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Life Cycle Management - A Growing Trend

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ABSTRACT: There has been a growing trend in the Mining Industry to opt for Life Cycle Management programs with major suppliers of equipment. This concept has grown from simple "parts supply contracts" to "full maintenance contracts" with provision of parts and labor, along with guarantees of machine availabilities and performance, in return the supplier is paid a cost per operating hour or cost per unit produced. The concept goes beyond that of the traditional "customer - supplier" relationship, risk sharing promotes the development of a partnership with both parties striving towards a common goal - lowest cost per unit produced. The trend of privatization in many countries provides an ideal situation for the implementation of LCM's, especially as the issue of "global player" accompanies privatization. This paper presents a synopsis of this growing trend of Life Cycle Management programs and highlights the mutual benefits derived from such relationships. The history of Life Cycle Management goes back quite a way, and not always in the form we currently know. The paper starts by outlining some of the steps taken in the past by the mining industry and it's suppliers to generate a workable solution in managing equipment throughout the lifecycle - moving from a situation of the OEM product being equipment and parts, to actually providing a something completely different - cost per unit of ore produced at the mine site, but more specifically - as a partner. Starting at the mine site, issues such as risk and the associated cost are discussed, guarantees offered by suppliers as well as the issue of skills availability and utilization. Optimal use of personnel as well as accessibility to maintenance practices utilized throughout the world is a key to the success of a full LCM contract. Procurement of parts, including non-OEM components can, in many cases be provided competitively by the OEM, especially with the capability to bundle everything into a LCM. The impact on infrastructure and associated costs is another key element and is also discussed. The discussion then moves to the provider of the LCM, usually il is the OEM. There arc obvious certain benefits to the OEM and its greater customer base and these will be expanded upon. Critical items such as the impact of forecasting, manufacturing lead times, inventory cost and control is all discussed. The importance of machine and major component databases is highlighted from an OEM as well as a mine site perspective. Finally, hurdles to the actual implementation of a LCM contract are discussed, liming, how to fit in with existing operations and coping with the threat of redundancy are just some of the elements that should be considered. There are limitations with implementation, especially with existing mines and these are discussed.

I INTRODUCTION

Our customers - want to be producing at the lowest cost and maximizing their profits for their shareholders. As suppliers - we want to be lowering our customer's cost and maximizing our profits for our shareholders.

Is this a dichotomy of objectives? We seem to be striving towards opposing goals, how can we both lower our costs and at the same time maximize our profits when the objective of one is achieved at the expense of the other? Or is it?

In order to produce cost effectively, the mine needs:

- Reliable equipment through
- Service and Support for that equipment
- Parts availability with realistic parts pricing

Traditionally the supplier would do exactly that:

- Design and manufacture reliable equipment
- Provide service and support on request
- Have parts available at a warehouse, ready for the mine to order

This may work if your mine is situated close to the center where the warehouse and service or support personnel are situated. But as we know, mines are generally not situated close to main centers and in many instances, have had to create their own communities and support infrastructures in order to operate.

The remote location of these mines resulted in the necessity for high inventory levels in order to keep the equipment operating with a high level of availability. These high levels of inventory had corresponding high carrying costs, and the mine was carrying inventory for ALL equipment. The alternative was to carry low inventory at a lower cost, but then there was the potential for associated drops in equipment availability, with resultant losses of production.

Adding to this problem is the decision as to what to actually carry in inventory, and in order not to get "caught short" the operations carried many unnecessary parts, parts that would not necessarily be needed at short notice. The extra cost of these parts just added to the cost.

A major contributor to the above was that the operation was taking all the risk with the supplier taking very little risk - other than that of warranty on parts. How best could the situation be improved? This saw the introduction of risk sharing.

2 RISK

What are the risks that we face on a day to day basis? From the mine perspective they are:

- Will my equipment be available to operate?
- Will I have the parts to repair it in the case of breakdown?
- Am I maintaining my equipment properly in order to minimize downtime?
- Are my operators making optimal use of the equipment?

From the supplier perspective they are:

- Is my equipment performing reliably?
- Are proper maintenance practices being followed to ensure ongoing reliability?
- Are my parts going onto the equipment to ensure optimal reliability?

Sharing can minimize the risks on both sides.

The operations have certain competencies, but their specialty is that of exploration and mining. Mineral exploration, mine planning, extracting and processing the ore and selling the resultant product is what they are good at.

Equipment manufacturers on the other hand design and manufacture equipment. They study component lives and try and build optimal life into these components. They also provide the service and support, the training of maintenance personnel. Does it not make sense for them to play a larger role in the performance of their equipment?

3 LIFE CYCLE MANAGEMENT (LCM)

Life Cycle Management is a very loose term, but as equipment manufacturers, we wish to support that equipment "from the cradle to the grave". That in turn generates parts and machine sales, which generates margin dollars, from which we take development dollars to improve the performance and reliability of our equipment for our customers. So as manufacturers we like to be directly involved in the support of our equipment.

Many other terms are applied to the principles of Life Cycle Management:

- R&M Contract - Repair and Maintenance Contract
- Parts supply contract
- Service contract
- MARC - Maintenance and Repair Contract

They all have one common goal, to increase the involvement of the supplier in the support of their equipment with resultant reliability and cost benefits.

The problem lies in that there is a reluctance to give up that control by the operation. A reluctance through lack of trust'? A reluctance through perceived loss of control?

Early contracts started with a small amount of risk sharing, in the form of "parts supply" contracts. These contracts were set up to guarantee parts availability for agreed on prices, in return, the operation guaranteed 100% usage of the parts in the agreement. This contract was somewhat limiting, as they were usually limited to the faster moving parts. Contracts did develop to include slower moving items.

Contracts grew to include availability guarantees by the supplier. This increased the risk on the part of the supplier, and at the same time lowered the risk from the operational perspective. But at the same time, the operation was better able to plan and it benefited while the OEM benefited from the increased business.

Some contracts involved just the supply of "Service", where a representative is placed - either in a consultative capacity or to perform a more "hands on" role. This minimized the need for certain training at the mine. OEM personnel along with guaranteed availabilities contributed towards a more efficient and cost effective operation.

Contracts further grew to include supply of parts at a guaranteed cost per hour, or even supply of parts on a flat cost per hour basis, where the mine pays the supplier a cost per hour, and the supplier provides the parts.

Today we have some contracts where we provide the entire support infrastructure for our equipment, parts supply, maintenance personnel, warehouse management, availability guarantees, support logistics for our personnel - transportation within the mine and even service vehicles for lubrication and maintenance purposes.

4 ECONOMY OF SCALE

This whole business is one of economy of scale. Could we provide the full suite of service options for one shovel in a remote location? The answer is "yes", but it would probably not be cost effective. So one has to be mindful of that.

Each LCM contract has to be judged on it's own merit, and if it's not good for BOTH parties - forget it! The fact that you have only one piece of equipment from one supplier does not necessarily mean that there is no chance of an LCM - in order to achieve economies of scale there is always potential to have an LCM that involves one supplier supporting the equipment of another. This is already being done in a number of areas in the world.

In addition to running LCM's on our own products, we represent other OEM's in various parts of the world, and we operate contracts on that equipment as well, and in some operations, ONLY on that equipment.

5 SKILLS

One thing to bear in mind is that skills are required to operate these contracts. They should not be viewed as a means of transferring a problem of employing, training or retaining the necessary skills. The OEM will also try to utilize local skills as they are the most cost effective. There are instances however where one may have to resort to the use of "imported" skills but these can be expensive and are generally avoided wherever possible.

6 INVENTORY AND RISK MANAGEMENT

The management of inventory is a huge expense, not only lor the skills and personnel to do it, but also for carrying costs. To minimize the carrying cost it is most desirable to have the required parts on a JIT (Just In Time) basis. Of course the logistics of getling parts to a remote site normally presents a problem and in a lot of instances the impact of NOT having a part should be evaluated. To this end the manager of the LCM conducts a risk analysis - "what's the current condition of the machine and what can potentially go wrong?" versus "what do we have m stock and where are the potential shortfalls?" In many instances more critical decisions are made in conjunction with the mine, to have a mutually beneficial support plan.

7 FORECASTING

The mining equipment business - particularly that of Draglines, Shovels and Drills is not a "fast moving consumer goods" business. Our equipment is unique, and in as much as we try to maximize parts commonality, we have to keep up with technological advancements. Our inventory therefore moves slowly in comparison to say, an automobile manufacturer. As part of our responsibility to our customers we have to manufacture and keep the required parts in stock, on the other hand our responsibility to our shareholders dictates that we minimize our stockholding. How does the LCM benefit this aspect of the business?

Since we are right at the user interface and involved with regular maintenance of the equipment with an LCM, we are in a position to determine imminent as well as longer-term parts requirements. These requirements are fed back to our manufacturing facility and we produce only the required parts as identified on the forecast. Accurate forecasting ensures we have no unnecessary parts in stock and we are able to minimize our overheads on inventory, which translates to us being able to offer competitive pricing on our parts and associated contracts.

8 CONCERNS

Perhaps the major concern with operations venturing into an LCM contract is the one of trust. There has to be a mutual trust between the parties involved. Contracts have to benefit both parties, and without the trust element the contract will be doomed to failure.

Since, as mentioned earlier, many of the mine sites have created their own communities, there is a second concern - and it is more social than business, but what will happen to the existing workforce if a contract is entered into? This may cause many operations to balk at entering into a full LCM agreement - in this instance the option could be to go for just a parts supply contract.

Many operations may view the handing over of the maintenance function as a loss of control. The trust element again comes into play here. There should be full confidence in each other for both parties.

"Greenfield" operations are perhaps a little easier to consider for LCM contracts. Communities have not been established and there is less "social" threat. Getting in at the beginning of an operation also ensures that infrastructure is set up according to the needs of the company who will be doing the maintenance.

9 SUMMARY

There can be significant advantages to LCM contiacts if properly implemented. There can be limitations to what agreements are concluded in the various areas of the world, but there should be some kind ot workable solution.

The operations get the benefit of reliable equipment through OEM trained personnel, inventory managed to levels appropriate with the operation and a predictable cost according to forecasts generated.

The OEM gets the benefit of long term parts and servuce income stream and accurate forecasting for loading their factory. Both companies benefit in the longer run.

10THE TREND

One further measure of the success of LCM initiatives is to look at what's already out there, and is there a growing trend? At time of writing this paper, P&H MinePro Services is involved in over 60 contracts involving more than 130 pieces of equipment - these contracts range from supply of parts or labor only, to full LCM contracts. The volume of equipment we are supporting in these contracts has more than doubled since 1999.

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Application of Web-based Knowledge Management Systems in The Mineral Industry

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ABSTRACT: Knowledge Management has been shown to benefit many large corporate companies by using the intellectual capital of the firm. The data, information and knowledge captured in a firm in terms of explicit and tacit knowledge can be used to gain competitive advantage. The use of electronic information systems using corporate intranets and the Internet can help facilitate the management of knowledge in terms of sharing best practise, globalisation, rapid change in technology, downsizing, managing information and communication overload experienced by many companies.

The paper discusses the application of web-based intranets linked to knowledge management systems to provide codification of knowledge and models for converting this knowledge into a corporate resource. A case study to demonstrate the application of multimedia using metaphors, models and narratives to exchange and synthesise knowledge is discussed with reference to the minerals industry.

1 INTRODUCTION

The value of knowledge as a commercial asset within an organisation is well known and the exploitation can give competitive edge and reputation to the organisation and is usually referred to as intellectual capital [Botktn 1999]. The knowledge could be through information collated by organization and expertise of their staff, built up over many year from custom and practise. Knowledge Management (KM) is a process that helps an organisation identify, select, organise, disseminate and transfer important information and expertise that is contained in the organisation usually in unstructured format. The structuring of knowledge in an effective way will assist in problem solving, dynamic learning, strategic planning and decision-making. The application of using this knowledge in a structured way to give commercial value through its reuse throughout an organisation is referred to as a Knowledge Management Systems (KMS) [Turbin et al 2002]. In information systems it is useful to distinguish between knowledge, data and information. Data tends to be simple observations, which are easily captured and consist of facts, measurements and statistics, while information is structured or processed data within a time frame of applicability. Knowledge is information with the most value, that is contextual, relevant and actionable and which is difficult to capture electronically, hard to structure and highly personal and tacit. Over time, information accumulates within an organization and tends to decay, while knowledge evolves and if managed correctly could produce intellectual capital [Davenport 1997, Turbin et al 2002]. The pharmaceutical company Hoffman-La Roche developed a web-based knowledge management system supplied by Skila Inc, called Global Healthcare Intelligence Platform (GHIP) to integrate documents from multiple repositories. This helped to reduce the filing time for Federal Drug Administration (FDA) approval for new drugs reducing the time from 18 months to 90 days [Shand 2000]. Ernest and Young, one of the big five accountancy and financial services companies, implemented a state of the art knowledge management system, which allowed the organisation to globally share leading practises and intelligence, and contributed to the success of the company increasing its US revenues by up to 24% in 1997. Nike one of the world's largest sportswear companies with sales of $9 billion in 1997 have implemented a system to recommend to their customers their shoe size and also allow them to design their own trainers. When Nike are extracting information about their customer shoe size and footwear design they are able to interpret this information to determine the most popular footwear size, colour, style and design etc which results in better sales revenues [http://www.fhwa.dot.gov/km/. www.nike.com]. In the US it is suggested that 55% of the labour force consists of knowledge and information workers and that 60% of the gross domestic products comes form the knowledge and information sectors of finance and publishing. Many US companies now have Chief Knowledge Officers (CKO) and Knowledge Engineers (KE) that are used to design systems through eliciting knowledge from specialists. Knowledge management has an important role within companies particularly in e-business since success is critically dependent on staff knowledge concerning the micro-environment concerning customers, suppliers, intermediates and competitors in order to shape internal processes to deliver customer value [Saunders 2000]. Some management theorists believe that these knowledge assets are as important for competitive advantage and survival, if not more important than physical and financial assets [Laudon and Laudon 2002, Barnes 2002].

2 EXPLICIT AND TACIT KNOWLEDGE

Knowledge in an organisation can be distinguished between two types explicit and tacit knowledge [Nonaka and Takeuchi 1995]. which is outlined as follows:

- Explicit knowledge maybe organised, expressed and communicated relatively easily and transferred through digital means. This has been a traditional aspect of IT and would include databases, manuals, financial reports and articles etc, basically some form of documentation.
- Tacit knowledge is usually in the domain of subjective, cognitive and experimental learning and includes experience, expertise, know-how. intuition, and trade secrets. It can be fuzzy, often complex and unrecorded. An analogy would be the ability to ride a bicycle, snow board or fly a jumbo jet but which would be difficult to explain to someone else via an email.

All organisations have both explicit and tacit knowledge, usually it is the tacit knowledge that tends to give sustained competitive advantage, because it is difficult to replicate by competitors and can produce a strategic knowledge management environment. However explicit knowledge may also produce competitive advantage in the form of patents and/or copyright and although it is in the public domain it is less easy for competitors to use because the originating company exclusively owns it. Some aspects of organisational knowledge cannot be captured easily or codified especially tacit knowledge and the information that organisations finally manage to capture may become outdated as environments change. A key challenge in knowledge management is to make appropriate tacit knowledge explicit and powerful. [Nonaka and Takeuchi 1995, Laudon and Laudon 2002, Lynch 200.3].

A widespread model on knowledge creation after Nonaka and Takeuchi [1995] is outlined in Figure 1.

		To: **Tacit Knowledge**	To: **Explicit Knowledge**
From	**Tacit Knowledge**	**Socialisation** Transferring tacit knowledge through shared experiences, apprenticeships, mentoring relationships, on the job training, 'talking at the water cooler'	**Externalisa tion** Articulating and thereby capturing tacit knowledge through use of metaphors, analogies and models
From	**Explicit Knowledge**	**Internalisation** Converting explicit knowledge into tacit knowledge; learning by doing: studying previously captured explicit knowledge (manuals, documentation) to gain technical know-how	**Combination** Combing existing explicit knowledge through exchange and synthesis into new explicit knowledge

Figure I Four Modes of' Knowledge Conversion [Nonaka and Takeuchi 1995] edge is called socialisation i.e. sharing experiences and observation, imitation and practise. This could include capturing knowledge using videotapes of a story for example in the use of dowels to control floor heave and improve stability of roadways etc. Tacit knowledge to explicit knowledge called externalisation is the sharing of metaphors and models during social interaction. Externalisation is considered do be difficult and costly because of the elusive nature of tacit knowledge. Explicit knowledge to tacit knowledge called internalization, for example after studying manuals or documentation to gain technical know how of a particular equipment an engineer with prior knowledge may combine this new knowledge to diagnose a mal-function that would have resulted in production failure. Explicit knowledge to explicit knowledge, called combination, to exchange and synthesis and to produce new explicit knowledge.

Capturing knowledge by itself is pointless it must be shared within the organisation to have any value. The process of representation of knowledge in a manner that can easily accessed and transferred is referred to as codification [Davenport and Prusak 1998]. While some explicit knowledge may lead itself to codification. Tacit knowledge tends to be subjective and difficult to transmit and some theorist suggests it would be difficult to be copied outside the human mind. A key challenge in developing knowledge management systems is to make appropriate tacit knowledge explicit to gain competitive advantage similar to the Hoffman-La Roche Global Healthcare Intelligence Platform to gain competitive advantage [Shand 2000].

3 STAGES OF IMPLEMENTING KNOWLEDGE MANAGEMENT SYSTEMS

In determining how knowledge management could be applied to an organisation the following four stages can be used as basis of knowledge management assessment and are outlined as follows:

1. Determine the business goals that knowledge management (KM) could assist, thereby providing competitive advantage.
2. Illicit and determine what knowledge management systems are available in and outside the organisation to achieve the business goals
3. Evaluate the knowledge collected to achieving the business goals of the organisation.
4. Codification of explicit and tacit knowledge so that it can be utilised by the company.

3.1. Determining Business Coals

Application portfolio matrix techniques have been widely used in Information System (IS)/lnformation Technology (IT), and are useful to formulate the application of knowledge management and to achieve a consensus on strategy. The matrix or Strategic Grid (SG) is a similar concept to Boston Matrix (BM) for product portfolios analysis [McFarlan 1984, Ward and Peppard 2002].

Figure 2 outlines a typical analysis that could be used to assist in identifying business goals of knowledge management for an organization. SG shows application of knowledge management systems that could contribute to the competitive advantage of a company particularly in strategic (sometimes referred to as star) and high potential areas (problem child) using entrepreneurial or technology driven innovation. In the support (traditional) areas these are typically systems operating as islands of technology whereas the key operational area (factory) forms the backbone of the production type of applications This concept can enable discussion on a consensus of strategy for the organisation in terms of KM positioning and allow management a pertinent number of discrete options for high-level direction. The matrix should be used in conjunction with internal Strengths, Weakness, and external environment based analysis namely Opportunities and Threats (SWOT) analysis to determine the position of current application and to assist future visionary competitive advantage of KM applications to an organisation. Just classifying current and future application into a 2*2 matrix is of little value unless it causes each application to be managed more effectively. In this context it is suggested that the strength and weakness where possible should be related to Critical Success Factors (CSF) analysis to link KM projects to business objectives. An example based on the information depicted in Figure 2 would be a Virtual Reality System (VRS) where KM could be achieved by building a 3-D graphical simulator of a longwall transfer operation. This would assist in achieving a reduction in transfer time particularly in situations where only one longwall is the main method of production.

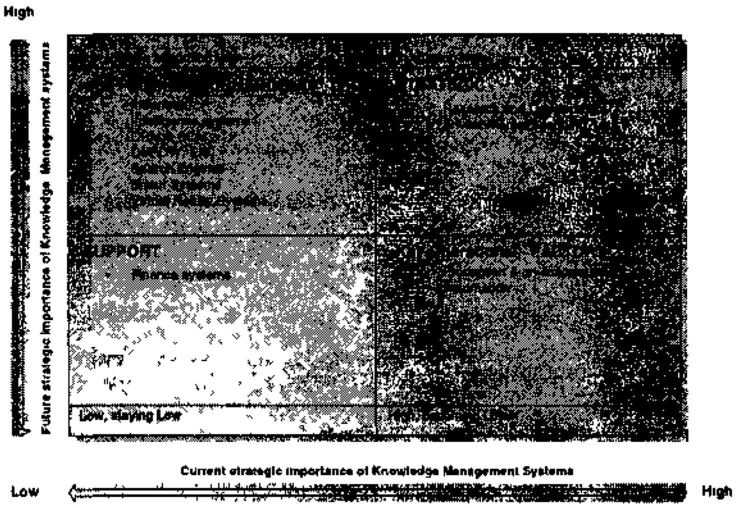

Figure 2 Application Portfolio Matrix for Knowledge Management (KM)

The SWOT in conjunction with CSF analysis on aspects such as performance of transfer, accident prevention etc in utilising knowledge management could be quantified in relation to the business goals of the company. This would be particularly important in situations where only one longwall is the mam method of production. In large organisations the operation can typically be composed of multiple Business Units (BU). in these cases, composite analysis using the matrix maybe needed for the enterprise.

3.2 *Illicit and Determine Knowledge Management Systems*

This would involve analysing and searching for appropriate KM systems available in the market either by developing or customising appropriate software systems [Vince 1999]. In the case of virtual reality systems (VRS) there are various 3-D systems in the market for 3-D layouts including tunnelling, process plants, office layout etc [http://www.web3d.org, http://www.parallelgraphics.com, http://www.virtex.co.uk]. Alternatively you could use Virtual Reality Modelling Language (VRML) enabling Internet browsers to interact with 3-D environments such as World Up and World2 World to develop appropriate systems [http://www.sense8.com/].

3.3 *Evaluating the Knowledge*

This can be achieved from an engineering first principles approach with the purpose of communicating between the technical and non-technical specialist and the needs of the stakeholders of the system. One useful tool is the rich picture developed from a soft system approach which can be used to outline the problem situation in a pictorial form using symbols and pictures with a minimum amount of text [Checkland 1981]. In developing a rich picture it is important to use the terminology and vocabulary applicable to the situation, and also to outline and identify any problems and sources of conflict affecting the knowledge management environment.

3.4 *Codification of Explicit and Tacit Knowledge*

The purpose would be to codify the explicit and tacit knowledge to determine 'best practice' which is the most successful solution or problem solving method that can be developed by an organisation. In addition to improving existing work practices the knowledge can be preserved as 'organisational memory' to train future employees [Laudon and Laudon 2002]. Anecdotal evidence suggests that case studies indicate a combination of 'hard' information (reports and memo etc) and 'soft' information (ideas gossip and opinion) in combination produce the best result. In the case of using a Virtual Reality System to convey knowledge, best practice could be to use a 3-D graphical representation of a mine layout incorporating multimedia features such as voice over narratives and alpha blending of video clips etc. This could then be linked in to corporate intranets to disseminate the knowledge throughout the organisation using e-Leaming.

4 APPLICATION OF KM IN MINING OPERATION

In mining and tunnelling industry, the stability of the .structure surrounding an excavation represent a major challenge to successful operation of the excavated facility. In coal mining operations the stability of roadways is important from both safety and productivity aspects of the operation. Thus implementation of an effective support programme is paramount for the economic viability of the mine. The cost of accident related compensations is on the rise and a significant component of such claim is attributed to falls of roadway roof /rib falls. For example, the Lost Time Injuries (LTD claims related to falls of rocks in 2001-02, in the category of fall, slip and trip mechanism, amounted to 28 % of all claims lodged in NSW, Australia. The underground coal mines use rectangular roadway shapes supported with bolts, and straps. The length of the bolts and their numbers per line of installation across the roadway is dependent on the prevailing geological conditions and manager's support rules. The manager's support rules is usually developed based on the past experience in the mine and in surrounding local mines and beyond. Thus it is abundantly obvious that the manager's rules falls with in the frame of tacit knowledge as being the information based on experience, know how, and trade secret. Publicising the manager's rules to the interested parties, represents a means of communicating this knowledge in an explicit manner. Advancing the process for wider circulation can best be made as a website which can be set up with the relevant information as demonstrated by the following case study.

Case Study: - Support Management Plan Application in an Underground Coal Mine, NSW, Australia

The homepage of the mine's strata management for this panel is shown in Figure 3. An underground coal mine was experiencing poor ground control conditions at the roadways serving a retreating longwall face. The mine (Mine X) was situated some 50 km south of Sydney, and mined coal form 3.3 m thick seam which was situated some 500 m below the ground surface. The longwall face was accessed by two entry headings from either side. Each twin entry heading was intersected by cross cuts at 75 m intervals leaving pillars of 35 m wide. Each road way was 5.5 m wide and 3.3 m high. The roadways and intersections were supported with a combination of bolts, steel straps (W Straps) and wire mesh, both at the roof and ribs. The primary roof supports consisted of six, 2.4 m long bolts per row. The minimum length of W straps installed was 4.6 m. The roadside (rib) reinforcement consisted of W straps, wire mesh and 1.2 m long bolts made from mild steel, fibreglass or plastic.

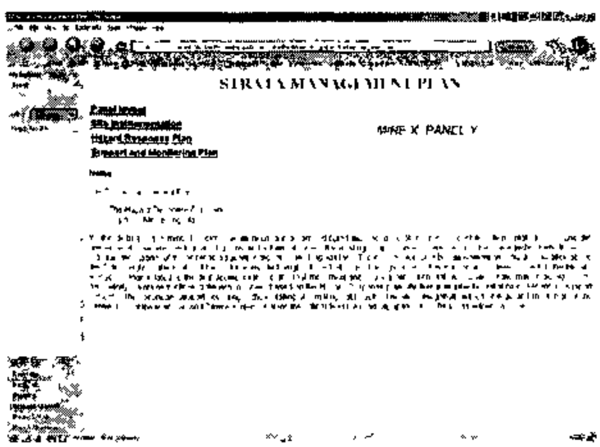

Figure 3 Homepage of Mine X Strata Management Plan (SMP)

During retreat mining of the longwall faces the panel entries were found to deteriorate significantly affecting mining operation, threatening the economic viability of the mine. As a consequence a programme of geotechnical investigation was undertaken to assess the effective support needs of the mine. A roadway section was appropriately selected to conduct the experimental study of the support integrity. The site was situated out bye of a retreating longwall panel. From the mine's SMP homepage (Figure 3), the panel layout can be accessed and displayed (Figure 4).

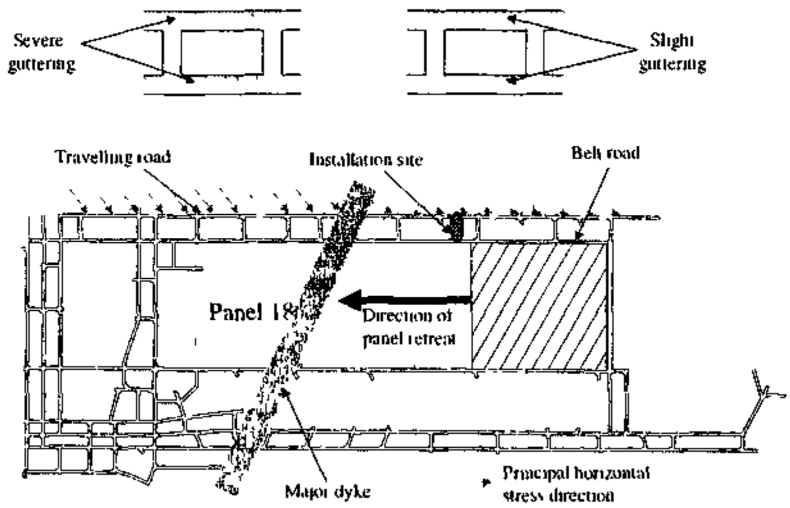

Figure 4 Mine X longwall panel layout

A major dyke intersected both roadways, and there was also a major high horizontal stress acting 45 ° to the direction of mining until it hit the dyke zone, thereafter it swung to run almost parallel to panel direction of the travel road. The magnitude of the high horizontal stress was estimated around 35 MPa, which was equivalent to almost 4 times the vertical stress. The width of the dyke varied between 0.8 m and 1.5 meters. The size and direction of the horizontal stress is shown in Figure 4. The programme of field instrumentation consisted of installing 6 x 2 4m strained gauged bolts (2 rows ot 3 bolts, a meter between rows) with corresponding extensometcrs in between and aligned with the two strain gauged bolts. The instrumentation was carried out in both the travel and belt roads and was close to intersection No 7. From the mine's SMP homepage (Figure 3), the details of the instrumentation of the site can be accessed and displayed (Figure 5).

An important aspect of the Strata Management Plan was that, it followed initially the mine management support rules and preceded with the recommendations, when the ground conditions deteriorated. It should also be emphasised that the support management plan would have the approval of the Mine manager as well as the District Inspector of the Mine, and at times the Check Inspector of the mine representing miners union.

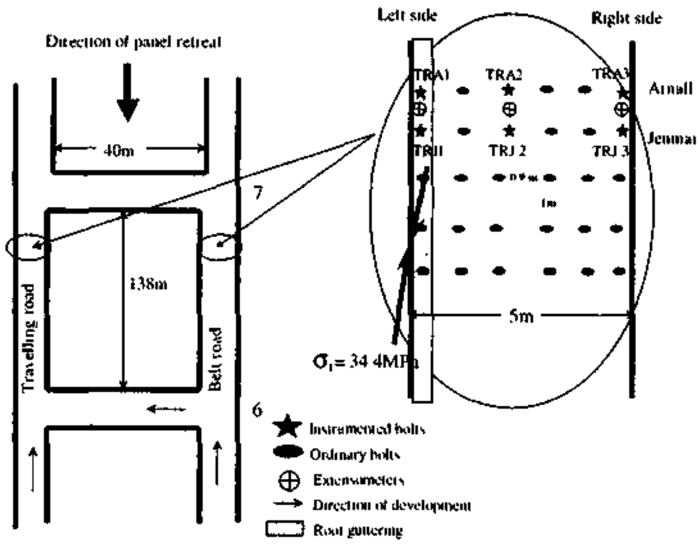

Figure 5. Details of instrumentation of site layout

From the homepage of the mine's SMP website, the following two PDF files SMP plans are hyper linked:

- Hazard response plan (Table 1)
- Support and monitoring plan (Table 2)

5 CONCLUDING REMARKS

The paper outlined the importance of knowledge management in minerals engineering and discussed how it can be used to give competitive advantage. Explicit and tacit knowledge has been discussed indicating the challenges posed in capturing and converting tacit knowledge for use within the organization. The paper has demonstrated how knowledge from the effect of a dyke can be linked to intranets for the benefit of the company via e-Learning. The major difficulty was to develop a culture in an enterprise to codify the knowledge to gain competitive advantage. The old adage that 'knowledge is power' needs to be moderated to benefit from knowledge management. Consideration of the benefits of knowledge management has prompted some visionary organizations to appoint a Chief Knowledge Engineer to champion the pursuit of competitive advantage through knowledge management. The mining industry has generally a good reputation of sharing knowledge particularly in relation to education and safety issues. Some companies have used incentives to capture and disseminate best practice and to ensure that the threats of sharing information were minimized in the organization.

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Table I MINE X - Strata Management Plan

HAZARD RESPONSE PLAN – Maingate 18 Panel
as per "Location Plan" SR-250D

HAZARD INDICATOR	HAZARD RESPONSE TRIGGERS	HAZARD RESPONSE
Dısplacement *Effective for monitoring tools installed within 20m of the advancing face otherwise refer to Displacement Rate triggers*	Total >100mm *if not supported with supplementary cables* *(@ 0 5m horizon or closest anchor within ± 0 2m)*	Site inspection & roadway mapping by area Coordinatoı &/or Mining Engineer to assess any immediate action(s) as required *(eg continue to monitor increase monitoring frequency install additional monitoring tools &/or supplementary support)*
Dısplacement Rate *Require at least 5 data sets to establish roof displacement trends*	Accelerating >2mm/week Linear ≥ 1mm/wk over 12 weeks ≥ 2mm/wk over 6 weeks ≥ 3-5mm/wk over 4 weeks > 5mm/wk over 3 weeks	Implement Strata Management team meeting to review roof monitoring trending & roadway mapping to determine appropriate action(s) as required Time frame to install supplementary support team member responsibilities and monitoring program to be signed off and implemented Set time for next meeting to assess response success
Roadway Mappıng	**Support Deformation:** Inversion of support bearing plates - failure of roof supports Strata Failure - fretting of roof &/or ribs roof sagging &/or guttering - tension fractures structures *(i e faults & dykes)* Roadway Width >5 5m	Deputy shall notify Shift Undermanager for inspection of deteriorating roof and slides UMIC shall be notified Site inspection & roadway mapping by area Coordinator &/or Mining Engineer to assess any immediate action(s) as required *(eg install monitoring tool monitor at increased frequency increase primary support &/or set supplementary support)*

Reaısteıed Mine Manager Date / /

N>

Table 2. MINE X - Strata Management Plan – Support and Monitoring

SUPPORT/MONITORING PLAN Maingate 18		
HEADING	**CUT THROUGH**	**INTERSECTIONS**
PRIMARY SUPPORT 6 x 2 4m roof bolts x 1 in row spacing to be installed with Machine mounted hydraulic bolting rigs (Wide head ABM 20 or I2CM20) 8 x 4.2m roof bolts x lm to be installed when using hand held bolters (CM72) Mesh modules or straps	*1.1.1 Primary Support* 6 x 2.4 roof bolts x Im row spacing to be installed with Machine mounted hydraulic bolting rigs (Wide head ABM 20 or I2CM20) 8 x 2.4m roof bolts x 1 m to be installed when using hand held bolters (CM72) mesh modules or straps	*1.1.2 Primary Support* 6 x 2.4 roof bolts x 1 in row spacing to be installed with Machine mounted hydraulic bolting rigs (Wide head ABM 20 or 12CM20) 8 x 2.4m roof bolts x Im to be installed when using hand held bolters (CM72) mesh modules or straps
Secondary Support Monitor and assess. Install as required by the Plan	*Set ondary Support* Monitor and assess. Install as required by the Plan	*Secondary Support* Monitor and assess. Install as required by the Plan
Monitoring Eveiy 100m of panel advance	*1.1.3 Monitoring* each cut through	*Monitoring* Each intersection where a breakaway is formed. As determined by Strata Management Team
Primary Support Notes: *refer to Manager's Support Rules for roof & rib bolt location 1* *all roof bolts to be installed with 120mm' x 10mm thick bearing plates or plates of equivalent strength* *ensure that bolts are pretensioned to the optimum speι ıfu allons*	***Secondary Support Notes:*** *all cable bolts to be encapsulated to >l.5m.* *all type & location of cable bolts will be determined by Sιrata Management Team & a Initialised plan shall be implemented.*	***Roof Monitoring Notes:*** *Additional monitoring tools installed as per SMT* *location of monitoring tools to be communicated on Weekly Planning Sheet.* *Extensometers or tell-tales / rockets are to be used for roof monitoring. Tools specified above must be installed within 20m of fat e unless spct ified otherwise.*

Registered Mine Manager........................Date:........./...../.

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SLO - A Program for Stope Limit Optimisation Using A Heuristic Algorithm

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ABSTRACT: Tlie heuristic *"Maximum Value Neighbourhood" (MVN)* algorithm is proposed to optimise stope boundaries. The algorithm provides a 3D analysis and can be applied to any underground mining method. The *MVN* algorithm uses a 3D economic block model to locate the best neighbourhood of a block, whtch guarantees the maximum net value. Neighbourhoods are restricted by the mine geometry constraints. A *Fortran 90* program, the *"Stope Limit Optimiser" (SLO),* has been developed to implement the algorithm. *SLO* integrates the *Fortran 90* code of the algorithm with the *Winteracter* user interlace features, to provide a Windows based interactive environment for defining the model specifications, stope constraints and mine economic factors and displaying the results as 2D plans/sections in a text mode view. Intermediate results are reported for each block and an on screen summary report is provided.

1 INTRODUCTION

Few algorithms have been developed for optimisation of the ultimate stope boundaries. The latest one, termed *"Maximum Value Neighbourhood" (MVN),* is based on a heuristic approach and benefits from its generality and simplicity (Ataee-pour & Baafi 1999). Few alternatives have been reported for the *MVN* algorithm including the Floating Slope Algorithm of Datamine (Alford 1995) and the Branch and Bound Technique (Ovanic and Young 1995). The *MVN* algorithm is implemented on a 3D economic block model, in which the problem of determining the best combination of blocks (for mining) that provides the maximum profit is formulated by Equation (1).

$$\left.\begin{array}{l}\text{Objective function :}\\ \qquad \textit{Maximise } SEV = \sum_{ijk} F_{ijk}\ BEV_{ijk}\\ \text{subject to :}\\ \qquad \textit{slope geometry constraints}\end{array}\right\} \quad (1)$$

where
SEV: total stope economic value,
BEV_{ijk}: the economic value of the block, B_{ijk}.
F_{ijk}: an indicator function showing whether the block, B_{ijk}, is mined or not. It is defined by Equation (2).

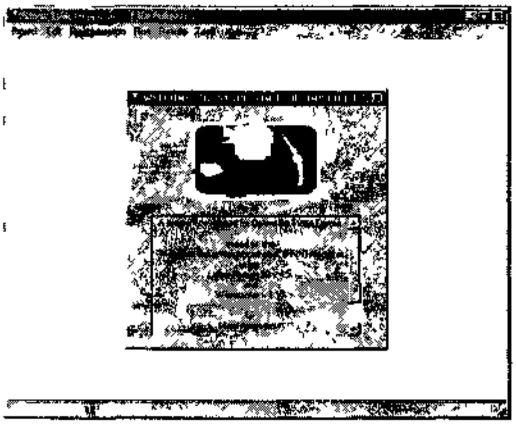
$$F_{ijk} = \begin{cases}1 & \text{if } B_{ijk} \text{ is mined,}\\ 0 & \text{otherwise.}\end{cases} \quad (2)$$

Stope geometry constraint is formulated by the *neighbourhood (NB) concept*, which is based on the number of mining blocks equivalent to the minimum stope size. The size of (he *NB* in terms of the number of blocks is defined as the *order of neighbourhood (0„i,).* The *MVN* algorithm constructs the set of possible *neighbourhoods for* each block, calculates the dollar value of each *NB*, locates the *NB* with the maximum dollar value *(MVN)* and flags the corresponding blocks for inclusion in the final stope. The *MVN* algorithm has been implemented on small sized examples, using *Excel Visual Basic* modules (Ataee-pour and Baafi 2000).

An application program, the *"Stope Limit Optimiser" (SLO)* was developed to implement the *MVN* algorithm on actual mine data. *SLO* integrates *Fortran W* code of the algorithm with *Winteracter* user interface features. This paper introduces the procedure, capabilities and limitations of *SLO* and illustrates how a project is manipulated in *SLO* and how the optimisation results are displayed.

2 GENERAL PROCEDURE

SLO considers jobs for optimisation as projects. A project is a collection of input files that specify the block model parameters, stope geometiy constiaints and economic factors An ıntelactıve environment is piovided tor the usei to define projects and import block data for the optimisation process Figure I shows the *SLO* welcome page with the main menu Using the '"Project" option ot the menu, the usei may create a new project or open an existing one, save the pioject and close it 01 exit from the *SLO* application



Fıguıe I The *SLO* welcome page

The general performance ot the *SLO* optimiser, tor any pioject, is divided into three stages, ı e the input, optimisation and output stages, as shown in Fıguıe 2 During the input stage, all input data, including the block model definition, stope constiaints, economic factors and the block data are edited and prepared using the "Edit" option Each input data type is saved in a separate file tor lurther use In cases where block data contain only grade values, block economic values *(BEV)* arc calculated tiom grade values by *SLO* through the data preparation phase ot the "Preoptımısatıon" option The final pioduct ot the input stage, however, is an economic-block model (a 3D array containing the economic values ot the blocks) together with the 3D order ot neighbourhood

The optimisation stage is the core ot the whole program Optimisation may be peilormed on the whole, or a sub-region ot, the block model Using the "Preoptımısatıon" and "Run" options, a (sub) region is specified, the corresponding block data are impoited and optimisation is performed in accordance with the oıder ot neıghbouıhood and based on the *MVN* algorithm The optimisation stage receives the 3D array ot block economic values, as well as the 01 dei of neıghbouıhood, and produces a 3D arıay of block flag data

The output stage includes all processes concerning the visualisation ot the optimisation results (using the "Results" option) At this stage, *SLO* receives the 3D array of block flag data, arranges them in a plan or section older, wiaps them in a table tormat with suitable annotations and finally displays the optimised stope layout in an ascn formatted file Alternatively, the *SLO* exports the flag data directly into an ascn formatted file, accessible to other computer packages, which have been developed tor 2D and 3D display ot lesults The output stage also includes the display of all the reports and intermediate results collected thioughout the optimisation stage

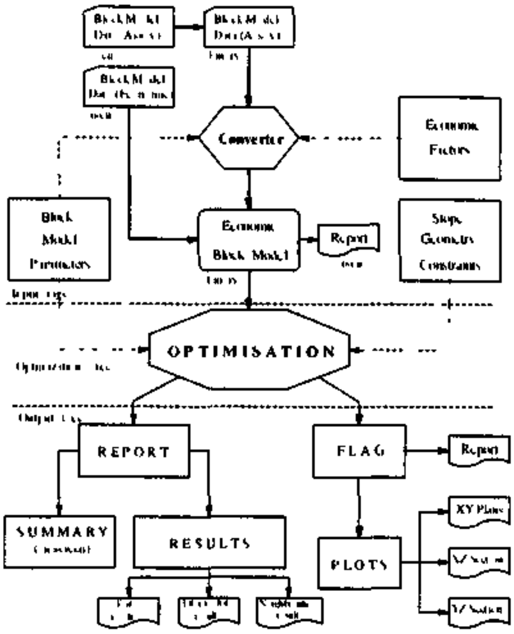


Figure 2 Geneial scheme of the *SLO* tlow-chart

3 PROJECT MANIPULATION

Projects are created, modified or deleted via three groups ot files, ı e model parameters files, stope constraints files and economic factors files

3 I Block Model Definition

The model parameters files contain information about the specifications ot the block model These include the co-oıdınates ot the origin and the maximum limit ot the block model, the extension and the numbei ot the blocks in X, Y and Z directions, as well as the definition ot possible sub-regions within the whole model Figure 3 shows the main dialog box tor the definition ot the block model

Block models may be defined in either the XYZ or UK mode In the XYZ mode, the model co-ordinates and the block dimensions are entered by the user The block volume, the number ot blocks in each direction and the total number of blocks within the model are automatically calculated. The UK mode requires that the user enter the number of blocks as well as the block's extensions in X, Y and Z dimensions. Then *SLO* calculates the minimum and maximum co-ordinates of the block model.

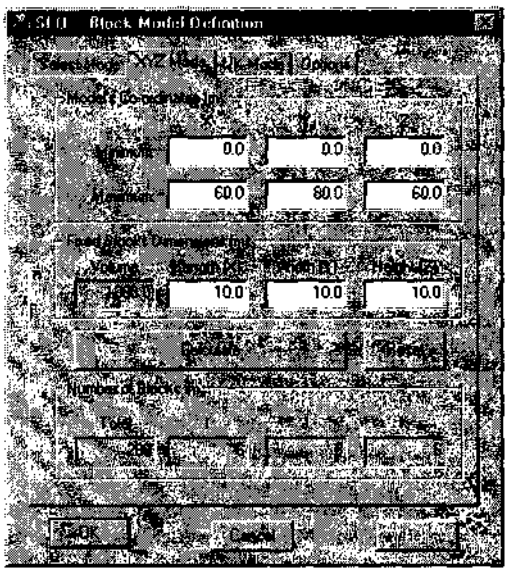


Figure 3. Defining the block model (XYZ mode)

It may be necessary to divide the block model due to geo-technical factors, different rock types or various mining methods. These impose different stope constraints to different parts of the block model. If this were to happen, various sub-regions may be defined to handle a variety of stope constraints and *orders of neighbourhood* within the model. When a deposit is too large and separate zones of mineralisation can be distinguished within the entire deposit, definition of various sub-regions may be helpful. In addition, the block model may contain either grade or economic values. *SLO* provides options to define both data types to a project. If economic values are used, the block data are directly used in optimisation. In cases where assay data are used, they should be converted into economic data before optimisation is performed. Through the "Options" sub-dialog in Figure 3, the user can state whether or not there are any sub-regions within the block model and specify the block data type.

3.2 Definition of the Stope Geometry Constraints

Information about the stope geometry constraints is collected in a separate file. The user should enter the minimum stope size, in terms of meters, for each of the three orthogonal directions. The stope block ratio *(SBR)* and the *order of neighbourhood* (O_{nh}), in each direction, is then calculated by *SLO,* based on the block size and the minimum stope size. O_{nh} is finally expressed in terms of the integer numbers, i.e. the number of blocks inside the minimum stope size in the X, Y and Z directions, respectively. The product of these three values indicates the total number of blocks within the minimum stope. It also indicates the total number of possible neighbourhoods for each block.

It is possible to define the stope constraints for the whole and/or sub-regions of the model. The existence of sub-regions often shows that there is not a consistent stope geometry constraint within the block model, so the constraints cannot be defined for the entire model but rather they should be defined for every sub-region. If the division of the model into sub-regions is not due to various stope constraints (e.g. it is because of a large deposit), then the stope constraints could be the same for all the sub-regions as well as for the entire model. This means that defining the stope constraints for sub-regions is not useful, unless the user is interested in performing the optimisation algorithm for a specified zone, instead of the entire model. *SLO* provides options to perform both cases. Figure 4 shows defining the slope constraints for a sub-region.

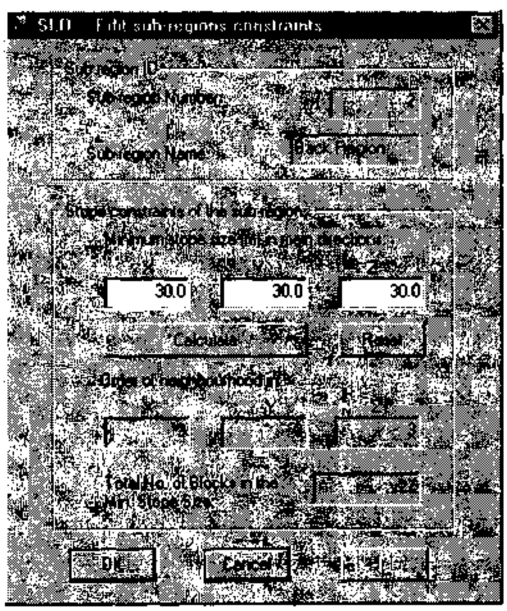


Figure 4. Defining the stope constraints (sub-regions)

3.3 Economic Factors

Another input to the system is information in respect of the economic parameters, which apply to mining the deposit. This information is used to help trans-loi m assay data into the dollar value of the blocks The tequired economic patametets include ınloıma-lion about the products of mining their puces, guides and price units costs of mining/processing of the pioducts, rates of recoveiy applied to products and the densities of the oie/waste Fıguıe 5 shows the economic paiameters dialog box thiough which pioducts ol the deposit may be defined Currently, *SLO* supports processing of economic factors tor multi product deposits containing up to tour by-products

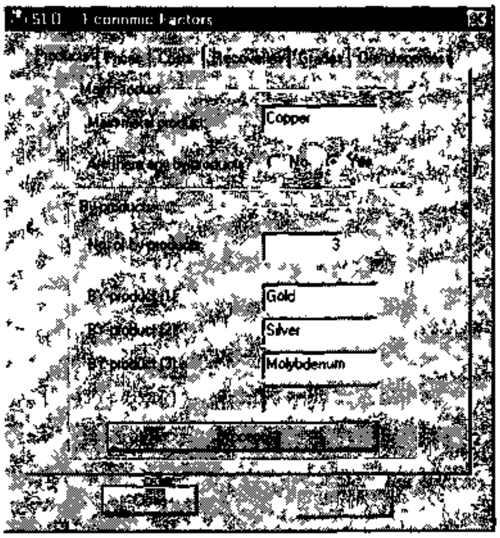


Figure 5 Defining the mine products

The puce and the price unit of each pioduct aie required The user may enter the price diiecliy in a real field in the supplied dialog box A list of three price units is available to select via a drop down menu which include dollar per tonne dollar per ounce and cents pei kilo Figure 6 shows the corresponding dialog box in *SLO*

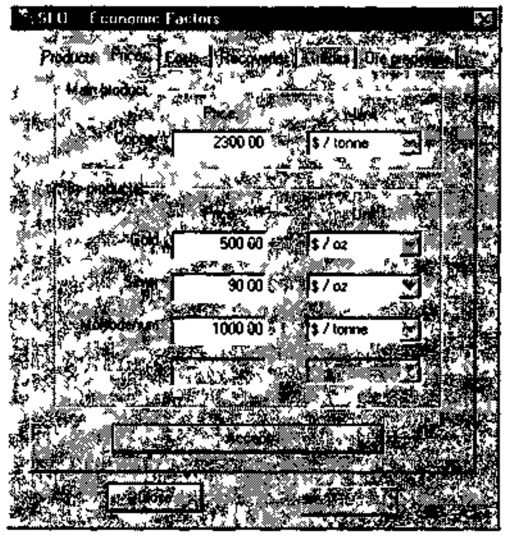


Figure 6 Defining prices and price units

There are two categories of costing applied in the optimisation of stope boundaries, ı e ore-based costs and metal-based costs Oie-based costs consist of all expenditures spent to extract the rock from the mine regaidless of whether it is ore or waste In order to obtain oie-based costs, all mining processes in eluding preparation, dulling blasting or haulage, should be consideied Then the average cost toi mining one tonne of rock (ore or waste) forms the ore-based costs Metal-based costs include all cx-pendituies to ıecover the metal pioduct from the mined ore Obtaining costs tor this category is by calculating the aveiage cost tor one tonne of the main product rather than the ore The metal-based costs may be broken down into a number of compo nents such as, the cost of smelting or pıocessıng, the cost ol letınıng, administration costs and other miscellaneous costs The user may enter these cost information thiough a dialog shown in Figure 7

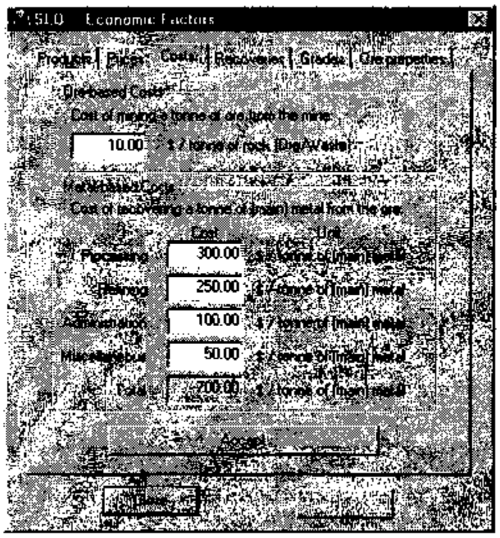


Figure 7. Defining various costs

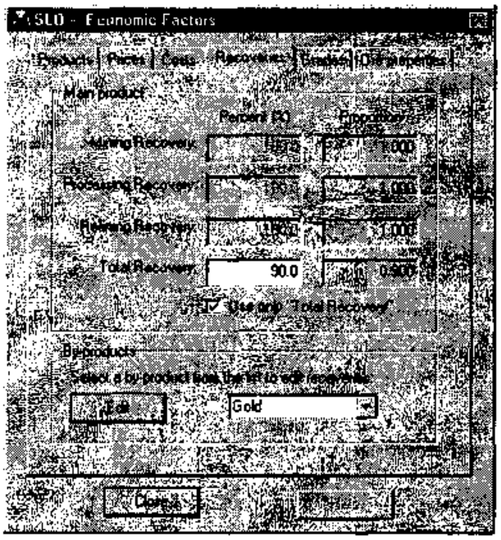


Figure 8. Defining various rates of recovery

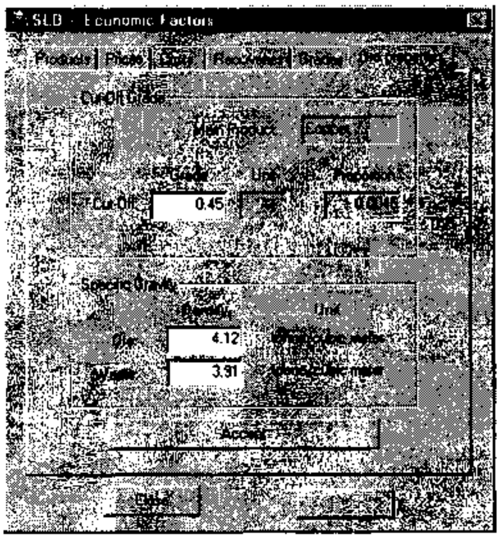


Figure 9. Defining ore properties

SLO provides definition of recovery of the metal contents in three stages, i.e. the mining, processing (smelting) and the refining stage. The user enters the rate of recovery at each stage, in terms of percentage, and *SLO* returns the total rale of recovery. However, the user may enter directly the total rate of recovery, if available, and ignore the sub recoveries. Figure 8 shows the dialog box for defining the various rates of recovery. If there are any by-products, the rates of recovery should be defined for each by-product using an additional dialog box.

The grade values are entered via the assay data file, however, the units of grades are defined through the project files. The grade units are defined for each by-product as well as the main product. A list of two items is available for the selection of grade units, i.e. the percentage (%) and grams per tonne (ppm). Ore properties are the last economic factor described in the project file. In order to obtain the weight of a block, the specific gravity of the rock is needed. Two distinct categories of density, i.e. the density of the ore and the density of the waste are supported. A cut-off grade should be defined for the main product lo discriminate between the ore and the waste. Figure 9 shows the *SLO* dialog box for defining the ore properties.

4 BLOCK DATA FILE

The main block inputs to the optimiser is the economic values of blocks stored in a data file. In cases, where economic values are not available, the data file may include the assay data of blocks. In either case, data files must contain information about the address and value of each block. For economic data files, the block value is a real number that represents the estimate of the dollar value of that block. Where the input is assay data, the block value consists of a group of, up to five, real numbers, which represent the estimated grade ot the main pioduct, as well as that ot a maximum ot torn possible by-products tor that block It is possible tor the usei to view and edit the data tiles in ascn format before peitoiming optimisation

In multi product projects where theie are multiple giade values tor each block, they are replaced by a single equivalent grade value, based on the main product giade In ordei to calculate the equivalent giade value, the prices grades and rates ot recovery tor each individual product are taken into consideration

5 OPTIMISATION AND RESULTS

After all data aie input in the îequired form, the usei may choose the entne model or select a sub-iegion to peitorm optimisation All required data corresponding to the blocks within the selected ıegıon aie then imported and optimisation is peiformed based on the *MVN* algorithm Finally, blocks ol the optimised stope aie flagged The flag data ot blocks are stoied in an output file

5 / *Plots*

The output tile may be impoited into other mine planning packages to display the end îesults in 2D and 3D views Howevei *SLO* provides utilities tor the usei to pioduce and view the plots ot the ultimate stope boundaries The usei may select the plot ling ol any ot the 2D views of the stope including X-Y plans X-Z sections and Y-Z sections, or all ot the plots It is also possible to specify a certain plan section, oi a range ot plans or sections ot the optimised slope tor plotting Figuie 10 shows the *SLO* dialog box tor specifying plans, or sections, ot the optimised slope to be plotted

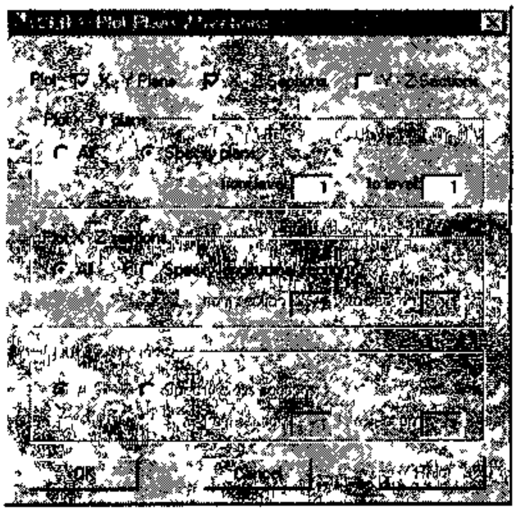


Figure 10 Spccılyıng plans/sections to plot

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PLAN   h   12
                                        I
        1  2  3  4  5  S  7  f1 9  10 11 12 13 14 15 16 17 İS İS

  29  * 0  0  0  0  0  0  0  0  0  0  1  1  1  1  1  1  1  1  0  ' 29
  28  * 0  0  0  0  0  0  0  0  0  1  1  1  1  1  1  1  1  1  0  * 28
  21  * 0  0  0  0  0  0  0  0  0  1  1  1  1  1  1  1  0  0  0  * 27
  26  * 0  0  0  0  0  0  0  0  0  1  1  1  1  1  0  0  0  0  0  * 26
  25  * 0  0  0  0  0  0  0  1  1  1  1  1  1  0  0  0  0  0  0  * 25
  24  * 0  0  0  0  0  0  0  1  1  1  1  1  1  0  0  0  1  1  0  * 24
  23  _ 0  0  0  0  1  1  1  1  1  1  1  1  1  0  0  0  1  1  0  * 23
  22  * 0  0  0  0  1  1  1  1  1  1  1  1  1  0  0  0  0  0  0  * 22
  21  » 0  0  0  0  0  0  1  1  1  1  1  1  1  0  0  0  0  0  0  * 21
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  13  * 0  0  0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  0  0  * 13
  12  * 0  0  0  0  0  0  1  1  1  1  i  1  1  1  1  1  1  0  0  * 12
  11  * 0  0  0  0  0  0  1  1  1  1  i  1  1  1  1  1  0  0  0  * 11
  10  * 0  0  0  0  0  0  0  1  1  1  1  1  1  1  1  1  0  0  0  * 10
   3  * 0  0  0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  0  0  *  9
   8  * 0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  1  1  0  0  *  8
   7  * 0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  1  1  0  0  *  7
   S  * 0  0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  0  0  0  *  6
   5  * 0  0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  0  0  0  *  5
   4  * 0  0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  0  0  0  *  4
   3  * 0  0  0  0  0  1  1  1  1  1  1  1  1  1  1  1  0  0  0  *  3
   T  * 0  0  0  0  0  0  0  0  0  0  0  0  0  0  0  0  0  0  0  *  2
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        1  2  3  4  5  6  7  S  9  10 11 12 13 14 15 16 17 18 19
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```

Figuie 11 An example of the plans/sections plotted in *SLO*

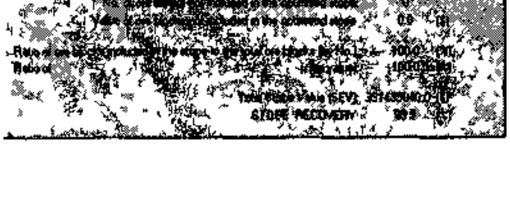
An example ot such plotted plans/sections is shown in Figure 11 Cunently, the plots of *SLO* are in text mode It is envisaged to modify the application to plot in giaphics mode

5 2 *Results Files*

SLO reports the intoimation obtained lor the optimised blocks at each stage This intoimation is updated as the optimisation progresses and is finally saved in some report files These include reports on the intermediate lesults, the neighbouihood results and a final on scieen summaiy report Information about the set ot possible neighbourhoods ot each block and their dollai values are saved in the specified file The intermediate results obtained tor each block contain the *neighbouihood (NB)* numbei within the set ot neıghbouıhoods ot that block, which piovides the maximum value, the maximum neighbourhood value, the maiginal value obtained fioin the *MVN* of the block, and finally the updated stope value These results obtained duiing the optimisation will help the user to see the vanation of the stope after the optimiser examines each block

5 3 *Summen ^ Repoı t*

Atteı the optimisation is completed, a summary leport ot the optimisation lesults is provided and appended to the end of the intermediate results file The summary report includes information such as the total number ot blocks within the region, number ot negative and non-negative valued blocks in the region numbei ot negative valued blocks included in the final slope, and then total values, numbei ot non-negative valued blocks excluded from the final stope and their total values, total stope value, and the percentage of the block values included in the ultimate stope Fıguıe 12 shows a typical on scieen summary leport



Fıguıe 12 A typical on scieen summaiy lepoit

6 CONCLUDING REMARKS

SLO is a Windows based application program which may be used to interactively optimise stope boundaries, implementing the *MVN* algonthm The major options available in *SLO* include XYZ and UK modes, defining sub-regions, accepting assay/economic value modes and supporting multi-product deposits (up to toui by-products) It is possible to produce nested stopes by changing the economic paiameters such as prices, costs, recoveries and cut-olt grades and multiple running ol *SLO* This parametensation is helpful in any decision making about the ore deposit including the feasibility study, prchminaiy mine evaluation and the mine closuie The variation in stope geometry constraints may also pioduce nested stopes which may help in mining method selection since different mining methods impose ditterent stope geomctiy constraints Running *SLO* tor alternative mining methods will help in the selection ot the method with the highest stope net value

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Artificial Neural Networks Provide a Toolbox for Analyzing The Pressure Transient Data Collected in Coalbed Methane Drainage Wells

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ABSTRACT: This paper addresses the need for the development of novel inverse solution methodologies with applications in the analysis of the pressure transient data collected from coalbed methane drainage wells for the purpose of characterization of the transport and storage characteristics of the coal. Typically, the transport and storage parameters are determined experimentally from the coal samples that are collected during the drilling operation. Due to the small size of the core plugs collected and the difficulty in preserving these samples in their virgin states as well as the challenges in restoring the original field conditions in the laboratory, it is proposed to develop *in situ* measurement protocols for the same purpose. The utilization of artificial neural networks (ANNs) as a potential tool in formation characterization using the *in situ* collected data is explored in this study. Several ANN models that are specifically constructed to analyze the pressure transient data collected from coalbed reservoirs are presented in an increasing order of complexity.

I INTRODUCTION

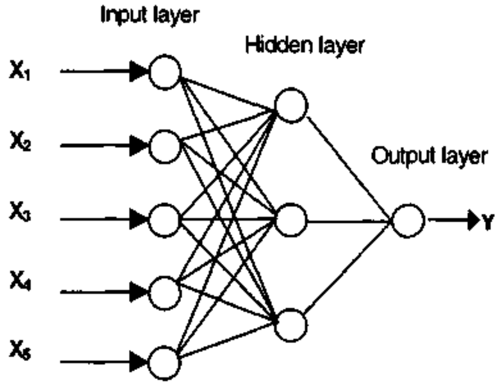
Coal seams are known to be source rocks for natural gas and are classified under unconventional gas reservoirs. A coalbed reservoir is different from its conventional counterpart in that it has a densely spaced natural fracture system and a good majority of the gas is found in the adsorbed state. Thus, conventional well test analysis techniques are not applicable in the analysis of well testing data collected from coalbed reservoirs. Anbarci & Ertekin (1991) developed an analytical forward solution model, which can be used in the analysis of the pressure transient behavior of coal seams. Type curves generated from this forward model can be employed to determine some of the coal seam properties. However, type curve matching analysis is limited to a relatively small range and a limited combination of these properties.

The principal objective of this paper is to demonstrate the efficiency and applicability of artificial neural networks in characterizing the transport and storage properties of coal seams, such as permeabilities, macropore porosity, and sorption parameters such as Langmuir volume and pressure constants and sorption time constant.

2 ANN AS AN INVERSE SOLUTION METHODOLOGY

ANNs have been used in a wide variety of fields to solve problems involving classification, function approximation, forecasting, control systems, etc. ANNs are considered as information-processing systems with certain performance characteristics that are in common with biological neural networks (Fausett, 1994). An ANN is made up of a large number of parallel-distributed processing units called neurons, which are simplified analogs of the human brain cells. These units store experiential knowledge and resemble the brain in certain aspects. Artificial neural networks acquire knowledge through a learning process and intemeuron connection strengths (known as weights) store the acquired knowledge. One type of ANN commonly used in petroleum and mining engineering applications is backpropagation network (BPN). A typical BPN architecture is shown in Figure I.

The number of neurons in each layer of this architecture is chosen for simple illustration, but can vary with problems. Training a network by backpropagation involves three stages: feedforward of the input data, calculation and backpropagation of the associated error, and adjustment of the connection weights (Fausett, 1994).



Figuie 1 A simple backpiopagation network (BPN) with one hidden layer

The inherent enormous parallel processing capability of ANNs makes them a promising tool in analyzing the well test data. The learning ability of ANNs can be effectively utilized in predicting properties of the coal seams.

In well testing, pressure transients (output) measured at a well represent the response of a coal seam to the conditions imposed at the wellbore (input). In an inverse analysis application using ANNs, pressure transient data and other known parameters such as reservoir temperature (T), wellbore radius (r_w), gas production rate (q), reservoir thickness (h), coal density (p_c) and reservoir initial pressure (p,) define some of the input neurons. At the same time, coal seam properties such as anisotropic permeabilities (kx, k_y), porosity (<])), Langmuir volume constant (V_L), Langmuir pressure constant (PL) and sorption time constant (t) constitute the output neurons. Figure 2 is a schematic representation of the forward and inverse solution procedures in system analysis. In this figure, "I" represents the input, "O" is the output and "S" is the system's characteristics.

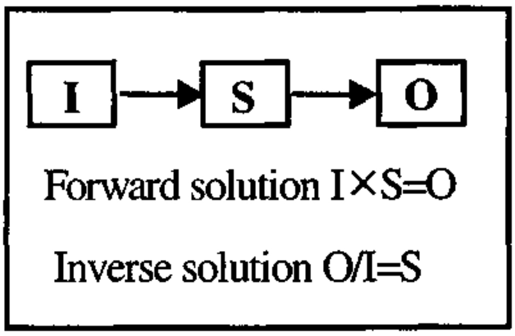


Figure 2. Forwaid .solution and inverse solution components of a system analysis piotoeol

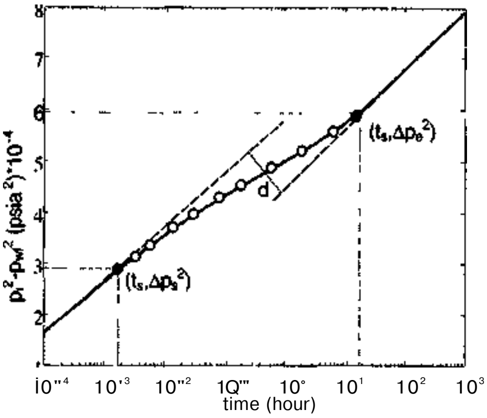


Figure 3 A typical pressin e transient behavior of a methane drainage well

3 DEVELOPMENT OF ANN MODELS

There are two important processes irç developing an ANN model: the training data preparation and the design and testing of an appropriate architecture.

3.1 Data preparation

Since the quality of the training data directly controls the ANN's behavior, its importance cannot be over stressed. The training data should provide a good representation of the problem within a large range of properties relevant to the solution domain. The working principle of an ANN is more like that of a human brain. With the help of biological neurons, one recognizes objects on the basis of their different characteristics. A similar convention is applied to ANNs. As they need to be taught of certain characteristics to distinguish and ultimately predict and associate different properties for various patterns.

Figure 3 shows the characteristic dual-porosity behavior of coalbed methane reservoirs when ($p_{,c}$"-p_w") is plotted against the logarithm of time. In this plot, two parallel straight lines represent the early and late time behaviors of the coalbed reservoirs, respectively. It should be noted that there is a transition period marked by circles when the pressure transient data shift from the first straight line to the second one. To characterize the overall behavior of such a signature, several data points matching some key events need to be identified within the transition zone as well as off the two straight lines. The slope of the straight lines, the vertical separation of the two straight lines and the time to reach the second straight line all contain information related to the transport and storage characteristics of a coalbed reservoir. The product of permeability and reservoir thickness (kh) can be calculated from the slope of these two parallel lines. In well test analysis, permeability (k) can be obtained if the reservoir thickness is known from geological, geophysical or drilling data. Porosity (())) can be inferred after permeability is calculated. The sorption time constant, x, can be obtained from the starting time of the second straight line. Finally, using the vertical distance (d) between these two lines, one of the sorption parameters V_L or P_L can be calculated.

The ranges of the data utilized in the training of the model are presented in Tables 1 and 2.

3.2 Designing and testing of the ANN architecture

The architecture of an ANN is not completely constrained by a given problem. Although number of input and output neurons utilized depends on the problem studied, functional links that are introduced to the ANN structure alters its topology. There exits no rigorous rules to guide the ANN practitioner in the choice of number of hidden layers and the number of neurons within the hidden layers. To obtain an appropriate architecture for a given problem, intensive testing of the prediction capabilities of the ANN must be conducted after the training of the model is completed. These two processes, training and testing, are revisited in a recursive manner until the prediction results are found to be satisfactory.

Table I- Ranges of ihe predicted parameters.

Pai ameter	Minimum value	Maximum value	Unit
Porosity	1	S	per cent
Face cleat permeability	0.1	100	mil
Butt cleat peimeability	0.1	50	mil
Langmuir piessure constant	15	200	psia
Langmuir volume constant	10	600	SCF/TON

Table 2-Ranges of the input parameters.

Parameter	Minimum value	Maximum value	Unit
Wellbore radius	0.25	0.5	ft
Formation thickness	2	20	ft
Flow rate	0.05	5	MMSCF/d
Coal density	1.30	1.40	g/cm'
Initial reservoir pressure	400	1500	psia
Reservoir temperature	60	160	°F

Since a neural network without a hidden layer can only solve linearly separable problems, at least one hidden layer is needed to solve the class of nonlinear problems. Two hidden layers are used in each of the ANN structures developed in this study. The purpose of using two hidden layers is to make the overall training process much more efficient.

J.J? *Model development stages*

Stage I: In this stage, an infinitely large reservoir with homogeneous and isotropic properties is considered. One producing well is placed at the center of the reservoir and each reservoir with different properties yields a different pattern. The analytical model of Anbarci & Ertekin (1991) is used to generate (he pressure transient data for training and testing the networks.

Step I: Prediction of three parameters (k, <) and t)
Figure 4 shows the architecture of an ANN model for predicting porosity (<>), permeability (k) and sorption time constant (x). In this ANN model, there are 44 input neurons including r_w, q, h, p_c, pj, T, |i, c_s, z, and 12 pressure-time pairs. Functional links such as slope of parallel straight lines, the vertical distance between the straight lines and the time and pressure differences between the beginning and the end of the transition period are also included as input neurons. It is observed that these functional links are extremely useful in improving the accuracy of the predictions. There are 40 neurons in the first hidden

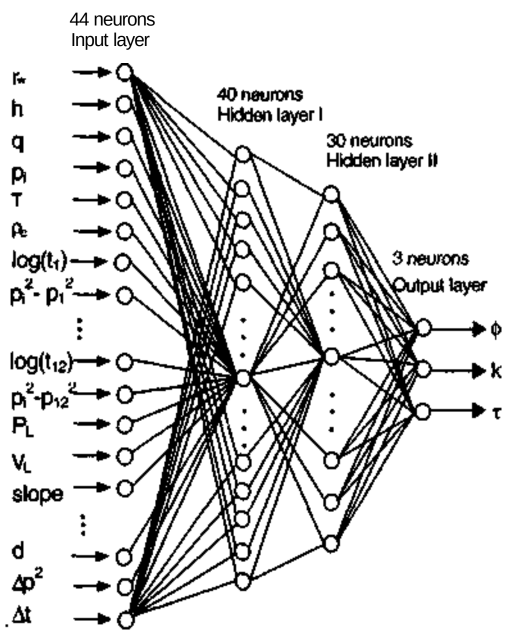


Figure 4. Netwoik architecture for prediction of <b k and T

layer, 30 neurons in the second hidden layer and 3 output neurons (k, φ, x) in the output layer.

Approximately 1000 training patterns are used during the training phase of the study. A total of 30 patterns is used to test the capability of the model. Figure 5 shows the test results of the ANN model. The figures on the left show the quality of the match between the predicted values and the actual values of porosity, permeability and sorption time constant and the figures on the right display the relative errors encountered during these predictions. The shaded bands in Figure 5 show that for more than 80% of the pressure transient data analyzed, the predicted values are found to be within the ±5% error margin.

Step II: Prediction of five parameters (k, φ, P_L, V_L and *x)*

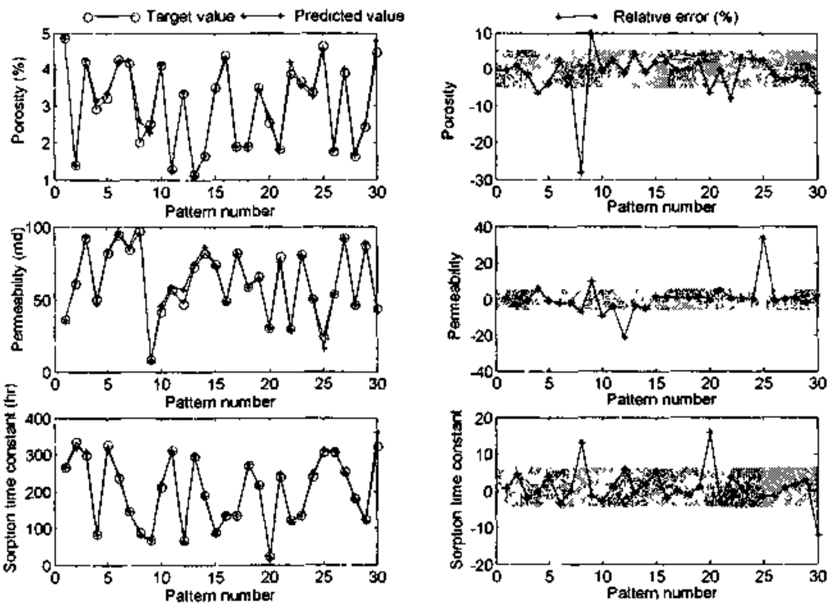
In this step, five parameters (k, φ, P_L, V_L, T) are to be predicted simultaneously. The architecture is similar to the one used in Step I. The difference in the structures stems from the fact that P_L and V_L neurons are moved from the input layer to the output layer. Figure 6 shows the prediction results. It is found that predictions of P_L and V_L are not satisfactory while the k, $ and T predictions match the actual values closely. Several networks were designed and tested, but none of them yielded a satisfactory simultaneous prediction of P_L and V_L. Various methods were also tried in presenting the training data to the network neurons, such as adding the spectral radius of the input matrix of pressure-time pairs and changing the order of output neurons tor V_L and P_L. The inability of the ANN in predicting P_L and V_u simultaneously will be discussed later in this paper.

Step III: Prediction of four parameters (k, φ, P_L or V_L, and T)

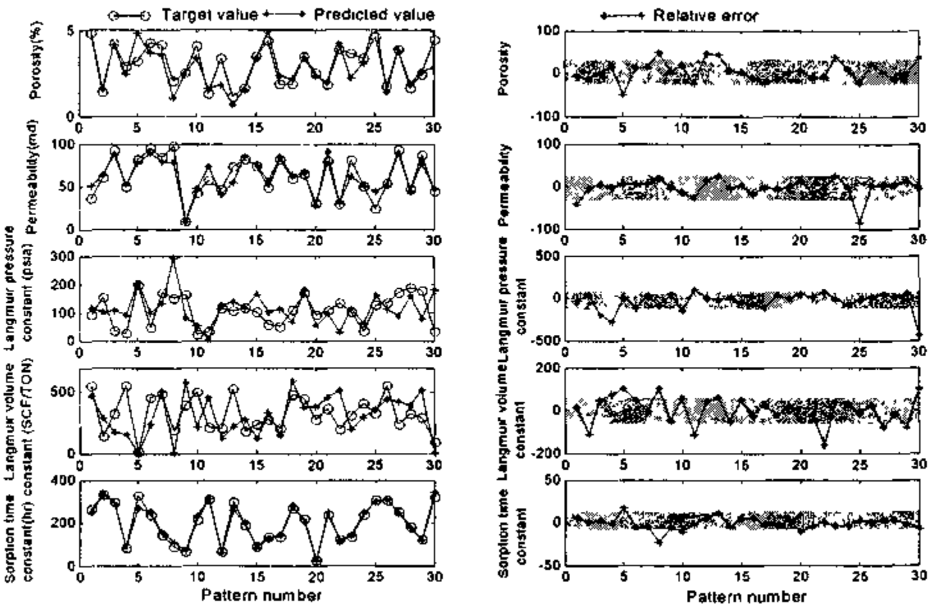
In this step, (wo cases are investigated as one of the Langmuir constants (P_L or V_L) together with porosity, permeability and sorption time constant is predicted. In both cases, it is found that predicted results match the actual values quite closely.

Case (a) Prediction ot k, *6),* P_L and *x* (V_L is treated as an input)

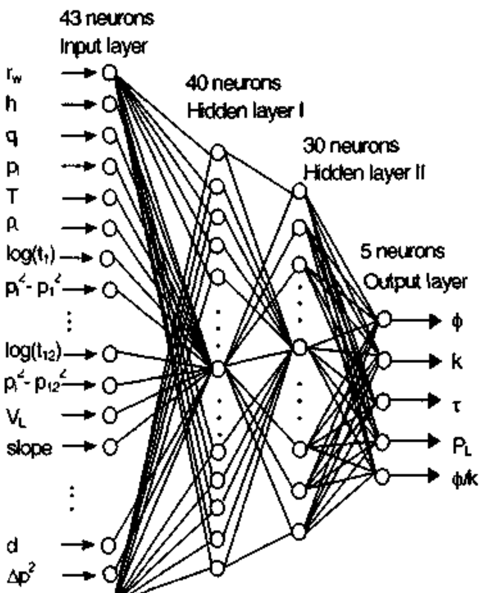
Figure 7 shows the architecture of the ANN developed to predict four parameters (k, φ, T, P_L). This architecture is quite similar to the one that predicts three outputs (k, φ, t). The difference is that P_L is treated as an output neuron instead of an input neuron. At the same time, one functional link output neuron (φ/k) is added to the output layer. It has been found that this functional link is very helpful in improving the accuracy of predictions.



Figuie 5 Prediction tesults of k. φ and T



Figute6 Piediction lesults ot k, ϕ P_L $V_L$anxlT



Figuie 7 Netwoik cuchiuxture tin piediclion ol k. *ty* P_L and

Figure 8 shows the predictions ot tour parameters, k, ϕ, P_L and T. The shaded bands in Figure 8 show that for more than 80% of the pressure transient data analyzed the predicted values are within the ±10% error margin.

Case (b) Prediction ofk, o). V, and T (P_L is treated as an input)

Figure 9 shows the ANN predictions ot tour parameters k, 0, V_L and x

The shaded bands in Figure 9 again indicate that tor more than 80% ot" the pressure transient data analyzed, the predicted values are within the ±20% error margin.

Step II and Step III clearly show that ANNs are capable of predicting only one of the Langmuir constants successfully In the forward solution protocol the P_L and V_L appeal together as a product Because ot the presence ot this product, a non-uniqueness issue is encountered in the inverse solution analysis In othei woids, the information available from the pressuie transient data is not sufficient to provide an accurate signature resulting from either ot these two parameters.

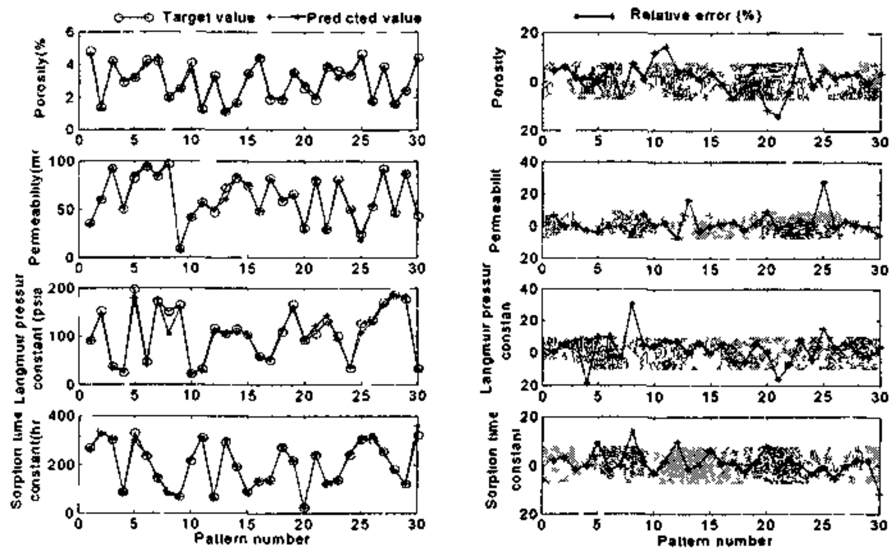


Figure 8 Prediction results of k φ P_L and τ

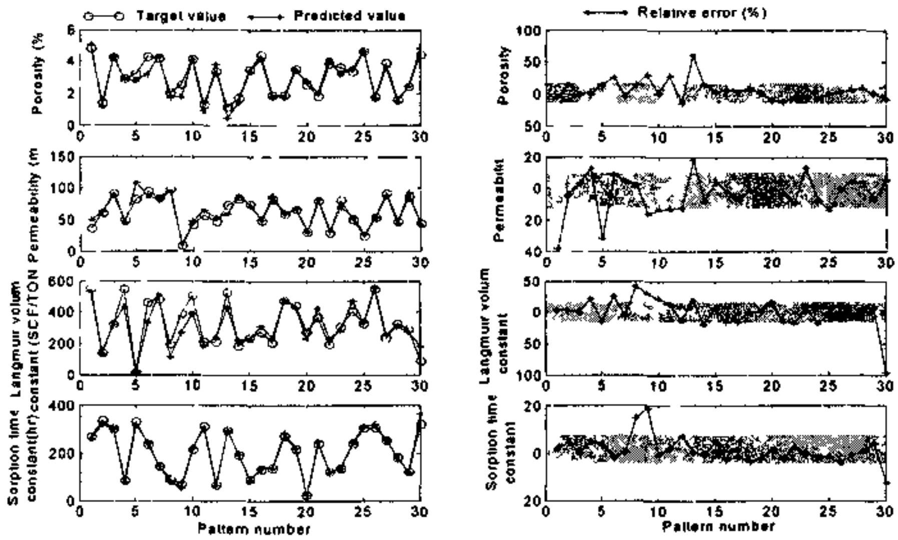


Figure 9 Prediction results of k φ V_l and τ

Stage II The butt and face cleat systems in coal res ervoirs aie usually orthogonal and they often exhibit anisotropic peımeabılıty values These anisotropic permeability characteristics cannot be obtained via analytical forward solution methodology In this stage a numerical sımulatoı (Manık et al 2002) ıs used to generate pressure transient data from an infinitely laige coalbed reservoir with homogeneous and anisotropic pioperty distribution It is obseived that the chaıacterıstıc two parallel straight lines disappeaied because of the anısotıopıc peımeabılıty Therefore neither vertical distance between these two straight lines nor slope of the parallel lines is available as input neurons However, some other characteristics such as the time and pressure differences between the beginning and the end of the transition zone still can be obtained from the data plotted as discussed in Stage I

Step I Prediction of four parameters (φ k_x, k_y and x) In this step, tour parameters (φ, k_x, k_y, T) are to be predicted simultaneously The ANN topology used in this stage is similar to the one presented in Stage I Fıguıe 10 shows the predicted results tor the atoiementioned foui parameters The prediction resuits are generally acceptable although the relative error is considerable larger than the prediction results ot Stage I. The bands in Figure 10 show that around 80% of the pressure transient data analyzed, the predicted values are within the ±50% error margin. The prediction accuracy of the sorption time constant is still ranked highest with more than 80% analyzed patterns falling within the ±20% error margin.

Siep II: Prediction of six parameters (ϕ, k_x, k_y, P_L, V_L and T)

In this step, six parameters (ϕ, k,,, k_v, P_L V_L, x) are predicted simultaneously. The purpose of this step is to investigate if the product effect of P_L and V_L still exists when data are generated by a numerical model. Figure 11 shows the prediction results. The prediction errors of Langmuir constants are observed to be much larger than that of other parameters.

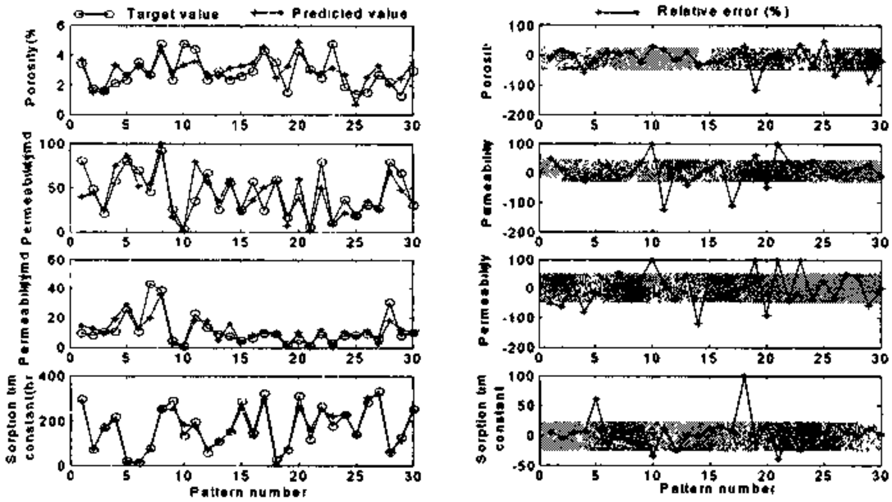


Figure 10 Prediction results of ϕ, k_x, k_y and τ.

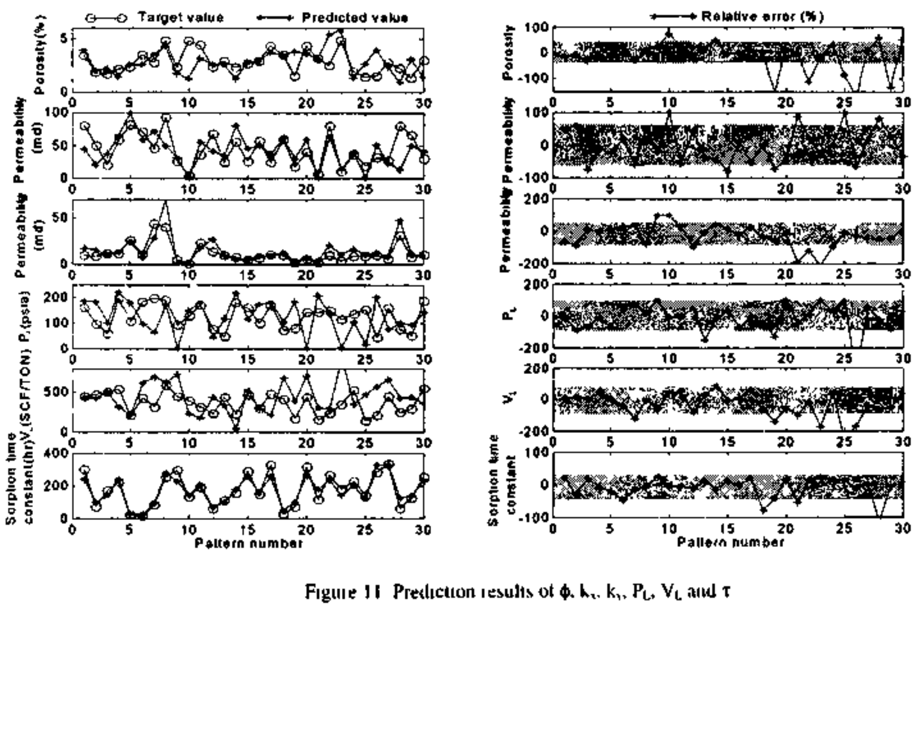


Figure 11 Prediction results of ϕ, k_x, k_y, P_L, V_L and τ

Step III Pıedıctıon ot five parameters *{§,* k_v k_y, P_L 01 V_L, and T)

Again two cases are tested in this step The directional permeabilities, porosity, soıptıon time constant with one ot the Langmuir constants are predieted Figure 12 shows the predictions ot ⌖, k_x, k_y, P_L t and Figure 13 shows the predictions ot ⌖, k„ k_y, V_L, x The relative error ot Langmuii constants in both cases decreases when one ot them is treated as an input neuron

By analyzing the testing results ot Step II and Step III, it is found that the product etlect on the pressure transient data still exists

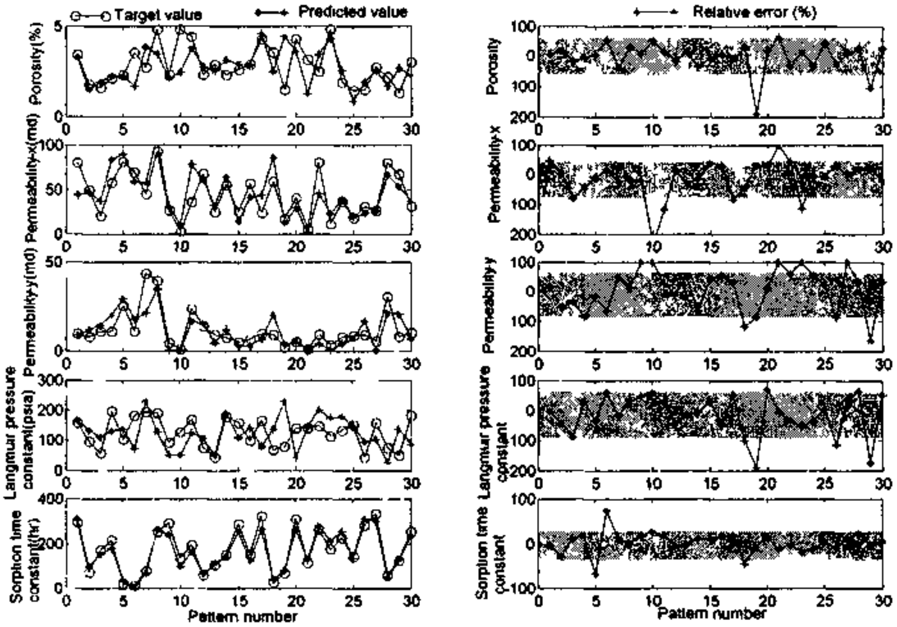


Figure 12. Prediction results of k_x k_y ϕ P_L and τ

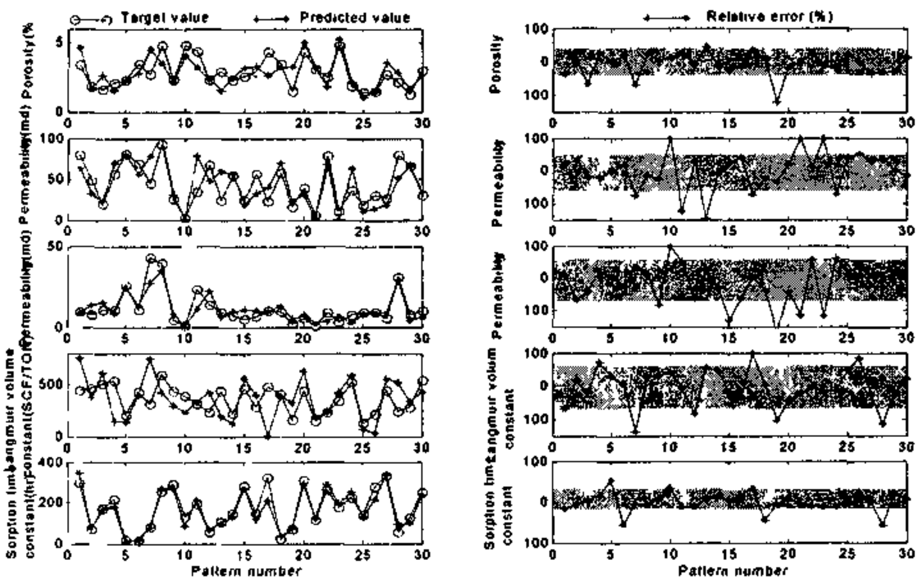


Figure 13 Prediction results of k_x k_y ϕ V_L and τ

Table 3- Error comparisons.

Variables predicted simultaneously	Error margin ϕ	k^*	k_x	k_y	P_L	V_L	τ	Relevant figures
ϕ, k, τ	±5%	±5%					±5%	Figure 5
ϕ, k, P_L, τ	±10%	±10%			±10%		±6%	Figure 8
ϕ, k, V_L, τ	±15%	±15%				±20%	±5%	Figure 9
ϕ, k, P_L, V_L, τ	±20%	±20%			±100%	±100 %	±10%	Figure 6
ϕ, k_x, k_y, τ	±25%	±50%	±40%	±50%			±20%	Figure 10
$\phi, k_x, k_y, P_L, \tau$	±50%	±50%	±50%	±90%	±80%		±30%	Figure 12
$\phi, k_x, k_y, V_L, \tau$	±50%	±50%	±70%	±80%		±80%	±40%	Figure 13
$\phi, k_x, k_y, P_L, V_L, \tau$	±60%	±50%	±70%	±90%	±100%	±100 %	±40%	Figure 11

Note: In an isotropic system, $k^*=k$ and in an anisotropic system $k^*=(k_x \times k_y)^{1/2}$.

The predictions in this stage are obviously not as accurate as that in the first stage. This is because some of the pressure transient data sets do not capture all of the characteristics of dual porosity reservoirs. The missing information may cause inaccurate predictions. Another reason for inaccurate predictions is evolves from the increasing complexity of the problem in Stage II in which coal seams are considered to be anisotropic.

4 DISCUSSION OF RESULTS

There ate three important considerations in creating a generalized network. One is the choice of the number of the hidden layers and the number of neurons in the hidden layers, the second one is selecting the training algorithms and the third one is the transfer functions used between the layers. More hidden layers and more neurons in layers are not always better than fewer, since more layers and neurons may result in over-training and make the architecture more complicated. During this study, conjugate gradient method is used as the principal training algorithm because of the less stringent memory requirements as well as its rapid convergence characteristics (Hagan et al. 1995). Transfer function between layers is also crucial in designing ANN models. Generally, the *purelin* (/(.») = .O is used in the output layer as the last transfer function while *tansig* (*d* ..., (,) or *logsig* (*j* ,.., ^) are used iff trfe'input layer or hidden layer tA'ydiiioglu et al. 2002). Furthermore, the convergence criterion should be chosen carefully, since while fine convergence criterion may lead to over-training, a coarse convergence criterion might result in incomplete training. Finally, providing a qualified data set and information to ANNs will increase the accuracy of predictions. It is also observed that in each stage of the development increasing the number of training patterns improve the accuracy of predictions.

Table 3 summarizes the error margins encountered for different models. The first four rows present the isotropic cases and the last four rows are for anisotropic cases. Anisotropic cases are more complex than the isotropic ones. The prediction accuracy of porosity is relatively satisfactory although it consistently shows a decrease in anisotropic cases. The prediction accuracy for directional permeabilities decreases when the number of output neurons is increased. However, the relative errors for the geometric average of the anisotropic permeabilities remain within an error margin of ±50%. The relative errors of the Langmuir volume and pressure constants reach the highest (+100%), when they are predicted simultaneously both in isotropic and anisotropic systems. However, the prediction quality of the Langmuir volume and pressure constants becomes better when they are predicted separately. Sorption constant is consistently the most accurately predicted (error is less then ±40% in anisotropic system) sorption parameter. It is noted that the predicted values still follow the trends made up of target values well in anisotropic system although the relative error is larger than that of the corresponding isotropic system.

5 CONCLUSIONS

Soft computing protocols such as artificial neural networks have potential applications in *in-situ* evaluation of the coal seam properties. The ANN models designed during this study for predicting the transport and storage characteristics of coal seams are found to be promising as they are functioning effectively. The ANN structures presented in this paper cannot simultaneously predict the Langmuir volume and pressure constants with a high order of accuracy.

This is attributed to the presence of the Langmuir volume and Langmuir pressure constants in the form of a product in the forward solution protocols used in the generation of the pressure transient data. Finally, it should be noled thai for any ANN application there is no perfect structure and a better structure can evolve by time. Following observations and conclusions are obtained from this study:

1. Peimeabihty, porosity and sorption time constant properties can be effectively predicted for both isotropic and anisotropic reservoirs using the artificial neural networks presented in this paper.
2. It is difficult to predict the Langmuir pressure and volume constants simultaneously.
3. Increasing the number of training patterns improve the prediction capacity of the ANN models.
4. The training data quality is critically important for accurate predictions.
5. Functional links plays a pivotal role in structuring an appropriate architecture for the desired ANN model.
6. Conjugate gradient method performs effectively as a training algorithm for the medium to large architectures.

NOMENCLATURE

t, = compressibility, psia"'
d = the vertical distance of the parallel lines
h = reservoir thickness , ft
k = permeability, md
/:, = face cleat permeability, md
k, = butt cleat permeability, md
p = pressure, psia
p, = pressure at the end of transition period, psia
/;, = reservoir initial pressure, psia
p, = pressure at the beginning of transition period, psia
P_L = Langmuir pressure constant, psia
q = flow rate, MMSCF/d

/„ = well bore radius, ft
T = reservoir temperature, °F
t = time, hr
;„ = end of the transition period, hr
U = beginning of the transition period, hr
V_L = Langmuir volume constant, SCF/TON
z = compressibility factor
A/r = pressure difference between the beginning and the end of the transition zone (Ap_c^2-$Ap_s^{;}$), psia"
At = time difference between the beginning and the end of the transition zone (t_L.-t_s), hr
0 = porosity, per cent
/.(= viscosity, cp
T = sorption time constant, hr
$p_{L'}$ = density of the coal seam, g/cnv

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UNIT CONVERSION

Field units		Metric units
I ft	=	0.3048 m
I md	=	1O'Vm^2
1 psia	=	6.895 kPa
I SCF/TON	=	2.86x10"'STD nrVkg
I MMSCF/d	=	2.86x10^{4}STDmVd

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Planning of Development of Mining Operations and Freight Traffics at an Open Cast by Analytic-Imitation Systems

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ABSTRACT: In the paper principle conception is presented of perspective and current planning of mining operations at open casts on the basis of economical-mathematical and imitating models of mining production. Worked out models provide interactive procedures when making technical and technological decisions. Offered methodology was evaluated when planning of mining operations at open casts of the Republic of Kazakhstan.

I INTRODUCTION

Industrial and financial-economical activities of mining and mining-and-proce.ssing enterprises in many respects depend on quality of planning and control of mining operations. When planning of mining operations volumes of useful minerals mining are determined with due account of their qualitative characteristics and volumes of overburden on within-year, years and stages of open cast existing. And extraction-and-loading operations must be ensured reliable freight-transport connection of working levels with points of receipt and dispatching of mineral raw material and overburden rock dumps.

Experience of perspective and current planning of development of mining operations at open casts of ferrous and poly-metallic ores and non-ferrous metals showed, that using of known methods of linear programming causes to considerable widening of domain and search, substantial increasing of computer time for their solving, increasing of errors in estimation of quality of mining ores and so on (Bukeikhanovetal. 2002).

Informational basis for mining-and-geological analysis and planning of mining operations is mathematical model of a deposit and an open cast, representing formalized description of form, structure and qualitative characteristics of a deposit and enclosing rocks, and also parameters of an open cast and its mining workings.

When .simulating open-pit field is divided into vertical sectors, which are limited of planes by different directions of studying development of mining operations. Within every sector at every open cast bench variants of technological blocks are separated. Ore blocks form as that qualitative characteristics of useful minerals, including in them, will be statistically indistinguishable. When planning, it is necessary to separate at a set of alternative variants of technological blocks for a period of planning such set of blocks in particular contours of mining operations, which will ensure receiving of production of given quality, which was taken for realization in every planned period of mining enterprise.

When simulating numbers of contours increase as advance of front of mining operations in a block. Here (' - number of a level (bench), at which block is located; *j* - number of a zone, in which block is located; *k* - number of contour of extraction in this block.

Scheme of separating section and blocks at one level and contours of variants of mining operations are presented in Figure l.

Here 1- position of /'" bench of open cast, where contour *of* mining operations is fixed; 2 - position of above located (/+//'' bench in studying k^{th} contour of mining operations; 3 - ore body within the limits of block; 4 - a part of ore body, falls within k'^{1}' contour of mining operations; 5 - boundaries of working bank of above located bench; 6 - opened-up part of ore body as of the moment of studying ot k''^{1}' contour of mining operations.

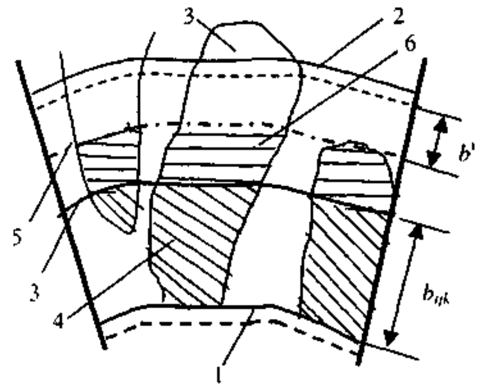

Figure 1 Scheme of separating of block into contours

For solving of this problem linear-integer-valued programming is used. As a controlling parameter Boolean variable of contour of mining operations x_{ijk} is taken. And A,,A=1, if contour has included into plan, in other case »,,<=().

Goal function of solving of the problem of annual planning of mining operations is specified by the following expressions:

$$\sum_{i=1}^{N}\sum_{j=1}^{s_i}\sum_{k=1}^{n_{ij}} S_{ijk}\, x_{ijk} \rightarrow \min, \qquad (1)$$

where S„i - annual costs on mining and processing of ore from a variant of contour of mining operations, tenge; tt,,, s,,, N_x - correspondingly a number of variants of contours of mining operations in a sector, zones (sectors) at a level and working benches in open cast.

$$S_{ijk} = [S_i^{or} + S_i^{opr} + S_i^{ora} l_{ij}^{ora} + (S^{ora} l_{ij}^{ora} + S_{sor}^{or})\cdot d_i^{ora} +$$
$$+ (S^{or} l^{cl} + S^{cl})\cdot n_i^{cl} d_i^{or}]\cdot A_{ijk}^{or}\cdot\gamma_p +$$
$$+ [S_i^{ov} + S_i^{ova} l_i^{ova} + (S^{ovr} l_i^{ovr} + S_{sor}^{ovr})\cdot d_i^{ova} +$$
$$+ (S^{ovc} l^{cl} + S^{cl})\; n_i^{cl} d_i^{ovc}]\cdot (Q_{ijk} - A_{ijk}^{or}), \qquad (2)$$

where i," - variable part of operating cost of ore mining without costs on transportation, tenge/f, .\"" - variable part of operating cost of ore processing into concentrate, tenge/t; y""- variable part of operating cost of transportation 1 t of ore by rail from /" level, tenge/tkm; /„ ' - distance of ore transportation by rail, km; ^ - variable part of operating cost of transportation of 1 t of ore by road, tenge/tkm; S'Z - costs per I ton of ore for construction of storehouse for ore re-loading from road transport to rail transport, tenge/t; /,; distance of ore transportation by road from contour of ore extraction up to nearest point of re-loading to rail transport or conveyor, km; d"" - Boolean variable, indicating ore transportation by road from faces to re-loading storehouses (1 - road transportation uses, 2 - road transportation does not use); S"'- variable part of operating cost of transportation of I ton of ore by conveyors, tengc/tkm; S"- costs per 1 ton of ore on 1 conveyor line, tcngc/t; /'' - length of one conveyor line, km; n" - a number of conveyor lines for haulage of rock mass from lower levels up to re-loading on rail transport; d'~ - Boolean variable, indicating using of conveyor transport for ore transportation from lower levels (1- uses; 0 - no); A_{lt} - reserves of commercial ore within limits of contours of ore extraction, m ; v„, - ore density, t/m ; ö,;n - reserves of rock mass in k'1' contour of extraction, m\

In the second part of the expression symbols y", A,"", /(/ and the others indicate costs, distances of transportation and other parameters for overburden from /'' sector of i'1' level. Conveyor lines service both ore flows and rock flows.

The following equations and inequalities present limits of economical-mathematical model of the problem:

$$(1 - f_{pl}^{l})\cdot A_{pl}^{or} / \gamma_{or} \le \sum_{i=1}^{N_k}\sum_{j=1}^{s_i}\sum_{k=1}^{n_{ij}} A_{ijk}^{or} x_{ijk}$$
$$\le (1 + f_{pl}^{h})\cdot A_{pl}^{or} / \gamma_{or}; \qquad (3)$$

$$\alpha_{min} \le \frac{\sum_{i=1}^{N_k}\sum_{j=1}^{s_i}\sum_{k=1}^{n_{ij}} A_{ijk}^{or}\alpha_{ijk} x_{ijk}}{\sum_{i=1}^{N}\sum_{j=1}^{s_i}\sum_{k=1}^{n_{ij}} A_{ijk}^{or} x_{ijk}} \le \alpha_{max}; \qquad (4)$$

$$b_{ij}^{wh} \le d_{ij} + \sum_{k=1}^{n_{ij}} b_{(i+1)jk} x_{(i+1)jk} - \sum_{k=1}^{n_{ij}} b_{ijk} x_{ijk};$$
$$i = 1, 2, \ldots, N_g - 1;\; j = 1, 2, \ldots, s_i; \qquad (5)$$

$$\left|\sum_{k=1}^{n_{ij}} b_{ijk} x_{ijk} - \sum_{k=1}^{n_{ij}} b_{i(j+1)k} x_{i(j+1)k}\right| \le b_{ij}^{wl};$$

$$i = 1, 2, \ldots, N_k;\; j = 1, 2, \ldots, s_i - 1; \qquad (6)$$

$$\left|\sum_{k=1}^{n_{ij}} b_{ijk} x_{ijk} - \sum_{k=1}^{n_{ij}} b_{i(j-1)k} x_{i(j-1)k}\right| \le b_{ij}^{wl};$$

$$i = 1, 2, \ldots, N_g;\; j = 1, 2, \ldots, s_i; \qquad (7)$$

$$\sum_{k=1}^{n_{ij}} b_{ijk} x_{ijk} \le B_{ij}^{wh} - b_{ij}^{wh};$$

$$i = 2, 3, \ldots, N_g; \; j = 1, 2, \ldots, s_i, \quad (8)$$

$$Q_{ij}^{min} \le \sum_{k=1}^{n_{ij}} Q_{ijk} \; x_{ijk};$$

$$i = 1, 2, \ldots, N_g; \; j = 1, 2, \ldots, s_i; \quad (9)$$

$$k_t^{min} \le \frac{\sum_{i=1}^{N} \sum_{j=1}^{s_i} \sum_{k=1}^{n_{ij}} (Q_{ijk} - A_{ijk}^{or}) \cdot x_{ijk}}{\sum_{i=1}^{N} \sum_{j=1}^{s_i} \sum_{k=1}^{n_{ij}} A_{ijk}^{or} \cdot x_{ijk} / \gamma_{or}} \le k_t^{max}; \quad (10)$$

$$\sum_{i=1}^{N} \sum_{j=1}^{s_i} \sum_{k=1}^{n_{ij}} A_{ijk}^{op} \; x_{ijk} \ge \frac{N_{op}}{12 \cdot \gamma_{or}} \; A_{pl}^{or}. \quad (11)$$

$$\sum_{k=1}^{n} x_{ijk} = 1; \; i = 1, 2, \ldots, N_g; \; j = 1, 2, \ldots, s_i; \quad (12)$$

$$x_{ijk} = 1 \text{ or } 0; \; i = 1, 2, \ldots, N_g; \; j = 1, 2, \ldots, s_i;$$

$$k = 1, 2, \ldots, n_{ij}, \quad (13)$$

where A_t - planned annual productivity of an open cast by ore, t: $;_{/il}$, J_{pl}- permissible standards of deviations of annual planned productivity by ore to less and bigger side, share of a unit; a_{ijk} - metal content in ore in k^{th} contour of mining operations ij block, %; a_{min}, a_{max} - lower and top limits of deviations of metal content in annual output from planned ore quality, $^c/<$; o_{ij} - minimum width of working bank of (/+//" bench at the top of // block, in: d_n - distance between i^{th} and /+/ benches up to the beginning of planning, m; o_{ijk} - advance ot mining operations at /"'bench in k^{th} contour, in;
"o + Dk - advance of (/'+/)"' bench from initial position when studying of k" contour of extraction ij block, m; ö$_{ij}$' - permissible mutual advance of lines of benches in adjacent zones (blocks); tf„ - determined distance from initial point of/"' bench up to temporary spoil bank in advance of ij block, m; ti_n - width of transport (safety) beim at a bottom of temporary spoil bank slope, m; u_{ij} - minimum necessary volume of mining operations in ij block, in : ki....., k'''''' - permissible boundaries of variations ot current stripping ratio to less and bigger side; A_{ijk} - volume of opened-up reserves between k'' contour in // sector and working bank of above bench, m; N_{op}, - standard ot opened-up ore reserves at open cast, months.

Balance limits of economical-mathematical model express the following. Deviations of plan by ore from required indexes are admitted in fixed limits (restriction (3)). Restriction (4) regulates metal content in market ore.

Technological restrictions lake into account characteristic property of carrying out of mining operations at an open cast. For upkeep of normal width of working banks at benches of an open cast rate of front of mining operations advance at above level must be not less than rate of front of mining operations advance at lower level (restriction (5)). At extracting bench in adjacent sectors (blocks) it is required to save smoothness of lines of front of mining operations. This requirement of technology of mining operations is taken into account by restrictions (6) and (7). Advance of a bench may be limited by temporary spoil bank in zone of ij block. This condition is taken into account by inequality (8). Possible volume of rock mass extraction in contour must be not more than minimum necessary volume of rock mass extraction in block (inequality (9)).Variations of current stripping ratio plans of mining operations are admitted in given limits (inequality (10)). Volume of opened-up reserves must ensure open cast operation with planned output during normative time (inequality (II)). Restriction (12) is caused by condition of contours forming. In every block only one contour is worked out or no one.

Calculating process of construction of boundaries of working bank includes the following operations. Having coordinates of limiting points/' zone on line (/+/)" bench from sequence of points U$_r$, y$_r$) (r = 1, 2 . . . .TJ by lower edge of bench point r(A$_r$, y$_r$) is separated by coordinate grid. Than some set of points is formed, abscissas of which on the main grid of coordinates lie between A$_r$ and A„, and ordinates - between v$_r$ and y„. In other words a number of points is produced, which lie in an area of the direct product of set A x B, $A_{r>l}$ = [\$_r$, A„], But = fvr. v,,]. And values A„, y„ are determined by boundaries ij block by front of mining operations. In received set of points A x B some subset M of points v(Ai- , v,) is selected for which the following condition is correct:

$$b_{ij}^{wb} \le \sqrt{(x_v - x_r)^2 + (y_v - y_r)^2} \le b_{ij}^{wb} + d_0, \quad (14)$$

where du - permissible value of variation from given value o$_{ij}$ distances from point, which must be found, r(.v$_r$, y$_r$) up to point v(.v„ y,.) of a set M.

In a set M new subset of points N cM is separated, distance of which up to points (A$_r$.ı, v$_r$.ı) and (A$_{r+}$i, w+i), adjacent with r(.i$_r$, Vr), not less than width of working bench o$_{ij}$.

If any point from direct product of sets A x B will be denote by (A;;',y ;'), and the shortest of distances from it up to adjacent with ri.v$_r$, y$_r$) points by R (A'; ,y$_;$ "), that as a result of all above-sited operations some number of points is formed:

$$\sigma_\rho^{(r)}(x_\sigma, y_\sigma) \; (\rho = 1, 2, \ldots, r)$$

under condition

$$\sigma_p^{(r)} \in E\{\pi\, x_p^{(r)}, y_p^{(r)}\, \phi \in M \mid R\,(x_p^{(r)}, y_p^{(r)}) \geq b_{ij}^{nk},$$ (15)

where *O],* - points of new line, lie opposite T[1'] point; /• - number of points of this line, falling within a sector oft[11'] point.

Further above-sited cycle is repeated with other values of rù„ y_r). until all given number T of points will be exhausted. As a result sequence of points will be received, which presents a boundary of working bank at a surface of *ij* zone. This procedure is present in all operations on verification of technological inequalities in economical-mathematical models. Since parallelism of adjacent benches is not caused by methods of carrying out of mining operations, standard of advance distance is taken between these benches in a point of their most approach.

Taking account of difficulty of adequate description of mining-and-geological conditions of deposit mining and their interaction with economical indexes, a problem of annual planning of development of mining operations is solved into two stages.

First of all, on the above-sited economical-mathematical model and solid-digital model approximate contours of extraction are determined by levels and blocks of an open cast. Then narrowing an area of a search of optimal plan of mining operations takes place. On the second stage search of solving the problem of planning is continued by man-computer procedures - in dialogue regime.

The main basis of search only solid-digital model of deposit and open cast becomes and system of automatic calculation of volumes and indexes of ore quality in given contours by sectors and open cast as a whole. Variants of contours of extraction "are drawn" by computer on plane of zone and level by all depth of open cast, and economic justification is carried out. And all roughness in configurations of ore bodies intersections, curvature of front of mining operations and so on are taken into account. If the best variant of plan does not bring to light, repeal calculation is possible with software complex of economical-mathematical model using. Only here boundaries of parameters and indexes, giving by inequalities, will be the most accurate and close real conditions. Carries out of planning, any person is carried out noted search procedures not blindly, but follow specific rules. These rules resemble algorithm of adaptation and teaching in automated systems of control by complex dynamic processes and objects. That is why here description is given of rules for optimal contours of extraction search in solid-digital model of open cast in terms and notions of these systems.

Search algorithm in recurrent form may be presented in the following form:

$$c[n] = c[n-1] - \gamma[n] \cdot \nabla J(c[n-1]).$$ (16)

where *c[n]* - realization of solving vector *c* as a result of *n'''* step of search; *c[n-\]* - value of this vector after preceding step of search; *y[n]* - some scalar, determining the next step of search; *J* - function vector *c = (c\. . . . .*CN). In given problem this vector conforms to criterion of optimum, that is to say total costs on mining and processing of ore when annual planning of development of mining operations. Gradient of function vector shows direction of changing of criterion index when searching of optimal decision on plan of mining operations. Vectors *c[n]* and *c[n-\]* conform to variants of combination of different contours of extraction on all blocks and levels of open cast. Value)L"1 defines quantity of the next step and depends on a number of a step and vectors *c[m] (in* = ii - I, n - 2, ...). Its quantity defines a set of steps of front of mining operations advance by the same blocks and levels that is to say by open cast. Minimum step conforms to distance between prospecting holes or, when quarterly-monthly and weekly-24-hourly planning, lo width of an excavator cut by pillar. In essence here iterative method is used, which we may call regular in difference with probable methods. These generalizations will be useful when we will change-over from search of optimal contours of mining operations "by hand" at the second stage to full automation of search of optimal plans of mining operations development. One of defining indexes in goal function of the model of annual planning is cost of loading and transportation of ore and overburden. In view of complexity of internal structure of mining-transport complex, and a fact that it is susceptible to influence of many casual factors, correct economical estimation of costs for loading and transportation of rock mass is carried out with a help of imitation simulation (Dzharlkaganov U.A. & Dzharlkaganov A.U. 2000).

Description of open cast's freight traffics it is the most convenient to carry out in terms and notions of multi-phases systems of theory of mass service with irregular channels and not equal in importance requirements. Forming of cost for loading and transportation of ore and overburden from open cast per time unit is described by the following assessing function:

$$W = \sum_{c=1}^{c1} C_c \cdot \sum_{n=1}^{N} n_c \cdot P_n + \sum_{g=1}^{g1} C_g^0 \cdot \sum_{m=1}^{M} m_c^0 \cdot P_{m^0} +$$

$$\sum_{g=1}^{g1} C_g^m \cdot \sum_{m_g=1}^{M} (m_g - m_g^n) \cdot P_{m_g} , \qquad (17)$$

where $C,$ - cost of being in line of application $e^{'h}$ type per unit of time; $g\backslash$ - number of types of applications; $n,.$ - number of applications $e^{'h}$ type in line and on service at all stages; f_a - probability of presence of n applications of $e^{'l'}$ type in line and on service at all stages; (. $_e$- cost of downtime of a channel of service $g^{'h}$ type per unit of time; g1- number of types of channels of service; (_ $_v$ - cost of operating of channel of service $g^{'l'}$ type per unit of time: M_v - number of channels of service $g^{'l'}$ type; m, - number of channels of service in downtime $g^{'h}$ type; r_m - probability of presence of application on service in channel $g^{'l'}$ type; , „- probability of downtime of channel $g^{'l'}$ type.

In expression (17) under many-types application loaded and empty trains or cars, transporting ore and rock are understood. Channels of service are all elements of transport network, sources and ends of open cast's freight-flows (face excavators, ore storehouses and rock dumps, re-loading points in open cast and at day surface and so on). In the first and the second components of the expression probable characteristics of excavator operations and transport flows and possible damage of downtime of the main equipment are determined by simulation. The third component presents costs on exploitation of this equipment and technological constructions. Cost of loading and transportation of a unit of ore volume and overburden, requiring for calculation of annual costs in annual planning by expressions (1) and (2) are determined with a help of assessing function (17) and other operations.

3 CONCLUSIONS

Distributions of volumes of rock mass by zones, technological blocks, and levels in annual planning are the basis of quarterly and monthly plans of development of mining operations. But in the last case the main purpose of planning is often achievement of the most stability of quantity of commercial ore.

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The Ground Control Management Plan of a Mine: Ovacık Gold Mine Example

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ABSTRACT: In order to reduce accidents, eliminate hazards and improving productivity it is essential to compile a best practice document whereby employees can perform their duties in a safe and healthy manner. The ground control management plan (GCMP) was drafted to describe the requirements of the rock mass control system in the Ovacik Gold Mine-Turkey. GCMP is regarded as part of mine's strategic plan to combat falls of ground and provide safer underground and open pit environment for the life of the mine. The principal aim of this study is to provide assistance to management in their formulation of strategies aimed at reducing the incidence, severity and damaging effects of possible rock-related hazards. This paper is briefly describes what kind of method of approach has been adopted at the Normandy Madencilik AŞ - Ovacik Gold Mine in terms of managing possible rock-related risks.

1 INTRODUCTION

The Ovacik Gold Mine is located 110 km north of Izmir-Turkey, and 12 km SW Bergama in western Turkey (Fig. 1).

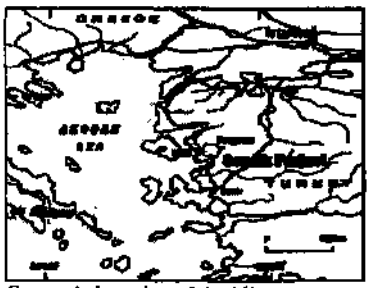

Figure 1. Location of the Mine

Past mining at Ovacik includes near-surface underground mining dated to Lydian and Roman times. The potential of the area was first discovered by Eurogold Madencilik in 1989, and economic mineralization has been identified on 2 outcropping epithermal veins, namely M and S vein. The mine is now owned by the Normandy Madencilik AŞ Company, a 100 percent owned by subsidiary of Newmont Mining Corporation.

2 GEOLOGY

The Ovacık area consists mainly of porphyritic biotite andésite with minor andésite breccia and debries flow, fluviaitle epiclastics followed by subaquesous shallow water dacite-rhyolite dome complex faciès. The dome complex faciès comprises a coherent lava hyloclastite mass flow and accrelionary lapilli, a subaerial andesitic dacitic lava dome complex faciès of coherent lava, autobreccias, volcanoclastics debris flows and fluvialile epiclastic debris flow with the latter fades hosting the Ovacık Gold deposit. The deposits comprise 4 outcropping epithermal veins transecting a large outcrop of silicified and argilised porphyritic andésite of early Miocene age. The veins dip steeply northwards and trend from NE-SW to E-W. At present M and S veins are known to contain economic gold bearing mineralisation. Average vein widths are 60m for M vein and 8m for S vein, however widths can occasionally exceed 20m(After Kara 2002).

The 4 epithermal veins have strike extends in the order of 400m and extend down dip for at least 300m. Orebody contacts are sometimes sharp but more often will be surrounded by a silicified transition zone or mineralized andésite of quartz veins.

3 GROUND CONTROL MANAGEMENT PLAN-GCMP

The main purpose of the GCMP in the Ovacik Gold Mine was to establish principles for the design and monitoring of the layout and support of mining excavations, so as to maximize their safety, stability and cost-effectiveness.

The GCMP is purely a description of the methods to be applied and procedures to be followed in conjunction with all aspects of mining and support strategies which can contributes towards the avoidance of rock-related accidents.

In order to maximize the clarity of he document, the main body has been broken into 6 main parts as follows:

- The mining environment, geology, seismological, and geotechnical properties of the rockmass enclosing the orebodies mined;
- The results of the rockmass response to mining as measured by an assessment of rock-related injury statistics;
- Rock-related hazards and risk management system based on all the above;
- Based on all the foregoing, the description of minimum performance standards by line management and the planning department to determine and enforce strategies to reduce and manage rock-related risks;
- A set of acceptable rock mechanics references that should be used in determining mining and support strategies to minimize and control rock-relate risks;
- Rock engineering services, monitoring quality controls and training

The GCMP for Ovacik God mine was intentionally non-technical, since it must be understandable by all employees of the mine. The technical aspects remained to province of the mining and planning departments, which will apply the acquired skills of their respective disciplines to the mining problems at hand. The main thrust of the GCMP was therefore at human systems: *management and communication.* These aspects require a lot of attention to implement and enforce good rock mechanics practice throughout the mine.

3.1 Fall of Ground Accident Analysis

The rock-related safety statistics for Ovacik Gold mine have been analyzed since underground and open pit came into operation and found that no rock-related accident has occurred during this period. Although the risk rating is very low under current conditions incidents may happen as the volume of mining operation in underground and open pit increase and poor ground conditions intersects.

Table 1. Rock-related accident statistic in Ovacık GM. 2002

Lost Time Injury Frequency Rate	**Serious Injury Frequency Rate**	**Fatality Rate**
0	**0**	0

3.2 Rock-Related Risk Management System

The rock-related risk management provides the basis for decision -making and enables management to create a safer environment. The principal purpose of rock-related risk management system in Ovacik God mine was as follows:

- to identify and assess the hazards to safety to which employees may be exposed while they are at work;
- to record the significant hazards to cerate safer and long term establishment.

Three types of rock-related risk management techniques were introduced for Ovacık GM namely:

/. Base line Risk Assessment

This will be done to identify major risk for future risk control such as analysis of historical data, accident reports, internet, information, sharing info between mines etc. These studies need to be comprehensive, and may well lead to further, separate, more in-depth risk assessment studies.

2. Issue Based Risk Assessment

As circumstances and needs arise, separate risk assessment studies will be conducted when, for example: a new support is introduced into the mine, after an accident or near miss incident, new knowledge comes into to light and information is received which may influence the level of risk to employees at the mine etc. The suppliers in Ovacık GM must ensure, as far as reasonably practicable, that the article is safe and without risk to health ad safety when used properly.

3. Continues Risk Assessment

This is the most important for all of risk assessment, which will take place continuously, as an integral part of day-to-day management of the mine. This will mainly be used by the front line supervisors in the Ovacık GM. for example checklists, audits, planned task observations, daily workplace inspection etc.

The baseline risk profile for rock engineering is presented in Table 2, followed by the risk-ranking scheme used for this overview. This profile will help define the objectives of the mine's mining department, against which achievement targets will be set and reviewed. It also highlights the rock-related safety issues that need to be addressed by other departments in the future.

After a baseline risk profile has been established highest priority risk areas are addressed in more detailed risk assessment. The method, which has been used widely in mines, is the ***WRAC (Workplace Risk Assessment and Control)*** technique. Using this technique, assessment can be done by a group or vertical slice of people from the workplace ranging from the person undertaking a given task to a higher level supervisor and is facilitated by personnel from Mine Planning Department. Hazards are identified by considering each step in the completion of a task and ranking the risk according to the probability of an incident happening and the likely consequence, as indicated by the rusk ranking matrix illustrated in Table 2.

Table 2 Risk Matnx-f (CxP)

	PROBABILITY				
CONS.	A	B	C	D	E
(1) Fatality-1					11
Serious Injury-2			8	12	
Disabling Case-3		9	13		
First Aid Case-4	10	14			
No Injury-5	15				

PROBABILITIES:

Common- (Dailv)-A. Likely- (VYeekly)-H, Happens- (Manlhly)-C, Unlikely- (Yearlj)-D. Rarely (1-3 >ears)-E

A risk ranking of 1 is the most serious and 25 the least serious. Thus all rankings from 1 to7 are critical and require urgent consideration, rankings 8 and 15 are serious and 16 to 25 are of lesser severity and should be addressed only when the more serious risks have been eliminated or controlled.

Table 3. Example of risk assessment using the WRAC techniques in Ovacık GM

Step	Hazard	P	C	R	Svstcm	RP
Application of support slandaids	Unsafe working areas	-	-	-	GCMP	PE

When the current system fails or is insufficient then recommended action needs to be developed in order to eliminate risk.

3.3 Rock-related hazards

Each rock type in Ovacık GM has been carefully identified and analyzed in order to develop strategies to combat rock-related hazards on site.

The hazards that have been identified in the geotechnical areas that were mined are summarized in Table 4 for decline development.

Table 4. Rock-related hazards associated with development rock types in Ovacık GM

ROCK TYPE	STRATEGIES
Hemalilic Quartz Breccia **(HBX)**	Hazaid levels are increased for all rock types where tunnels and decline aie developed or mined through faults and dykes due to presence of ground disturbed by lomting or smaller scale fracturing on the margin of these discontinuities
Hydrothermal Breccia composed predominantly of andésite in a siliceous to silica clay matrix **(ADX)**	Steelarch units in conjunction with 2.4m grouted swellex + 50mm fihrecrete with wire mesh to be applied wheie poor ground and self-mining conditions are intersected

4 EXCAVATION DESIGN

The underground mining method will be by decline access to the ore zones and the ore is then probably mined out by mechanized cut and fill sloping methods. The decline has a gradient of 1:7 and a cross-sectional profile of 5,2mx5.2m. The total length of the decline is estimated to be 1100m. Ore is accessed by crosscuts from the decline at 20m intervals. There are at least 2 crosscuts per level to enable multiple work areas and facilitate productivity. Other developments include vent drives, stockpiles and drill cuddies. Waste development drives have a cross sectional profile of 4mx4m and total length is 1200m.

Although *no final decision* has been made with regards to mining method, Ovacık mine intends *to* ***trial longhole open sloping using up-hole benching.*** The purposed sub-level interval is 20m (backs to backs), resulting in an approximately 25m stoping height (for a single sub-level). No final assessment has been made on the mining method. The principal extraction mining method will be based upon 100% extraction with complete recovery while allowing no perceptible surface subsidence. A crown pillar is planned to be left in order to protect underground environment from flooding.

According to up-hole benching method, the ore is developed by driving strike access drifts with a cross section of $25m^2$ along the hangingwall or footwall or in the center of the ore body to the boundaries. Stope preparation is carried out by driving sill drifts across the strike to the hangingwall or footwall. The dimension of the sill drifts can vary, which depends on the thickness of the ore body, and the location of the strike access drifts (example: 7m wide by 5 m high 35m length).

Slop production comprises the extraction of the for instance 25 high bench between two sill drifts. A drop raise is driven between sublevels at the end of the sill drifts. The raise is widened out to as slot, to create free breaking surface. The remainder of the beeches is blasted towards the open slot. The ore is mucked from the lower sill drift using LHDs. The open stope is then backfilled to the floor level of the upper sill drift. The backfilled floor becomes the mucking floor for the next lift. Once two adjacent primary stopes are completely backfilled, the intermediate primary pillar can be mined as a secondary stope. The primary slopes then become backfill pillars. The secondary stopes are also backfilled uppermost sill drifts in each main level are tightly backfilled to the back to support the back.

All other possible mining options are also being evaluated in order to determine optimum mining method.

4.1 Design considerations

The following issues were taken into account when determining method of mining: -

- Define Ground condition: Geology, drill hole data, lithology, mapping, ground water regime, seismicity
- Mineral occurrence: Continuity of ore zones within mineralized strata, grade, resources and mining reserve
- Ore body configuration: Dip, strike and shape
- Safety & regulatory: Labor intensity of method, degree of mechanization, ventilation, requirement, ground support regime, dust, gas and noise control, subsidence, air & water control
- Economics: Mineable ore tonnage, ore body grade, mineral value, capital cost, operating cost
- Political risks

4.2 Stope Ground Support

The objective of support system is to provide sufficient confinement to prevent management on joints and limit the growths of the fracture zone around the excavation so that displacements remain within stable limits during initial tunnel development and subsequent possible stress changes that characterize sub-level mining. The fracture zone will be protected by the support system from erosion by mining activities.

Support on the extraction level will be designed to ensure excavation stability during tunnel and draw points and as the sloping is moves backward. Thereafter, support must withstand erosion by mining activities over a period of several years.

The ground support system for in-stope development may consist of fibrecrete (FC-100mm), galvanized diamond mesh (M-5mm x5mm), standard rock bolting (RB) fully grouted split set (SS) 2,4m or Swellex (S) and 7m cable bolting (CB) in large excavations and intersections.

All waste drifts to be supported with FC or FC+M and Swellex. The hangingwall, footwall and central ore development access drifts may be supported with SS or S and additional RB. In addition, 7m long fully grouted CB must be installed in a pattern of 3-4 bolts per row depending ton the size of the excavation.

4.3 Blast design and practice

Blasts are designed by the mine's mining and planning departments. Notes with the detail of all blast designs will be supplied to the responsible shift boss. Smooth wall blasting techniques is highly recommended in jointed and fracture rockmass condition. In extreme cases JCB-rock breaker or rod header to be used for rock breaking purpose.

4.4 Large excavations

Such excavations must be sited in competent andésite where they will not be affected by stress changes associated with mining operations or adjacent excavations. Such an excavation should be at least 50m from the vein contact or 15m from an adjacent, large excavation. Such excavations are usually in use for several years and support must be designed to ensure the long-term safety of men and equipment in the excavation. Support design is aimed at preventing block fallout. Where excavation width exceeds 2,4m-rockbolt lengths must be increased to 3m with bolts installed on 1 m spacing. Fully grouted steel rope anchors should be interspersed with the rock bolts and installed on 2m spacing. Rockbolts and cable anchors (250kN) must be fitted with faceplates. If deemed necessary the area can be additionally supported with wire mesh, steel tendon straps and fibrecrete.

5 DECLINE DEVELOPMENT

The decline portal was established in a shallow 'box-cut' into the side of the hill. The portal floor elevation is I045RL.

The decline development takes place in jointed and clayed andesitic formation. A 5,2m x 5,2m wide decline access was developed and fully meshed and shotcreeted (100mm) in conjunction with swellex

(2,4m) bolts. Steel sets (2m toi, 5m apart) were also installed and encapsulated in shotcrete.

Cross cut development (lateral) in the decline system will commence once decline system is sufficiently advanced to permit initial driving on ore and establishment of underground exploratory drilling positions. The main ramp system is currently developed at 1:7 gradients down with a cross section of 25 m^2.

5.1 Support Design considerations

All designs for the support of excavations will take into account the following 4 factors: -

- Site investigation: geological data, rockmass characteristics, area! coverage, dynamic loading requirements-yieldebility, stiffness, support resistance concept and economics
- Initial design: Nominate designed units
- Installation: excavation, site inspection and design check
- Monitoring: feed back from production personnel

5.2. Ground control analysis in Ovacık site

5.2.1 Life of excavation

All long term establishment will be supported by fully grouted CT or Swellex bolts or other corrosion resistance units in conjunction with fibrecrete. Split sets will be installed in all temporary excavations in conjunction with 100mm thick of fibrecrete. Steel arch units will be installed when intersecting bad ground conditions.

5.2.2 *Discontinuities*

Support density, areal coverage (wire mesh, fibrecrete and washer for rockbolts) and resistances will be increased when approaching weak ground and mining through dyke structures.

5.2.3 Potentially unstable blocks

Any potential wedge and block failures are stabilized with a combination of fully grouted CT bolts, fibrecrete and mesh and RSJ sets. For large excavations and intersections 7m long cable bolts will be used to stabilize potential large failures.

5.2.4 Ground water control

Corrosion will be associated with ground water within the rock mass, particularly geological structures such as faults and dykes. In these circumstances consideration will be given to use special corrosion resistance steel materials or protective coalings. All rock bolts will be fully grouted to reduce the effect of corrosion. Additional drainage holes, sumps and pumping system will be used for this purpose.

5.2.5 Failure mechanism

Support units will be capable of eliminating the risk of both sidewall slabbing due to clay nature of the formation, wedge and block fall outs, tie and hold fractured or broken rocks and dynamic loading absorption (seismicity and blasting).

5.2.6 Monitoring and quality control

In order determine the actual performance of the support units pull test and visual inspections shall be carried on rock bolt units. Static and dynamic load tests on fibrecrete system will also be carried out to determine shear and axial strengths.

The pull tests will be performed on rockbolts that are as close to right angles with the development face. The supplier along with a mine's representative will carry out all test works, and all records will be kept in the office of the Mine Planning Engineer.

Table 5. Pull test work in Ovacık GM

Support unit	Bolts /month	Number to be tested
Split set	-	1%
CT bolt	-	1%
Swellex	-	1%
Cables	-	1%

Table 6.Recommended failure & test loads m Ovacık GM

Unit	Failure loadkN	Application kN
SS-39 (2.4m)	!30kN	80 % of FL
Super Swellex-4.1-52mm	200kN	80% of FL
CT bolt- (2.4m)	250kN	80 % of FL
Cables- (6m)	260kN	80 % of FL

Regular shotcrete tests are carried by the local university and supplier as part of quality management plan of the GCMP. The testing procedure was also designed in accordance with the provisions of European specification for sprayed concrete in order to achieve best result.

Table 7. Recommended test works on shotcrete Afttr every 50m³ of shotcrete sprayed in the excavation

Type of **control**	**Optimum value**
Compressive .strength	IOMPa24houis. 17MPa7days, Min25-30MPa28days
Energy absorption	Min. 500J
Quantity	Aggregate, accelator. cement, water and fibie content
Tensile strength	Min.2 MPa-28 days
Bond	Equal to tensile strength
Thickness	As designed--!-lOOmm

Table 8 Short summary of geomechanical classification in Ovacık GM

(1) Parameter	**(,2) Range of values**					
Strength of intact rock material- UCS-Mpa	> 25(1	11» 250	51)- KH>	25-50	5-25	1-5
Rating	15	12	7	4	2	1
RQD-%	90-1(H)	75-90	50-75	25-5(1	<25	
Rating	2(1	17	n	S	1	
Spacing of discontinuities-mm	>2	(16-	2(H)	r,0-2(KI		
Rating	2(1	15	10	«	5	
Condition of discontinuities						
Rating	.1(1	25	20	10	0	
Ground water-L/min	C-dry	<KI Dj mp	10-25 Wc (	25-125 Du ppı ng	>125 Flowing	
Rating	1'	1()	7	4	0	

Table 9. Risk rating for rockmass

RATING	CONDITION
100-R3	Excellent
75-90-R3	Good
50-74-R2	Fair
26-49-R1	Poor
0-25-R1	Very poor

It should be noted that the performance of a support element within a support system is highly dependent on the interaction between the support element and the rock mass.

In Ovacık Gold Mine, the tunnel support systems will be made up of reinforcing elements, (such as grouted split sets, swellex or CT bolt and cable bolts) that act directly with the rock mass to increase its inherent: and support elements, fabric support or coatings (such as steels sets, mesh and shotcrete) which act to contain the inherently unstable rock mass between the reinforcing units.

Table 10. Design table based on RQD values

Condition	**Primary Support** System	**Secondary Support System**
(2) **RQD** < 50-R1	lOOmin fibrecrele and mesh +2,4m fully grouted SS with combi washer	CT Bolt + steel arch where dead weights are expected
RQD-50-75-R2	70mm fibrecrete +split sets	2.4m tully grouted rockbolts where necessary + steel arch
RQD > 75-R3	50mm fibrecrete	2.4m fully grouted rockbolts

6 OPEN PIT STABILITY

In order to improve the stability of a slope in the open pit area. Ovacık mine has established a reliable prediction of the slope behaviour up to including the final failure so that appropriate action can be taken to minimize the danger to men and equipment.

There arc different systems of measurements to measure displacements in the open pit area, which will depend upon the magnitude of the anticipated movements, local site conditions and availability of staff and equipment. Ovacık GM has sufficient resources to establish the system of displacement monitoring. Mine planning department in conjunction with the survey department will carry out monitoring of displacement measurements in the pit area.

(n order to determine surface movements as a result of mining activity 35 ground control monitoring stations on permanent critical locations-*hot spots* have been established, and topographic record of each station.

Monitoring shall be carried out once a month or such shorter intervals, as the mine-planning department may deem necessary.

7 CONCLUSION

The Ovacık Gold Mine is the only operating role model gold mine in Turkey. In terms of rock engineering point of view all safety concerns (both employees and environment) have been taken into account when designing excavation and support requirements. The current ground control management plan-GCMP was purely designed to combat and eliminate possible rock-related hazards on site. Therefore, the GCMP must be implemented and annually reviewed by the mine management at all time. The same approach needs to be designed for all other mines as well.

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Dry Gas Injection or Underground Gas Storage

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ABSTRACT: In almost depleted oil reservoir the high gas injection can help to produce additional oil, if the process is in immiscibility or miscibility conditions. By using this reservoir for underground storage of natural gas the substantial quantities of oil can be recover. For that reason, the simulation both, high gas injection and underground gas storage has been performed. The two studied possible ways of future oil field management result by enrichment of dry gas with valuable components as underground storage of gas, rather than oil recover by high gas injection.

I. INTRODUCTION

When the oil field is almost depleted, there are some ways of using it. One way is gas injection for miscible displacement and to enhance oil recovery, or using it as underground gas storage.

By the process of gas injection under miscible conditions, the total oil recovery also includes vaporized hydrocarbon from residual immobile oil, in addition to the oil produced by direct displacement. The process is complex and involves the influence of the interaction of extracted hydrocarbons and reservoir oil at the displacement front. Therefore, the final oil recovery under miscible conditions is higher than "conventionally" displaced oil.

By using the reservoir as underground gas storage of natural gas it possible the substantial quantities of oil can be recover.

In this paper are two studied ways of future oil Held management, as additional recovery by gas injection and is the reservoir potential candidate as underground storage.

2. FIELD HISTORY

Zutica oil field is located 45 km SE of Zagreb. It is asymmetric anticline sticking from the northwest to southeast.

The sandstone layers of "A" series cover practically the entire area of the Zutica field. All layers have a common gas-water contact, and oil water contact in the deepest part. This deepest section of reservoir A|.i contains a large gas cap and an oil ring. The oil ring covers a relatively large area of heterogeneous litho logic composition, causing variations of the net pay thickness in different parts of the ring.

Three intervals were separated within the oil reservoir A|.j defined as independent exploitation objects. The production history was short and can be divided in two phases:

Phase 1 - test production, which was short due to very fast movement of gas from the gas cap into oil ring, which simultaneously reduced the productivity of wells.

Phase 2 - maintaining reservoir pressure by injection of dry gas in the gas cap, obtaining the immiscible conditions of the injection gas with the reservoir oil.

3. METHOD OF DRY GAS INJECTION

The aim of this research was to simulate the process of oil production in the oil field named Zutica, by maintaining reservoir pressure, and to define is the process in miscibility conditions. To simulate the process, a one-dimensional reservoir simulator COMP 3 was used. Only, the thermodynamic aspect of the process is investigated.

Methods of calculating multiple vaporization contacts with an Equation of State determine the miscibility conditions by simulation processes as a vaporizing or condensing gas drive.

Changes of Zutica oil properties by maintenance of constant reservoir pressure calculated for various pressures ranges from the initial saturation pressure (128,5 bar up to 2(X) bar shows that injected dry gas is poorly dissolved in already saturated oil. Swelling of oil is about 6%, while the oil density decrease by 3%. Fig.1.

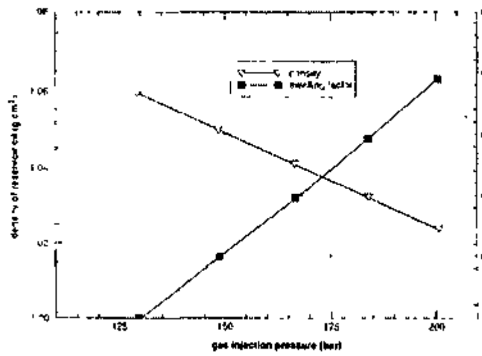

Figure I Changes in properties of reservoir oil during gas iniection

According to the criterion (Yelling &Metcalf) the minimum misbility pressure (MMP) is that particular gas injection pressure when 1,2 P.V. of injected gas displaced over 90% of present oil. To defining a characteristic multiple contact miscibilily between injected fluid and oil the "slim tube" tests has been performed.

Results of these tests are shown in Fig.2. which shows that the miscible condition in the system saturated reservoir oil and methane, respectively dry natural gas, can be achieved only after application of very high pressure (MMP of a system is 500 bar). In other words, in oil production from the Zutica field, the regime of pressure maintenance should make modest contributions by multiple contact vaporization to oil displacement, since in the interval of real applicable pressures of gas injection the process will proceed under immiseibility conditions.

To produce more oil, the pressure in the reservoir must be maintained by injecting another fluid. Oil displacement by maintaining reservoir pressure by dry gas injection at an actual pressure of 130 bars occurs under immiscible conditions in accordance with the expected phase behavior of methane - oil system, because the minimum miscibility pressure (MMP) of injected gas in reservoir oil is about 500 bars. If the process should be performed at higher pressure (up to maximum possible reservoir pressure of 200 bar), it cannot be expected to produce a greater contribution to miscibility displacement in the total production.

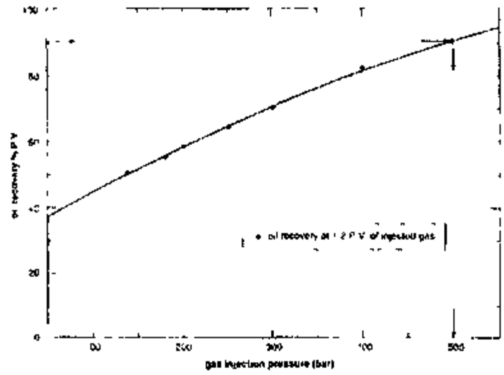

Figure 2. Determination or minimum miscibility pressure (MMP)

4. UNDERGROUND GAS STORAGE PROCESS

During the process of storing gas in depleted gas reservoirs there are practically no differences in composition between the injected gas in the fill phase, and from the gas that was produced during the production phase of the underground storage. However, by using partially or completely depleted oil reservoirs for gas storage, gas composition in reservoir changes. When injected gas comes in contact with residual oil, changes in phase behavior of new hydrocarbon mixture occur. The result of changes in phase behavior is vaporization of one part of hydrocarbons from liquid (oil) to gas phase so a change in content of C:+ component in produced gas occurs. This enrichment is desired effect, because it increases energetic and potentially economic value of stored gas.

Simulation of gas storage has been conducted using following parameters or assumptions:

- Reservoir fluid of oil ring is presented by composition of oil well samples, sampled under static condition.
- Present oil/gas ratio in gas cap is about 1:45 (vol./vol.)
- Maximum pressure variation of fill-in/emptying phase is 50 to 200 bars.

In the gas injection phase, the final storage pressure is systematically increased to its maximum value.

Empting phase has been simulated as differential vaporization in one stage

Equilibrium composition of liquid phase at lowest pressure of empting phase is initial composition for the next cycle.

During the fill-in phase, partial dissolving of gas in oil occurs in every cycle. Composition of oil changes after each cycle of fill-in/empting, due to step-by-step vaporization, and regarding that swelling factor and density of remaining oil also changes.

Table 1.

	injected gas	produced gas		
Component		cycle 1	cycle 2	cycle 3
ethane	1.3	9.0	5.3	3.5
propane	0.2	5.5	4.2	3.0
butanes	0.07	2.0	1.8	1.6
pentanes	0.02	0.56	0.53	0 50

In the first cycle, injected gas comes in contact with saturated oil under reservoir pressure (ph=l 29 bar), and dissolving of methane is small, which is represented by small values of swelling factor during the pressure increasing from 129 to 200 bars. In second and third cycle, swelling factor is twice the one in first cycle, while differences in these cycles are small.

Vaporization process of the light hydrocarbons from oil with mechanism of change in phase equilibrium at static condition is especially noticeable in change of oil density.

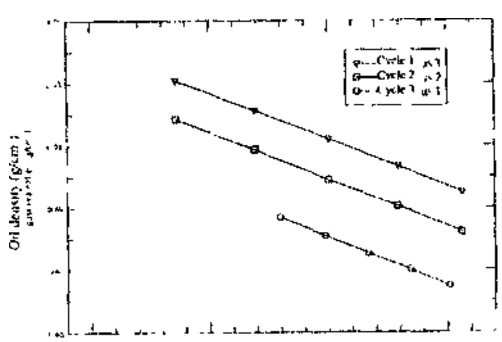

Figure 3. Change of reservoir oil density during gas storage

Effects of change of thermodynamically phase equilibrium are most responsible for enrichment of gas with C2+ components. In that process of enrichment, tendency to vaporization of individual component is proportional to its partial pressure at given condition of pressure and temperature, or, inversely proportional to its molecular weight. In concordance with that, increase in concentration in gas phase is greatest for ethane, smaller for propane, etc. It can be concluded that level of enrichment of injected gas, or, change in quality of vaporized hydrocarbon in each cycle of storage, is mostly function of material balance, i.e., total initial quantity of higher hydrocarbons.

5. CONCLUSIONS

- Optimal Peng-Robinson equation of state model of phase behavior of Zutica oil was used in 1-D reservoir model for simulation studies of oil production process for maintain reservoir pressure, as well as simulation study of gas storage in same reservoir.
- By processes of gas injection into reservoir oil, the composition *of* fluids in the area near critical point distinctly differs from the composition of the original reservoir fluid.
- Oil displacement by maintaining reservoir pressure by gas injection at an actual pressure occurs under immiscible conditions, and the contribution of multiple contact mechanism of hydrocarbon vaporizing in total oil displacement is negligible.
- Results of gas storage simulation showed: that during the first few years of storage it can be expected enrichment of injected gas due to favorable effects of phase behavior change within working pressure range.
- Using the reservoir for underground gas storage, in this case has additional benefit in increasing value of stored gas due to (1) increase of caloric value, (2) possibility of extracting valuable hydrocarbon components.
- Out of two studied possible ways of future management of Zutica oil field, results of this study favors using the reservoir for gas storage rather than continuation of oil production with high-pressure gas injection.

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Optimized Open Pit Mine Planning and Scheduling

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ABSTRACT: A complete Mine Planning solution often requires the integration of different "specialist" packages. It is necessary to understand what each package brings into a system and how they relate to each other in the mine planning and scheduling process. In isolation, each package may fall short of the final solution. However, when integrated with strategy, they form a powerful Mine Planning and Scheduling system. A step-by-step procedure is presented as a "model" for the mining engineer to follow in arriving at an optimized mine plan. The optimization process presented in the paper is capable of achieving all possible schedule objectives given the limitations in design and deposit. Obviously, as the deposit is depleted, the possibility that certain constraints can be met is reduced. This requires careful consideration in the scheduling optimization cycle.

I INTRODUCTION

l. I *The Concept*

In recent years, it has become a common approach in mine planning to use multiple software packages to achieve better results. The approach described in this paper consists of a general mining package and two mine scheduling and optimizing packages that operate on different stages of the optimization and scheduling process (Fig. I).

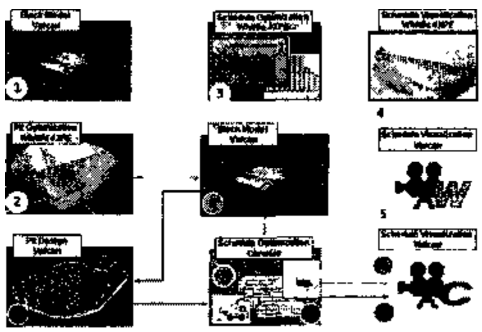


Figure I Optimized mine planning and scheduling model

1.2 *Software Components*

Three major software packages are discussed and illustrated here:

Vulcan's Envisage program is used as the General Mining Package (GMP). This software is used in basic block model preparation, manipulation, as a data transfer medium, and for schedule visualization;

Pit Optimization and first-pass Schedule Analysis is performed using Whittle 4X Multi-Element (Analyzer). Various features are showcased including the user friendly Proteus GUI and the new Blending Module. The Milawa Scheduling algorithm is used in an example case study.

Final Schedule optimization and blending is performed in Vulcan's Chronos Scheduling and Optimization package.

1.3 *The Model*

There are three "streams" in the mine planning and scheduling model:

Conceptual Design and Scheduling in Whittle: This involves block model preparation, variable export/import, pit optimization, mine design, scheduling and visualization;

Optimizing and Scheduling the Conceptual Whittle Design in Chronos: This involves passing the pit design back to Vulcan through push-back variables, reserving the block model against these variables, building a Chronos scheduling workbook in Chronos, and scheduling using the Chronos Optimization module. Results are transferred back to Vulcan for visualization.

Final Pıt Design in Vulcan and Final Schedule Optimization in Chwnoi This is the most detailed and time consuming stieam in the model It involves the use ol all previous results to i) cıeate a final phased pit design in Vulcan, n) transtet to a Chronos scheduling woikbook in) optimize and schedule in Chronos and iv) visualization in Vulcan

In the lollowing paıagıaphs these stieams will be discussed in detail using an example case study from a Banded Iron Formation (BIF) deposit in the US (Slade 2001)

2 CONCEPTUAL DESIGN AND SCHEDULING IN WHITTLE

2 1 *Pi cpaıatıon of the Bloc k Model m VULCAN*

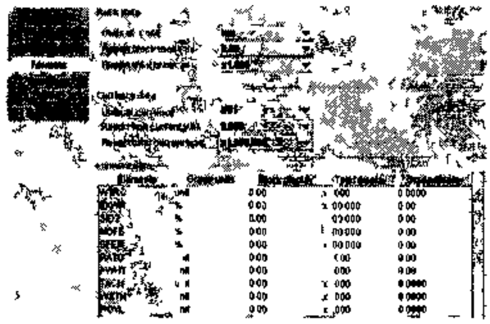
Whittle lequires a numbei ot vaııables to exist in the VULCAN block model These vaııables necessary tor pit optimization aie the following

1 A pit slope variable containing 'slope zone" numbers can be used Alternatively slopes can be set by rock type
2 A lock type variable ıs necessaıy tor the ditteient matenal types eg OVB WST ORE HGR, LGR
3 Whittle 4X Multi Element does not lequire a Net $ Value Product element guides (eg Au Ag, Cu, Fe and Recovery) are passed directly from Vulcan to Whittle and used within the progiam to calculate 'block values
4 Ten element variables can be set up toi just about any function you require The obvious pioduct elements might include Au, Ag Cu, or Fe Less obvious elements might include PIT (used for haulage calculations), ROYL (used to calculate loyalty costs), RECV (recoveiy variable used to determine recovered pioduct) The advantage ot this Multi-Element teature is that it is easy to perform What-if and sensitivity variations without having to go back to the ouginal block model

2 2 *Vulcan Mode I Cxpoi I to Whittle*

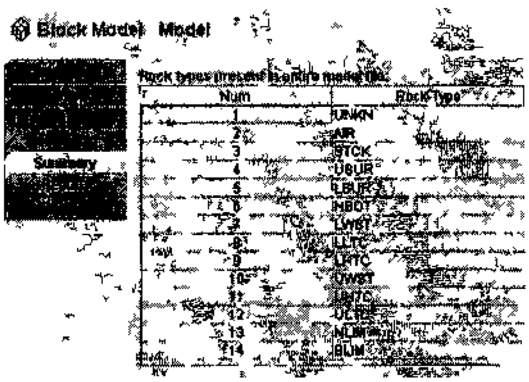
In this step we cxpoit pit optimization variables to Whittle MOD format The Foimats tab in 4X shows the 10 elements expoited from Vulcan (Fig 2)

In our BIF example the pnmaiy product grade is WTRC and is described as ıecoveıed pioduct tons divided by piocessed ore tons ' Rock types LLTC and LHTC correspond to the mineable low giade' and high grade ores respectively UHTC and ULTC are un-leased ore giade matenal and therefore excluded from the ultimate pit and scheduled mining All other rock types are waste matenal (Fig 3)



ELEMENTS
WTRC = Product Wt Recov
EXWR = Exploration Wt Recov
SIO2 = Silica in Concentrate
MGFE = Mag Suscept Crude
SFER = Indication of Oxidation
RATO = Indication of Oxidation
KWHT = Grindability measure
TACH = Ore Haul unit Cost
WSTH = Waste Haul unit Cost
ROYL = Royalty Cost

Figure 2 Foi mats tab in Whittle 4X



Fi gui e 1 The Summaıy lab shows the 14 rock types ex potted

2 3 *Pit Optimization m Whittle 4XPE*

Setting up ot the relevant expressions, cost model and revenue factors is required to ıun pit optimization in Whittle 4X Proteus Environment Once the setup is complete optimization can take place in a step by step procedure that is displayed graphically in a tree-like graph (Fig 4)

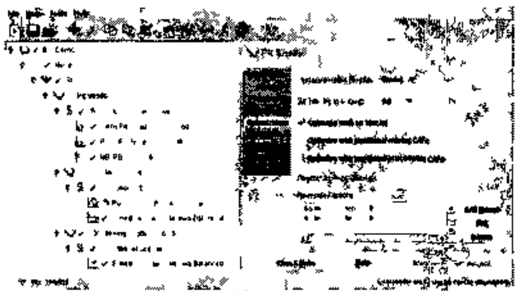


Figure 4 Optimization in 4X

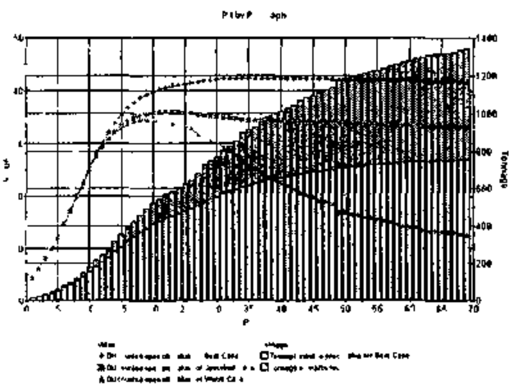
Push Back	Original Tonnage	Revised Tonnage	Variation	Percentage
	x1000	x1000		
1	148802	270093	121290514	61 5
2	244410	264073	19662842	8 0
3	232560	208783	-23776476	-10 2
4	331450	214346	117104150	-35 3
Total	957222	957294	72730	0 0

2 4 *Whittle Pit Design Selection*

This step constitutes the "conceptual" mine design sequence

1 Ultimate Pit we select an Ultimate Pit from the Pit-by-Pit Giaph There are many ideas on how to achieve this Some include

- Max Best Case Value,
- Required Tonnage,
- Product Price Point,
- Max Specified Case Value



Figuie 5 Pit by pit graph in Whittle 4X

2 Push Backs (Phases) we select a sequence ol Push Backs (Pit Phases) Irom the Pit-by-Pit graph (Fig 5)

3 Minimum Mining Width Whittle 4X allows the user to apply the concept ot 'Minimum Mining Width' (MMW) This has the effect ot "redistributing' tonnage between the chosen Push Backs m oidei to accommodate the MMW

Also, using MMW renames the Push Backs tiom Pit Number to Push Back numbei staiting from 1 (Fig 6)

The above leport shows the tonnages present in each Push Back - originally atter selecting the pits and atter the MMW is applied It is a "juggling" act to apply MMW and find a balance between the Push Backs In this case, a balance was found using Pits 9, 16, 25, and 37 A MMW ot 300 tt was used in this instance

Care must be taken when applying the MMW function, especially when optimizing existing pits Re-distributing tonnages can have the unexpected result ot "covering" up exposed inventory present in operating benches

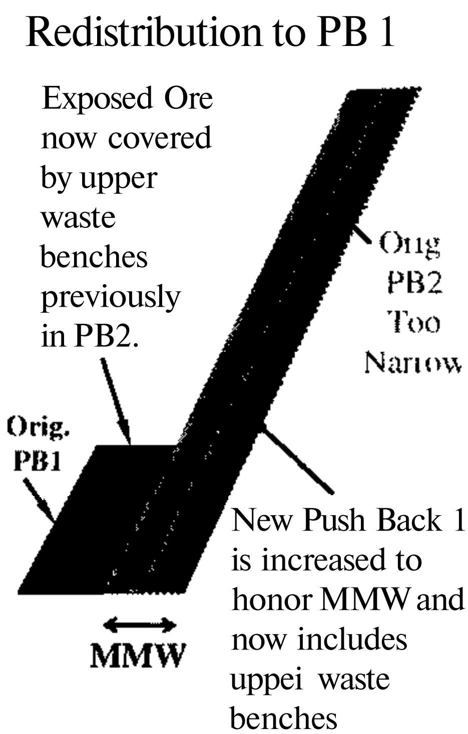


Figure 6 Pushback tonnage redistribution

2 S Sc hedulmg m Whittle

Using the Push Backs (Pit Phases) selected trom Pit Optimization, schedule using three techniques available in Whittle 4X Analyser In the case ol the BIF Pioject scheduling objectives wete

1 Must achieve stated product output - X MT,
2 Minimize and balance total mining (eliminate stripping spikes)
3 Limit the oie throughput to a maximum ot Y MT
4 Blend the High Low grade ores to **70% W%**
5 Maximize NPV

Whittle allows the scheduling ol the conceptual mine design inheient in the Push Backs and Benches The physical problem is set up. with all the spatial integrity ot pit slopes and Bench-Push Back Precedence automatically in the Whittle model

I Fixed Lead

This schedule mines out the Phases sequentially, with a hxed lead between benches in adjacent Push Backs (PB) A zero lead specified in Whittle mines out and completes each PB sequentially This technique is often teimed Best Case' (Fig 7)

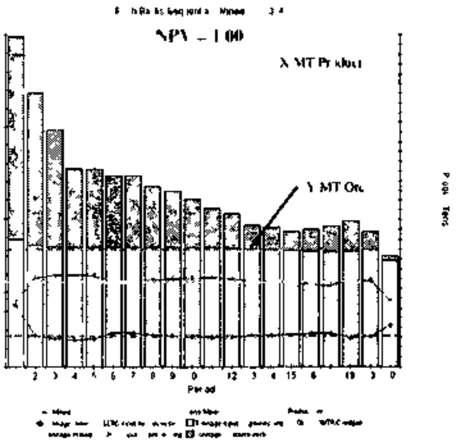


Figure 7 Fixed lead scheduling

2 Milawa NPV Maximizer Algorithm

This scheduling method mines die benches in an optimum sequence maximizing NPV Note the inuease over 'Best Case" is +36% (Fig 8)

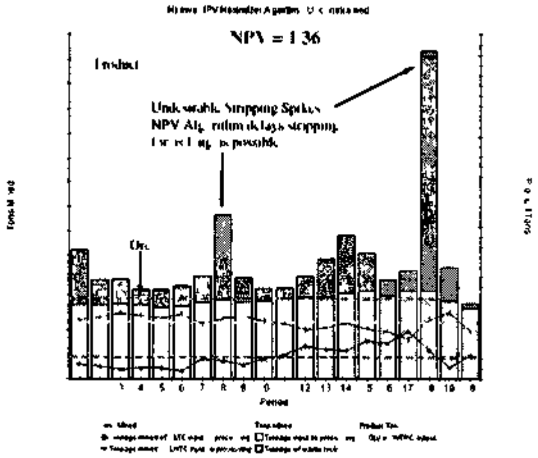


Figure 8 Milawa NPV maximizer .Ugonlhm

3a **Milawa Balanced**

This technique mines the benches in an optimum sequence, attempting to balance out mining requirements, while trying to maximizing NPV Note that this has come at a cost ot 21% in NPV compared with the Milawa NPV method (Fig 9)

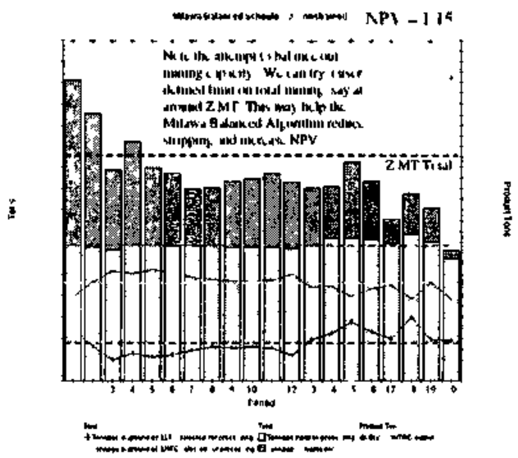


Figure 9 Milawa balanced scheduling algorithm

3b **Milawa NPV and Sequential Lead Schedules with a Mining Limit**

Apply a Total Mining Limit (Fig 10)

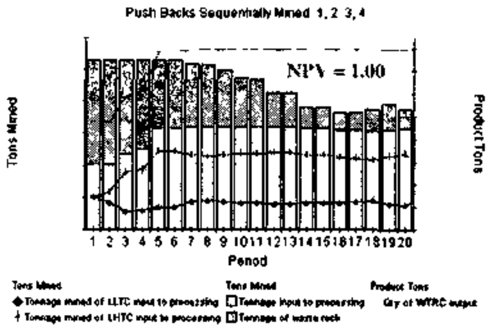


Figure 10 Milawa NPV and Sequential Lead schedules

Sequential Lead

- Fails to meet Product tons in early years;
- Fails to meet ore tons in early years;
- Fails to blend ore types in early years:
- Unworkable in the critical early years.

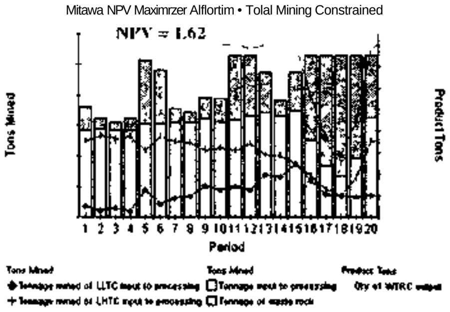


Figure 11 Milawa NPV

Milawa NPV

- Stripping minimized in early years;
- Fails to meet ore production in later years;
- Fails to meet Product tons in later years:
- Ore type blend is very poor;
- Basically unworkable - yet makes the most money? (Fig. 11).

3c. **Milawa Balanced with Mining Limit**

Limit total mining to Z MT. Note that we now have a "balanced schedule" which meets the #1 objective ot making required product tons. Objectives #3 (ore mining) and #4 (ore type blend) are still off target (Fig. 12).

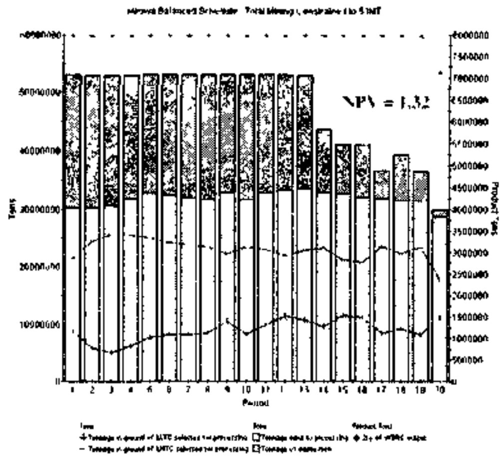


Figure 12 Milawa balanced with mining limit

2.6 *Blending in Whittle*

In 4X Version 2.2, Whittle have added a new blending module. The parameters for the example BIF problem were entered and the module applied. We are able to make the 30:70 Low.High grade ore blend. However, it is at the expense of other schedule objectives, i.e., Product and Ore Tonnage targets (Fig. 13).

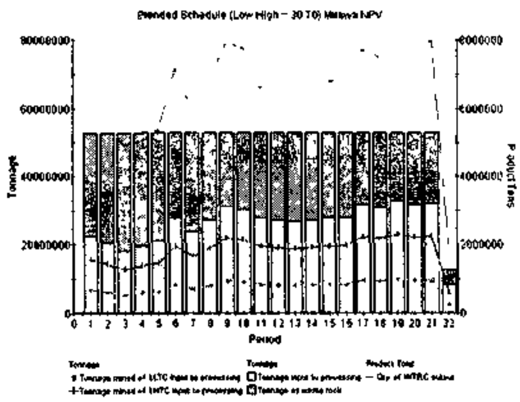


Figure 13 Blending in 4X

2.7 *Visualization in Whittle*

Whittle 4X Proteus Environment includes a 3D Visualization module. A "Mining Sequence and 3D Visualization file" is specified on the *Definition* tab for Whittle Schedule graphs.

The mining sequence file produced (.MSQ) can be read by the 3D Visualization module. A pit shell representing the mining progress to the end of each scheduled period is displayed and can viewed interactively. It ıs also possible to display colored block model slices along with the pit shell suiface (Fig. 14).

Fıgure 14 Visualization in Whittle 4X

2 8 *Whittle Schedule Visualization in Vulcan*

The Mining Sequence File (MSQ) generated in Whittle contains all the information necessary to create a Vulcan Animation. However, currently the relevant tntormation must be extiacted using CShell and other programming utilities. Pre-processing is as follows:

1 Export the MSQ file from Whittle 4XPE.
2. Remove all the "parcel" records using GREP with a reverse search on the PROCESSING path held (last field).
3 Run the "penodspht.pl" PERL script on the "PP37_MW300.rep" tile to produce a series of mining "touch" files: "fileOl.rep", "file02.rep", "file03.rep", "file04.rep". The number of files produced, which contain data, represents the maximum number of bench/block "mining touches" found in the schedule (benches mined over a series of production periods). This procedure only handles up to 4 mining "touches" per bench.
4 Run the "convert_ijk.csh" CShell script which executes the "schedule.awk" AWK script to convert the **ijk** coords to xyz coords. The "schedule.awk" is customized to the translation implied in the Block Model header used and adjusted for the **ijk** offsets to block centroids
5 Import the ASCII files into the GMP block model. You will need 2 block model variables per "touch" file. One to contain the penod in which a block is first touched (eg, msqOl), another to contain the % mined in that period, and so on, for touch 2, 3, and 4.

Once the import is complete, the engineer can contour (Grade Shell) the "first touch" blocks variable. This shows the first period in which a block is "touched" by mining and gives the engineer some idea of the sequential order of "bench by bench" mining (Fig 15).

Hgure 15 Visualization ol Whittle schedule in VULCAN

2.9 *Conclusions*

Whittle is a very poweilul tool when the LG pit optimization tools are combined with the outstanding time-value-of-money (DCF) and scheduling analysis routines. They are very easy to set-up and use within the new Proteus Environment.

Many different types of schedules can be run on the chosen conceptual pit design (Push Backs or Phases). These allow the user to focus on the various scheduling objectives The execution of the various schedules is so fast that a user can run as many as necessary to diaw initial conclusions on the value of the prospect in hand.

The "Milawa Balanced" schedule has been shown to get closest to the schedule objectives for our example Banded Iron Formation project. Whittle 4X now provides a simple, yet powerful method for visualizing the Pit design and Schedule.

The scheduling output from Whittle can be imported into Vulcan loi animation.

Whittle's new blending routine enables the user to quickly establish the effect ot blending on the schedule.

Whittle scheduling does not provide a final solution to multi-variable scheduling constraints. Theretore, the next Model Stream is to export the Whittle design to Chronos where it is possible to apply constiaints to multiple variables, divide the conceptual design up into multiple pits, and solve complex blending problems.

3 OPTIMIZING AND SCHEDULING THE CONCEPTUAL WHITTLE DESIGN IN CHRONOS

3.1 *Preparing the Vulcan Block Model for Chronos*

This step provides Chronos more choices as it seeks an optimum solution to the multiple constraints applied. The actions included are:

1. Import the Whittle Pit Optimization results into the Vulcan model. This constitutes the Conceptual Mine design.
2. Add other "key" variables (in addition to **pushback)** required by Chronos for reserve block location. These variables are **pit** and **bench.** This will allow the engineer to separate the logical pits and allocate blocks to the benches identified in whittle.
3. Divide the deposit into pits using *solids flagging* in Envisage.
4. Add the block **"value"** variable (Cash Flow). Use the same formulation as that used in Whittle 4X. This must be verified.
5. Generate a *Reserve Inventory* for Chronos using *Block - Reserves Advanced.* This will include the key breakdown fields: **Pit_Pushback_Bench,** the **products** to be scheduled, the **grades** to be reported, and the **cashflow** resulting from the mining of each bench.

3.2 *Set up a Chronos Scheduling Workbook*

A brief outline of the steps required in setting up a workbook follows:

1. Start Chronos and open a new Workbook.
2. Import the Vulcan Reserve Dump file (.DMP) into a Reserve Sheet.
3. Insert derived columns into the Reserve Sheet.
4. Format the inserted columns and set the type and weighting fields.
5. Generate a Period Calendar and set the scheduling periods.
6. Create a single Chronos Destination called MINE. *This is a key step in that all mission critical variables must he defined here.*
7. Create a single Chronos Process called EXCAVATE.
8. Create a Process Lookup Table called TABLE.
9. Create a Push Back Ratio table called RATIO. This table sets, i) the minimum proportion of any one bench to be mined in any one period, ii) the proportion of the bench above to be mined in advance of the current bench.
10. Create a General Sheet and populate the lower half with an automatic Period Summary.
11. Set Block Precedences and check in the Reserve Sheet. Convert Precedences to Block Ordering Numbers.

3.3 *Run Optimization and Schedule the Results*

The secret of successful Optimization and Scheduling in Chronos is to approach it as an iterative "hands on" procedure. It is a cyclical process in which the engineer:

1. Identifies a set of schedule constraints, according to the schedule objectives, and saves them in named Constraint Sheets.
2. Builds a list of available mining benches in a Chronos Task.
3. Sets up the problem, specifying the periods to be optimized, the constraint tables to be used in each period, and other conditions on the optimization, eg, whether to allow partial mining, Push Back Ratio tables to be used, etc.
4. Creates the "bei" and "bco" optimization files which contain the problem (bei) and provide a file to record the result (bco).
5. Solves the optimization.
6. Loads the result (bco) into Chronos Result Tables (CRT).
7. Schedules the result (from the CRT) and observes the schedule in the Summary General Sheet.

Schedule Optimization in Chronos is an iterative process. The engineer needs to be constantly reviewing and testing the result while optimizing. Rarely is the problem solved by pushing a button and walking away. (Fig. 16)

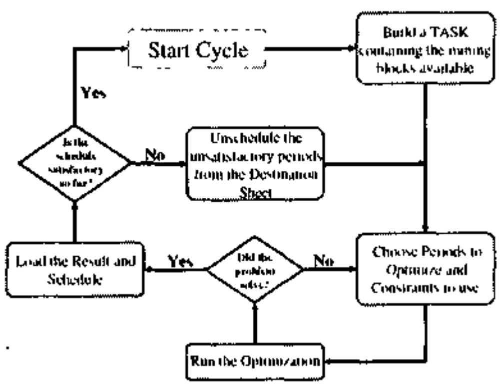


Figure 16 Chronos schedule optimization cycle

It appeared that there was further potential to reduce the upper mining constraint, thereby increasing NPV. The only constraints used on this schedule are the range of Gold Metal production, and Ore less than 4.75Mt. The objective is to test what the project can theoretically create in NPV.

Attempt to balance the Total Mining tons by carefully reviewing the existing schedule and finding the "hurdle" points of the schedule. The result of some trial and error optimization mns, using constraint variations, is a set of Total Mining constraints which solve. The illustration below shows the Optimized Schedule 02 (Fig. 17).

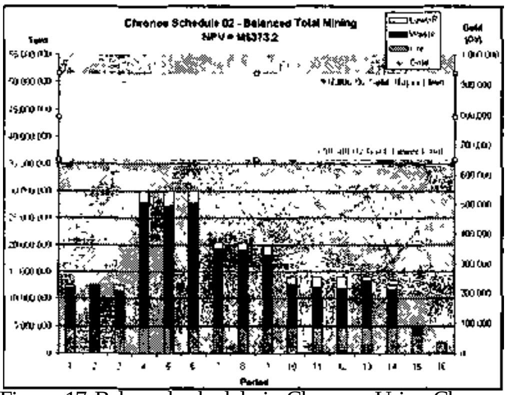


Figure 17 Balanced schedule in Chronos. Using Chronos Optimization the schedule can be balanced for only 0.3% loss in NPV

3.4 *Create an Animation in Vulcan for Schedule Visualization*

VULCAN provides functionality for creating schedule sequence animations. In our example we need to grade shell each individual mining bench per pit, pushback, and bench. Because the Chronos reserve was taken directly from the block model, keying off the three variables: **pit, pushback,** and **bench,** triangulations do not exist for the scheduled entities. A CShell script can be used to batch the Grade Shelling routine which creates the triangulations all in one step.

The next step is to rename the Pit-Pushback-Bench triangulations resulting from the Grade Shelling process to conform to the key field naming convention in the Chronos Reserve Sheet. The naming convention used on the BIF project is: **P<pit#>_<pb#>_<bench#>** e.g., P01_01_01.00t.

Finally we run a Perl script which builds a Vulcan animation file by matching the Chronos Reserve Sheet keys with the triangulation and displaying it in the period in which it was mined. The *Display Attributes* option in Chronos is used to color the bench triangulations by *Period.*

3.5 *Conclusions*

Using the designs generated in Whittle and the schedules generated in 4X Analyser, schedules can be optimized using any number of constraints in Chronos. The up-front scheduling work in Whittle is invaluable in setting the scene for the final optimization work in Chronos.

Vulcan provides powerful tools for easily subdividing the Whittle Push Backs into Multiple pit models. This provides more alternatives for optimization as illustrated in the BIF project.

The Chronos Optimization-Scheduling process is cyclic and should be viewed as an iterative process. Chronos Optimization is capable of achieving all of the Schedule objectives given the limitations in design and deposit. Obviously, as the deposit is depleted, the possibility that certain constraints can be met is reduced. This requires careful consideration in the Scheduling Optimization cycle.

Vulcan provides the engineer many ways in which to animate schedules from Chronos for visualization, checking and presentation.

4 FINAL PIT DESIGN IN VULCAN AND FINAL SCHEDULE OPTIMIZATION IN CHRONOS

All the previous schedule optimization is done to give the engineer an understanding of the problem, the possibilities, and the result to expect once this "stream" is complete. The results illuminate the path ahead so the detail design work can follow previous work based on sound principles (Rodriguez, 2001).

The user can maximize NPV (Net Present Value) and at the same time it can follow blending constraints, keeping stripping ratio at a constant rate until the later years in the life of the mine where it drops off and keep positive cash flow.

Figure 18 Graphical display of schedule in VULCAN

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la" International Mining Congress and Exhibition of Turkey-IMCET 2003, & 2003, ISBN 975-395-605-3

Designing of Information Model of Mines

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ABSTRACT: Examination of modern conditions of a process of simulation of separated systems (mines) shows that received results broadened our theoretical knowledge and were useful in designing. But they don't give full insight about behaviour of systems as integrity structure. In this connection it is necessary to carry out system-information examination of particular systems which will allow creating of model of ventilation system of a mine as an integral structure. For this reason, creating of information that supports the computer-aided models of conditions of mines' ventilation (CAM CMV) is necessary, which are constantly in operation. CAM CMV systems are sub-systems of systems of higher level, where information is received related to objects of mining production.

I INTRODUCTION

In process of working out in detail requirements to system of computer-aided models (CAM), a problem arises of designing of data base structures they include. The most important stage of this problem solving is information simulation of application domain. Information models are interface facilities between different categories of users and designers of the system. That is why, as a rule, they are worked out ignoring characteristic properties of physical representation of data.

2 DETAILS OF THE PROBLEM

Designing of information model is a multi-stage interactive process including design and identification of objects, determination and description of their properties; separation and description of relations; analysis of the model for fullness, redundancy and inconsistency; and its reduction to canonical form.

Designing of objects is carried out according to hierarchical principle by a way of step-by-step decomposition of complex objects into simpler ones. Detailing is carried out up to separation of classes of objects, which will henceforth study as elementary objects. Functions of computer-aided models of conditions of mines ventilation, which are typical of CAM, allow carrying out of designing of models of complex ventilation system from a set of elementary domains, air losses and designing of schemes of solving of problems of heat-mass transfer by means of a set of basis operations. Structure of CAM CMV is formed as informational-logical system and includes data banks, procedures, command processor and system of control of process of simulation. Data bank is intended for storing computer models of ventilation objects, numerical models of their schemes, aerodynamic parameters and archive information. Description of definitions of problems, separate sub-systems, from which graph of problems solving is designed, is carried out on formalized language. System of control of a process of simulation implements of users requests, plans and coordinates operation of other components. Command processor is a source of exchange of information between users and CAM CMV.

In a process of solving of problems, interconnection of adjacent ventilation levels is taken into consideration by a way of air moving through zones of caving, interaction with surface ventilation equipment and also interaction with surface zones of caving, which extend immediately to air workings and main workings. There is a possibility of imitation of ventilation equipment in adequate regime: districts, where mining operations are being carried out,' and zones of cavings in filtration regime of air moving from day surface into a mine at the expense of pressure differential. And as a base, aerodynamic scheme of ventilation of any objects (mines) with all their technological and technical parameters is taken. For implementation of so wide functional opportunities besides powerful software, statistical and mathematical apparatus working group is formed including specialists on mine ventilation, mine workers, programmers, mathematicians and physicists. This group carries out preparing and entering information into system, maintaining of data banks and procedures in a satisfactory state of operation, solving the system subject problems together with external users, evaluation and analysis of their results. One of the important

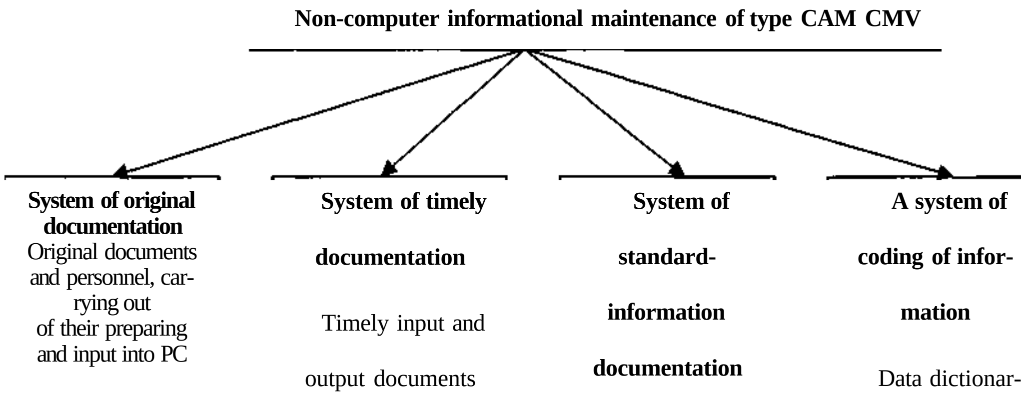

Figure 1. Structure of non-computer informational maintenance of type CAM CMV

problems of the working group is development and improvement of methods of problems solving, technology and exploiting of computer-aided models of conditions of mines' ventilation.

The foregoing information gives a common insight into the mathematical and organizing-methodical maintenance of the system. Now we will present characteristic properties of its information maintenance. In Figure 1 a structure is presented of non-computer informational maintenance of type CAM CMV. As seen, a system of original documentation includes input graphical and text documents, used for creation and identifying of model and personnel, carrying out of their control, record-keeping and monitoring, preparing and loading them into PC.

A system of timely information includes input and output documents, which are used in a process of a model operation for its revision and correction, description of solving problems. Output timely documents include results of problems decisions and also information about data, holding in a system. A system of standard information documentation defines sequence of development and function of CAM. The system's designer carries out entering, accumulating and applying changes for it in consultation with the working group. A system of coding is a set of data dictionaries. A special staff carries out their management and replenishing.

So, non-computer informational maintenance of type CAM CMV forms data, which are presented as graphical and text documents using or forming in a process of the system operation (non-computer information base). Administrative group carries out maintenance of the information base.

3 CONCLUSION

As seen, CAM CMV will ensure forming and solving of direct and reverse problems of simulation and also forecast control of conditions of ventilation regime of mines. Now, creation of three-dimensional models is provided for three-dimensional simulation of complex situations.

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System Dynamics Applications in The Mining Industry

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ABSTRACT System dynamics is soft operational research (OR) technique, used for reflecting information feedback characteristics of complex systems. The methodology identifies causal relationships between the variables of a system and investigates the effect of these relationships to the overall system behavior. Such representation of the system enables providing an insight to the system and allows construction and testing of alternative policies in order to achieve objectives of the system. In this manner, the methodology differs from conventional OR techniques, for the properties of complex systems, such as order of the system, loop multiplicity and nonlinearity that lead to dynamic behaviors in the system are reflected and explained within the models. The methodology has been applied to many areas, ranging from business, environmental and industrial policy research to medicine, energy policy research and engineering. This paper concerns with application of the methodology in the mining industry.

1 INTRODUCTION

System dynamics is a methodology that reflects the information-feedback relationships of the complex systems. The methodology has been under use since I950's and been applied to many business and engineering cases where an information-feedback system exists."(Gielen and Yagita, 2002; Stave, 2002; Anderson, 2001; Lyneis, 2000; Abbot and Stanley, 1999) The methodology, besides providing a better understanding of the system, also allows the design and testing of alternative policies for better managerial decision making. Like most other systems, mining systems can be represented as complex information-feedback systems, in a manner that the result of each mining operation affects itself back at the next point in time. Such a causal representation provides an insight for the understanding of the system and tests how alternative information inputs can change the performance of overall operations.

System dynamics is a widely applied OR technique in the energy sector, where the interactions between variables within the sector are dynamic and complex, and therefore, hard to identify and represent with other OR techniques. The methodology has been applied as a tool for energy policy research, both in smaller scale models, which try to determine the firm's policy to meet the energy demand, and in larger scale models, in which the energy policy of a country is tried to be evaluated. On the other hand. despite the existence of complex relationships between the variables of a mine, system dynamics is not a widely used modeling technique in mining industry and the applications of the methodology in the mining industry is restricted to production planning and finding the optimum conveyor belt capacity for underground collieries, so far.

The aim of this paper is to provide examples for the usage of system dynamics in the mining sector to illustrate system dynamics as an alternative approach to system investigation and policy construction and testing in the mining industry.

2 SYSTEM DYNAMICS METHODOLOGY

2. / *System Dynamics*

System dynamics, developed by Jay W. Forrester during the mid 50's, is a methodology used for reflecting the information-feedback characteristics of complex systems through the system's structure, the amplifications and delays contained within the system, such that it provides an insight to the system behaviour.

An information-feedback system arises whenever a decision leading to an action affects itself in return. Hence, the three characteristics of an information feedback system are:

- The structure of the system
- Amplifications within the system

- Time delays

The structure of the system consists of levels and rates. Levels are accumulations within a system, resulting from the difference of inflows and outflows thai take place in the level. They describe the state of the system upon which the decisions are made. A level may be inventory of goods, number of equipment or personnel, average demand or sales.

The decisions, which control the rate of change in the levels, are referred to as rates. The rate of change of the levels lake place as the flows between levels in a given time interval, DT.

Rates are calculated by using the information about the current state of the levels according to the rules defined by the decision functions. Decision functions are the equations that define how the system behaves. In return, rates determine the preceding state of the levels.

Amplifications, which occur in most systems, are the actions that affect themselves more forcefully than they seem to have at first glance in return. The reason of amplification is generally the policies, which define the flow rates.

Time delays are the periods of time during which the flow from one level to another is delayed. They may occur due to external factors (transportation delays, mailing delays, etc..) and during decision-making processes. In fact, a delay is a special kind of level within which incoming flows are stored.

2.2 *Model Construction in System Dynamics*

System dynamics models must reflect cause-effect relationships, be simple in mathematical nature, be able to handle large number of variables and the continuous interactions of these variables. There are six steps of model building in system dynamics. (Forrester, 1994)

2.2./ *Define the problem and identify the structure of the system:*

This first step of model construction involves identifying and defining the problem, setting the purpose of the model and specifying the borders. Next, relationships between the parameters of the model is built, which is achieved in two ways (Wolstcnholme, 1990):

If the behaviour of the system is known to the modeller, a causal relationship is constructed first and then the levels and rales within the system are identified.

If the behaviour of the system is not known to the modeller, resources within the system are identified fust, then the levels contained in the resources will be determined followed by the rates, which relate levels to one another.

2.2.2 Construct the Model Using Equations

Once the structure of the system is identified, the equation construction process starts. In system dynamics, there are mainly three types of equations:

Level Equations
Rate Equations (Decision functions)
Auxiliary Equations.

Equation construction is an iterative process, which changes over lime.

Level Equations

Let J, K, L be respective points in time, separated by an time interval DT. Then, value of the level at time K becomes the resulting difference of inflow and outflow rates during the time interval JK, plus the value of the level at time J.

LEV. K = LEV. J+ DT (Ratejn JK - Rate_out JK) (Forrester, 1961)

Rate Equations

Rates are calculated by using the information about the current state of the levels according to the rules defined by the decision functions. Decision functions are the equations that define how the system behaves. In return, rates determine the preceding state of the levels.

Auxiliary Equations

Auxiliary equations are equations, which convert flows from one type to another, and provide information to change or control rates. These equations are used for breaking down the rate equations into manageable parts so that ease in computation of other equations is achieved.

While evaluating these equations, delays become important too, since a delay is a special class of level where the outflow is determined only by the internal level stored in the delay. Time delays are represented by packages, consisting of combinations of level and rate equations that are inserted in a flow channel.

2.2.3 Simulation and Testing of the Model

In order to perform simulation, all terms of the model must be expressed in quantitative form. After constructing the model in such a manner, simulation is performed by various system dynamics softwares, such as DYNAMO, POWERSIM, VENSIM AND STELLA.

After building the model, its validity is tested. Here, the most important criteria is whether the model serves for the purpose/objectives or not. Besides that, the validity of the model is based on two foundations;

- The acceptability of the model as a representation of separate organisational and decision making details of the actual system.
- Correspondence of the total model behaviour to the system behaviour. There are two aspects in

model validation, firstly a philosophical aspect that concerns with the internal structure of the model, but it can not be judged in an objective and formal way for it is an qualitative approach, and secondly there are structural and behaviour tests.

Here, the former set of tests check whether the model is adequate to represent real world by comparing model equations with the relationships within the system and available theory, while the latter set of tests compare the model's behaviour with real world behaviour system under concern to check if a major structural error exists.

2.2.4 Construct Alternative Policies And Structures

Policy is defined as the relationship between the information inputs and resulting decision Hows. (Forrester, 1992) Once the model is constructed, alternative policies are simulated on the model to determine the policy with greatest benefit. The policies are generally generated by insights, from experience of the analysts and from proposals of the people involved in the real world system. In fact the major aim in constructing a dynamic model is studying the effect of alternative policies on system behaviour.

2.2.5 Educate and debate

After constructing a policy and performing the simulation, it is necessary to get people involved with the new policy so that they make contributions, reflect their ideas and experiences via education and debate sessions. In most organisations it is harder to gain people's confidence in policies than constructing the policy itself, so this step is of crucial importance.

2.2.6 *Implementation*

Once the education and debate is completed, the phase of implementing the results begins. The constructed and revised policy is implemented on the system. The current state jof the system is redefined after policy implementation.

3. SYSTEM DYNAMICS APPLICATIONS IN MINING INDUSTRY

Despite the wide application areas of system dynamics, it is not a commonly used methodology in the mining industry. Although any mine is a complex system, existence of uncontrollable factors like geology, climate., etc make it difficult to implement system dynamics in mining industry, therefore, the attempts for applications of the methodology in mining engineering context is limited.

3.1 Use of System Dynamics in Mine Planning

One of the initial works considering the system dynamics applications in mining engineering covers optimization of an underground colliery, in the paper 'The Design of Colliery Information and Control Systems' (Wolstenholme & Holmes, 1985)

The model developed in the study, identifies the colliery as a feedback system, in which the coalfaces and their associated development works are operated to reach a target output, under geological and manpower availability fluctuations.

In the colliery under concern, coal is extracted using retreat longwall mining technique, and the colliery is described at as a system for converting coal reserves buried underground to mined coal on the surface. (Wolstenholme & Holmes, 1985) The description of the system in this manner helps identifiying the boundaries of the system and the states in which coal exists. Such a description is given in Figure 1.

It is obvious from Figure 1 that, the developed capacity which is generated by the development rate, can be converted into production capacity after whole face is developed. Therefore, both developed capacity and production capacities are defined as levels. The production capacity, is then consumed by the production rate. The two controlling rates in this system are, the development and the production rates, and both of these rates are controlled by manpower, and external factors such as changing geology-

For controlling the development of coal face and hence, coal production, the desired state of the face and its divergence from this desired state in actual life , is needed to be known. Once this information is achieved, necessary corrective control actions can be implemented. Control is applied by defining the target states of cumulative development and production, and according to the information coming from the colliery, various control policies can be implemented, if necessary.

The control policies used within the colliery arc constructed under two assumptions: Semi-Integrated policies, which assume that, only a subset of total information is available, while the Fully-Integrated policies suggest that, all the information about ongoing operations in the mine are available. Under these two assumptions, two policies are constructed: Manpower allocation policies and machine shift allocation policies.

When there are insufficient number of men to carry out the work, manpower allocation policies, under the assumption of fully integrated policies, suggest that, the information on the discrepancy between the actual and desired state of each coalface, is chosen as the basis for manpower allocation. The coalfaces, which most lags behind the schedule should receive the most men.

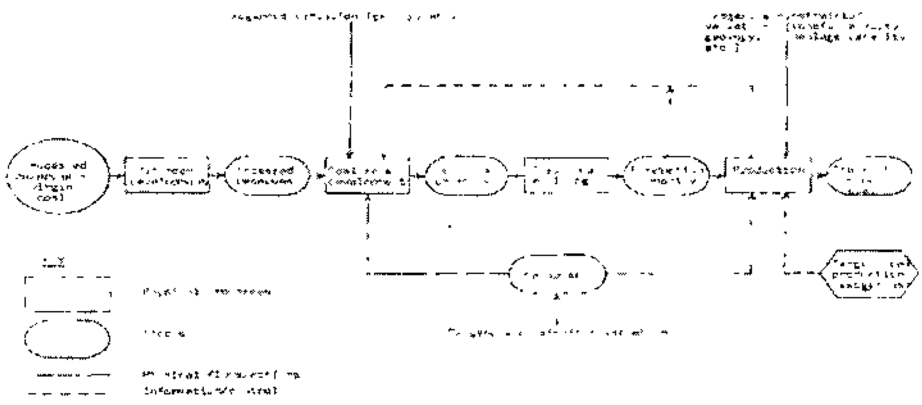


Figure I Description of the System (Wolstenhulme and Holmes 1985)

Machine shift allocation policies, are too defined on the basis of discrepancies. When the production is lagged behind the target to an extend which is considered to be critical, machine shift allocations is increased. This is achieved either by a the usage of spare man or by canceling a development shift for each extra production shift allocated.

The policies are tested tor two exogenous shocks: A changing geological situation, which slows down the work and causing a reduction in face production and a change in manpower availability. The results indicate that, for the latter shock, continuous allocation of manpower while achieving fully integrated information is the best mechanism, while machine shift allocation policies give better results under geological shocks.

3.2 Use of System Dynamics in Equipment Selection

In another system dynamics application to mining systems " Control of a Coal Transportation System', Wolstenholme (1990) tries to choose the optimum blinker- conveyor belt system for coal haulage in a three laced, longwall mine. Main levels contained within the system are. the level of bunkers, cumulative coal production rates and cumulative coal haulage rates. The bunkers discharge coal over the conveyor belt whenever there is available room on it. There are three policy alternatives for the system: The first policy charges coal at the bunker at zero or its maximum discharge rate, and the latter is used as long as there is coal available in the bunker and room available on the conveyor belt. The second policy sets discharge rate at any point between zero and maximum discharge rate in the same proportion as the bunker level to the bunker capacity. Finally, the third policy guarantees that there is no shortfall from the conveyor belt capacity and that the bunker levels are not exceeded (i.e. fallen into negative terms) when they discharge at the maximum rates.

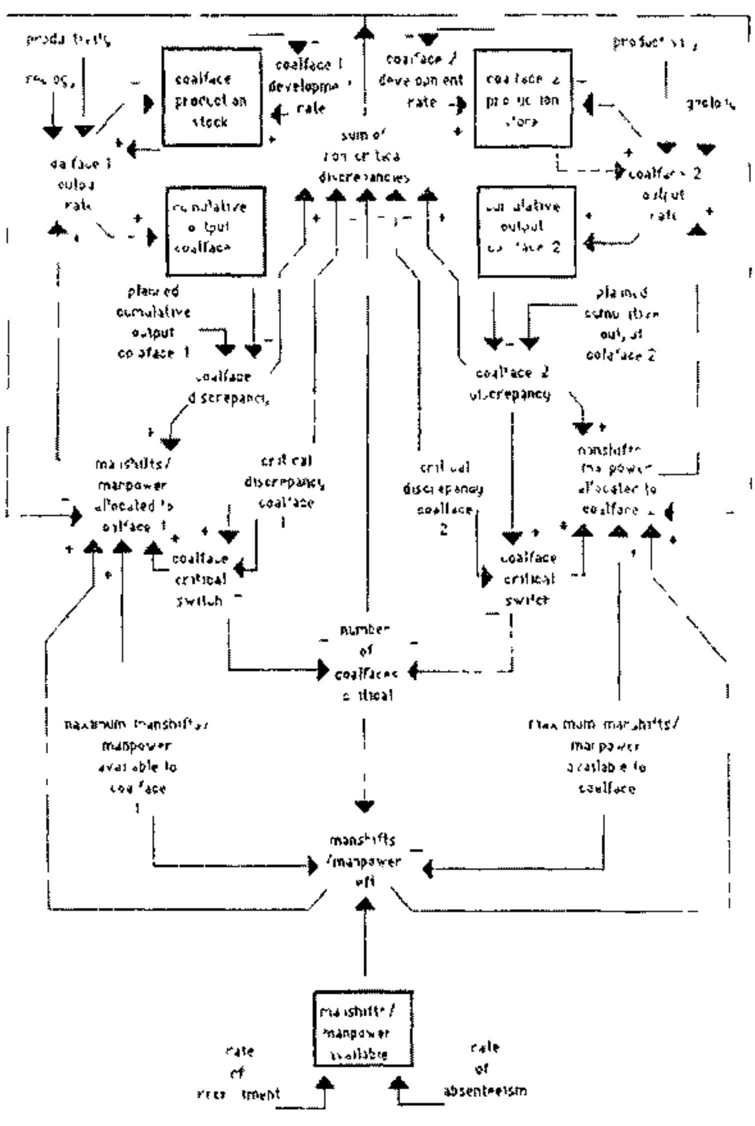
The model experiments, simulated with STELLA software tor these three different policies, determines the efficiency of the system when all determinants of the system, namely the coal production rate, bunker capacity and conveyor belt capacity are changed, and among the three policies, third is observed to be the most efficient.

The influence diagram for the underground mine of concern is shown in Ficure 2.

4 CONCLUSION

System dynamics is not a widely used technique in mining industry. Despite this fact, the relationships between the variables of a mine is mostly nonlinear and dynamic, and such a description of the system not only allows better understanding of the shortfalls of the system but also facilities the simulation of operating policies.

The two research work described in this paper, suggest that, a mine can be described as a dynamic feedback system. The description of the mine in such a manner helps identifying the relationships and interactions between the variables of the mine, and therefore such a representation allows a better understanding of the system behavior than conventional techniques.



Figuie 2 Influence diagram of the undeigiound coal mine (WoKtenholme 1990)

Anothei advantage of the methodology is that, it allows simulation ot vaiious policies on the model, and the lesults of different policies under different conditions can be observed easily In this sense, the constructed models act as manaaeual decision tools

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A Guideline for DSS System for Underground Mining Method Selection

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ABSTRACT: Underground mining method selection is an important planning phase which affects project costs. In this phase, there is a need to deal with multiple criteria related to mining method selection and decision makers have some difficulties to reach the right decision in the multiple criteria environment. In this paper, a decision support system for underground mining method selection has been designed to lake into account all problem criteria in a scientific base, to research all effects of different scenarios related to the determined criteria and to carry out all necessary sensitive analysis. To produce an acceptable solution alternatives have provided by using Analytic Hierarchy Process (AHP), which is one of the multiple criteria decision-making methods.

I INTRODUCTION

Optimal underground mining method should be primarily selected to make use of underground resources optimally. The basic priorities are performing underground working safely, rising productivity and eliminating production costs and losses. Besides, controlling works on the mining centers, making ventilation be easy, decreasing repairing and maintaining costs of gallery, making mining panels be ready and making underground works be with a good time study and in a good order are also direct related to the selection of underground mining method. Because of those factois, underground mining method selection process is extremely important.

All available criteria should be analyzed carefully for selection of optimal underground mining method. In the method selection process, many factors are available. Some of them are controllable but some of them are not. Known conventional methods may generally be inefficient to reach the optimal choice. Once one method is selected, it is nearly impossible to change that method because of rising costs and mining losses.

in the last two decades. Analytic Hierarchy Process (AHP) has been frequently used for helping to solve the problems in multiple criteria environment and also used in selection of underground mining method. In this paper, a decision support system (DSS) is designed to help the decision makers who have to solve underground mining method selection problem that is one of the multiple criteria decision making problem. To produce a solution, the DSS uses AHP.

2 ANALYTIC HIERARCHY PROCESS

Analytic hierarchy process (AHP) is a framework of logic and problem-solving that spans the spectrum from the instant awareness to fully integrated consciousness by organizing perceptions, feelings, judgments and memories into a hierarchy of forces that influence decision results. The AHP is based on the innate human ability to use information and experience to estimate relative magnitudes through paired comparisons. These comparisons are used to construct ratio scales on a variety of dimensions both tangible and intangible. Arranging these dimensions in a hierarchic or network structure allows a systematic procedure to organize our basic reasoning and intuition by breaking down a problem into its smaller constituent parts. The AHP thus leads from simple pairwise comparison judgments to the priorities in the hierarchy (Saaty 2000).

3 THE DECISION SUPPORT SYSTEM FOR UNDERGROUND MINING METHOD SELECTION

Designed decision support system (DSS) includes three subsystems; Database Management Subsystem, Model Management Subsystem and Dialog Subsystem. Database management subsystem is responsible for managing a database in which the data concerning with all technical and processing criteria to be used for selection of underground mining method is stored. Model management subsystem which has an interaction with the database management subsystem provides a decision maker an opportunity to evaluate all available solutions alternatives with the help of Analytic Hierarchy Process (AHP) which is one of the methods foi making a decision in a multiple criteria environment, according to the user inputs. Dialog subsystem is the last component of the DSS and includes a user interface which provides an interaction with the decision maker in the decision making process. Before the details of the subsystems of the DSS , underground mining methods and related decision factors will be primarily examined. Basic decision factors related to method selection process are given below.

The basic factors affecting the selection of underground mining method (Saltoglu. 1979):

- The shape of the orebody and wall rock condition,
- Mineral dissemination,
- The distribution of the high grade portions within orebody,
- The mineral and chemical composition of ore,
- The physical properties of the orebody and wall rock,
- Extraction depth,
- The roof wall condition,
- The subsidence,
- The inclination angle in the vein type deposits,
- The thickness of the orebody,
- Market price of the ore,
- The presence ol the methane,
- Orebody condition.

The most important factor affecting mining method selection among these factors is the shape of the orebody. The mining methods that can be used in the tabular type orebodies is given below;

- Advance longwall with caving,
- Advance longwall with backfill,
- Retreat longwall with caving,
- Retreat longwall with backfill,
- Room and pillar with caving,
- Room and pillar with backfill,
- Rill stopes,
- Robbing-caving,
- Top slicing,
- Slicing caving,
- Shortwall.

The mining methods can be used in the massive or irregular type orebodies are given below;

- Shrinkage stoping,
- Cut and fill,
- Square-set stoping,
- Underhand open stoping,
- Pillar caving,
- Pillar mining with filling,
- Top slicing,
- Sub-level caving,
- Breast stoping,
- Room and pillar,
- Chamber mining,
- Sub-level stoping,
- Underground bore hole,
- Mitchell slicing,
- Block mining with square-sets,
- Block caving.

The basic factors concerning the underground mining method selection can be classified, depending on the field condition and geomechamcal properties of the orebody and wall rock. These factors can also be classifed as environmental and economical factors. In this classification, decision factors are defined as (Karadoğan, et al.. 2001);

1. Geometrical shape of the orebody,
2. Vein thickness,
3. Vein inclination,
4. Extraction depth,
5. Physical properties of the orebody and wall rock,
6. Strength of the orebody,
7. The condition of the orebody and wall rock contact,
8. The strength of the roofwall,
9. The strength of the footwall,
10. The subsidence effect,
11. Mineral dissemination,
12. The distribution of the high grade portions within orebody,
13. The mineral and chemical composition of ore
14. Support requirements,
15. Market price of the ore,
16. The chemical properties of the orebody and wall rock,
17. Hydrological conditions,
18. Capital cost,
19. Operating cost.

3 I. Database Management Subsystem

The most important component of database management subsystem is the database in which the data related to different mining methods and decision factors which are classified under 19 entries, is stored. Those factors should be paid attention by the decision makers in the decision process of method selection. Database management subsystem is responsible for managing the database.

3 2. Model Management Subsystem

Model management subsystem with the interaction of database management subsystem, provides the decision makers a help to evaluate different underground mining methods according to the inputs of the decision makers by dialog subsystem. To pei form this task, model management subsystem uses AHP. All pairwise comparisons related to 19 decision factors is primarily performed and all available mining methods are ranked in AHP. This rank is only decision proposal for a decision maker but not a decision. The DSS only guides to the decision makers to make a decision easily and is not a replacement of a decision maker.

3.3. Dialog Subsystem

Dialog subsystem is also named user interface is an application interface to the decision makers can interact with the DSS for providing inputs and outputs and performing necessary tasks in the underground mining method selection process. This user interface should be helpful and useful for pei forming the process for decision makers when one decision makers is unavailable. By using dialog subsystem, it is possible to interact with database management subsystem and model management subsystem and to produce most suitable underground mining method subject to 19 decision factors.

4 RESULTS

In this paper, a decision support system for underground mining method selection has been designed to eliminate the difficulties in taking into consideration the decision factors in the method selection process and to guide the decision maker to select the optimal underground mining method. This decision support system uses analytic hierarchy process which is one of the methods to make a decision in a multiple criteria environment. With the DSS, other decision makers when one decision maker is unavailable, can now evaluate the underground mining method selection problem according to the 19 decision factors and derive a solution subject to the shape of the orebody

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Reduction of Sulfur Oxide Emissions by Means of Mechanical Preparation of Hard Coal

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ABSTRACT: A possibility for the reduction of the worldwide increasing emission of sulfur dioxide is the bencficiation of coal with high sulfur content before its utilisation. Reduction of sulfur content by means of piocessing methods is possible by cleminaling pyrite. The success of this way depends above all on the sulfur distribution in raw coals and on the selected specialised processing methods. In the last decades RAG and DMT carried out extensive tests in laboratory and pilot plants with a large number of different raw coals with high sulfur content. Under our assistance another pilot plant was designed and operated for sulfur reduction of extremely high-sulfurous lignites in Spain. All test results lead to the construction of numerous large-scale preparation plants for sulfur reduction in Europe (Germany, Poland, Spain). Particularly in case of three new preparation plants in Poland, more than 90% of the pyrite contained in the raw material was removed. At the same time, the coal quality as a whole has been improved, which resulted in a reduction of carbon dioxide emissions when the coal is being burnt. To reach the same reduction of sulfur dioxide, the investment for the mechanical separation is only I *OVr* of the costs necessary for *a* Hue gas desulfurisation plant.

1 SULFUR DIOXIDE EMISSION AND SULFUR IN COAL

Sulfur dioxide results primarily from the combustion of sulfur-bearing fuels, smelting of sulfur-bearing metal ores, and industrial processes. Major sulfur dioxide emission sources are power plants, refineries, some types of chemical plants, primary metal smelters, and cement plants. The problem of acid rain is becoming serious as energy consumption and use of fossil fuels is increasing. Although in most industrialised countries, sulfur emissions are in sharp decline over the past decades, it is in the coal dependent developing countries including China and India, where the concern for the future is potentially great. In the absence of widespread sulfur abatement measures, SO_2 emissions in South and East Asia would triple by 2010.

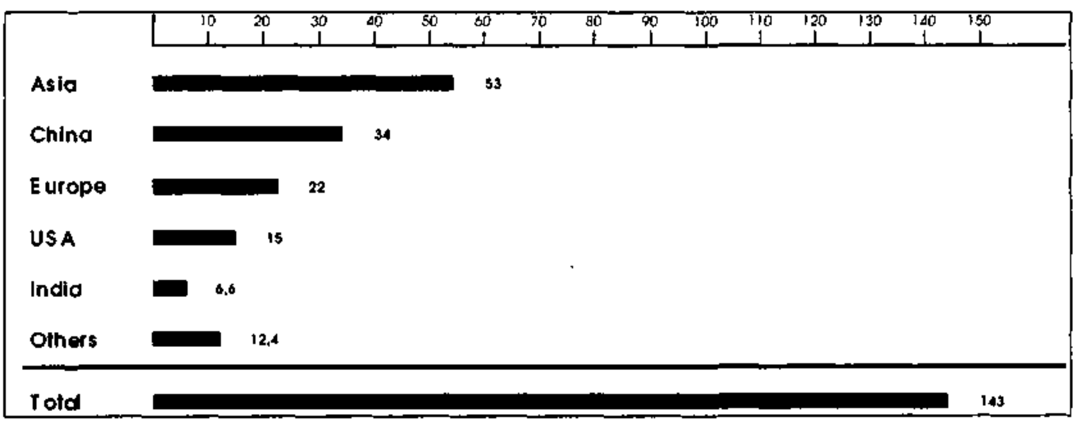

Figure 1. Assessment of global sulfur dioxide emission in year 2000 11,000,000 t/a]

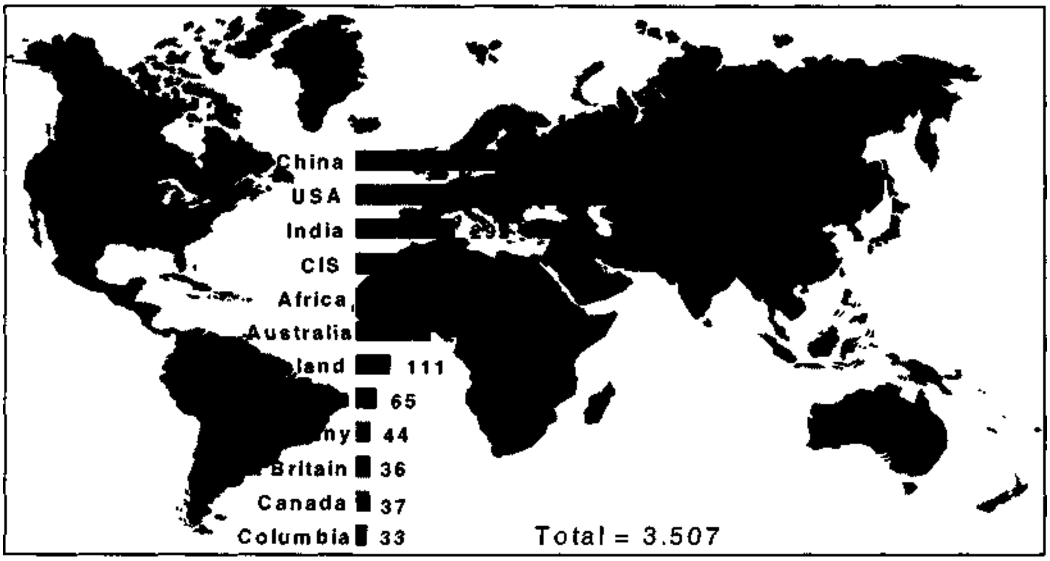

Figure 2. World coal production in year 1999 [1.000.000 t/a]

The People's Republic of China as the world's leading hard coal producer is also the world's leading consumer of coal as primary energy for the generation of electricity. During the maximum production in 1996, China consumed about 1,374 million tonnes of coal which is more than 70 % of the primary energy. Approx. 150 million tonnes of the total coal production, i.e. 12.5 % of the total sales, have medium or high sulfur contents. The average sulfur content of raw coal is 1.72 %.

From the total amount of sulfur dioxide (SO:) emissions in the year 1996 of 18.25 million tonnes, about 80 % or 14.6 million tonnes of SO_2 came from coal combustion.

2 SULFUR DISTRIBUTION IN RAW COALS

It is difficult to check emissions of sulfur dioxide during burning of coals totally because sulfur is generally present in all the coals either in small or in large proportions,

Seen world-wide, sulfur contents in hard coals and lignite can be from 0,2 %, which is very low, up to extremely high 10%. Usually, coals which are referred to as sulfurous coals contain between 1,3 % and 2,8 % sulfur. These are total sulfur contents.

In general, we find sulfurous coals in two types of formation:

2. / *Organically bound sulfur*

This form of sulfur occurs in all biological primary substances and is imbedded in the molecular structures of all living organisms (plants, animals, human beings). Coals, which developed from plants, contain organically bound sulfur of about 0,1 % up to 1,2% as regards the dry substance, very rarely they have a higher sulfur content. In case the total sulfur content is higher than the organically bound sulfur, this indicates that there is the second form of sulfur which is

2.2 Anorganic Sulfur, Pyrite (FeS,)

Pyrite has either been formed during the carbonification process or was incorporated into the coal seam by natural influences. Pyrite can occur in different forms, but it usually occurs in the macrocrystalline (epigenctic) or microcrystalline (syngenetic) form.

Reduction of sulfur content by means of processing methods is only possible by eliminating pyrite. First of all, a few microscopic slides as examples for different pyrite formations are shown: