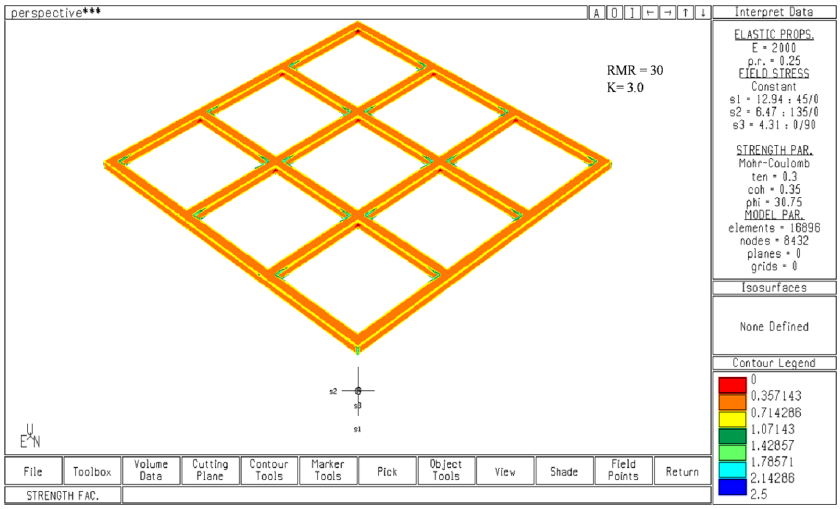


Figure 6 FoS plot for development galleries of Class-C oriented at 45° to horizontal stress components and the compass coordinate (see online version for colours)

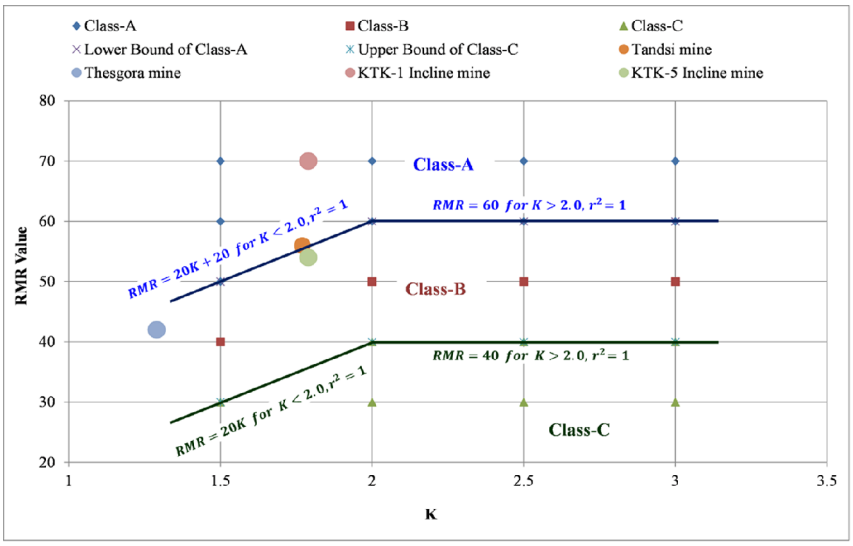


2.4 Influence of RMR and in situ stress ratio 'K' on stability of roof rock in development galleries of bord and pillar mines

Based on the parametric study by varying the RMR value from 30 to 70 in steps of 10 and varying the *K*-value from 1.5 to 3.0 in steps of 0.5, a plot depicting relationship between *RMR* and in situ stress ratio, *K*, is shown in Figure 7. The figure also indicates the class of development workings in the legend. This cross plot with different class of development working reveals that one can distinguish between different class/category of development galleries easily using regression analysis. It is pertinent here to mention that Class-A galleries does not require re-orientation. Class-B galleries can be made stable by

re-orienting them with respect to the major principal stress by 45 . Class-C galleries are worst case where the roof rock is weak as well as the stress condition is also high indicating requirement of heavy support during development.

Figure 7 Plot of case studies overlapping the classification of development galleries for strata control severity based on roof rock competence and in situ stress ratio *K* (see online version for colours)



As observed from the class plot in Figure 7, Class-B workings follow a trend that can be best addressed through bilinear curve. Physical interpretation also supports the fact that as the in situ stress ratio increases, the galleries will experience strata problem but there must exist a cut-off in terms of rock competence characterised by *RMR* at which the Class-A or Class-C can be clearly distinguished. To distinguish between the three classes, the following procedure was adopted:

- The data in Class-A which forms its lower bound was characterised by a bilinear curve. Any point lying above this curve will fall into Class-A. With a similar reasoning, any point lying below this curve will either fall in Class-B or Class-C.
- The data in Class-C which forms its upper bound was considered for bilinear curve fitting. Any point lying below this curve will fall into Class-C and any point lying above this curve will either fall under Class-B or Class-A.
- The region below the bounding curve of Class-A and that above Class-C will form a common region. This common region will be characterised by Class-B.

The curve for the upper bound of Class-C and lower bound of Class-A are given by equations (8) and (9), respectively, and shown graphically in Figure 7.

$$\left.\begin{array}{ll} RMR = 20K & \text{for } K < 2.0 \\ RMR = 40 & \text{for } K > 2.0 \end{array}\right] r^2 = 1.0 \tag{8}$$

$$\left.\begin{array}{ll} RMR = 20K + 20 & \text{for } K < 2.0 \\ RMR = 60 & \text{for } K > 2.0 \end{array}\right] r^2 = 1.0 \tag{9}$$

3 Validation of equations developed for classification of galleries in respect of severity of strata problem based on roof rock competence and in situ stress ratio K

In order to validate the equations (8) and (9), the plot the *RMR* vs. in situ stress ratio *K* is superimposed on the plot shown in Figure 7. This gives a pictorial representation of the class or category of development gallery. Thereafter, one has to verify physically in the field if the mine gallery experiences problems due to directional influence of in situ stress.

The equations developed were validated in respect of four mines namely, Tandsi mine, Thesgora mine, KTK-1 incline and KTK-5 Incline. The pictorial representation of the same is given in Figure 7.

The *RMR* values and the measured in situ stress ratios of the aforesaid four mines are summarised Table 3.

Table 3 RMR values and in situ stress ratio of the mines under case study

	Tandsi mine	*Thesgora mine*	*KTK-1 Incline mine*	*KTK-5 Incline mine*
In situ stress ratio, *K*	1.77	1.29	1.79	1.79
RMR value	57	43	70	54

Source: Sengupta et al. (2004) and Venkateswarlu et al. (2007)

From the plot as shown in Figure 7, the following observations related to the validation of the classification system emerge:

i The KTK-1 Incline mine come under Class-A.

ii The other two mines namely Tandsi and KTK-5 Incline fall, respectively, on the lower bound of Class-A and upper bound of Class-B. Though as per classification Tandsi mine should have no roof problem but in reality it had severe roof problems in some pockets of the development workings. This clearly indicates role of directional influence of in situ stress. KTK-5 Incline did not experience any strata problem in reality though a moderate problem is expected as revealed by plot in Figure 7.

iii Thesgora mine falls under the Class-B. As per classification, Thesgora mine should not have strata problems if the direction of the major principal stress makes an angle of 45 with the direction of galleries. However, in reality it had severe roof problems in some pockets of the development workings.

iv Keeping in view the fact that the real observation is contrary to the prediction made through plots in Figure 7 as mentioned in point (ii) it perhaps requires to carry out detailed investigations for class of mines falling at the boundary of two classes of the proposed classification.

A brief explanation to highlight the cause of strata problem in Tandsi and Thesgora mines which are contrary to the predicted one is given as under:

One of the possible causes of strata problem could be the unfavourable orientation of joint planes with respect to roadway directions. Orientation of the discontinuities in the above case with respect to direction of roadways was found to be largely favourable. Indicating possibility of one or more of other causes of failure such as inefficacy of support system, delay in the installation of support system, improper blasting and structural instability. These possible causes were also examined on site and it was found that dip and level galleries were supported with roof bolts and rope stitches. Anchorage tests were carried on these bolts and stitches which showed load-bearing capacity of more than 10 tonnes thereby negating the possible cause of roof failure due to ineffective support system.

Coal due to its low density and high value of elasticity constitutes a good roof and can tolerate some time delay in support installation. However, no such delay in the installation of roof bolts and rope stitches was reported in the above case. This proves that the roof failure in the above case was also not due to delay in installation of support system.

The system of blasting was examined in respect of pattern of holes, type of explosives, total charge per round of firing and delay time of detonators being used. The effects of blasting were examined at different places and found to have caused no over breaks at any place, thus negating the possible cause of failure due to improper blasting.

In view of disagreement of aforesaid possible causes, unfavourable orientation of these roadways with respect to direction of high horizontal in situ stress is suspected to be the likely cause of the roof fall. But, it is again observed at the same time that roof falls do not occur throughout the roadways (level/dip galleries) in spite of no change in the orientation of these roadways. The reason for dissimilar behaviour of roof may be due to different orientation of maximum compression with respect to direction of roadways. This re-orientation of the horizontal stress might have been caused due to the influence of discontinuities such as fault. By elimination process, it may be concluded that the probable cause of roof failure in the above case may be due to unfavourable orientation of roadways with respect to direction of maximum compression.

3.1 Tandsi mine

In situ stress was measured using hydraulic fracturing at two sites namely, 26D/8X and 13D in Tandsi mine. The site 26D/8X was far away from geological discontinuity such as fault. The other site 13D was chosen close to a geological discontinuity like fault. A summary of the stress regime in Tandsi mine as revealed by hydraulic fracturing at the aforesaid two sites namely, 26D/8X and 13D is given in Table 4.

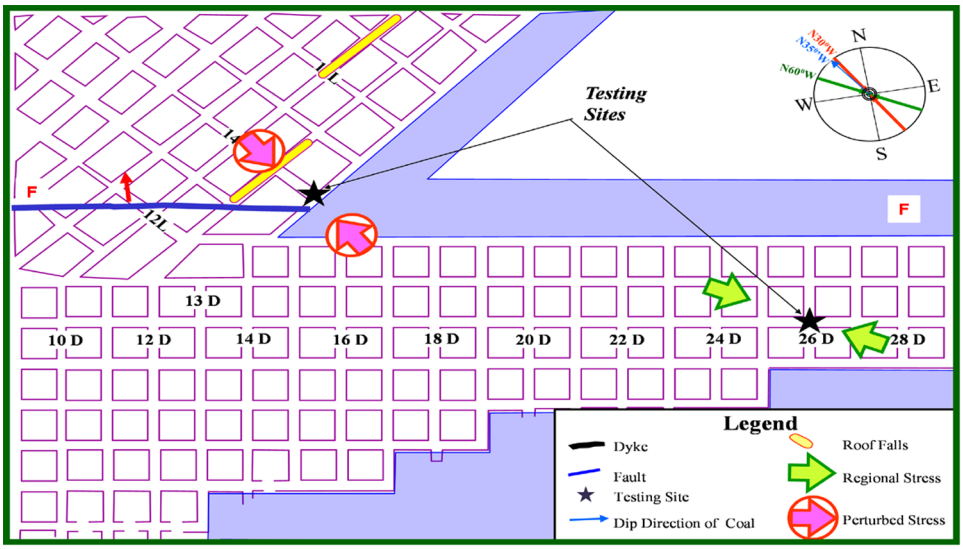
The stress regimes evaluated at the two sites in Tandsi mine are, respectively, N120 (N60 W) and N150 (N30 W) resulting into a perturbation of the principal horizontal stress direction due to the influence of the fault to the extent of 30 .

Now, a close look at Figure 8 reveals that the roof fall does not take place in all galleries having similar geo-mining conditions; rather it is localised near the fault. At this location, the perturbed stress is oriented across the dip gallery. On the other hand, the stress direction at 26D/8X is oriented neither along nor across the galleries. This stress perturbation provides the clue as to why two stretch of level/dip galleries having similar geo-mining conditions do not have a similar nature of roof problem.

Table 4 Stress regime indicated by the tests at two sites in Tandsi mine

	Regional stress at the first site i.e., 26D/8X having 230 m thick overburden	*Perturbed stress at the second site i.e., 13D having 255 m thick overburden*
Maximum horizontal principal stress (σ_H), MPa	9.00 ± 0.618	11.20 ± 0.906
Direction of maximum horizontal principal stress (σ_H), degree	N 120 (N 60 W)	N 150 (N 30 W)
Minimum horizontal principal stress (σ_h), MPa	4.5 ± 0.309	5.60 ± 0.453
Vertical stress (σ_v), MPa (with density of rock = 2.2 g/cm^3)	5.06	5.61
Ratio, $K = \sigma_H/\sigma_v$	1.77	1.99

Figure 8 Part plan of Tandsi mine depicting locations of testing sites for in situ stress measurements by hydraulic fracturing method at 13D and 26D/8X sites (see online version for colours)



At Tandsi Mine roof falls were observed in level numbers 12–15 lying between 11 and 12 dip. In situ stress measurement at the junction of 14th level and 13th dip revealed the direction of major principal stress as N 30 W. This location is in close proximity to a fault that gives perturbed stress. The direction of the level gallery was found to be along the direction of the major principal stress.

Under these circumstances, the dip drive is bound to be overstressed leading to roof failures (Figure 8). The perturbed stress also has a magnitude higher than the unperturbed regional stress (Table 4).

The other investigation site at 26 Dip in Tandsi mine was far away from geological discontinuity such as fault. The measurement at this location shows unperturbed direction

of the stress and its magnitude. In this zone, no roof fall was observed as the direction of the dip gallery is aligned at 40 to the direction of the major principal stress.

3.2 Thesgora mine

In situ stress was measured using hydraulic fracturing at two sites, namely 20L/3D and 19L/3D. The site 20L/3D was chosen far away from any site of fault and another site 19L/3D close to a fault. A summary of the stress regime in Thesgora mine as revealed by hydraulic fracturing at the aforesaid two sites is given in Table 5.

Table 5 Stress regime indicated by the testing at two sites in Thesgora mine

	Regional stress at the first site i.e., 20L/3D having 212 m thick overburden	*Perturbed stress at the second site i.e., 19L/3D having 187 m thick overburden*
Maximum horizontal principal stress (σ_H), MPa	6.04 ± 0.0679	6.00 ± 0.0162
Direction of maximum horizontal principal stress (σ_H), degree	N 100° (N 80° W)	N 170° (N 10° W)
Minimum horizontal principal stress (σ_h), MPa	4.03 ± 0.453	2.82 ± 0.0108
Vertical stress (σ_v), MPa (with density of rock = 2.2 g/cm^3)	4.66	4.11
Ratio, $K = \sigma_H/\sigma_v$	1.29	1.46

The stress regimes evaluated at the two sites in Thesgora mine are, respectively, N100 (N80 W) and N170 (N10 W) resulting into a perturbation of the principal horizontal stress direction due to the influence of the fault to the extent of 70 .

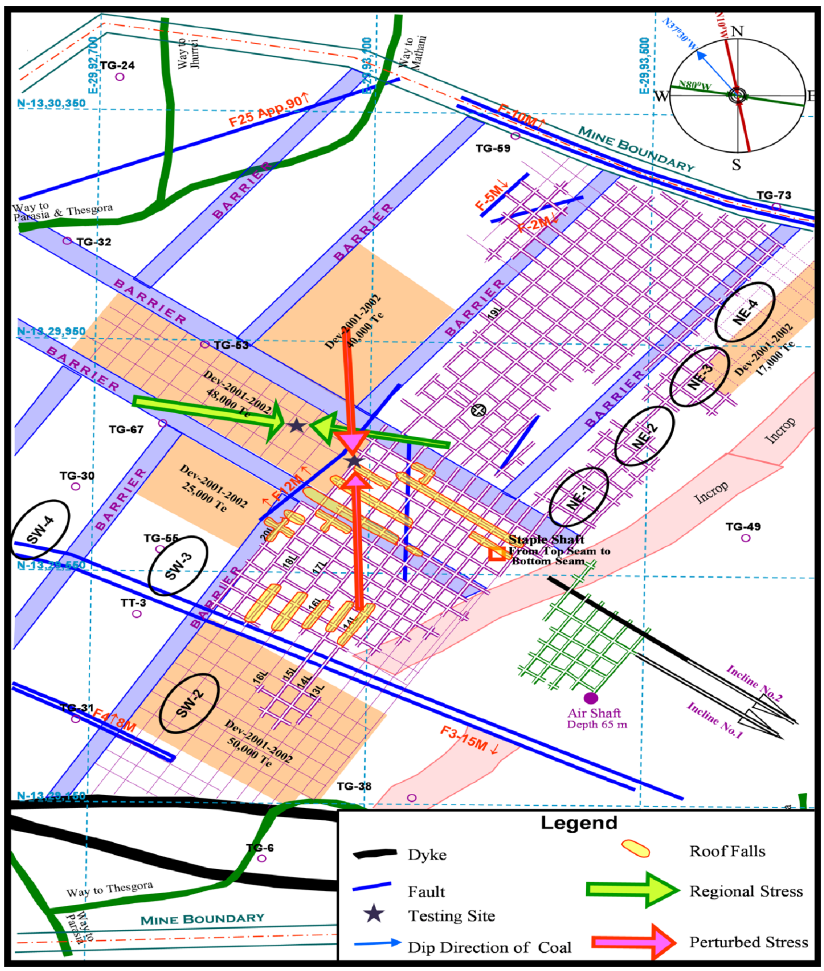
Now, a close look at Figure 9 reveals that the roof fall does not take place in all galleries characterised with similar geo-mining conditions; rather it is localised in certain pockets. A close look of stress perturbation (Figure 9) reveals the problems of roof being associated with the direction of horizontal stress.

In situ stress measurement at the site three dips away from the 20th level revealed the unperturbed stress as it was away from the zone of influence of the fault. Hence, no roof problem was there in this region.

However, major roof problems were marked in between 14th and 19th levels. A close inspection of these areas reveals that they are surrounded by faults on three sides that creates a highly complex stress regime. One of the faults close to 20th level is convex in S–E direction. However, the measured direction of in situ stress at convex part of the fault make an angle of 45 with both level and dip galleries. At this point there was no roof problem. But, in the southeast direction of the fault, direction of the dip galleries approaches orthogonal to the stress direction. The dip galleries in this area have a strong influence of shear stress which is due to interaction of other faults also (Figure 9). The influence of existing discontinuity surfaces on the distribution of maximum shear stress are best understood through experiments on photo-elastic models (after Gzovskii 1971). It can be observed that in case of straight discontinuity oriented at an angle of 45 with

greatest principal stress, there will be a zone created at the middle of the discontinuity characterised with strong reduction of shear stresses, but zone of high shear stress will be created at the two ends of the discontinuity. In case of wavy discontinuity oriented at 45 to the greatest principal stress, zones on the concave side of the discontinuity will remain uninfluenced by discontinuity surface, whereas zones on the concave side will be influenced by strong increase of shear stress. This clearly explains as to why regions beyond 20th level on northwest side of the convex fault do not experience any roof problem.

Figure 9 Part plan of Thesgora mine depicting locations of hydraulic fracturing tests for in situ stress measurements (see online version for colours)

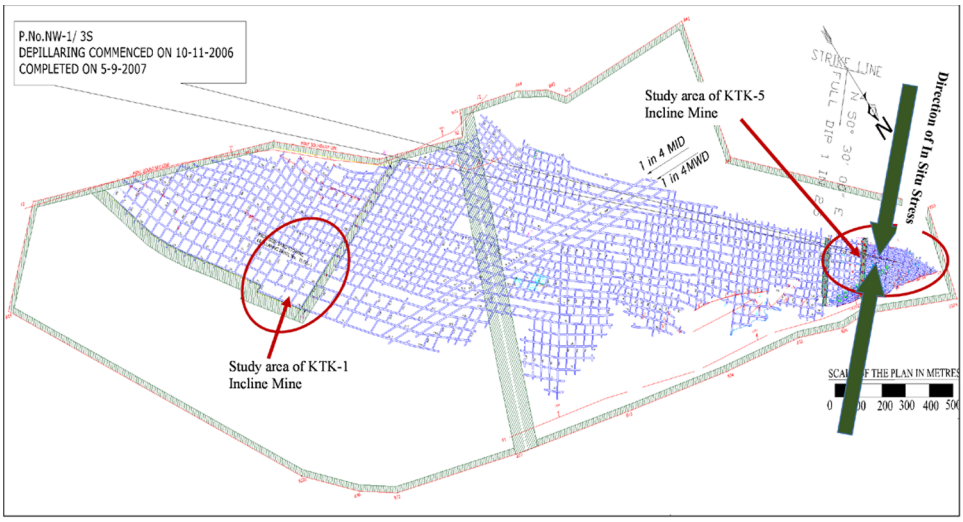


3.3 *KTK-1 and KTK-5 incline mines*

Two mines in Bhupalpalli Area namely, KTK-1 and KTK-5 Incline mines were chosen for validation of the classification. The Kakatiyakhani Number 1 (KTK-1) Incline mine is situated in the Bhupalpalli Area, in Andhra Pradesh. The mine having an area of 7.2 km^2

is located in Sector B of Bhupalpalli Block-1, close to Bhupalpalli village on northwest side of Parkal-Mahadevpur road in the Warangal District of Andhra Pradesh. KTK-1 Incline mine lies between the latitudes 18 26′52′ and 19 28′30* N and longitudes 79 50′00* and 79 52′30* E on Survey of India Topo Sheet No. 56N/15 (Figure 10). Mining through this incline started in the year 1988.

Figure 10 Key plan of KTK-1 and KTK-5 Incline mine showing locations for hydraulic fracturing test for in situ stress measurements (see online version for colours)



The KTK-5 Incline mine (Kakatyakhani Number 5) lies in the vicinity of KTK-1 Incline mine. It falls between latitudes 18 25′00″ and 18 26′30″ N and longitudes 79 50′38″ and 79 52′32″ E on Survey of India Topo Sheet No. 56N/15 (Figure 10). Bhupalpalli is well connected to Warangal by an asphalt road (about 65 km). Kakatiyakhani Incline mines forms a part of the Bhupalpalli Block I of Mulugu coal belt, which starts from Maneru River at the north to Lingala village in south of Warangal district. Bhupalpalli Area forms a part of this coal belt. The Mines in this area are named as Kakatiyakhani and is abbreviated as KTK.

The data of in situ stress measurement as shown in Table 6 is collected from the S&T project report of NIRM. In situ stress measurement was carried out in 17th and 19th Level of KTK-5 Incline mine of Bhupalpalli area and the value of K and direction of the maximum compression were found identical at two sites. These values being identical and unperturbed may be taken as regional stress.

From the results of the measured in situ stress at KTK-5 Incline mine of Bhupalpalli area it is concluded that the ratio of maximum horizontal to vertical stress is 1.79 and its direction is along the true dip of the seam (N 50 E). It is also observed from the physico-mechanical properties of the coal-measure rocks at KTK-1 and KTK-5 Incline mines that the roof rocks are strong and competent as depicted by RMR value. There are no major discontinuities traversing through the mines and no major problem related to stress was observed during drivage of level galleries, which is contrary to accepted norms. Thus, it can be concluded that the directional influence of stress in competent rock mass subject to moderate level of horizontal stress is insignificant (equations (8) and (9)).

Table 6 Regional stress field around the site at 19th level in KTK-5 incline

Regional stress field around the site 19th level in KTK-5 Incline mine with 246 m thick overburden	
Maximum horizontal principal stress (σ_H), MPa	9.52 + 0.76
Direction of maximum horizontal principal stress (σ_H), degree	N 500 E
Minimum horizontal principal stress (σ_h), MPa	3.81 ± 0.30
Vertical Stress (σ_v), MPa (with density of rock = 2.2 g/cm^3)	5.3
Ratio, $K = \sigma_H/\sigma_v$	1.79

4 Conclusion

Failure of any material is not only due to the stress acting on it but it is also reliant on its competence to counteract the stress prior to failure. This competence is an inherent property of all materials and applies to rocks as well. The competence of the exposed rock, i.e., the roof rock, is characterised by RMR in the present paper. In situ stress is inherent in nature which is also beyond the control of the designer. It is the designer's job to negotiate between the in situ stress and the competence of the roof rock in designing an optimal and successful bord and pillar development layout. This paper has attempted to address the re-orientation of bord and pillar development galleries with respect to the direction of in situ stress keeping in mind the competence of the roof rock. The classification of the bord and pillar layout discussed in this paper brings in objectivity with respect to the re-orientation of the galleries.

The classification spells out clearly three categories of mine, namely, Class-A, Class-B and Class-C. According to the classification developed for bord and pillar galleries, the Class-A galleries come under category of mines having highly competent roof rock and subjected to low levels of in situ stress. Such kind of galleries will be stable under adverse direction of mining also indicating that they do not require re-orientation. Class-B galleries come under the category of mines which have moderately strong roof rock and are subjected to moderate levels of in situ stress. Such kind of galleries if reoriented by 45 to the in situ stress direction can remain stable during development. The Class-C category of mines come under the category of mines which have least competent roof rock and are subjected to high horizontal stress. Such kind of galleries will not be stable even if they are re-oriented. Mining in Class-C galleries require heavy supports even during development.

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Study on mining method of cretaceous coal seam under the aquifer of outcrop area in golden concord coal mine

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Abstract: As coal seam of Golden Concord coal mine occurs in cretaceous, outcrop mining area is threatened by tertiary gravel aquifer. This paper determines the water abundance of outcrop area through pumping test, analyses the physical mechanical properties and composition of cretaceous strata, and also researches the damage characteristics of cretaceous rock by field measurement and similar simulation experiment. The final conclusion includes: 1) the water abundance of tertiary gravel aquifer is weak; 2) the cretaceous rock strata belongs to weak type; 3) the ratio of caving zone and mining height, and the ratio of fracture zone and mining height are very close through field measurement and similar simulation experiment, and using the measured data to predict the 'two zones' height; 4) using different mining methods in different outcrop areas. This research could liberate amount of coal in the outcrop area, and guarantee the continuous safety production of the coal mine.

Keywords: mining method; cretaceous coal seam; aquifer of outcrop area; pumping test; field measurement; similar simulation experiment; overburden failure law; caving zone; fissure zone.

Reference to this paper should be made as follows: Li, J., Xu, Y. and Gu, W. (2015) 'Study on mining method of cretaceous coal seam under the aquifer of outcrop area in golden concord coal mine', *Int. J. Mining and Mineral Engineering*, Vol. 6, No. 3, pp.276–293.

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1 Introduction

Coal resources are rich and distributed in different regions in China, however, the hydrogeological conditions of coal mine are quite complex and the process of mining usually faces threat from different kinds of water bodies (Jiang et al., 2007). According to statistics, the reserves of coal resources under the threat of water amount to more than 25 billion tons, among which nearly 10 billion tons are under the threat of roof water (including surface water, Cenozoic loose aquifer, roof bedrock aquifer, etc.). The northern, northeastern and eastern plain areas, where the shallow coal seams face the mining problems under aquifers, are generally covered by loose aquifers in China (Wang and Ti, 2007; Xu, 2007).

Fully mechanised top-coal caving mining method has many advantages, such as low drivage rate, high efficiency, strong adaptability and tendency to realise high yield, and has made rapid development in China (Yang, 2010). Application of this method in mining under aquifers has good prospects. However, the bigger the mining height is, the severer the overburden damage is. Because the mining height of the fully mechanised top-coal caving is obviously bigger than slice mining, the effect intensity of mining damage and the development rules of overlying rock destruction are also obviously different (Shen and Kong, 2000; Unver and Yasitli, 2006). In general, the height and extent of damage will significantly increase, especially when the fissure caused by mining damage easily spreads to the water body located in the overlying strata, and can result in the increase of the rate of water and sand inrush accidents (Hu and Zhao, 2001; Xu and Liu, 2011).

Previous study on destruction rules of overlying strata is mainly carried out in the mining areas like Anhui, Shandong, Hebei and Henan province where the coal seams exist in Carboniferous-Permian strata (Fan et al., 2009; Jin, 2009; Lan and Lou, 2010; Li, 2001). However, inner Mongolia has become the key area of China where amounts of coal will be produced and many coal mines will be found at present. In some areas of inner Mongolia, such as Golden Concord coal mine, the coal seam is located in the cretaceous strata where the bedrock shows distinct difference in physical and mechanical properties through comparing the Carboniferous-Permian strata. The coal mining is also under the threat of overlying aquifers. Therefore, only by conducting thorough research on fully mechanised mining rules of cretaceous strata under aquifer can the amount of coal be released and the coal mine achieve safe and efficient production.

2 Mine profile

Golden Concord coal mine is located in Duolun county – the middle area of Inner Mongolia, with a design production capacity of 1.2 million t/a. No.7 is the main minable seam which is buried in the cretaceous strata – 125.70 to 224.49 m depth. The coal seam

thickness varies from 1.01 m to 55.07 m, average 15.80 m, and the average inclination angle is 13°. The No.7 coal seam is divided into two layers – the top layer is 7^{-1} and the bottom layer is 7^{-2}, by the middle dirt band – packsand and siltstone. The average thickness of top layer is 8 m, and the bottom layer is 7 m. Currently only the top layer is mined with the method of fully mechanised top-coal caving mining. The thickness of the mining faces overburden rock which is located at the outcrop area of the first mining area in the north wing of the mine field is thinner than other areas, and the mining faces are under the threat of the tertiary sand gravel aquifer of unconsolidated formation, which can result in amount of coal lost. To further ascertain the hydrogeological situation of the outcrop area, pumping test on 09-S1 and 09-S2 hydrogeological drillings was carried out in the supplement exploration of hydrology. The pumping tests results are shown in Table 1. From Table 1 it can be known that the units inflow of tertiary gravel aquifer is 0.0146 ~ 0.0517 L/s·m, which belongs to weak watery (State Safe Production Supervision Administration, 2009).

Table 1 The pumping test results of hydrological drilling

Borehole no.	*Aquifer*	*Groundwater type*	*Ground elevation* (m)	*Borehole depth* (m)	*Osmotic coefficient* (m/d)	*Units-inflow* (L/s·m)	*Radius of influence* (m)
09-S1	Tertiary gravel aquifer	Pore and fissure confined water	1270.12	171.63	0.0520	0.0146	621
09-S2	Tertiary gravel aquifer	Pore and fissure confined water	1264.96	172.38	0.340	0.0517	677

3 The lithology features of cretaceous strata

3.1 Physical and mechanical properties of rock

Bedrock of Permian-Carboniferous strata has high unidirectional compressive strength, such as 24.70 ~ 24.70 MPa in Zhaogu No.1 coal mine located in Jiaozuo city (Fan et al., 2009) and 92.4 ~ 136.1 MPa in Haizi coal mine located in Huaibei (Xu et al., 2010a), while the unidirectional compressive strength of bedrock of the cretaceous strata in Golden Concord coal mine is only 1.4 ~ 11.6MPa. Comparing with the Carboniferous-Permian coal-bearing strata, due to the short deposition time, the cretaceous coal-bearing strata has some features, for instance, higher softening coefficient, low apparent density, low elasticity modulus and it is very easy to be weathered because of the low degree of consolidation. The physical and mechanical properties of cretaceous strata in Golden Concord coal mine are shown in Table 2.

3.2 Lithology composition

By collating and analysing 27 core logging data (ZK–1, ZK–2, etc.) in this coal field, the overburden rock strata are composed of mud rock and sand rock. The bedrock is divided into four sections which are 0 ~ 20 m, 20 ~ 40 m, 40 ~ 60 m and >60 m from the top of the bedrock to the top of the No. 7 coal seam. The proportions of mud rock are,

respectively, 50%, 49%, 50% and 48% of each section. So mud rock and sand rock account for half each, the overburden rock strata belongs to moderate proportion. However, due to the low strength of cretaceous rock, the roof is classified as weak type (Peng et al., 1989).

Table 2 The comprehensive statistics table of the physical and mechanical properties of cretaceous strata

Index / *Lithology*	*True density* (kg/m^3)	*Apparent density* (kg/m^3)	*Compressive strength* (MPa)	*Softening coefficient*	*Elasticity modulus* (MPa)	*Poisson's ratio*	*Internal friction angle*	*Cohesion* (MPa)
Gritstone			3.7~6.9	0.21~0.43			37°04′~41°38′	0.099~0.61
			4.8	0.32			39°21′	0.35
Packsand	2523~2746	1855~2620	3.8~7.2	0.41~0.48				
	2635	2238	6.8	0.46				
Sandy mudstone	2482	1739	4.9~9.0	0.6			41°11′	0.5
			7					
Mudstone	2305~2601	1625~2200	1.4~11.6	0.24~0.60	3.28~7.44	0.19~0.29	34°41~42°57′	0.4~2.6
	2508	1753	5.8	0.34	5.36	0.24	36°50′	1.9
Carbon mudstone			3.6~7.1					
			5.6					
No.7 coal			2.7~9.1					
			6.2					

Each segment of overburden rock has mud rock with good water-resisting property, of which the dry saturated water absorption rate is 37.2%. The mud rock has strong resistance-to-deformation ability and a series of other specific features such as the easiness to close and compact with initial fissures and mining-induced fissures, which have a strong inhibitory effect on the upward development of mining-induced fracture. At the same time, the bedrock has a strong water-resisting property, which can effectively prevent water and sand located in aquifers of unconsolidated formation from inrushing into working face (Zhang et al., 2009). However, because of the low strength and small hulking coefficient of cretaceous strata in this area, the goaf needs higher rock strata to cave to support the roof, which in return leads to the increase of caving zone height.

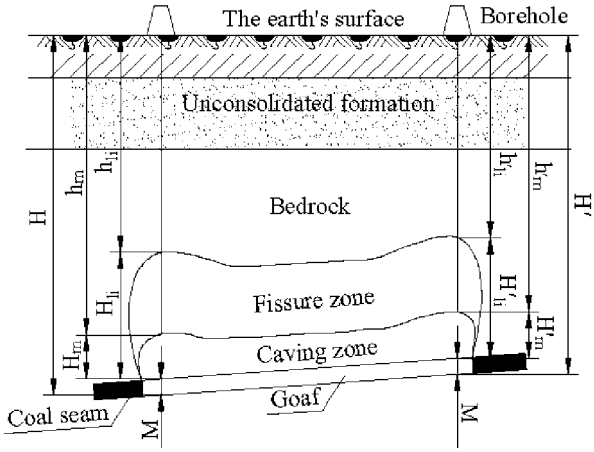
4 The damage characteristics of cretaceous strata

The damage characteristics of overburden rock refer to the movement, deformation and destruction of overlying strata after mining (Qian and Shi, 2003; Singh and Jakeman, 2001). The height of the roof caving and water flowing fractured zone (hereinafter referred to as 'two zones') will directly determine the safety and reliability of mining under water body, and the way to design adequate coal and rock pillars to provide the safety in the outcrop area (Guo et al., 2008). Therefore, accurate prediction of the damage height of overburden rock is the key point to liberate the coal under the water body and guarantee the safety of coal mining.

4.1 The field measurement of the damage height of overburden rock

At present, the heights data from actual measurement of 'two zones' mainly stem from Carboniferous-Permian strata, on the contrary, the actual measurement data of cretaceous are few. Therefore, only by directly on-site measurement can the accurate and reliable data of the damage height of overburden rock be obtained, which will provide reliable basis for safe mining under water body. Because the buried depth of No.7 coal in Golden Concord coal mine is shallow, designing specialised drilling to observe the heights of 'two zones' on the ground has technical feasibility and economic rationality (Wang et al., 2005; Xu et al., 2010b). The diagrammatic sketch of 'two zones' observation is shown in Figure 1.

Figure 1 The diagrammatic sketch of 'two zones' observation



4.1.1 The drilling location

The 1703^{-1} working face which is under the threat of the tertiary gravel aquifer is the first mining face in the coal mine. Because the thickness of the coal and rock pillars is more than 140 m, the working face can be designed to use fully mechanised top-coal caving mining with a total mining thickness of 9.5 m and a length of 120 m. According to the existing data and site conditions, two 'two zones' observation holes (No. 09 – SD1 and 09 – SD2) were assigned, which are located in the 'saddle' peak area of 'two zones' above the 1703^{-1} working face. The sketch map of 'two zones' location is shown in Figure 2. To obtain the maximum height of overlying rock destruction, the working face should have passed the location for a month and with a distance of more than 100 m before the bottom of the holes reaches bedrock interface during drilling. If the construction is delayed, the heights of the 'two zones' may become small because of the compaction of broken rock.

4.1.2 Calculation of 'two zones' height

By using the method of observing the leakage of washing fluid during drilling, the 'two zones' heights of the overburden rock were measured. The observation result of 09-SD1 borehole is shown in Figure 3. Clean water was used to drill, and when the borehole

depth reached to 190 m, the leakage of washing fluid increased obviously. When the depth reached to 195 m, the circulation of washing fluid stopped, no returning water, borehole had slight suction phenomenon and the core had longitudinal cracks (as shown in Figure 4(a) and (b)). By comprehensive analysis, it is considered that the position is the top of water-conductive fissure zone. When the depth of drilling reached to 248 m, the borehole water level cuts down obviously (the depth of water level was more than 100 m), drilling speed was sometimes fast and sometimes slow, and drilling tool shook more intensively, and "falling and sticking drills" phenomenon occurred. Borehole had distinct suction phenomenon, and the core had the following features: high fragmentation degree, low extraction ratio, many stagger cracks and disorganised stratification and angles (as shown in Figure 4(c)). On the basis of the above analysis, it is judged that this position reaches to the caving zone.

Figure 2 The location sketch map of 'two zones' observation holes (see online version for colours)

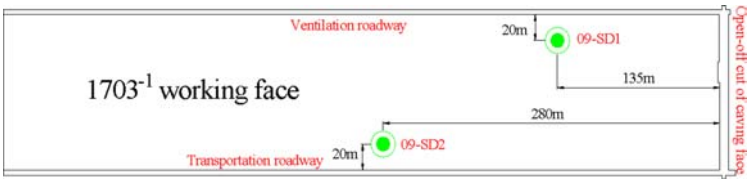
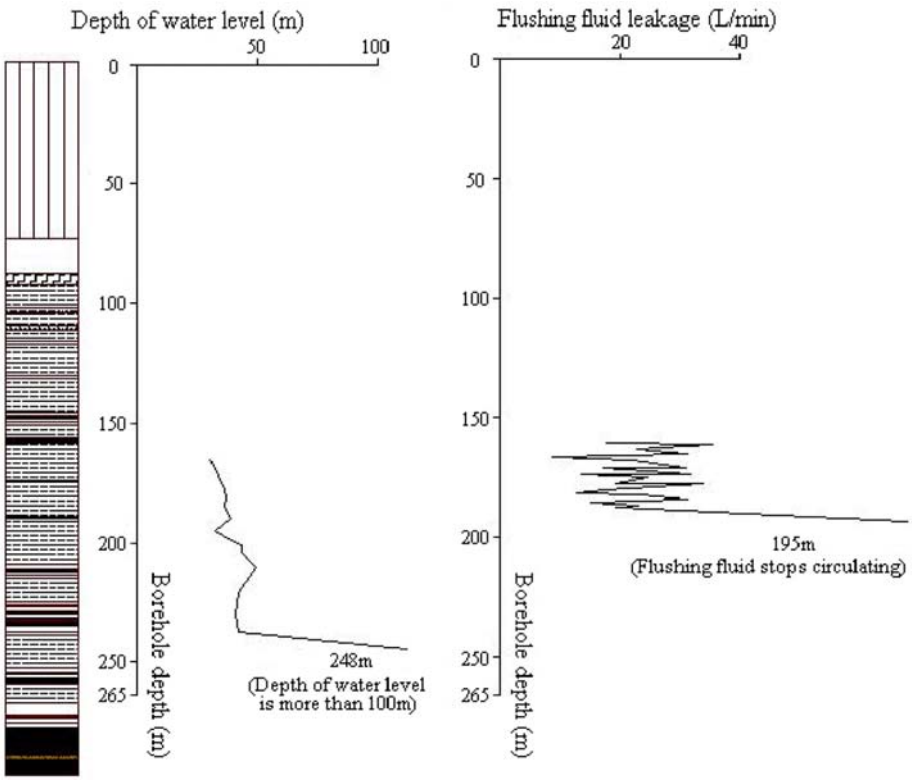
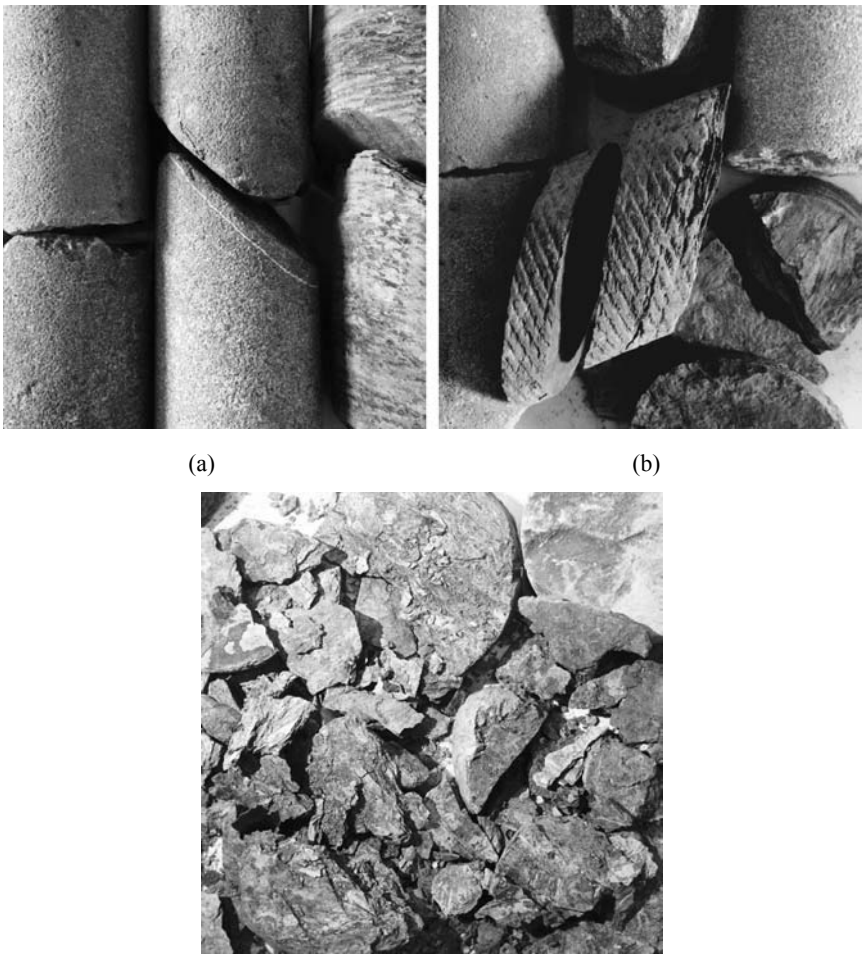


Figure 3 The variation of water level and the flushing fluid leakage of borehole 09-SD1 (see online version for colours)



The observation result of 09-SD2 borehole is shown in Figure 5. Using the same method, when the borehole depth reached to 201 m, the leakage of washing fluid increased obviously, the circulation of washing fluid stopped, no returning water, borehole had slight suction phenomenon and the core had longitudinal cracks. It is considered that the depth is the peak of water-conductive fissure zone. When the depth of drilling reached to 261 m, the depth of borehole water level was more than 100 m, drilling speed was sometimes fast and sometimes slow, drilling tool shook more intensively, and "falling and sticking drills" phenomenon occurred. Borehole had distinct suction phenomenon, and the core had high fragmentation degree, low extraction ratio, many stagger cracks, and disorganised stratification and angles. On the basis of the above analysis, it is regarded as that this position is the peak of the caving zone.

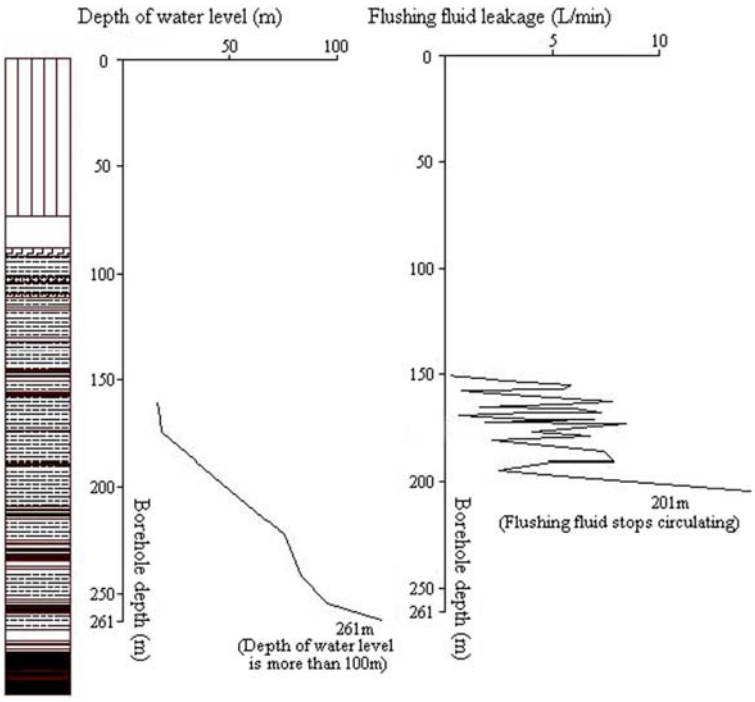
Figure 4 The damage condition of core in borehole 09-SD1: (a) and (b) The core with longitudinal cracks and (c) the core with disorganised stratification and angles



(a) (b)

(c)

Figure 5 The variation of water level and the flushing fluid leakage of borehole 09-SD2 (see online version for colours)



The ratio between the height of caving zone (H_m) and mining height (M) is recorded as 'a' (a = H_m/M), and the ratio between the height of water flowing fractured zone (H_{li}) and mining height (M) is recorded as 'b' (b = H_{li}/M), which are the most common parameters to predict the height of the 'two zones'. The mining height can be calculated by the coal production after a certain time of the working face passing by the 'two zones' drill. According to the situation of 1703^{-1} working face, the average mining height is 9.58 m when mining face reaches to the position of 09-SD1 drilling, and the mining height is 9.09 m when mining face reaches to 09-SD2 drilling. Formulae (1) and (2) which are obtained from Figure 1 are used to determine the actual measurement height of the 'two zones':

$$H_m = H - M - h_m \tag{1}$$

$$H_{li} = H - M - h_{li} \tag{2}$$

where H_m is the height of the caving zone, H_{li} is the height of the fissure zone, H is the vertical depth of the coal seam floor, M is the mining height, h_m is the vertical depth of the peak of the caving zone and h_{li} is the vertical depth of the peak of the fissure zone.

According to the formulae above, the heights of the 'two zones', ratios 'a' and 'b' can be calculated. The results of calculation are shown in Table 3.

Table 3 The heights of 'two zones'

Drilling no.	H (m)	M (m)	h_m (m)	h_{li} (m)	H_m (m)	H_{li} (m)	a	b
09-SD1	311.58	9.58	248	190	54	112	5.60	11.69
09-SD2	321.09	9.09	261	201	51	111	5.61	12.21

From Table 3, it can be found that the heights of 'two zones' in the position of the two drillings are very close, and both 'a' and 'b' are within other statistics scope of the damage height of overburden rock, which indicates that the observed results are reasonable.

4.2 Similar simulation experiment of the damage of overburden rock

Although the 'two zones' observation can determine the damage height of overburden rock, it is short of intuitive understanding of overburden rock fracture morphology (Chen et al., 2012; Liu et al., 2011). Morphological characteristics of strata movement can be acquired through similar material simulation test (Zhang et al., 2008; Cui, 1990).

4.2.1 Scheme of similar simulation experiment

- Scheme and purposes of simulation experiment

The 1703^{-1} working face is taken as the prototype and the thickness of coal is designed as 15.0 m (the upper layer is 8 m, and the lower layer is 7 m). Besides, with considering the boundary effect and the thickness of rock stratum, the length is taken as 200 m, the width is 16 m and the height is 134 m (As shown in Figure 6(a)). The thickness of unconsolidated formation is 181 m, with applying surface force of 4.53MPa on the top of the model. By using long-wall and layered fully mechanised mining method, the destruction rule of overburden rock, migration rule and change law of displacement during mining through simulation technology were researched.

- Ratio and size of simulation

According to the simulation areas, the nature and structure of roof and floor strata in the 1703^{-1} working face as well as the purpose of simulation research, the geometric ratio (a_L), volume-weight ratio (a_γ) and strength ratio (a_σ) of the model are determined:

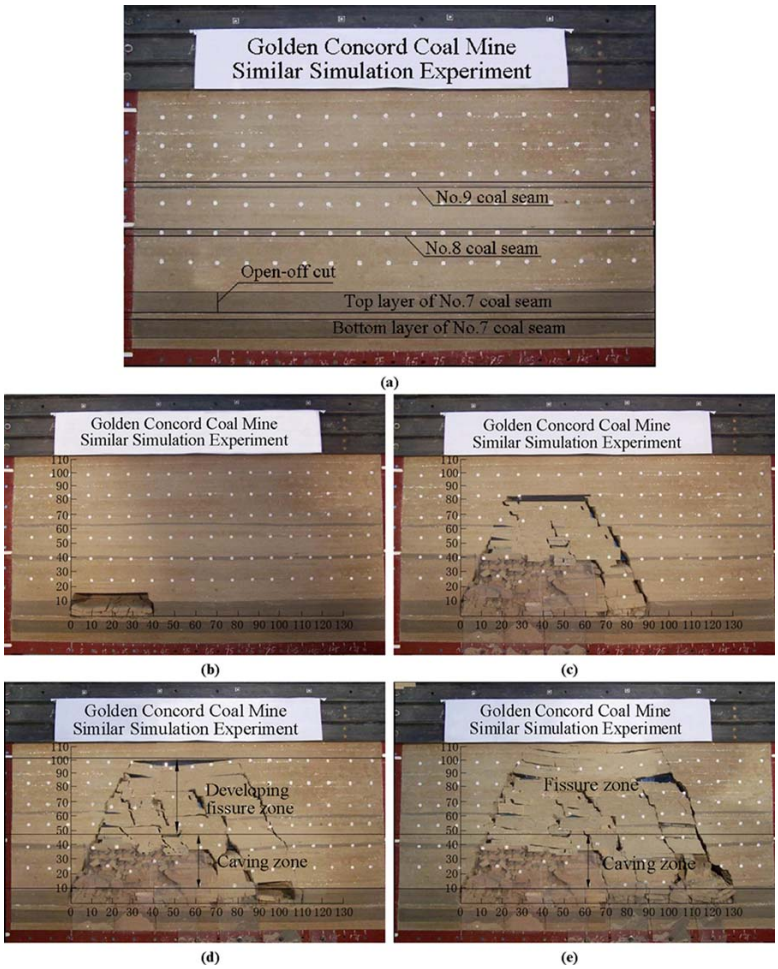
$$a_L = \frac{L_P}{L_M} = 100,\ \ a_\gamma = \frac{\gamma_P}{\gamma_M} = 1.6,\ \ a_\sigma = a_L \cdot a_\gamma = 160 \text{ (Schuring, 1997).}$$

The conversion relation of strength parameters between the prototype and the model can be deduced by dominant similarity criterion, that is

$$[\sigma_c]_M = \frac{L_M}{L_P} \cdot \frac{\gamma_M}{\gamma_P} [\sigma_c]_P = \frac{[\sigma_c]_P}{a_L \cdot a_\gamma} = \frac{[\sigma_c]_P}{a_\sigma}$$

where L_P is geometry size of prototype, L_M is geometry size of model, γ_P is volume-weight of prototype, γ_M is volume-weight of model, $[\sigma_c]_M$ is uniaxial compressive strength of model, $[\sigma_c]_P$ is uniaxial compressive strength of prototype.

Figure 6 The damage variation of roof rock strata with mining: (a) the model before excavation; (b) top layer excavation of 40 m; (c) top layer excavation of 90 m; (d) Top layer excavation of 110 m and (e) top layer excavation of 130 m (see online version for colours)



The uniaxial compressive strength of No.7 coal varies from 2.7 to 7.7 MPa in natural state, on average 6.1 MPa, and the bulk density is 1.5 g/cm^3.

According to the datum above, the thickness (h_M), uniaxial compressive strength ($[\sigma_c]_M$) and bulk density (γ_M) of the upper No.7 coal in the model can be calculated:

$$h_M = \frac{h_p}{a_L} = \frac{8}{100} = 0.08 \text{ m} = 8 \text{ cm} \quad [\sigma_c]_M = \frac{[\sigma_c]_P}{a_\sigma} = \frac{6.1}{160} = 0.038 \text{ MPa}$$

$$\gamma_M = \frac{\gamma_P}{a_\gamma} = \frac{1.5}{1.6} = 0.94 \text{ g/cm}^3$$

In a similar way, the similar geometric, physical and mechanical parameters of other rock strata in the model can be calculated. The results are shown in Table 4.

Table 4 The physical and mechanical parameters of rock strata

Thickness of strata (cm)	*Gross thickness* (cm)	*Lithology*	*Compressive strength* (MPa)	*Volume-weight* (g/cm^3)	*Internal friction angle* (°)
1	182	Coal 12	0.038	0.94	40
16.5	198.5	Siltstone	0.063	1.31	34
0.5	199	Coal 11	0.038	0.94	33
14	213	Siltstone	0.063	1.31	34
1	214	Coal 10	0.038	0.94	40
20	234	Siltstone	0.063	1.31	34
1	235	Mud rock	0.05	1.19	33
3	238	Siltstone	0.063	1.31	34
1	259	Coal 9	0.038	0.94	40
2	240	Gritstone	0.03	1.13	40
11	251	Siltstone	0.063	1.31	34
2	253	Packsand	0.073	1.25	40
5	258	Siltstone	0.063	1.31	34
2	261	Mud rock	0.05	1.13	34
2	273	Coal 8	0.038	0.94	33
2	263	Packsand	0.073	1.19	33
4	267	Siltstone	0.063	1.31	34
2	269	Mud rock	0.05	1.13	33
2	271	Siltstone	0.063	1.31	34
1	274	Carbonaceous mudrock	0.044	1.19	34
8	282	Siltstone	0.063	1.31	34
5	287	Packsand	0.073	1.25	40
1	288	Mud rock	0.05	1.13	33
2	290	Packsand	0.073	1.19	40
2	292	Carbonaceous mudrock	0.044	1.25	34
8	300	Coal 7	0.038	0.94	40
1	301	Packsand	0.073	1.19	34
1	302	Siltstone	0.063	1.31	34
7	309	Coal 7	0.038	0.94	40
6	315	Siltstone	0.063	1.31	34

4.2.2 Damage characteristics of overburden rock

Mining starts from open-off cut of top layer which is 30 cm away from the model boundary. Mining length of the model is 5 cm each time. Through careful observation and experiment phenomenon analysis, the following information can be obtained:

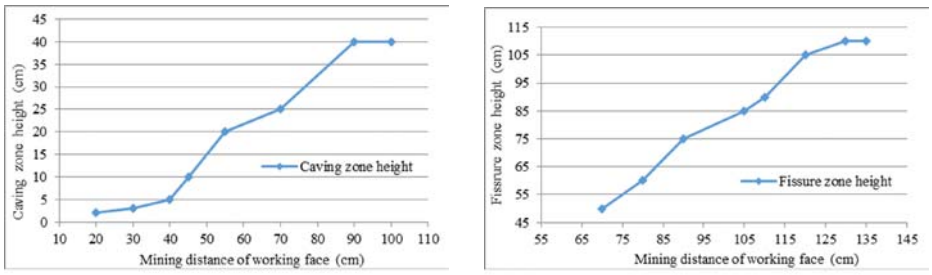
With the increase of mining length, damage occurs in roof strata which develops upward gradually, the height of caving zone and fissure zone increases and turns stable finally (as shown in Figure 6). When the roof strata collapses the first time, the mining length is 40 cm (equivalent to 40 m of actual field) which is the initial caving step distance of immediate roof, and the height of caving zone is about 20 cm (equivalent to 20 m of actual field) (as shown in Figure 6(b)). Owing to the large mining height, continual collapse occurs in roof strata with the increase of mining length instead of forming self-bearing structure. When excavating length is 90 cm (equivalent to 90 m of actual field), the damage height of roof strata is 80 cm, and the height of caving zone is about 40 cm (equivalent to 40 m of actual field) (as shown in Figure 6(c)). The height of fissure zone keeps increasing with continual mining. When mining length reaches 110 cm (equivalent to 110 m of actual field), delamination crack occurs in roof strata and the damage height of roof strata reaches to 100 cm (equivalent to 100 m of actual field) (as shown in Figure 6(d)), but the height of caving zone no longer has obvious change. When mining distance reaches to 130 cm (equivalent to 130 m of actual field), the bedrock is separated from clay layer (water-resisting layer), and at last the height of fissure zone is 110 cm (as shown in Figure 6(e)). From the mining process, it can be obtained that the height of delamination increases slowly at the beginning. However, with continual mining, the height decreases gradually and tends to stabilise finally. In addition, the middle-upper part of the fractured zone appears flat and the height of fractured zone has few obvious changes.

4.2.3 Analysis of similar simulation experiment results

By observing the experimental phenomena, the fracture of the overlying strata which bases on bed separation fracture and expands gradually appears continuous development regularly with the upper coal excavation. The fracture reaches the bottom of the clay layer when the excavation finishes. However, due to the obstruction of clay layer, bed separation appears between clay and rock layer when rock layer deformation occurs slowly. The fracture presents trapezoid distribution from bottom to top. Clay has expansibility by absorbing water and will suffer small deformation when mining height is small. Although slight fracture appears in clay layer, it still can prevent fracture development effectively as long as it has a certain thickness (generally not less than 3 m) and mining design is reasonable (the length of working face is not more than 130 m).

The variation of caving zone height and fissure zone height with the increase of mining distance is shown in Figure 7. It indicates that the caving zone height increases gradually with the increase of mining distance. When mining distance reaches to 90 cm (equivalent to 90 m of actual field), the caving zone height is in a steady state, essentially unchanged, and then reaches to the peak of 40 cm (equal to 40 m of actual field) (as shown in Figure 7(a)), the ratio ('a') between caving zone and mining height is 5. It also demonstrates that the bedrock destruction is in a non-fully development condition and the fracture height increases gradually before mining distance reaches 130 cm (equal to 130 m of actual field). When it reaches 130 cm, bedrock destruction develops fully and the fracture runs through the whole rock strata, the fissure zone height is 110 cm (equivalent to 110 m of actual field) (as shown in Figure 7(b)), and the ratio ('b') between fracture zone and mining height is 13.75.

Figure 7 The variation of caving zone height (a) and fissure zone height (b) with the increase of mining distance (see online version for colours)



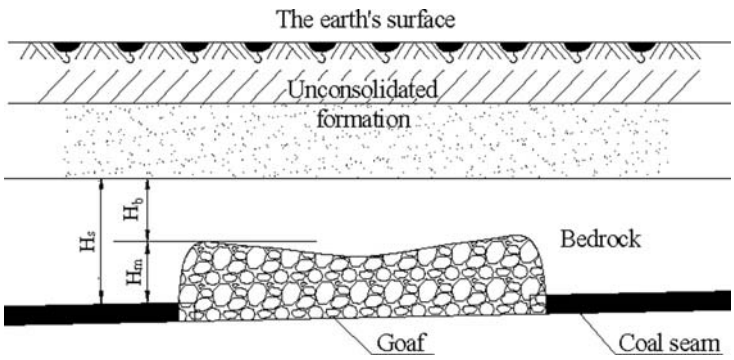
(a) (b)

The height of the 'two zones' is close between the above two methods, 'a' varies from 5.00 to 5.61 and 'b' varies from 11.69 to 13.75. A similar simulation experiment demonstrates the measured data are accurate and reliable. Because the measured data can truly reflect the characteristics of overburden destruction, the larger measured datum 5.61 and 12.21 (from 09 – SD2 hole) is chosen as numerical basis to set up safety coal and rock pillars in the upper layer of No. 7 coal fully mechanised mining.

5 Setting up safety coal and rock pillars in outcrop area

The tertiary gravel layer in outcrop area belongs to weak aquifer, which meets the requirement of setting up sand control safety coal and rock pillar. It means the actual thickness of rock pillar is not less than the minimum thickness of sand control safety coal and rock pillar while mining the top layer of No. 7 coal seam with fully mechanised top-coal caving mining (National Coal Industrial Bureau, 2000; Wu et al., 2013). The design of sand control safety coal and rock pillar is shown in Figure 8.

Figure 8 The design of sand control safety coal and rock pillar



Setting up the thickness of sand control safety coal and rock pillar is according to the following formula:

$$H_s = H_m + H_b = M \times a + H_b \tag{3}$$

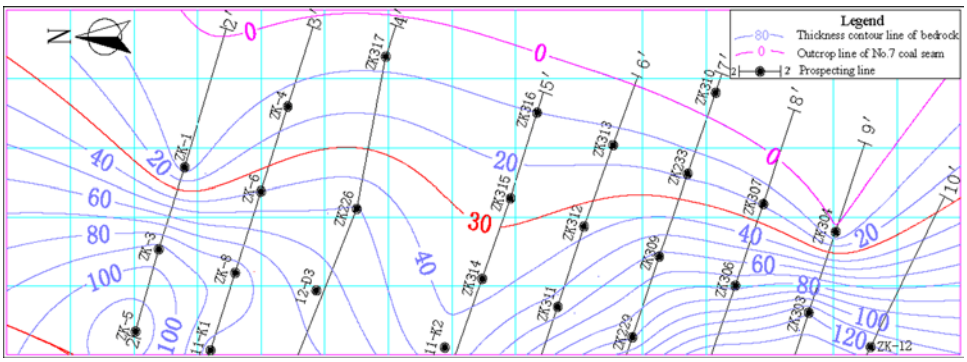
where H_s is the height of sand control safety coal and rock pillar; H_m is the height of caving zone; H_b is the thickness of protective layer (the selection method is shown in Table 5, selecting 2.8M) (Li, 2001; Xu, 2005; Wu et al., 2013); M is mining height; a is the ratio between caving height and mining height (selecting 5.61).

Table 5 Thickness of protective layer with fully-mechanised caving mining

Types of safety coal and rock pillar	*Types of roof*	*Aquifer*	*Negative error rate*	*Conversion mining height*	*Safety factor*	*Thickness of protective layer*
Water control	Intermediate-strength	Medium	–7.73	0.97M	2M	3M
		Strong	–7.73	0.97M	3M	4M
	Weak roof	Medium	–5.20	0.68M	2M	2.7M
		Strong	–5.20	0.68M	3M	3.7M
Sand control	Intermediate-strength	Weak	–1.55	0.08M	2M	2.1M
	Weak roof	Weak	–13.38	0.83M	2M	2.8M

On the basis of tidying the materials of 27 boreholes (11-k1, 11-k2, et al), *Suffer* software is used to draw the thickness contour map of bedrock. The thickness contour map of outcrop area bedrock in Golden Concord coal mine is shown in Figure 9 and the thickness of the bedrock in every borehole is shown in Table 6.

Figure 9 The bedrock thickness contour map of outcrop area (see online version for colours)



From Figure 9 it can be seen that the thickness of outcrop area bedrock is 0 ~ 120 m. *Preliminary Design* stipulate No. 7 coal of the area where the thickness bedrock less than 30 m cannot be mined. When the thickness of bedrock is from 30 to 60 m, the safe mining thickness can be calculated on the basis of the minimum value of 30 m.

$$H_s = 5.61\text{M} + 2.8\text{M} = 8.41\text{M} \le 30 \text{ m}.$$

Table 6 Thickness of bedrock in boreholes

Borehole No.	*Coordinate*			*Thickness of bedrock* (m)	*Borehole No.*	*Coordinate*			*Thickness of bedrock* (m)
	X (m)	*Y* (m)	*Z* (m)			*X* (m)	*Y* (m)	*Z* (m)	
11-K1	4679759.79	39456212.12	1250.19	75.33	ZK303	4677878.23	39456322.20	1260.77	119.34
11-K2	4679020.87	39456220.50	1273.95	41.00	ZK304	4677793.67	39456556.92	1259.38	0.78
12-D3	4679428.32	39456383.95	1269.30	64.30	ZK306	4678109.10	39456400.65	1262.30	76.70
ZK-1	4679841.29	39456742.88	1264.25	5.24	ZK307	4678020.94	39456637.91	1260.22	21.07
ZK-3	4679922.54	39456506.19	1255.93	94.53	ZK309	4678348.38	39456485.24	1262.58	62.81
ZK-4	4679516.79	39456919.83	1276.03	15.28	ZK310	4678170.91	39456959.98	1258.41	No coal
ZK-5	4679995.79	39456267.35	1257.41	119.37	ZK311	4678667.94	39456338.74	1266.12	54.42
ZK-6	4679601.29	39456674.13	1271.19	40.12	ZK312	4678585.14	39456572.81	1263.75	38.23
ZK-8	4679682.39	39456437.75	1264.94	85.98	ZK313	4678493.39	39456807.44	1263.59	14.76
ZK-12	4677683.40	39456222.12	1265.43	120.93	ZK314	4678905.48	39456420.61	1274.41	35.04
ZK226	4679299.78	39456623.87	1285.03	50.75	ZK315	4678816.43	39456654.30	1280.45	26.33
ZK229	4678433.06	39456250.44	1264.68	88.51	ZK316	4678732.73	39456901.80	1280.57	10.08
ZK233	4678259.06	39456724.10	1260.53	17.67	ZK317	4679208.48	39457065.06	1282.00	6.91

Then, M ≤ 3.5 m, so the maximum thickness of the safety mining is 3.5 m, and fully mechanised coal mining is the chosen technology in this area.

When the thickness of bedrock varies from 60 m to 90 m, the thickness of safety mining will be calculated on the basis of the minimum value of 60 m.

$$H_s = 5.61\text{M} + 2.8\text{M} = 8.41\text{M} \leq 60 \text{ m}.$$

Then M ≤ 7.1 m, so the maximum thickness of the safety mining is 7.1 m, and the fully mechanised top-coal caving mining is chosen.

When the thickness of bedrock varies from 90 to 120 m, the thickness of safety mining will be calculated on the basis of the minimum value of 60 m.

$$H_s = 5.61\text{M} + 2.8\text{M} = 8.41\text{M} \leq 90 \text{ m}$$

Then, M ≤ 10.7 m, so the maximum thickness of the safety mining is 10.7 m, and fully mechanised top-coal caving mining is chosen.

6 Conclusions

- Through hydrological supplementary exploration and pumping test in the outcrop area, it can be obtained that the units-inflow of tertiary gravel aquifer is 0.0146 ~ 0.0517 L/s·m, which belongs to weak watery.
- Through the analysis of physical and mechanical properties and lithology composition of the cretaceous strata, and comparing with the Permian-Carboniferous coal-bearing strata, it could be concluded that the cretaceous strata belongs to weak type.
- Through field measurement and similar simulation test, this paper analyses the damage characteristics of overburden rock in cretaceous strata. it could be concluded that the heights of the 'two zones' are close between the two methods, and the larger measured datum 5.61 and 12.21 (from 09 – SD2 hole) are chosen as numerical basis to set up safety coal and rock pillars in the upper layer of No.7 coal fully mechanised mining.
- The outcrop area of Golden Concord coal mine meets the requirement of setting up sand control safety coal and rock pillars. When the thickness of bedrock varies from 30 m to 60 m, the maximum thickness of the safety mining is 3.5 m, and fully mechanised coal mining technology is chosen in this area; when the thickness of bedrock varies from 60 m to 90 m, the maximum thickness of the safety mining is 7.1 m, and the fully mechanised top-coal caving mining is chosen; when the thickness of bedrock varies from 90 to 120 m, the maximum thickness of the safety mining is 10.7 m, and the fully mechanised top-coal caving mining is chosen, which can liberate amount of coal in the outcrop area and guarantee the continuous safety production of the coal mine as well.

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