

**THE DESIGN AND DEVELOPMENT OF A MECHANISED  
SUPPORT SYSTEM FOR TABULAR STOPES**

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## ABSTRACT

The single largest cause of injuries and fatalities suffered by the workforce in South African gold and platinum mines is rockburst and rockfall hazards. The majority of these rock-related fatalities ( $\pm 56\%$ ) occur in the immediate vicinity of the stope face. Relatively few fatalities ( $< 5\%$ ) are associated with the back areas. The strike gully is associated with the second highest number of fatalities ( $15\%$ ) (Daehnke et al., 2000). This project is a continuation of a previous SIMRAC (Safety in Mines Research Advisory Committee) project, namely GAP 708, which was initiated as part of a research thrust to develop an alternative support system technology to combat the hazards of rock mass instabilities.

The objective of GAP 708 was to develop a stope support system concept that addressed the deficiencies in current stope support systems, and that would significantly reduce the fatalities in the stope face area in the short to medium term. Rock engineering and operational specifications were detailed for the purpose of clearly identifying the requirements for a support design concept. These specifications are detailed in Appendices B and C of this report. Eleven concepts were generated and during a process of evaluation the remotely advanced headboard system was selected as the most viable system that could be implemented.

The continuation of GAP 708 (namely SIM 020204) entailed developing the remotely advanced support system to prototype level, at which stage it could be trialled on surface and in an underground environment. The system was developed from a basic conceptual format to manufactured prototype, which entailed a rigorous mechanical design process. At its current prototype level, the system became known as the 'walking beam' system. The system consists of two steel beams that are linked to each other, with each beam mounted on a hydraulic prop. The combination of beams and props was automated to enable the operator to control the machine safely from a remote area.

The walking beam underwent a series of surface and underground trials. Design enhancements were made to the system as the trials progressed. The system concept was proved from the successes attained with the functionality tests in surface rigs, notably at the Savuka test facility where the system withstood drop tests that simulated rockfall scenarios. The underground trials were conducted at Randfontein Estates and East Driefontein Gold Mines.

The surface and underground tests performed on the system from the beginning of its development added up to a thorough evaluation of the system. The operational shortcomings were identified and recommendations were made for improving on the system's implementation with respect to present and future developmental stages. In its current state, the system is not yet suitable for implementation.

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# 1 INTRODUCTION

## 1.1 Some background on underground mining and mine support systems

Prior to defining the design problem, it is essential to provide the reader with background information on the mining process, mechanisation, system implementation and the support systems that are generally utilized in the mining environment. This is also important for understanding the process of integrating a mechanised support system into the respective mining cycle, which is discussed further in Chapter 3 in the context of the prototype support system that was designed and manufactured. This prototype support system was called the 'walking beam stope support system' and is the main focus area of this dissertation.

The issue of mechanised mining system implementation is touched upon for the purpose of highlighting the importance of having proper structures in place before actually trying to implement new technologies in the mining environment, especially the gold mining sector of South Africa.

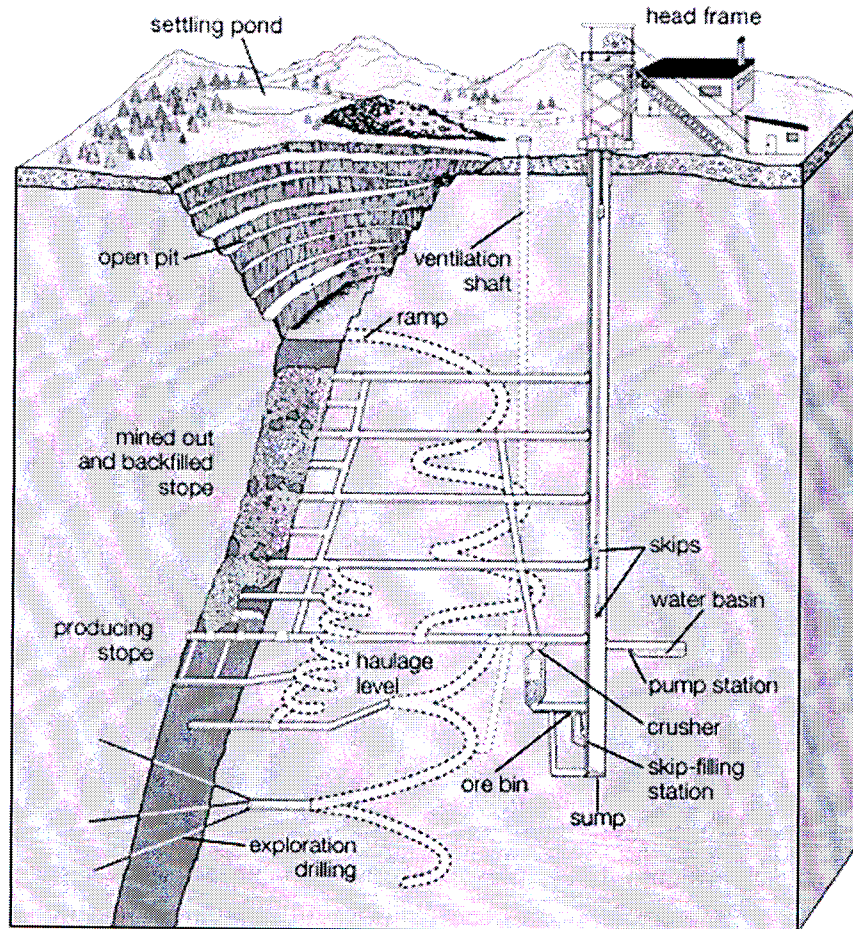
### 1.1.1 The mining process, mechanisation and system implementation

#### 1.1.1.1 The mining process

Underground mining operations incorporate a variety and combination of different processes that require a range of scientific and engineering disciplines for the planning, design and execution thereof. These processes include mineral exploration, mine layout design, mine excavation, mineral extraction, mineral transport and ventilation. The activities encompassed in these processes vary according to site conditions and the geological and rock mechanical conditions of the orebody.

The location of the orebody plays a fundamental role in determining the mining or excavation method that is required. As an example, if the orebody is close enough to the surface, mining may start with an open excavation (also known as 'quarrying' or 'open pit' mining, see Figure 1-1) and continue underground at a later stage (Puhakka et al., 1997).

A typical mine layout (Figure 1-1) would entail a straight vertical shaft being drilled and blasted to a specified depth (A mine can have a depth as high as 3 km's in some cases). This would involve the use of shaft sinking technology that are generally pneumatic or hydraulic driven drilling rig systems. Blasting involves loading the drilled holes with a specific type of explosive and igniting the explosive with an initiating device.



*Figure 1-1 A typical mine layout (After Hamrin, 1997)*

Vertical shafts are usually used for ventilation, access and hoisting. Once the shaft is produced, development work can begin. A structural frame called a 'head frame' is constructed for the purpose of housing rope winding systems and also to allow access for men or material into the lifts that are used for transporting men and material to the respective underground levels. These lifts are called 'cages' or 'skips' and are raised or lowered by means of the rope winding system.

Development work consists of a whole range of activities and includes all the excavation that precedes mining itself, and is aimed at preparing the orebody or stoping area for productive mining operations. The mine design and excavation plan would dictate the construction of tunnels, travelling ways for men and material, and access paths (including ramps for vehicles and machinery) to the stoping area. Stopping areas are cavernous openings that have been mined out, where minerals are excavated. The stope face is the area in the stope where actual excavation operations take place, such as drilling and blasting operations.

A concurrent activity to mine excavation and mineral extraction processes is that of supporting the excavated area, which is discussed in section 1.1.2. Support is installed as mining progresses, and with this progression the need arises to transport the broken rock back to the respective shaft from where it can be hoisted to surface.

The blasted rock in the face area is conventionally moved by means of scraping systems. 'Scrapers', as they are generically called, are large mechanical spades that are driven by rope-winch motors along the face area, and move the broken rock into adjacent passageways known as gullies. From the gully the rocks are moved by another system of scrapers into a hole that is called a 'box-hole'. At the bottom of the box-hole lies a locomotive into which the rocks are dropped. The locomotives are mounted on rails and transport the ore to the shaft area or skip-filling station (Figure 1-1) from where it can be hoisted to the surface.

The process of removing the broken rock from the face area is known as 'cleaning'. Aside from scraping systems there are alternative methods of cleaning the face such as the use of powerful water-jetting systems to entrain the rock along pathways. A new development in ore removal is a method known as hydraulic hoisting. This involves the use of crushing machinery to crush the rock into finer particles and mixing it with water to form a slurry. The slurrified rock can then be transported by means of a powerful sequential pumping system via a special piping configuration that leads to the surface. Research work that has that been undertaken for the implementation of hydraulic hoisting systems has produced some valuable outcomes, proving the technical feasibility of this technology (Van Den Berg, 2004).

A vital requirement for all underground mines is ventilation. Ventilation is usually provided by large powerful fans with ducts that are designed for positional efficiency near the working areas underground. The three main functions of ventilation are:

- The supply of air for human respiration and diesel-engine powered equipment;

- Dilution and removal of contaminants resulting from the mine production processes such as strata gas, dust and products of combustion (eg. from diesel engines or the use of explosives); and
- Provision of a suitable working climate with respect to heat through either the distribution of heated/cooled air and/or the removal of strata/process generated heat (Davies et al., 2005).

### **1.1.1.2 Mine mechanisation**

The key driver behind the utilization of mechanised systems on the mines is improved safety for workers and higher production rates. Gold mining operations are currently very labour-intensive, with mining personnel having to work at very deep levels where safety becomes the primary concern. There are many safety hazards associated with working in the deep levels of a mine including the high seismicity levels, uncomfortable air temperatures and unstable rock masses, to name a few.

The process of mine excavation and mineral extraction could involve mechanised methods, non-mechanised methods or a combination of these methods. The conventional method of excavation and extraction is the drilling and blasting method which has been developing and advancing since the early 1900's. Research and development initiatives for drilling and blasting technologies is an ongoing effort and with the introduction of autonomous control and computerized drilling, major leaps have been made by the mining sector towards automated mining systems for different mining applications and processes. New drilling technologies include the advent of microwave drilling systems, acoustic drilling and plasma drilling systems to name a few.

Non-explosive rock breaking methodologies that improve production and safety are also being investigated by research institutions and the mining industry. An example of such a system is the water-pulse gun that was developed by the CSIR's mining division, which involved the release and propagation of a high-pressured wave from high-pressure water into low-pressure water filled, drilled hole. The pressure wave then causes the rock to fracture by imparting stresses into the rock (Leong et al., 2002).

Other non-explosive rock breaking systems that are currently being researched and developed include the impact ripping system and hard rock cutting machines such as the ARM 1100 that is

being developed by Sandvik. The area of rock cutting technologies is a whole research thrust on its own. One of the major challenges that new technologies and mechanised systems face, especially in the South African gold mining sector, is that of system implementation.

### **1.1.1.3 Implementation of new technology**

There have been many initiatives over the past years for the implementation of mechanised equipment in both the platinum and gold mining sectors. South Africa's gold mines possess some of the most difficult and harsh conditions in comparison to the rest of the world's mining sectors. Due to the tough hard rock conditions and unique geological conditions in South Africa, most equipment has to be invented, designed, tested and mass-produced locally. The gold mining industry does keep abreast of new techniques and technologies that have evolved in other parts of the world to see how these can be applied to local conditions. New ideas and machinery have to be properly evaluated and adopted in "development projects" before operational testing. They have to then be launched in "application projects" under the testing production conditions in the South African gold mines (Kebble et al., 1975).

The author was involved in an earlier study conducted by the CSIR Mining Technology Division as part of the Futuremine collaborative research programme. The study entailed an audit of the drilling technologies utilized on South African gold and platinum mines. One of the major conclusions derived from the research was that the success in all the systems audited was highly dependent on the implementation planning process for equipment, as any deficiency in this area would be the primary cause for system failure or rejection by the mining industry. This was an outcome based on systems that were already in the product design stage.

A ratio of 40:40:20 was proposed as the optimum combination of design (40%), implementation (40%) and management buy-in (20%) for successful equipment implementation. The senior management team must have the belief for success as they play a key role in shaping the rollout of equipment, and it is also critical that there is a top level agreement with unions to eliminate fear of job losses (Rapson et al., 2002).

A lack of understanding with regard to the benefits of a new or mechanised system is a fundamental reason for the failure to implement these systems, as the workforce perceives the new technology or implementation of mechanization as threatening job security. A paper by Rupprecht (2001) highlights the need for a list of universally accepted benefits to motivate the development and implementation of mechanised equipment in the mine. These include:

- Improved labour productivity;
- Improved working conditions; and
- Improved quality of life for the worker (safety, physical effort).

## **1.1.2 Stope support systems**

### **1.1.2.1 Stope support function and layout in underground mines**

A section of a typical stope layout is shown in Figure 1-2. The plan and side views are shown for the purpose of highlighting the different parts and areas of a stope. The stope consists of timber support units and cement packs that are conventionally used in South African gold and platinum mines. In the stope area, as material is mined out, support needs to be installed to provide a degree of stability to the mine roof which in mining terms is named the 'hangingwall'. Similarly, the mine floor is termed the footwall.

Two other common terms used in mining scenarios particularly with regard to the stope area are 'dip' and 'strike'. The dip of a stope is the angular orientation of the stope relative to the horizontal and is dictated by the geometry or dipping angle of the orebody. The strike direction refers to the advance or direction in which the stope is being mined out.

The face area is the area where excavation operations actually take place. This is an area that is generally classified as highly unsafe because as the stope is mined out ('advanced') the face area remains unsupported or insufficiently supported for a certain period, during which activities are ongoing. As a result mine personnel would be present in the area performing different activities, which includes installing temporary support, and are exposed to the risk of rockfall and rockburst hazards. The back area is usually classified as the area where permanent support units (eg. cement packs) are installed.

The support standard on some mines requires the installation of blast barricades on a specific row of support units. A blast barricade is a shield that is installed close to the stope face before the commencement of blasting operations, in order to contain the fly-rock. These barricades serve to speed up the cleaning cycle and protect installed support. Grave (2003) conducted a test program to evaluate a range of different types of blast barricades.

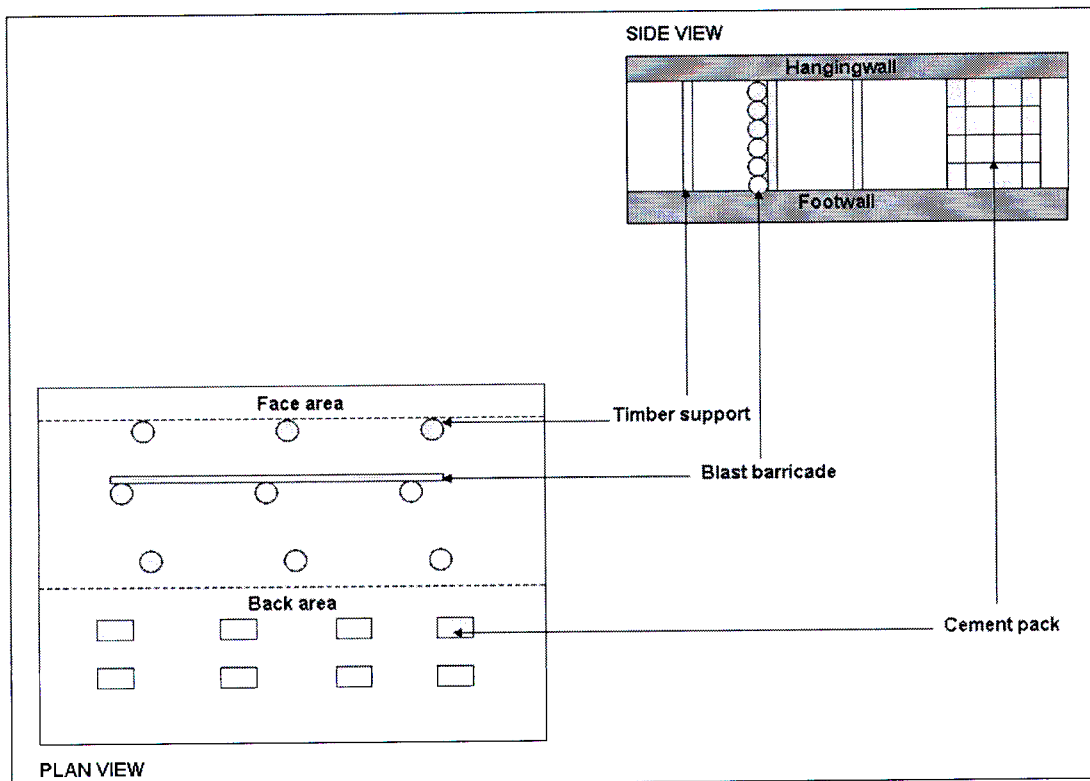


Figure 1-2 Schematic showing section of a typical stope support layout

There are many factors that influence the stability of a stope, which includes the mining depth, mining method, the blasting design, type, and spacing of the installed support etc. The installed support must serve the following functions:

- Prevent local rockfalls;
- Reduce the damage caused by rockbursts;
- Allow ready access for men, materials, and rock during the useful life of the stope;
- Allow for acceptable uninterrupted production; and
- Be cost-effective (Spearing, 1995).

As mining progresses to increased depths, rock becomes more fractured or discontinuous. The design and selection of support systems becomes a critical need for the alleviation of rockfall and rockburst occurrences. Cook (1960) states that stope support effectiveness depends on the extent to which the fractured rock, especially in the hangingwall, can be kept in place to reduce the damage and production delays resulting from rockfalls and rockbursts. The primary function, therefore, of a stope face support system is to maintain the integrity of the fractured hangingwall beam and thus prevent falls of loose rock for a specific period (Daehnke, A. et al., 2000).

In between the installed support units there exist areas of unsupported hangingwall. When the spacing of fractures in these regions is smaller than the spacing between individual support units in both the strike and dip directions, a block of rock is defined as that which can potentially fall (Stacey, 1989). This scenario represents a potential rockfall hazard.

In deep stoping conditions, the support is required to exert clamping forces on the immediate fractured hangingwall, before loosening of any inherently unstable blocks can take place. An opinion shared by a number of researchers (Gay and Jager, 1987; Roberts *et al.*, 1993) is that it is therefore generally required that, irrespective of mining depth, initial forces be generated rapidly by support systems. Another critical requirement of a support system is that it provides sufficient areal coverage to prevent falls of loose rock (Daehnke, A. *et al.*, 2000).

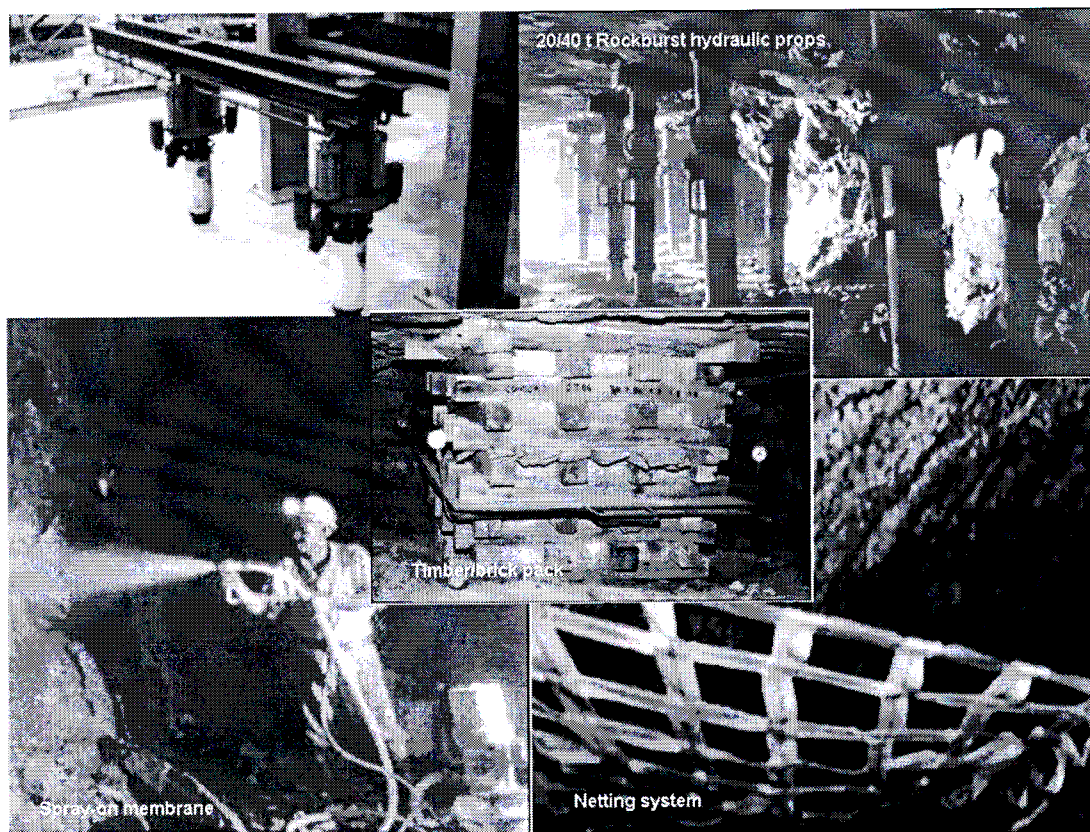
A rockburst is characterised as a seismic event that radiates sufficiently intense shockwaves to cause visible damage to an excavation (Heunis, 1980). The stress waves interact with mining excavations, leading to interface and surface waves, energy channelling and wave focussing (Daehnke, 1997). The rock is subjected to rapid accelerations, resulting in rock fabric failure, ejection of blocks of rock and stope closure (Ryder *et al.*, 2002). Stope closure is basically the convergence (or 'narrowing') of the hangingwall and footwall due to the re-distribution of stresses within the rockmasses as the stope is advanced.

#### **1.1.2.2 Types of stope support systems**

Many different types of support systems are used in the stoping environment. These could typically include combinations of props, packs, tendons and backfill. Early mining methods and support practice is summarised in a paper by Pretorius (1973). The primary in-stope support used were mine pillars and from the 1920's onwards, timber support was used on a large scale.

Stope support can generally be classified as face support, back-area support and gully support. Face support usually consists of temporary or removable support mainly in the area where drilling operations take place. Some examples of face support include: rapid yielding hydraulic props (RYHP's), mechanical props, timber sticks or sprags, propsetters and reinforcing members. Back area support is permanent support that is installed and left in the stope. These include cement and grout filled packs. Some examples of gully support types include mechanical rock bolts, cone bolts and meshing type support.

With regard to the issue of sufficient areal coverage, the CSIR: Division of Mining Technology was involved in the development of alternative support systems that offered extensive coverage. These included the twin beam support system, netting systems installed between prop units and spray-on membranes that provide 100% coverage in the applied area (Ryder et al., 2002). Figure 1-3 shows photos of these systems and additionally, a stope installed with rockburst props, and a timber/brick pack.



*Figure 1-3 Group of photos showing different support system types*

All support types, whether used in stopes or tunnels, fall into one of two categories namely, passive and active support. Passive support units (eg. timber support) are always unstressed on installation and begin to function only when the surrounding rock dilates, causing the unit to deform and thereby a reactive force is induced in the support. Active supports (eg. hydraulic props) are always installed stressed and the support exerts and maintains a restraining force on the rock, the magnitude of which depends on the stiffness of the support. Active support systems tend to control the deterioration of the rock, and limits the initial fracturing (and failure) of the surrounding rock mass, better than a passive support does (Spearing, 1995).

## **1.2 The design problem**

Despite many years of intensive research and development, current stope face support systems and technologies in South African gold mines are inadequate for significantly reducing the number of rock-related injuries and fatalities. It was found that the majority of these rock-related fatalities ( $\pm 56\%$ ) occur in the immediate vicinity of the stope face (Daehnke et al., 2000). A critical need was thereby identified to design and develop an alternative support system that would provide protection for workers who perform operations in the face area.

The stope face area is particularly difficult to support. Some deficiencies of current stope support systems include (i) poor areal coverage, (ii) poor installation or support not installed timeously, or at all, and (iii) poor face area support during cleaning. It is necessary that alternative support systems provide high levels of areal coverage in the stope face area, to significantly reduce rock-related hazards (rockfalls, rockbursts etc.) during stoping operations, and provide improved levels of support immediately after the blast (Daehnke et al., 2000).

It is also vital that any alternative support system offers protection to workers during all phases of the production cycle, i.e. drilling, charging, cleaning, making safe and face preparation. It is further important to integrate the application and use of alternative support technologies into the production cycle. Thus, the proposed support system should take into account the space requirements and other factors related to barring (process of removing rock from the hangingwall usually with a pinch bar), cleaning, drilling and blasting operations (Daehnke et al., 2000).

## **1.3 Initiative to address the design problem**

A number of stope support concepts were developed by means of reviews, workshops and brainstorming sessions as part of phase 1 of the SIMRAC (Safety in Mines Research Advisory Committee) project that was labelled as GAP 708. These concepts were evaluated for safety, practicality, and research and development requirements. The large-scale implementation of each system in the short, medium and long term was indirectly addressed as part of the evaluation of the system's research and development needs.

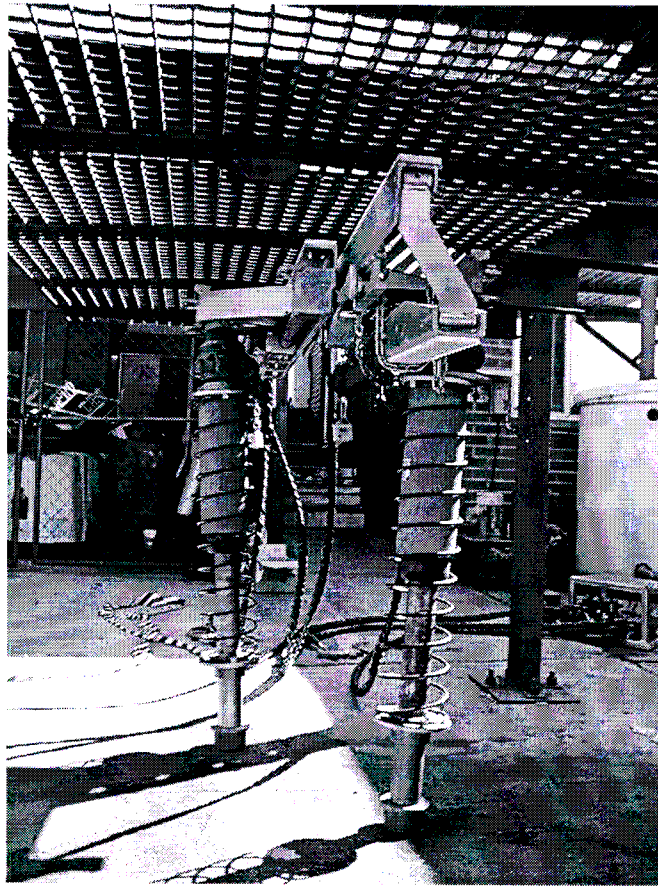
On the basis of the rating system developed as part of this project, three systems were found to have the best potential for reducing the rock-related hazards in the vicinity of the stope face. The recommended systems were:

- Remotely advanced headboards (based on reusable props and headboards that can be moved forward remotely);
- Rockbolt reinforcement (based on a dense array of short rockbolts, which reinforce the hangingwall beam); and
- Longhole drilling (a modified mining method, which allows all personnel to be concentrated in well-supported gullies, and in which no people enter the panel).

For general use in most gold and platinum mines in South Africa, the remotely advanced headboard system was considered the most suitable of the three systems.

The original conceptual design for the remotely advanced headboard system was one that entailed a link-bar mechanism between the individual headboards/beams. When the design of this mechanism was found to be impractical, a new system was developed which incorporated a crank-sliding configuration in place of the initial link-bar concept.

The eventual result of these initial design concepts was a first prototype stope support system, which was named the 'walking beam stope support system', and is shown in Figure 1-1.



*Figure 1-4 The walking beam stope support system*

The system comprises a set of beams and hydraulic props, which contain a number of auxiliary components. The beams are mounted on the hydraulic props and secured to the props with special attachments called 'torsion bars'.

The beams and hydraulic props are remotely operated by means of a hydraulic control system. Each beam with its attached prop is moved forward through hydraulic action, relative to its adjacent beam-hydraulic prop counterpart. In this manner the hydraulic props are 'walked' towards unsupported areas – hence the name 'walking beam stope support system'.

This dissertation is based on the work performed for the SIMRAC project labelled SIM 020204 that is a continuation of GAP 708 and encapsulates the design, testing and implementation stages for the prototype 'walking beam' system. Project SIM 020204 was valued at 2.4 million rands in total and the work was performed by numerous individuals since its initiation in the year 2000. The developmental phases of the 'walking beam' are discussed in Chapter 3.

#### **1.4 Project objectives and deliverables**

The primary objective of SIMRAC project SIM 020204 was to develop an optimised face area support system to a stage where it could be manufactured and implemented. Eventually, the improved stope support system will be used as face area support in shallow, intermediate and deep-level mines (for, primarily, rockburst conditions).

This system is destined for use by underground mining personnel on both gold and platinum mines. The main criteria for use of the system are that it should provide high levels of safety, while being practical and cost-effective. The critical problem area on which such a system could potentially make an impact is the high fatality and injury rates in the face area of tabular stopes.

The following enabling outputs were listed for project completion:

- To manufacture three to four prototype units, conduct tests and identify a champion mine;
- To optimise the design and to manufacture enough units to equip an underground panel, which had been modified by SIMRAC to include an initial underground trial of two units;
- To conduct a full underground trial in one panel with the purpose of identifying shortcomings and necessary improvements to the design of the support system, and to investigate the full integration of the system with other in-stope equipment;
- To workshop with experts and stakeholders to assess the results of the underground trial; and
- To compile a final project report, a CD and a video for demonstration to the mines.

#### **1.5 Industry partner for development of the support system**

Novatek Systems had been selected by CSIR: Mining Technology as an industrial partner for the design, development and manufacture of the prototype support system. A brief overview of the company is given below, highlighting the primary reasons for their selection.

“Novatek is a company that designs, manufactures, supplies and services innovative mining and drilling equipment for the mining industry. Novatek is ISO9001:2000 accredited and strives continuously to improve the quality, service and support of its products, and to provide its customers with better-value products and services. The result of the continuous improvement

drive is to make mining safer and more profitable. Novatek has the necessary in-house resources and experience for effectively designing, manufacturing and maintaining mining and drilling equipment. Nestek, the manufacturing arm of Novatek, is also ISO9001:2000 accredited and has the in-house capability to produce prototype and mass-production items for Novatek, utilising NC lathes and milling machines, as well as grinding and welded fabrication items. Therefore, Novatek and Nestek together have the experience and facilities for designing, developing and producing equipment from prototype to final product implementation” (Frangakis, 2003).

## 2 DESIGN OF THE WALKING BEAM STOPE SUPPORT SYSTEM

### 2.1 Support system design requirements

It is important to define the requirements of the system before going into the more detailed component-by-component design process. The conceptual design phase was an earlier initiative (GAP 708) as mentioned previously and the final design concept selected was the remotely advanced headboard system that was designed and developed to a stage where it became known as the walking beam stope support system. This concept was selected as it was concluded that its structural design would be the most suited to meet the requirements for most of the gold and platinum mines in South Africa. The rock engineering and operational requirements were generated through GAP 708 initiative. The reader is referred to Appendices B and C of this dissertation for a comprehensive description of rock engineering and operational specifications for the proposed support system. The critical/fundamental design requirements are briefly discussed in this section.

An important requirement for stope support is the ability of the support to prevent rockfalls in the working area between the stope face and the back area. The primary criterion for rockfall prevention is the support resistance capability (measured as force per unit area) to stabilise the blocks of rock that make up the hangingwall of a stope. The scope of GAP 708 constrained the support resistance requirement to a value of  $40 \text{ kN/m}^2$  due to the impracticality of designing a support system that caters for each of the many reef types.

For rockburst conditions Roberts et al. (1993) identified the capacity of a support system to do work and absorb energy as the most appropriate design criterion. Again, the project scope constrained the value of this criterion, as a general energy-absorption criterion of  $40 \text{ kJ/m}^2$  was considered as appropriate.

The main cause of rockfalls in stopes is inadequate areal coverage and the majority of these falls of ground (FOG) occur between the support units. Work performed by various authors including Klokow (1999) and Daehnke et al. (1999a) have found that a high proportion of rock mass instabilities occur between support units often in the unsupported areal spans between the

stope face and permanent support. In general, support units fail purely mechanically only under severe dynamic loading (Jager et al., 1999). The maximisation of areal coverage is therefore critical for reducing FOG casualties (Ryder et al., 2002).

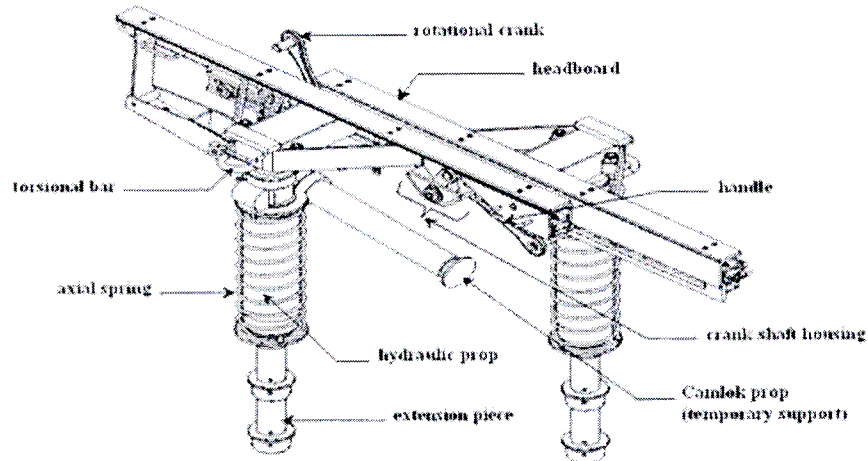
The other important parameter that is related to areal coverage is the spacing between support units. The spacing between support units is determined by a number of factors. The support spacing must be sufficient to accommodate blasted rock, rock handling and cleaning and manoeuvrability of drilling equipment. Work by Daehnke et al. (1999b) on the zones of support influence of individual support units indicate that, in general, a dip span of not greater than 1.5 times the strike span is appropriate (Daehnke et al., 2000).

The main operational requirements for the support system include the following:

- Safety: the system should protect workers adequately during all mining operations;
- Integration into the mining cycle: the system should not interfere or compromise stope face area activities;
- Handling: the system should be easy and safe to handle. Component masses should be minimal;
- Installation and removal: the system should be safe and easy to install/remove. A degree of automation is required;
- Blast resistance: the system should be able to withstand blasting effects;
- Mining height of stope (stope width): the system should be easily installed and function effectively in stope widths between 0.9 m to 2.5 m or if limited to lower widths, have a facility for accommodating the abnormal higher widths;
- Reef dip: the system should be able to accommodate dips of up to 30°;
- Faults and reef rolls: the system should be able to negotiate an additional change in dip of 20° over 5 m;
- Production system: the support installation rate should fit into the production rate. The system should not obstruct the flow of ventilation; and
- Maintenance: the system should entail minimised maintenance requirements to follow a simple standard procedure. Replacement parts should be easy to install (Daehnke et al., 2000).

## 2.2 Mechanical design

### 2.2.1 Description of the walking beam stope support system



*Figure 2-1 Main components of the system*

The walking beam support system (see Figure 2-1) comprises two headboards/beams that are mechanically linked, with each headboard attached to a hydraulic prop. The headboards contain hydraulic cylinders that can be controlled from a remote, safe area. The hydraulic control system is linked to a custom-designed pump that requires an air pressure of about 4 bar or better for optimal operation. The hydraulic cylinders enable this support system to be advanced forward by means of a rotational cranking motion followed by a linear sliding advancement, with simultaneous extensions and retractions of the corresponding prop.

The advancement of the support unit can be varied to correspond to the advance achieved by a blast. Thus, the advance and setting of the units are carried out remotely with no physical effort being required of the production personnel.

In basic terms, the motion of the system involves:

- Activating one beam against the hangingwall;
- Lowering the adjacent beam and advancing it forward;

- Activating the adjacent beam against the hangingwall; and
- Repeating the process for the beam that was first activated.

## 2.2.2 Components of the system

This section discusses the engineering design and functional specifications for the main components of the system. All components were modelled in 3D by means of a computer aided design (CAD) package named Autodesk Inventor. The mechanical drawings for the components were derived from a drawing package namely Mechanical Desktop that interfaces with Autodesk Inventor.

### 2.2.2.1 Headboard/beam design

The headboards are 8 mm angle-iron sectioned mild steel beams that have nominal lengths of 1.4 m and 1.5 m respectively (Figure 2-2). The angle-iron beam cross-section was selected because it enables the hydraulic cylinders to be attached efficiently to the beams and thus provides greater dynamic flexibility. The main idea behind having long beams is to give greater areal coverage for the hangingwall and hence extra support. The long axis of the beam is oriented on strike (perpendicular to the face area) such that face-parallel fractures are covered (refer to Appendix B for more information on rock engineering issues).

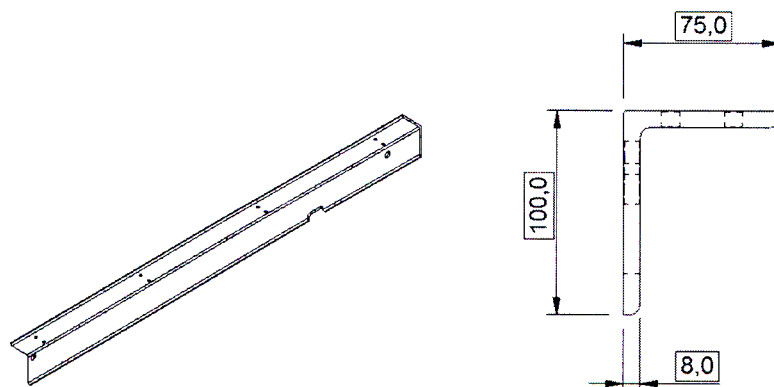


Figure 2-2 Beam design showing angle-iron cross-section

The load characteristics of the beams were determined from deflection tests. See Section 3.3.1 for more details on these tests. Note the closeness of the results from the basic finite element model and the actual physical load test. High-density polyethylene (HDPE) strips were secured to each beam for better energy absorption between the mine hangingwall and the beams during system pressurisation.

The headboard system is based on a headboard developed by the Chamber of Mines Research Organisation (COMRO) in the early 1990s. It makes use of a force-apportioning lever (Figure 2-3). The principle of force apportioning is to share the prop force between the beam and a metal pad so as to limit the force carried by the relatively weak beam. The relative lengths of the lever arms determine the proportion of force carried by the beam and the pad. The beams carry a sufficient load for providing rockfall protection only, while the aggregate beam and pad are sufficient for controlling rockburst energy (Taggart *et al.*, 1992).

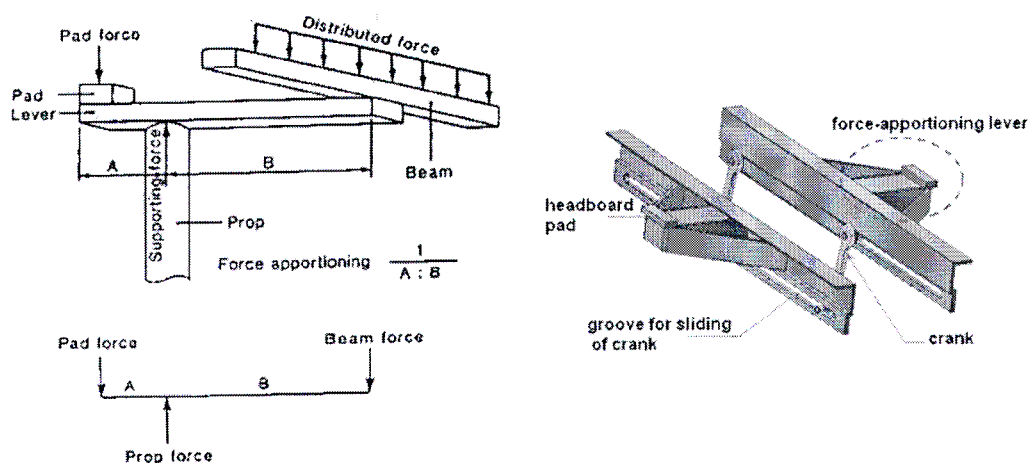


Figure 2-3 Principle of force-apportioning (After Taggart *et al.*, 1992)

At all times during the operation of the system, at least one headboard must be in adequate contact with the hangingwall. It is imperative that the beam pad and at least two other points on the beam involved have contact with the hangingwall, otherwise the system will become unstable and could possibly fall over. Some extreme testing was performed during the first underground trials and it was found that for certain hangingwall profiles, when the pad itself was in adequate contact and the rest of the beam did not contact at all, the system remained stable. However, these are unique cases and the general rule of three-point contact must always be adhered to. Ensuring proper contact would be the responsibility of the operator who would have to have the necessary training and experience to operate the machine.

Figure 2-4 shows the beam configuration and its orientation relative to the stope face. Note that the beam on the left contains a rotational cylinder, while the beam on the right contains a linear actuating cylinder. The hydraulic cylinders are designed to provide two specific dynamic actuations of each beam, namely rotation and translation.

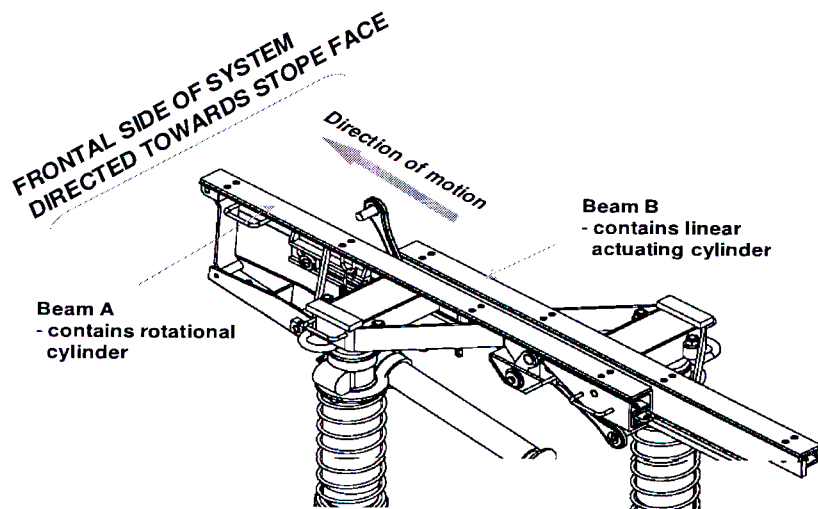


Figure 2-4 Headboard set-up

#### 2.2.2.2 Rotational cylinder assembly

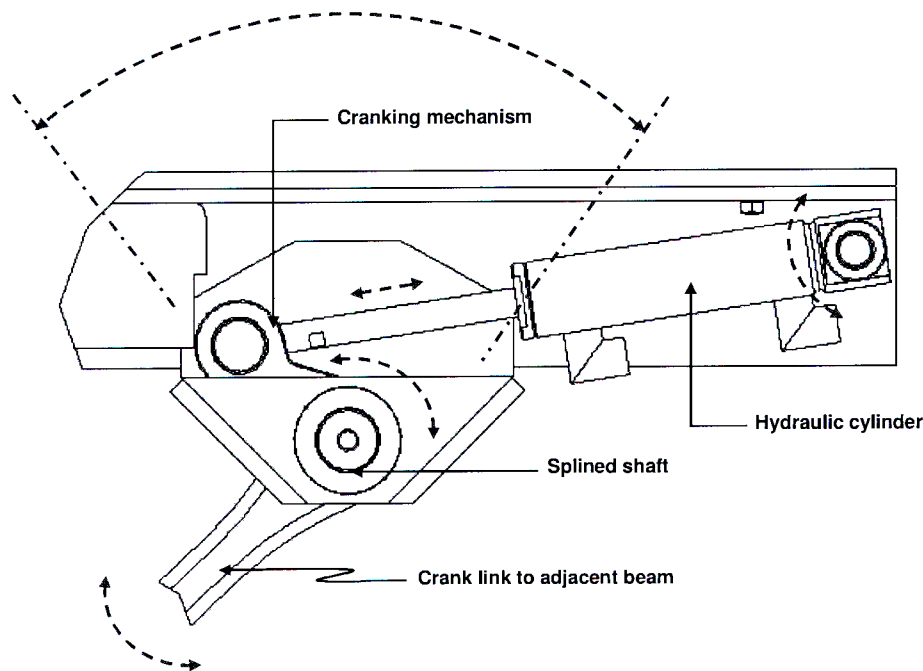
The rotational cylinder has a two-fold function. Firstly, it provides assistance to the hydraulic props for the setting of the headboards against the hangingwall. Secondly, it imparts a pivotal mode of motion to the beams, which assists them to negotiate rolls and protrusions in the hangingwall during advancement of the system.

The rotational cylinder assembly has the following main components:

- 1) Hydraulic cylinder and associated parts;
- 2) Internally splined crank;
- 3) Splined shaft; and
- 4) Crank linkage.

Figure 2-5 shows the layout of these components and how they interface with each other. The hydraulic cylinder is linked to the internally splined cranking mechanism via a cylindrical pin

connection. A splined shaft interconnects with the cranking mechanism and the crank linkage is TIG (tungsten inert gas) welded to the opposite end of the splined shaft. All components of this assembly are dimensionally and dynamically constrained by the angle-iron beams and therefore adequate tolerances had to be incorporated into the design. The dynamic functioning of the rotational cylinder system can be visualised from Figure 2-5, from the arrows that indicate the respective directional dynamic motions.



*Figure 2-5 Rotational cylinder components and dynamics*

The hydraulic cylinder and its associated components are shown in Figure 2-6, which also shows the positions of the respective seals and O-rings. The final diameters of the cylinder bore and piston rod were specified as 34 mm and 16 mm respectively. Initially, the diameter of the rod was specified as 20 mm. A number of factors had to be taken into consideration for the rotational cylinder design. These included the manufacturing process, the modularity of the assembly, the sealing constraints and the critical issue of buckling failure. It was determined from previous experience and studies on hydraulic mining systems that a rod diameter of 20 mm would be satisfactory as an initial design specification. The final dimensions were determined from a systems-based perspective, co-relating, by means of an iterative process, the load-bearing requirements of the splined shaft and the optimal cylinder dimensions. Also, the threads on the cylinder endcaps had to be checked via standard stress calculations, for each iterative step, to ensure the integrity of the endcap when subjected to the specific forces.



stronger than keyed shafts. A keyed shaft has an inherent weakness in its construction as slots are cut into it, which consequently reduce the torque-transmitting capacity (Oberg, 1996).

The first step in the design process for the splined shaft involved an analysis of the forces and moments that the system would be subjected to. This analysis was conducted primarily to establish the strength requirements of the splined shaft and ultimately to determine the optimal shaft diameter. The method that was used for the force analysis is presented below.

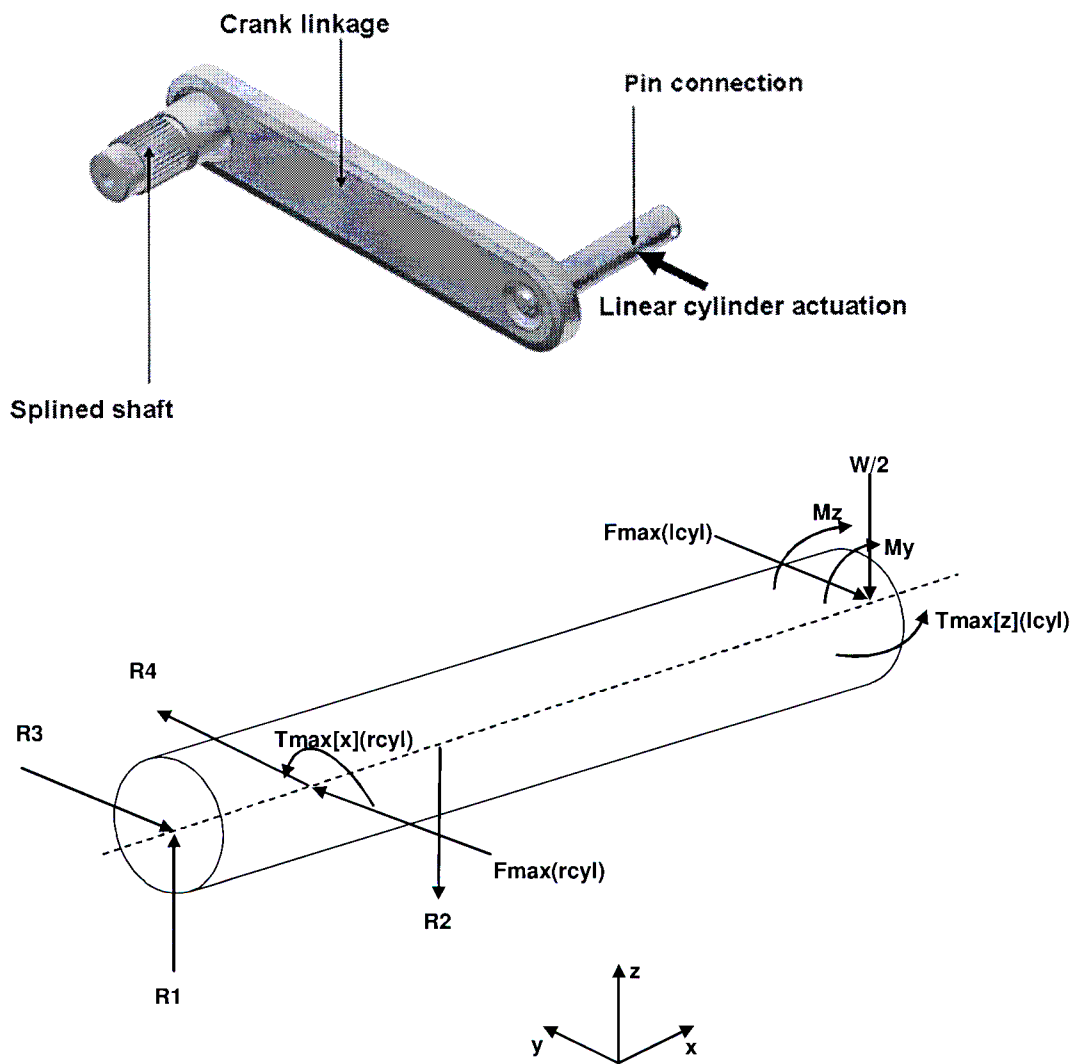


Figure 2-7 The derived force diagram for the splined shaft

The forces R1, R2, R3 and R4 in Figure 2-7 represent reactional forces that are induced in the splined shaft.  $F_{max}(rcyl)$  is the maximum force that is applied by the rotational cylinder to the actual spline.  $T_{max}[x](rcyl)$  is the maximum torque that is applied by the rotational cylinder via

the internally splined crank mechanism and oriented about the x-axis. Similarly,  $F_{\max}(l_{\text{cyl}})$  is the maximum force exerted by the linear actuating cylinder and  $T_{\max}[z](l_{\text{cyl}})$  represents the maximum torque applied by the linear cylinder via the crank system about the z-axis.

$M_z$  represents the bending moment that is applied to the shaft about the z-axis due to the actuation of the linear cylinder, as indicated in Figure 2-7.  $M_y$  is the bending moment that is applied to the shaft with respect to the y-axis and is a function of the weight  $W$ . The weight component of the adjacent beam is taken as half of the total applied weight as it is assumed that one half of the total applied weight is carried by each crank linkage's pin connection.

The reactional forces were calculated by applying the equilibrium condition, namely:

- 1) The summation of bending moments about any point along the axis of the shaft is equal to zero; and
- 2) The summation of all forces perpendicular to the shaft axis is equal to zero.

The shaft is now treated as a beam system, and the equilibrium condition is applied in both the x-z and x-y planes. As mentioned previously, the rod diameters were initially specified as 20 mm and the system pressure was selected to be 20 MPa. The respective forces and torques applied by both the linear and hydraulic cylinders were thus calculated and applied in the equilibrium calculations. The vector sum of shear forces and bending moments applied to the shaft was calculated via shear force and bending moment diagrams with respect to the x-z and x-y planes. In this way the maximum shear force and bending moment were calculated.

It is clear that the splined shaft would experience a combination of different stress regimes. When faced with these circumstances, it is common practice to utilise failure theory for making predictions of how strong the system under investigation will be under conditions of static loading. The 'theory' behind some of the various failure theories is that whatever is responsible for failure in the standard tensile test will also be responsible for failure under all conditions of static loading (Juvinal et al., 1991). Maximum-distortion-energy theory was utilised to determine the optimal spline shaft diameter. In summary, the theory involves calculating an 'equivalent stress' for the system concerned and comparing it to the yield strength of the system. If the calculated equivalent stress exceeds the yield strength value for the given material, yielding is predicted.

For the splined shaft, the equivalent stress was calculated using equation (1-1) below:

$$\sigma_e = (\sigma_b^2 + 3\tau^2)^{1/2} \quad (1-1)$$

where  $\sigma_e$  is the equivalent stress,  $\sigma_b$  is the maximum bending stress and  $\tau$  represents the total stress induced in the splined shaft due to both shear and torsion. The left-hand side of equation (1-1) was equated to the yield strength of the material and the diameter for the splined shaft was thereby determined. A safety factor of 2 was used for the calculated shaft diameter. The calculated shaft diameter proved useful as a guideline for establishing the final design specification for the nominal diameter of the splined shaft. A number of additional factors had to be taken account for this dimensional specification, namely the sizes of locating bearings, the splined shaft assembly housing and the actual spline sizes themselves. The size of the splines depended on the load transfer requirements, and the most important criteria were material selection, the bending stresses induced and the minimum splining width. The involute spline design involved an iterative process of trial and error to determine the optimal dimensions required, as there are a number of parameters that have to be considered. An example calculation for the splined shaft diameter that includes the force analysis is presented in Appendix E.

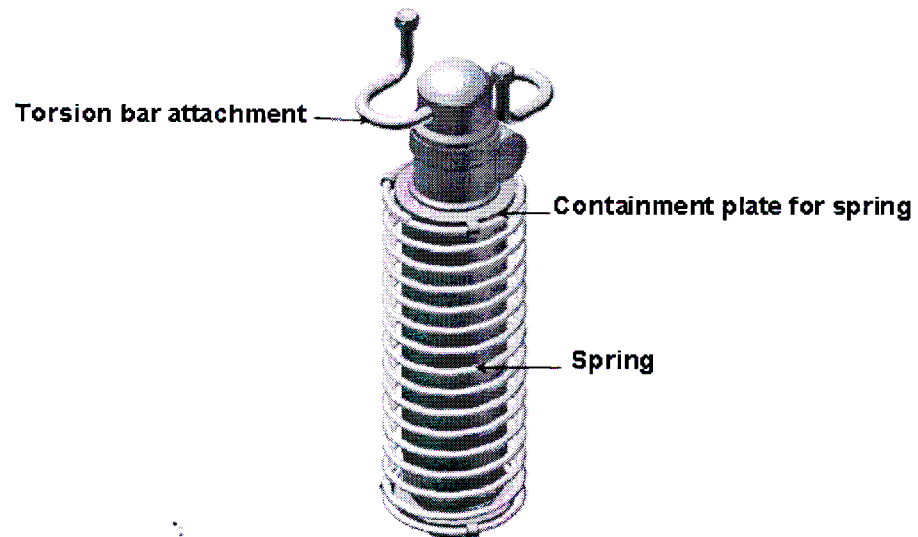
### **2.2.2.3 Linear actuating cylinder assembly**

The linear cylinder serves to slide the beams forward relative to each other. When the piston is at its full stroke, the beam is advanced approximately 0.6 m. The length of the cylinder and stroke had to be optimally designed to enable this advance as a two-step process.

### **2.2.2.4 Hydraulic props**

The hydraulic props used are 20/40-ton props and operate at a pressure of 40 MPa. The internal valve system for the props was initially one with a standard release valve. This system was modified by Unique Engineering to an internal combo-valve system comprising rapid-yielding and slow-release valves. The rapid-yielding capability is essential for the absorption of rockburst energy underground. A special filling valve was designed for pressurisation and depressurisation of the hydraulic props (see Figure 2-13). The hydraulic prop heads were machined flat to accommodate spherical steel seatings, which act as ball-joints for the prop-headboard interface. The prop heads were also cross-drilled to accommodate the torsion bar components that secure the props to the beams.

The hydraulic props used were single-acting props. A co-axial spring was designed to provide retractional dynamics to the props. The spring set-up was a temporary solution for the prototype as it was envisaged that double-acting hydraulic props could be developed at a later stage.



*Figure 2-8 Hydraulic prop assembly*

#### **2.2.2.5 Torsion bars**

Refer to Figure 2-8 above. The torsion bars are spring steel bars that interface with the hydraulic props and beams. These bars are designed to secure the props to the beams in such a way that the props are constrained to two degrees of dynamic freedom relative to the beams (in the advance and transverse directions of the beams). In the advance direction, the torsion bar allows the beam to swing backwards thereby aiding the motion of the beam during the advancement process. In a transverse orientation, the torsion bar permits a very small degree of motion or 'play'. This allowance was incorporated for the purpose of easing the assembly process.

#### **2.2.2.6 Hydraulic control system**

The main requirements for the control system design were the following:

- 1) Incorporation of safety mechanisms to prevent excessive pressure build-ups in the respective hydraulic lines;

- 2) Independent control of the rotational and linear cylinders, and also for each hydraulic prop;
- 3) Catering for separate pressure requirements, namely 20 MPa for the beam hydraulic cylinders and 40 MPa for the hydraulic props;
- 4) A suitable pumping system with a good range of pressure and delivery rate specifications capable of meeting the requirements of the hydraulic props and beam cylinders; and
- 5) Independent pressure relief on the hydraulic lines for the beam cylinders and hydraulic props to enable individual attachment or detachment of hoses from the respective components in case the need for trouble-shooting occurs or maintenance needs to be done. Independent pressure relief is also important after the system is fully installed and blasting operations begin. (Before blasting operations commence, the pump, control system and hydraulic hoses need to be detached and stowed safely, away from the rock blasting zone.).

To meet these requirements, the system depicted in Figure 2-9 below was designed. The valve system comprises four main units and sets of ball valves that perform specific functions. The four main units consist of two spool valve arrangements (Figure 2-9, item 1) for control of the hydraulic props (stabiliser cylinders) and beam cylinders, and two pressure-relief valves (Figure 2-9, item 4) to reduce the pressure to 20 MPa for the beam cylinders. The type of spool valve used is basically a four-way directional-control valve system. This valve type contains four ports, namely a pressure port, a return or exhaust port, and two cylinder ports. The pressure port is connected to the main system's pressure line and the return line is connected to a reservoir. The two cylinder ports are connected to the rotational and linear cylinders respectively. For more detail on the operation of the control system and the different pressure-relief valves, refer to Appendix D.

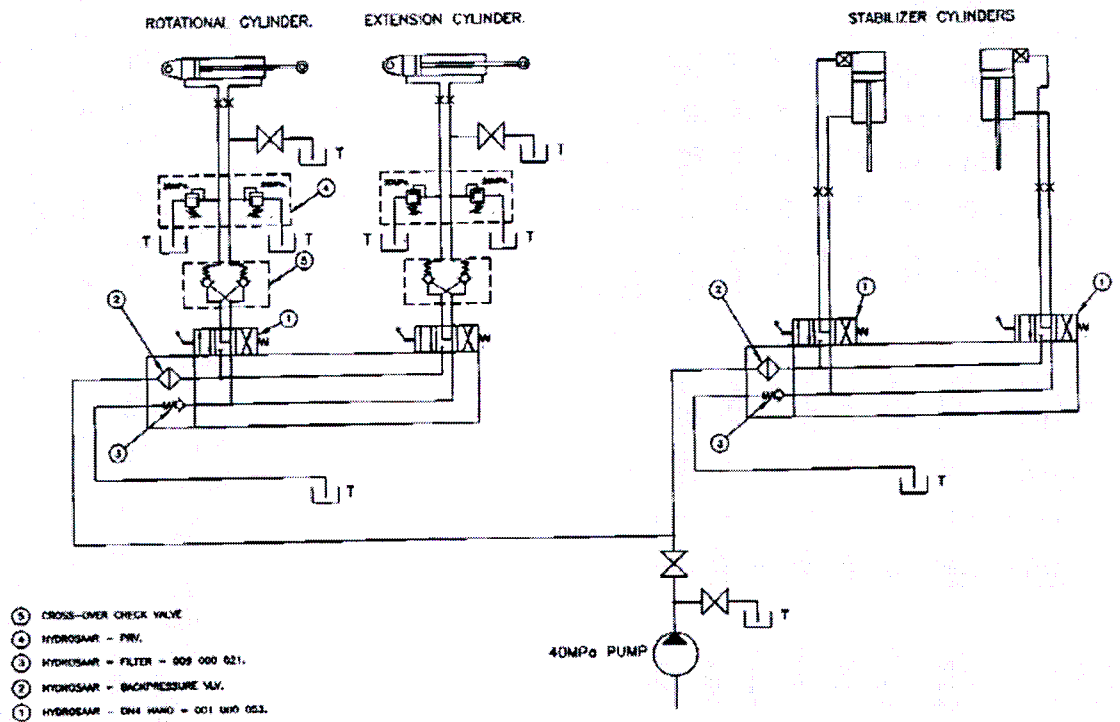


Figure 2-9 Hydraulic control circuitry

The control valves are mounted in a skid frame, which provides for ease of movement of the valves along the footwall and protection for the valve components. The pump unit (Figure 2-10) is a double-acting unit with an air inlet and a hydraulic output. It is encased in a stainless steel box (to protect the internal components against damage, contamination and corrosion, which are common problems underground) with angle-iron skids for movement along the footwall. The pump was custom designed at PF Hydraulics (Villani, 2003).

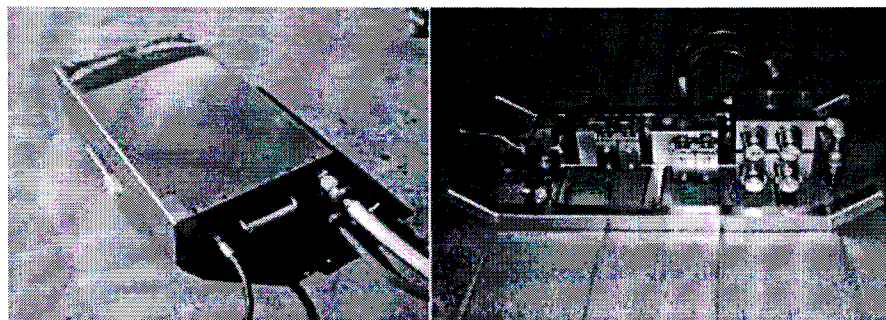
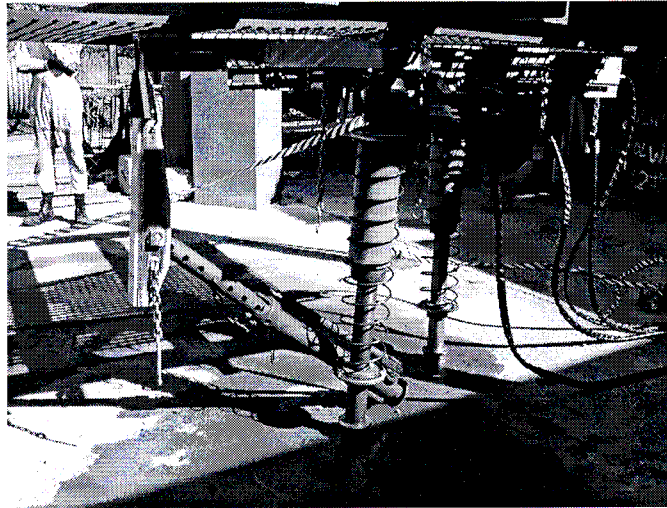


Figure 2-10 Pump and control system

### 2.2.2.7 Camlok prop

A Camlok prop rated at 10 tons is attached to the front of the beam on the left, which contains the rotational cylinder. The attachment plate for the Camlok is so designed that the Camlok can be stowed away beneath the beam during blasting operations, and reinstalled before drilling operations commence. The attachment plate is a robust steel component that also serves to deflect material during blasting.

A special headboard has been designed for the Camlok to provide more areal coverage in the face area during drilling operations (Figure 2-11). The headboard is made of 8 mm U-section mild steel and is approximately 0.7 m in length.



*Figure 2-11 Camlok prop attached to front of headboard*

## 2.3 The final prototype assembly

Refer to Appendix D for detailed information on the operational procedures for the different component assemblies. This section provides a brief description of the assemblies and how the main components come together to form the final prototype support system.

The beam assembly was designed such that the beams could be assembled with ease in the underground environment (the rotational and linear cylinders are pre-assembled on surface). This entailed making the components modular and utilizing a minimal number of components to

manually assemble the individual beams. The assembly is shown in Figure 2-12. The components for the beam assembly include retaining rings, spacers and sellock pins for securing the assembly.

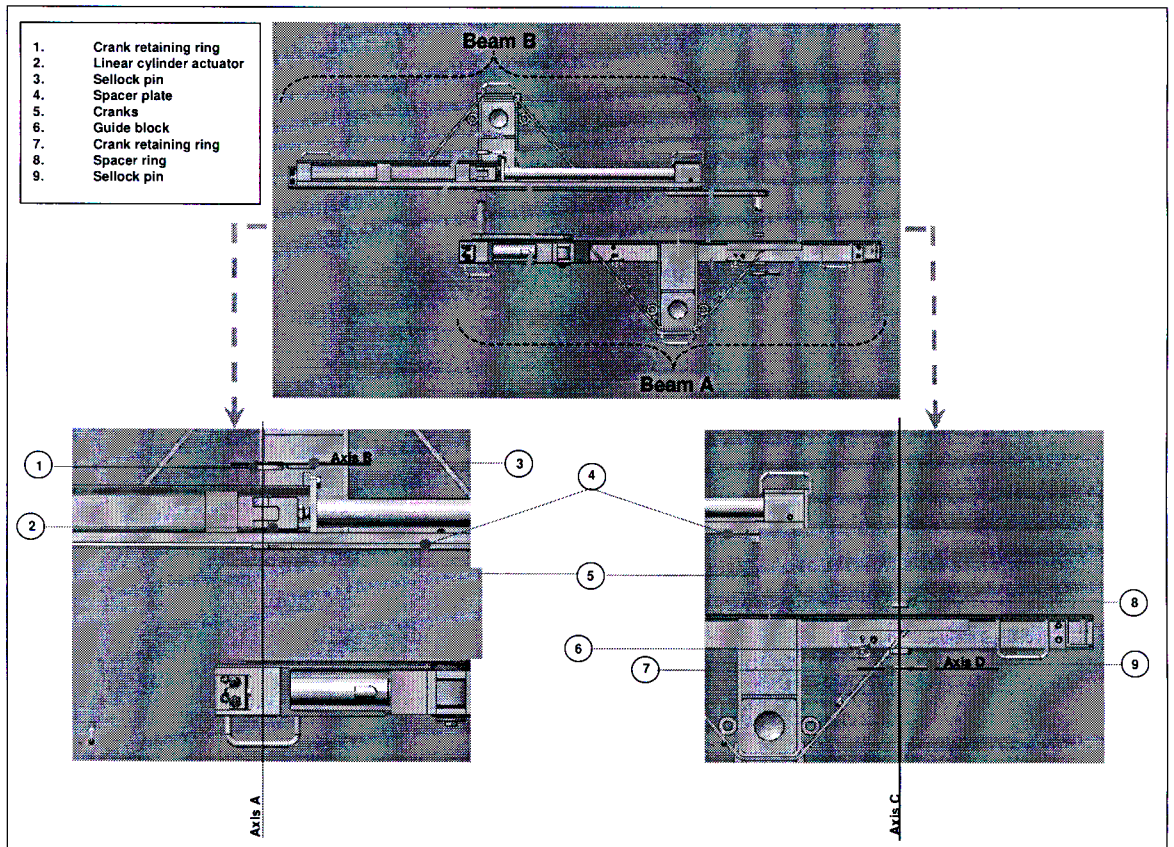
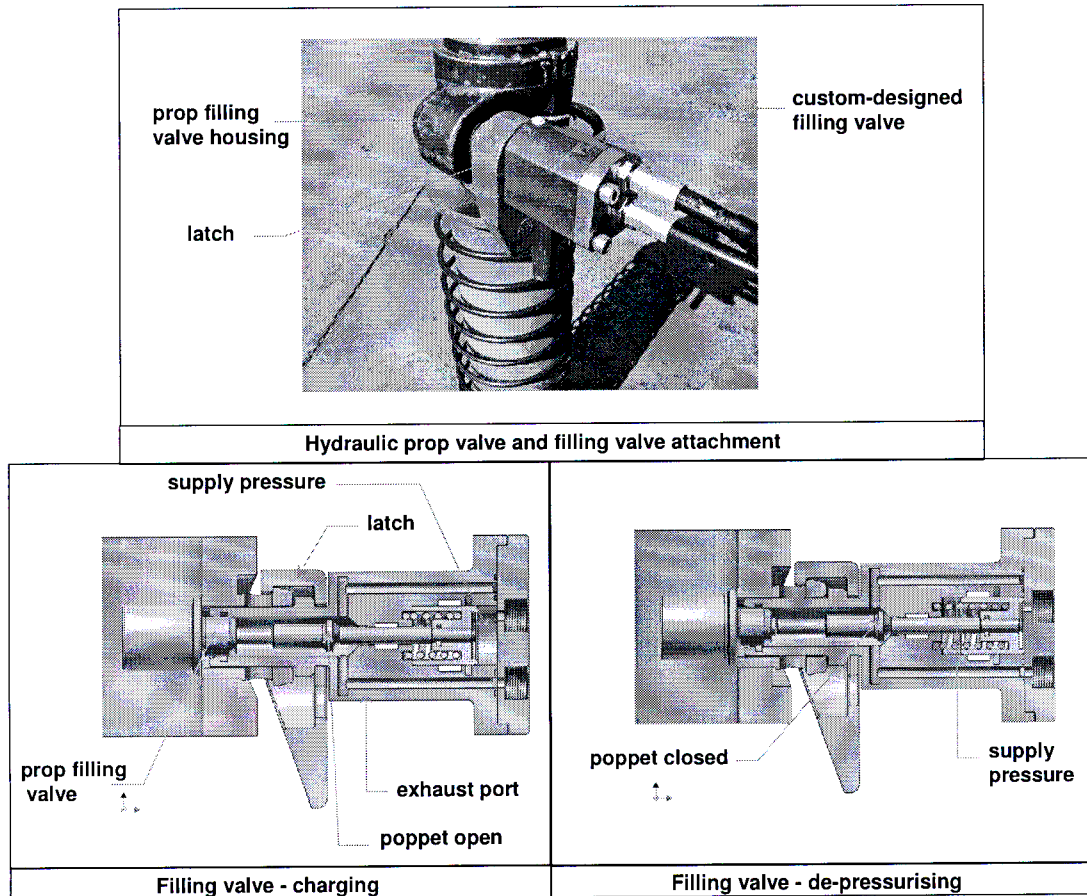


Figure 2-12 The components for the assembly of the individual beams and their assembly axes (plan view)

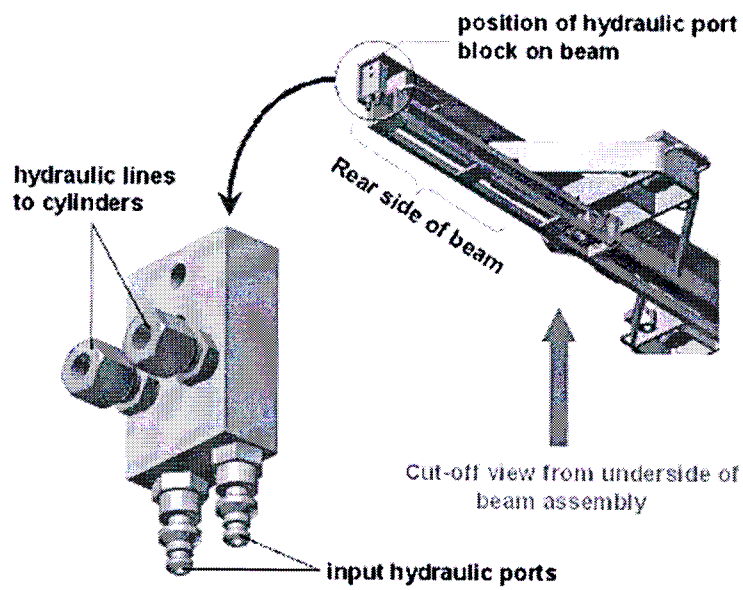
Once the beams are assembled the hydraulic props can be attached to the beams with the torsion bars. As mentioned previously, a special filling valve (Figure 2-13) was designed for the pressurisation and depressurisation of the hydraulic props and is connected to the prop filling valve after the beams and attached hydraulic props are lifted in position by means of an installation system (An installation aid (refer to sub-section 3.5.1.4) had to be developed to aid in lifting the beams and hydraulic props in position before pressurising the props against the hanging wall.) The valve has a spring-loaded poppet type valve mechanism that opens and closes supply and exhaust ports for the charging and depressurisation of the respective hydraulic prop. A custom-designed filling valve is inserted into the filling valve of each of the hydraulic

props and is kept in place with a latching device. The charging and discharge hydraulic lines of the filling valves are connected to the hydraulic control system.



*Figure 2-13 Hydraulic prop and filling valve interface*

The hydraulic props are then pressurised by means of the hydraulic control system, to set the beams against the hangingwall. Once the beams are set, the hydraulic lines from the control system to the beam cylinders are connected via the port blocks located on the respective beams (Figure 2-14). The system is now ready for operation.



*Figure 2-14 Hydraulic lines for operation of beam cylinders*

A simplified version of the separate component assemblies to form the finalised prototype system is depicted in Figure 2-15. A photo of the system being tested in the underground mining environment represents the final prototype system. Note that the arrows on the assembly lines are used as an aid to show the connections between the different components/assemblies.

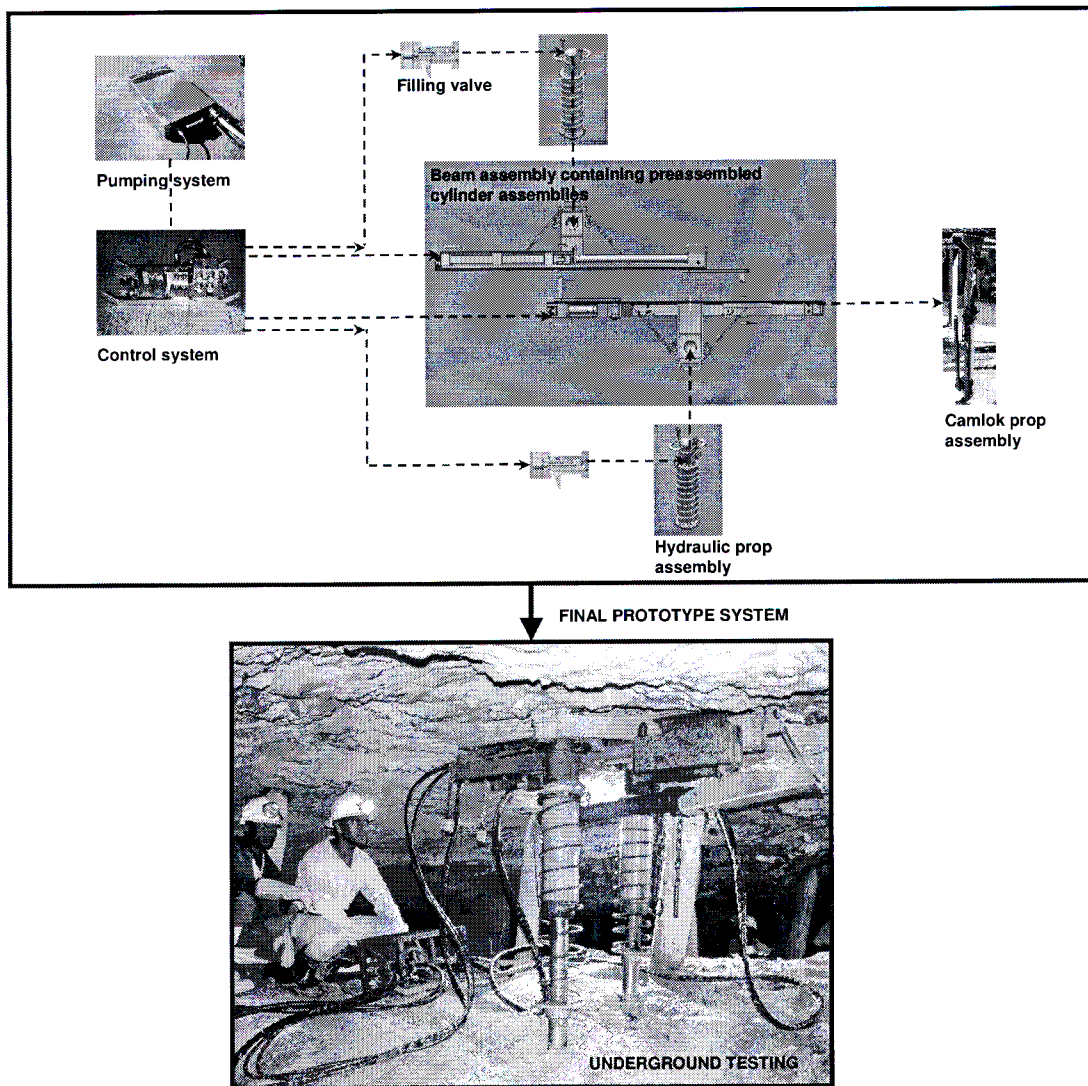


Figure 2-15 Final prototype support system

#### 2.4 Risk assessment and operational procedures for the finalised prototype system

A comprehensive risk assessment was formulated by Snowden Mining Industry Consultants in coordination with CSIR: Mining Technology. This assessment was performed in June 2004 and entailed a complete risk analysis for the operational processes involved in the implementation of the walking beam stope support system. The risk assessment is reproduced in Appendix A.

From the risk assessment a coherent set of procedures was drawn up, by the author, for the operation of the walking beam stope support system. These procedures must be strictly adhered

to in order to ensure that the system is used correctly and safely and poses no threat to people working in the vicinity of the machine. A compulsory requirement for operating the walking beam system is that only personnel who are specially trained by designated trainers to use the machine in accordance with the operational procedures are permitted to operate the machine.

These operational procedures were divided into the following groups:

- Handling procedures;
- System assembly;
- System installation;
- System operation; and
- System removal.

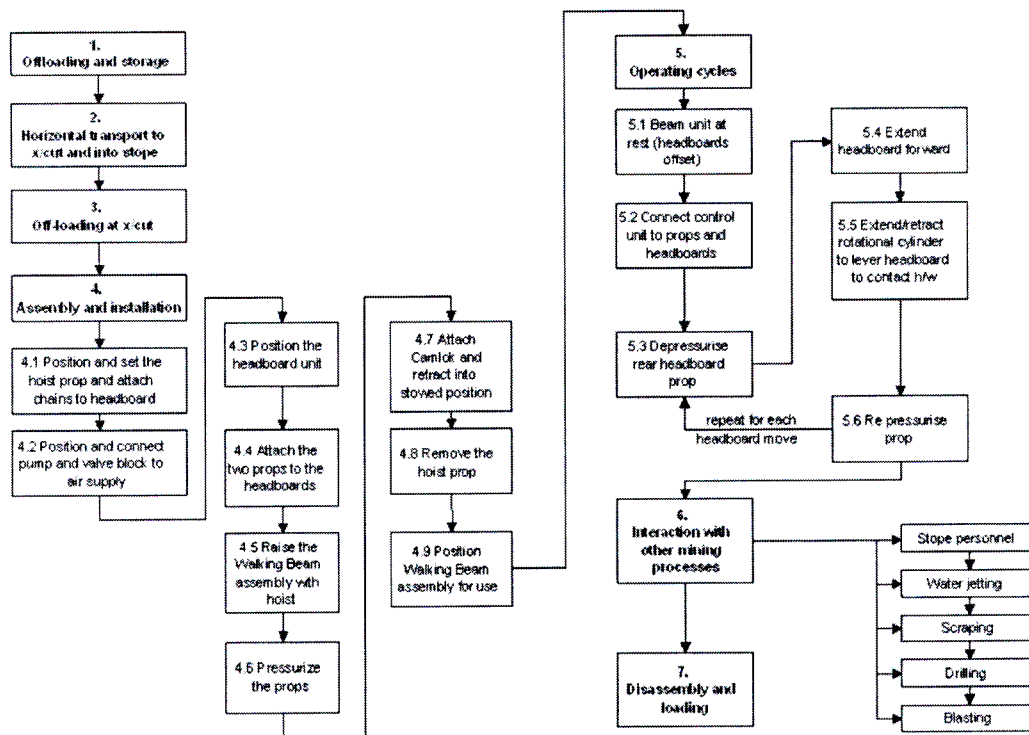


Figure 2-16 Process flow diagram of the use of the walking beam stope support system (After Forbes, 2004)

The process flow diagram in Figure 2-16 shows a summary of the primary steps involved in the use of the walking beam stope support system in sequential order. More detail on the operational procedures is provided in Appendix D.

### 3 DEVELOPMENTAL PROGRESSION OF THE SYSTEM

#### 3.1 Year 1: 2000

The SIMRAC project GAP 708 was initiated in this year. GAP 708 was undertaken as an initial phase to the project and entailed specifying requirements for an alternative support system to current support technologies and the conceptualisation of potential system designs.

The remotely advanced headboard system, currently known as the 'walking beam stope support system' was conceptualised, evaluated and selected as the best alternative stope support system, in comparison with ten other conceptualised systems. Current support system technologies were used as a benchmark for the system evaluations.

The following systems were generated from the workshops and brainstorming sessions:

- Remotely advanced headboards (the 'walking beam');
- Longhole drilling;
- Rockbolt reinforcement;
- Twin-beam support system;
- Walking beam wishbone support system;
- Modified spiling system;
- Remote miner;
- Safety cell;
- Powered shields;
- Pneumatic support system; and
- High-pressure stope system.



Towards the end of 2001, the designs and machine drawings for all the components were finalised, which meant that the manufacturing of prototypes could now proceed.

### 3.3 Year 3: 2002

The manufacture of components began in January 2002 and by the end of March 2002 two prototype support units had been manufactured. The headboards were deflection tested and the system dynamics were inspected at CSIR: Mining Technology. The prototype units were then taken to Kloof Gold Mine for further functionality tests and, thereafter, to the Savuka test facility for drop-load testing.

#### 3.3.1 Headboard deflection tests

The headboards (mild steel) were deflection tested in a press at CSIR: Mining Technology's Cottesloe facility. They were placed under a press and loaded sequentially to 5, 10, 15 and 20 tons. The test setup for the headboards was not documented at the time and could therefore not be detailed in this section. It is known, however, that an SABS standard for beam testing was adhered to.

The loading was carried out three times and an average deflection reading was taken. After each headboard had been loaded, the load was removed and the headboard was checked to see if the deflection had returned to zero. On all occasions, the deflection readings returned to zero after removal of the load. This showed that no plastic deformation had occurred. The results of the tests are given in Table 3-1.

Table 3-1 Load and deflection data for the headboard

Load [tons]	Load [kN]	Deflection [mm]			
		1st Reading	2nd Reading	3rd Reading	Average
5	49.05	1.46	1.48	1.35	1.43
10	98.1	2.42	2.43	2.53	2.46
15	147.15	3.64	3.63	3.61	3.63
20	196.2	4.7	4.54	4.51	4.58

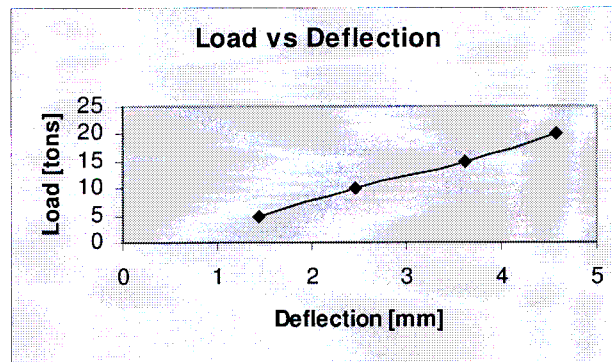


Figure 3-2 Load vs. deflection for the CSIR: Mining Technology headboard (After Ismail, 2002)

A basic finite-element analysis was performed and the results of the analysis matched well with the results from the actual tests performed. From Figure 3-3 it can be seen that the finite-element model gave a maximum deflection of 4.66 mm (at 20 ton loading) as compared with the actual test, which gave a maximum deflection of 4.58 mm (Figure 3-2) (Ismail, 2002).

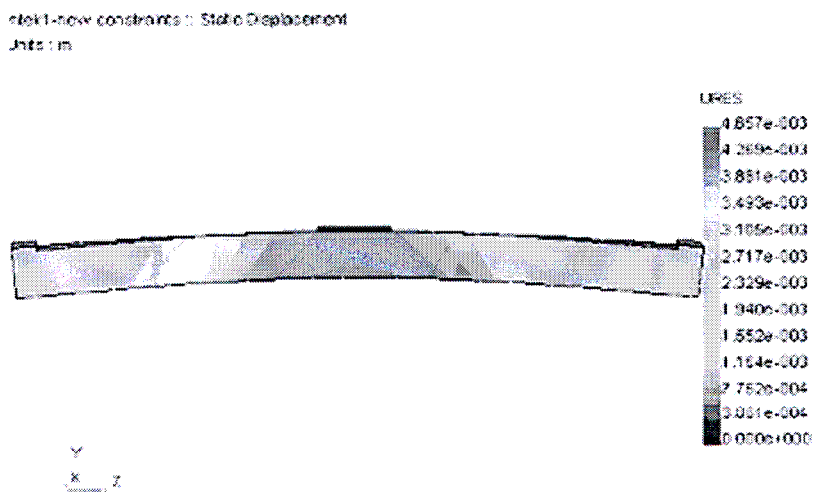
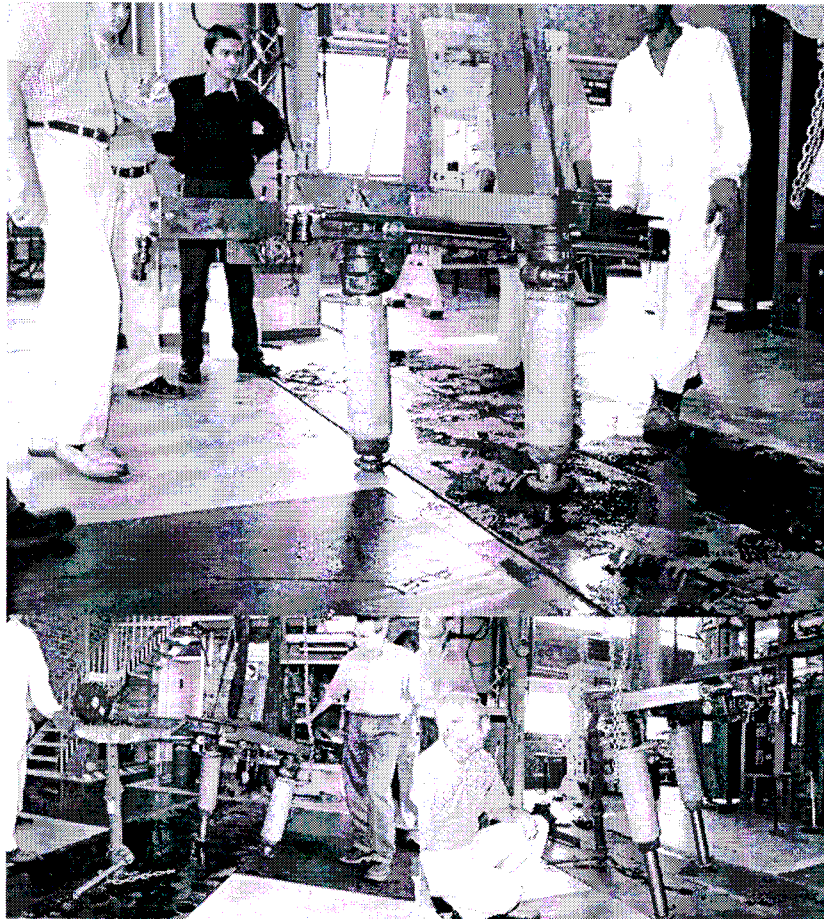


Figure 3-3 Front view of headboard showing the position of maximum displacement (After Ismail, 2002)

### 3.3.2 Inspections at CSIR: Mining Technology's support testing lab

Inspections were performed on the units at CSIR: Mining Technology's support testing laboratory (Figure 3-4). The hydraulic props were attached to the unit and it was hoisted into position. The hydraulic props were then pressurised and depressurised and the system dynamics were checked from a functional perspective in order to identify hazards and to check whether there were any problems with the critical mechanical areas of the machine. Possible design improvements were noted after the inspections and these were finalised on the basis of the testing at Kloof Gold Mine, which is discussed in Section 3.3.3.



*Figure 3-4 Photos showing prototype unit being inspected by CSIR: Mining Technology*

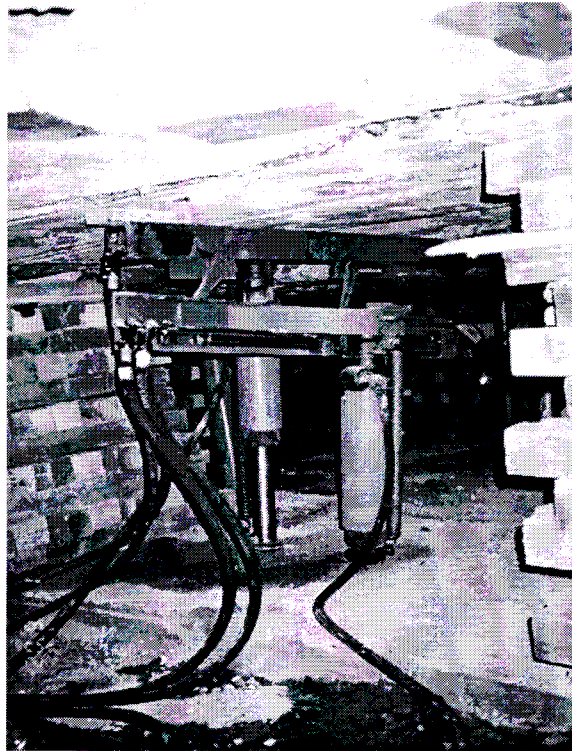
### 3.3.3 Surface tests at Kloof Gold Mine

The units were taken to a mock-up stope at Kloof Gold Mine (see Figure 3-5). The main objectives of the testing here were to evaluate the system's functionality and to identify potential operational shortcomings and design improvements.

At this stage of the system's development, the hydraulic control system consisted of sets of ball valves. This valve system was used to control the fluid flow through the beam's hydraulic cylinders and the pressurisation and depressurisation of the hydraulic props. The control of the beam cylinders and props entailed opening and closing sets of valves in the correct sequence. This method of operation was not a preferred solution as it was very cumbersome and did not incorporate a high degree of safety. However, it provided a preliminary solution for the system's functionality testing.

The units were advanced forward and backward successfully. The following system optimisations were concluded from the test:

- An improved control system, entailing a proper valve housing and incorporating mechanisms to check pressure in the system
- An attachment to the hydraulic prop for constraining the prop in a mode of torsion and linear displacement
- A reduction in the mass of the headboards
- Hydraulic prop extensions to cater for a range of stope widths
- Hydraulic prop modification to allow retraction of the prop, as the props were single-acting.

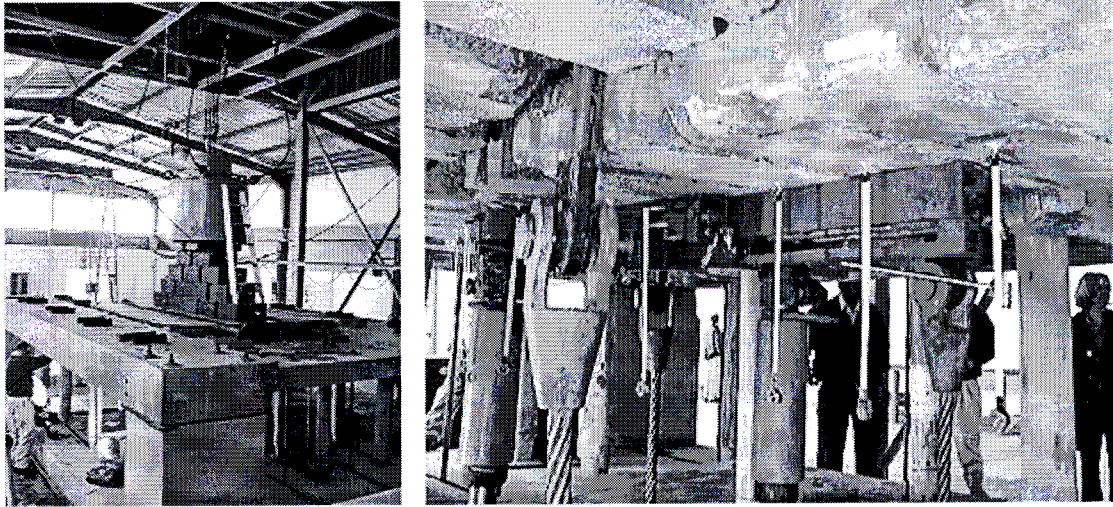


*Figure 3-5 Prototype unit installed in mock-up stope at Kloof Gold Mine*

### **3.3.4 Drop test at Savuka Test Facility**

Drop tests were conducted on the prototype units at the Savuka Test Facility. The tests entailed dropping a 10 ton weight onto a mock-up stope set-up, beneath which the walking beam was installed (Figure 3-6). The purpose of these tests was to test the support capabilities of the system.

The walking beam supported the simulated hangingwall successfully after successive dynamic impacts from the 10 ton weight. SIMRAC members, mine personnel and manufacturers of underground support equipment attended the tests.



*Figure 3-6 Drop test*

### **3.4 Year 4: 2003**

Design and conceptualisation work was undertaken to establish how the system could be optimised as concluded in Section 3.4.1. The designs were finalised and two new prototype units were manufactured. A rig was built on CSIR: Mining Technology's premises to perform tests on the system. After the surface tests, minor refinements were made and the units were then taken to Randfontein Estates for underground trials.

#### **3.4.1 Design improvements**

The following design optimisations were made to the walking beam support system to improve performance:

- The main mass-contributing components (making up approximately 80% of the total headboard masses) were reduced in section in order to reduce the overall weight of the headboards;
- Handles were attached to the headboards for lifting purposes. HDPE plastic strips were attached to the headboards for increased friction between the headboards and the hangingwall, and also to absorb energy during pressurisation;
- The rotational and extension cylinders for the headboards were resized for increased capacity and the sub-components were standardised;

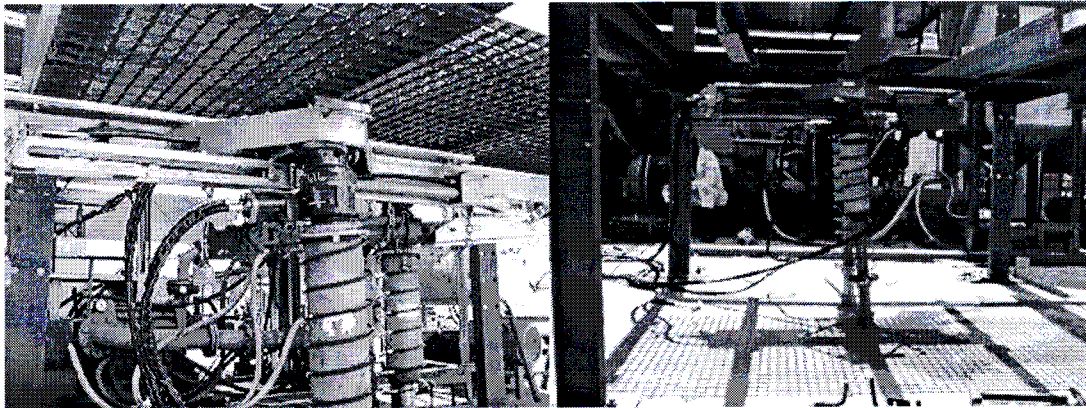
- Hydraulic prop torsion bars were designed for securing the hydraulic prop to the headboards and to constrain the props, as discussed in Section 3.4.2;
- A co-axial spring was designed to provide retractional dynamics to the hydraulic props;
- The mass of the temporary headboard for the Camlok prop was reviewed and a latch arrangement was designed for ease of operation and stowage of the Camlok prop; and
- The hydraulic control system was completely redesigned to give a more compact system configuration.

### **3.4.2 Surface tests**

After the completion of the design modifications, surface tests were performed at CSIR: Mining Technology to evaluate the system's functionality before underground trials began.

The test rig for the surface tests consisted of steel channel sections secured to each other to form a structural framework. Vertical steel sections were secured to the ground with sets of rock bolts rated at 10 ton load resistance for the purpose of anchoring the rig to the ground. The bolts were anchored with a rockbolt resin that bonds to form a tight, solid bond anchorage. The rig had to be able to withstand cumulative vertical loads of up to 40 tons, as the hydraulic props for the walking beam were 20/40-ton props.

The headboards of one prototype unit were assembled and hydraulic props were attached to each headboard with torsion bars. The unit was then manually lifted into position and pressurised against the steel frame. The hydraulic props were observed to be not standing perpendicular to the ground as a result of the indentations and plastic deformation on the mild steel torsion bar connections. New spring steel torsion bars were then designed and manufactured for increased strength and elasticity in order to avoid these problems. The new torsion bars were tested and found to perform significantly better. Figure 3-7 shows photos of the unit installed in the test rig.



*Figure 3-7 Prototype unit installed in test rig at CSIR: Mining Technology*

The prototype was then advanced forward in the test set-up. The unit's hydraulic components functioned well and the dynamic operations for the unit's advancement proved successful.

The surface tests also enabled conceptualisations to be made for designing blast-protective mechanisms for the system. Special blast-protective shields were designed for the purpose of protecting components that would be susceptible to damage by blasted rock. These shields are made of mild steel and have angle-iron reinforcements along their lengths for increased buckling resistance. The shields have encased sleeves for bolt attachments onto the headboards. Figure 3-8 shows one of these shields.



*Figure 3-8 Blast-protective shield*

### **3.4.3 Preliminary underground trial at Randfontein Estates (Cooke 2 Shaft)**

The prototype units were taken to Randfontein Estates for a preliminary underground trial. Randfontein Estates' Cooke 2 Shaft had kindly allocated for the trial a special short panel that was not subjected to production pressures. The stope was on the Upper Elsburg 1A (UEIA) reef, which has a strong, glassy hangingwall of quartzite with no parting planes close to the reef. The hangingwall of the stope had a factory-roof profile with brows of up to 250 mm depth.

The stope width in the back area where initial mobility tests were done varied from 1.1 m to 1.5 m. In the area that was blasted during the trial, the width was reduced to approximately 1 m. The footwall was irregular, had humps and dipped both towards the face and away from the gully at angles of less than 10 degrees.

The preliminary trial was conducted on the units in order to assess their overall performance and to identify problem areas, which would then be rectified before going ahead with the full underground trial in a production stope.

The main objectives of this initial underground trial were to assess:

- 1) The ease of transportation of the units into the stope panel in two configurations of sub-assembly;
- 2) The time required to assemble the units in the stope and the ease with which this could be done;
- 3) The mobility and supporting effectiveness of the units in various conditions of hangingwall and footwall roughness; and
- 4) The susceptibility of various components to blast damage.

The units were transported to the stope panel in two assembly configurations. One prototype was taken fully assembled and the other with the headboards linked to each other and with the hydraulic props detached. Although the units were heavy, they presented no serious problem during the transportation process. There is, however, room for improvement in this area. Figure 3-9 shows some of the workers lifting the headboards out of the cross-cut into the centre gully.

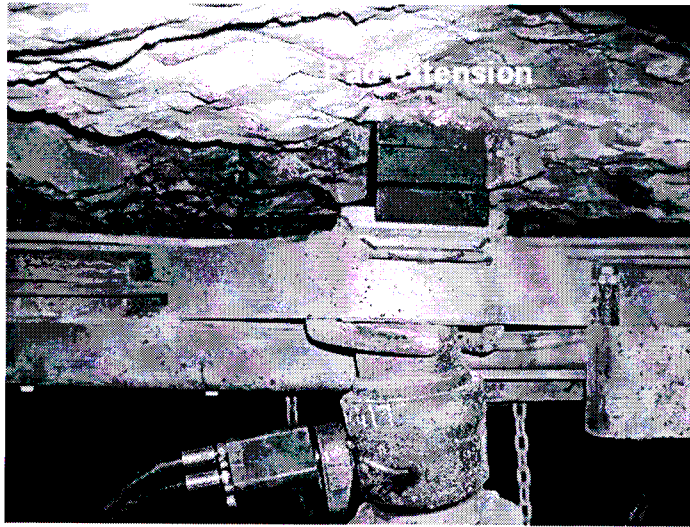


*Figure 3-9 Moving the units towards the test panel*

Assembly of the units proved to be difficult and took approximately 30 minutes to complete. Installation was also difficult, primarily because of the weight of the units. It was concluded that an auxiliary system consisting of a jack and a winch or pulley to rotate and stabilise the system would be required to aid installation.

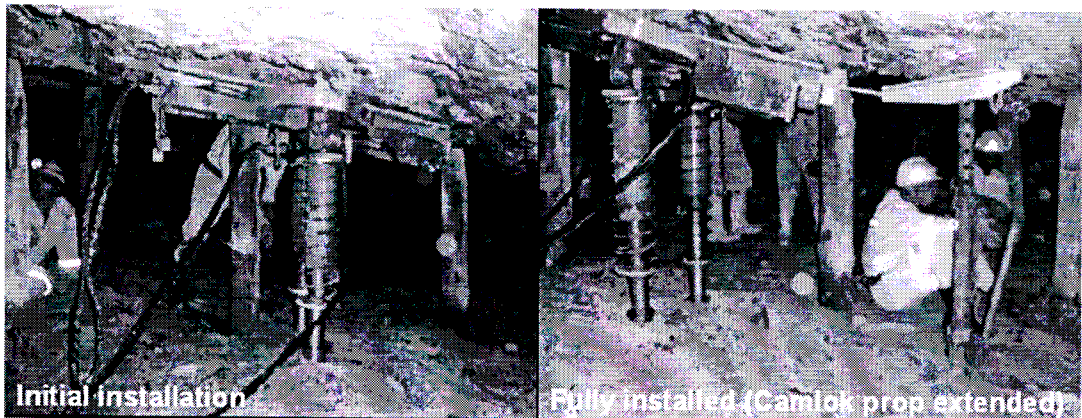
Functioning of the system was ultimately successful. The units were tested in the back area with variable stope widths up to 1.4 m, with up to two extension pieces on the hydraulic props.

For certain areas along the hangingwall, the headboard pads did not contact. As a result, a rotational moment was induced in a plane parallel to the headboard, causing the unit to rotate. To alleviate this problem, special pad extensions were manufactured (Figure 3-10). These pad extensions were made in various sizes to accommodate irregularities in the hangingwall.



*Figure 3-10 Extension slid over pad to enable contact with hangingwall*

It was concluded that further tests would have to be done on the units on surface to evaluate their performance for steeper dip angle conditions. On a relatively shallow dip, the unit had proved its functionality and had 'walked' satisfactorily. Figure 3-11 shows photos of the initial and final installations of the unit in the test stope.



*Figure 3-11 System installation*

The units were subjected to four blasts without the use of barricades. The distance from the tip of the headboard to the stope face varied from 1.6 m to 2.8 m before each blast. The face

advance per blast was approximately 0.6 m. The units did not sustain any serious damage and the trials proved successful for the given blasting conditions (Figure 3.12).



*Figure 3-12 Condition of unit after being subjected to blasted rock*

*Note the roughness of the hangingwall*

From the underground tests, the following improvements/modifications were identified:

- The hydraulic prop retraction springs needed to be redesigned as the original spring was too weak to retract the piston fully;
- Improvements/enhancements for installation of the units needed to be investigated;
- The hydraulic hose adaptor blocks needed to be repositioned for easier attachment; and
- The crank linkages between the headboards needed to be strengthened as they were slightly bent due to the irregular contact between the headboards and the hangingwall (see Figure 3-13).



*Figure 3-13 Underside of headboards showing the position of the hose adaptors at the back of the headboards, and the slightly deformed crank*

### **3.5 Year 5: January 2004 – March 2005**

The two prototype units that had undergone the preliminary underground trial in 2003 were modified for improved performance. Two additional units with these new modifications were manufactured.

A concrete ramp and a framework with an inclined, steel roof surface were built on CSIR: Mining Technology premises. These structures were built for the purpose of simulating an underground stoping scenario with a dip angle of approximately 24 degrees. The units were

tested in this rig set-up to evaluate system performance in a steeply-dipping stope. An installation system was designed to make the initial installation process for the walking beam support system easier. It entailed the use of hydraulic props, chain hoists, cables and attachments. This system for installation was also assessed in the test rig at CSIR: Mining Technology.

An underground site was established at Goldfields East Driefontein 1 Shaft for the purpose of evaluating the walking beam support system's integration with mining processes. The units were transported underground, but problems were experienced that resulted in critical time delays in the project. It was also found that the stope width of the test panel had been widened beyond the operating range of the walking beam system. As a result, a second site was needed.

The second site was identified and arrangements followed for getting the equipment into this area. This new site was found to be a more appropriate area for testing, as the access ways were better and the stoping conditions satisfactory for the operational specifications of the walking beam system. Some key problem areas were identified during the process of setting up the prototype units in the stope; design enhancements/modifications were formulated to address these problem areas. Certain components were damaged during the testing process and were taken to Novatek Systems for repair.

However, testing of the units was again hampered as a result of the reef width being widened beyond the operating range of the walking beam system. As a result, the primary objectives of the underground trials could not be fulfilled. At this stage, a decision was made to move the equipment to Goldfields East Driefontein 4 Shaft as recommended by mining personnel.

The units were repaired at Novatek Systems and taken to East Driefontein 4 Shaft so that underground testing could be continued. They were transported and assembled underground in December 2004. In 2005 the assembled beams were carried to installation positions in the face area. A unit was installed, but the hydraulic prop spring was damaged. The installation of additional units was hampered by conditions in the panel and the stope width had increased to 2.1 m, which is beyond the machine's capability.

However, all the tests that had been performed at this stage highlighted the system's operational shortcomings and its development needs, negating the need for further tests.

### 3.5.1 Design improvements after initial underground trial

The modifications identified from the preliminary underground trial were done on the two units. Two additional units were manufactured for use in the next set of underground tests in a production stope.

The modifications done on the units are described in the sections that follow.

#### 3.5.1.1 Stronger crank linkages between headboards

The manner in which the individual headboards adapted to the hangingwall in the initial trial resulted in deformation of the cranks, which hampered operational performance since at times the cranks became caught on the bottom edge of the adjacent headboard. The cranks were therefore strengthened for better force transference and bending resistance. The width of the cranks was increased from 16 mm to 20 mm and the height from 30 mm to 70 mm. (See Figures 3-14 and 3-15.)

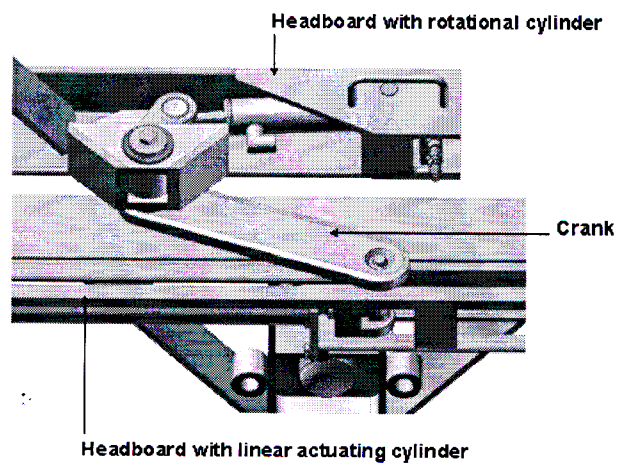
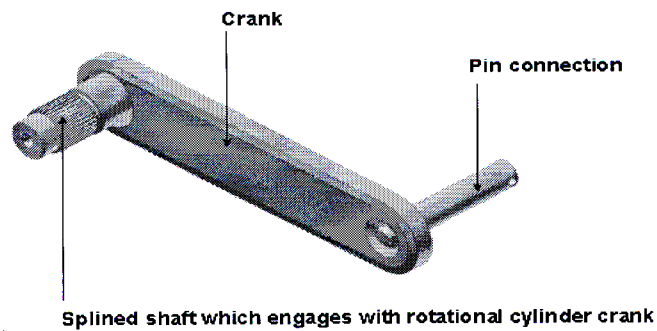


Figure 3-14 Side view showing crank connection between headboards

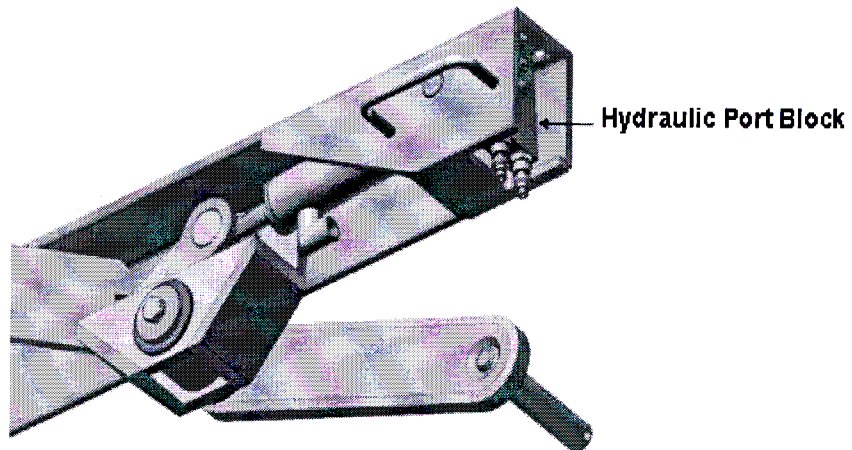


*Figure 3-15 Crank with adjacent connections to respective headboards*

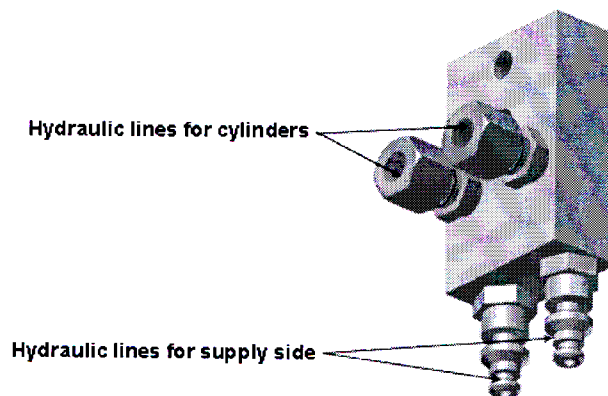
### **3.5.1.2 Re-designed hydraulic port block**

The original hydraulic port blocks were positioned in such a way that the operator had to slide them horizontally into the respective headboards, parallel and close to the hangingwall. This was seen as a safety hazard because attaching the blocks in this manner could lead to injury if the profile of the hangingwall permitted only minimum clearance between the operator's hand and the hangingwall.

The new hydraulic port blocks were oriented in such a way that the operator could attach the blocks from the underside of the headboards safely and more effectively. (See Figures 3-16 and 3-17.) The flexible hosing from the linear actuating cylinder to the interface with the hydraulic port blocks was changed to hard piping, making it more robust.



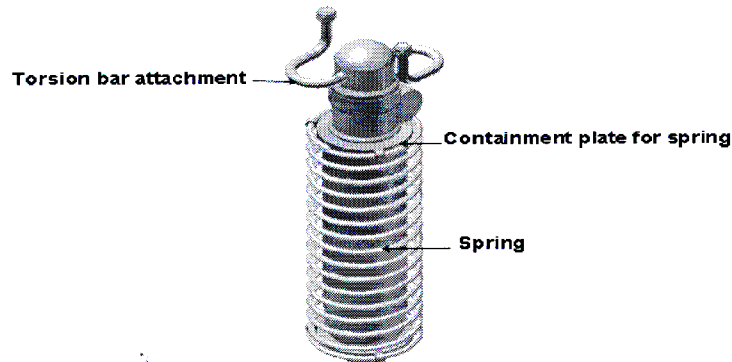
*Figure 3-16 View showing position of hydraulic port block in headboard*



*Figure 3-17 Hydraulic port block and fittings for flow to cylinders and supply*

### **3.5.1.3 Hydraulic prop springs**

The original springs were not strong enough and provided a very slow retraction rate for the prop. New springs were designed to have increased stiffness and a greater reaction force for retracting the prop effectively so that the respective headboard could be moved forward without the prop leg or extension dragging on the footwall. (See Figure 3-18.)

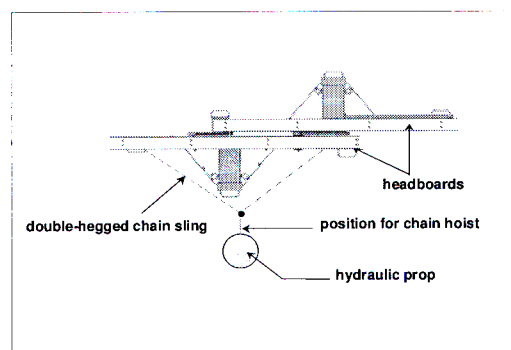


*Figure 3-18 Hydraulic prop and its spring assembly*

#### 3.5.1.4 Installation aid

An installation system was devised to aid the initial assembly, for lifting the walking beam into position and setting the system against the hangingwall. This system was developed under the guidance of Yale Lifting and Mining Products (Davis, 2004).

The system consisted of a hydraulic prop placed on the updip side of the walking beam, and a chain hoist attached to plates that fitted around the head of this installation prop (Figure 3-19). The chain hoist had a sling attached to it that would hook onto attachments on the headboard. In this way the walking beam could be hoisted into position and, thereafter, each of its props could be pressurised to make contact between the beam and hangingwall.



*Figure 3-19 Installation system components*

### 3.5.2 Surface tests on dip

The prototype units were tested in a test rig that was built on CSIR: Mining Technology premises (see Figure 3-20). The main objective of these tests on surface was to evaluate the system's functional performance on a dip angle of 24 degrees. Also, the system dynamics and hydraulic components were checked to ensure sound operations.

The test rig comprised a concrete ramp and structural frame. The frame consisted of an inclined roof structure that was bolted around the ramp to simulate a hangingwall scenario. The area around the ramp had to be concreted to a thickness of approximately 25 cm to accommodate the bolts that prevented the frame from pulling out as a result of the upward loading of the walking beam hydraulic props.

The installation system, which was designed to reduce the excessive manual effort involved in the initial setting up of the system, was also assessed in the rig set-up. After these surface tests, it was concluded that the overall performance of the system was satisfactory, mechanically sound and ready for the underground tests to proceed.



*Figure 3-20 Collage of photos showing building of concrete foundation and testing of system in the rig*

### **3.5.3 Underground trial at East Driefontein 1 Shaft**

The primary objectives of this underground trial were to evaluate the system's functionality, blast resistance and integration with mining activities.

An underground site was established at Goldfields East Driefontein 1 Shaft for this purpose. The mine was chosen because the mining and rock engineering personnel were keen to be involved in the application of the walking beam support system.

Meetings were held with mine personnel to clarify the objectives of and requirements for the underground implementation of the walking beam system. The support standards for the mine were acquired and from this the optimal integration of the system with the mining cycle was formulated.

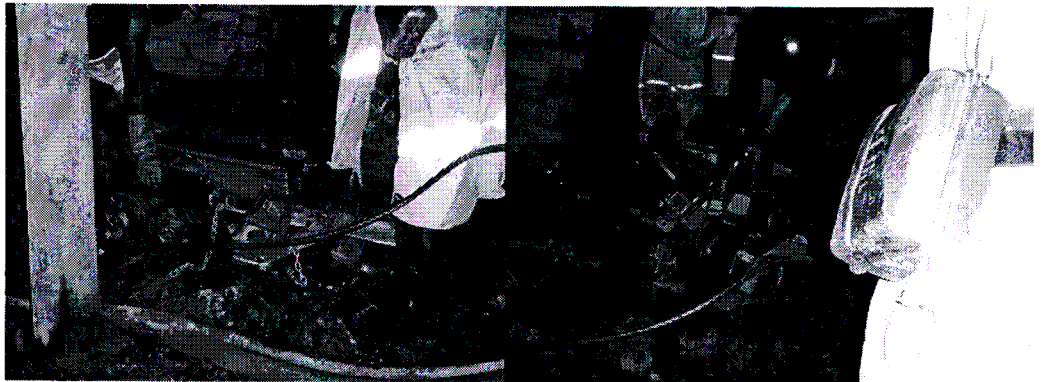
Problems arising from logistical complications associated with the shaft conveyance process resulted in unforeseen delays in moving the equipment to the designated test area. When the equipment had eventually been delivered to the test site, the project team were then notified that the stope width had been increased from 1.5 m to 2.2 m because of the increase in the reef width. This new stope width was beyond the operating range of the walking beam system, which meant that a new site had to be found for the trials.

The need to identify a new site resulted in the project being set back by another week. However, the new site proved to be a more suitable area for the testing. The access ways and transportation route were more convenient, and the stope width (at the time) was more suitable than that of the previous test panel. The key problem areas were identified during the process of setting up the units in the stope.

#### **3.5.3.1 Key results from the trial**

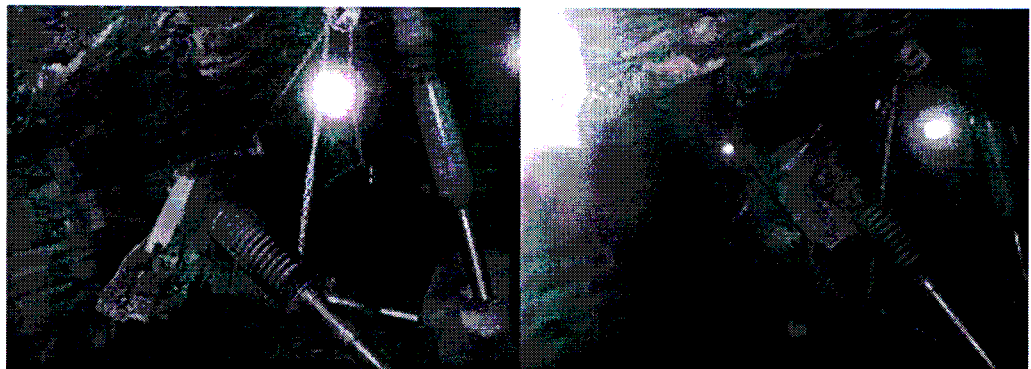
The assembled beams and hydraulic prop counterparts were transported separately to the test position in the stope behind the first row of pack supports. The individual beams were transported already assembled to each other. This was done as a possible solution to the 'transportation' problems experienced during the initial trials at Harmony's Cooke 2 Shaft in Randfontein in October 2003.

Four people carried the beam assemblies into the stope, but they experienced difficulty in this process as a result of the combined weight of the beams, the confinement within the stope and the need to negotiate the undulations in the footwall. It took approximately 30 minutes to move one unit into position. Figure 3-21 shows the transportation of the assembled beams into the stope.



*Figure 3-21 Transportation of beam assembly into the stope*

The system for aiding in the installation of the walking beam system was tested to evaluate its functionality. A major shortcoming encountered with the system was that it did not lift the walking beam up sufficiently to allow the props to be installed at 90 degrees to the dip of the reef, as shown below in Figure 3-22.



*Figure 3-22 Testing of installation system*

Some of the components of the walking beam were damaged during the testing process. One of the hose fittings was broken, but the manner in which this occurred was uncertain. Also, one of the hydraulic props was found to have broken due to non-axial loading that had resulted in plastic bending of the piston. At the time of the installation, three extension pieces were used. The damaged components are shown in Figure 3-23.

It is important to note that the reef width in this second test panel had also increased from 1.5 m to 2.0 m. Once again, this was beyond the operating limit of the walking beam system. To compensate for the increase in stope width, three extension pieces were used on the hydraulic props. The system was designed to function best with a maximum of two extension pieces, however, it was necessary to test the unit with three pieces to fundamentally establish the limitations of the unit while in operation.



*Figure 3-23 Damaged hose fitting and broken hydraulic prop*

As a result of the increase in the stope width, the walking beam system could not be effectively assessed. It was concluded that in order to fully evaluate the system, a third test site would have to be located.

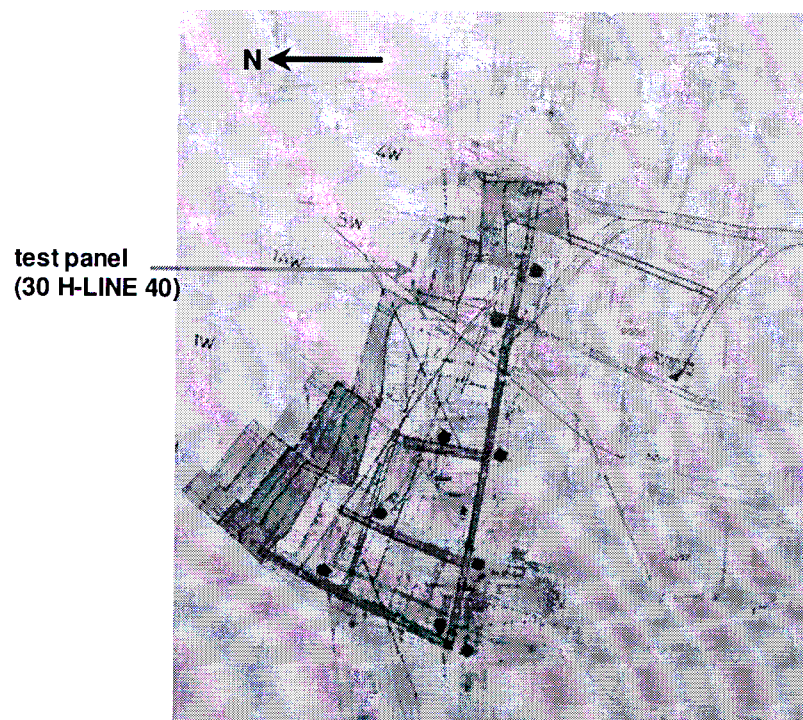
#### **3.5.4 Underground trial at East Driefontein 4 Shaft**

A new site was established at East Driefontein 4 Shaft.

The underground site was identified as shown in the mine plan in Figure 3-24 below. This site was chosen on the basis of the following factors:

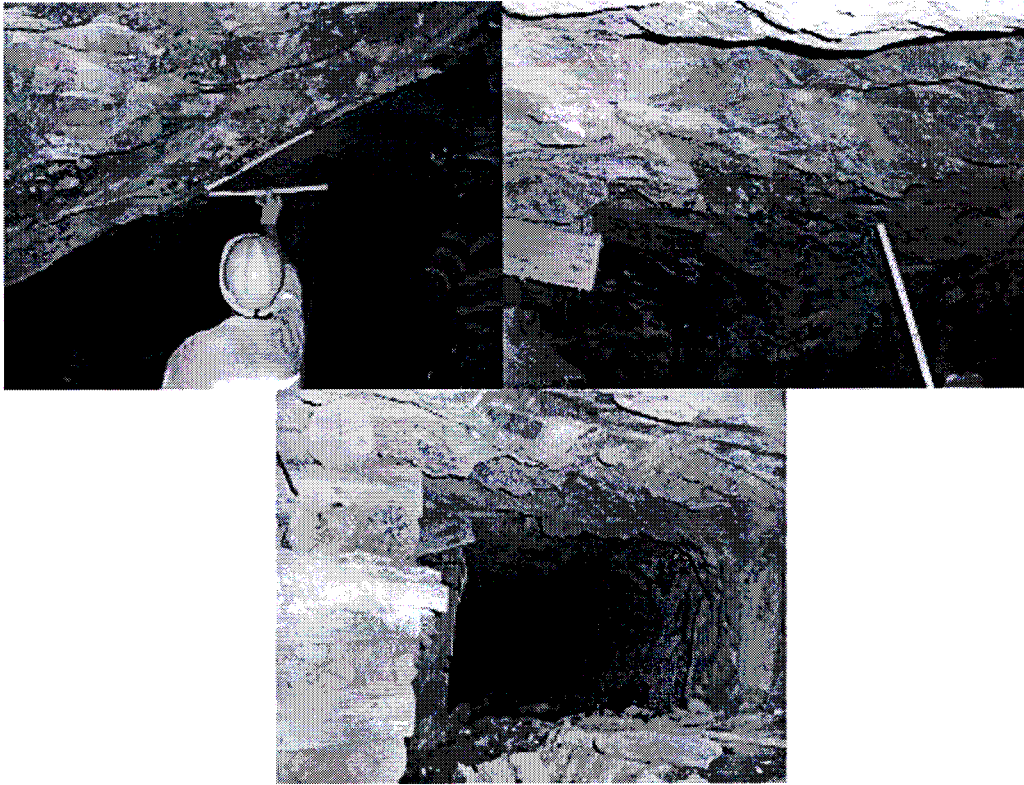
- The average stope width of the panel was 1.2 m;

- The reef dip (on average) was 24 degrees;
- The supply air pressure was approximately 4 bar;
- The distance from the shaft to the test panel was suitable for transportation of the units;  
and
- It was decided that the travelling ways were also suitable for transportation.



*Figure 3-24 Mine plan for East Driefontein 4 Shaft*

An underground site visit was undertaken to assess the actual conditions and to ensure that the system requirements were met (see Figure 3-25). The hangingwall was found to be fairly rough with substantially large recesses/brows in many areas. In this respect the test would be a good representation of how the system adapted to the hangingwall. Factors such as the stope width and reef dip were also verified during the visit.



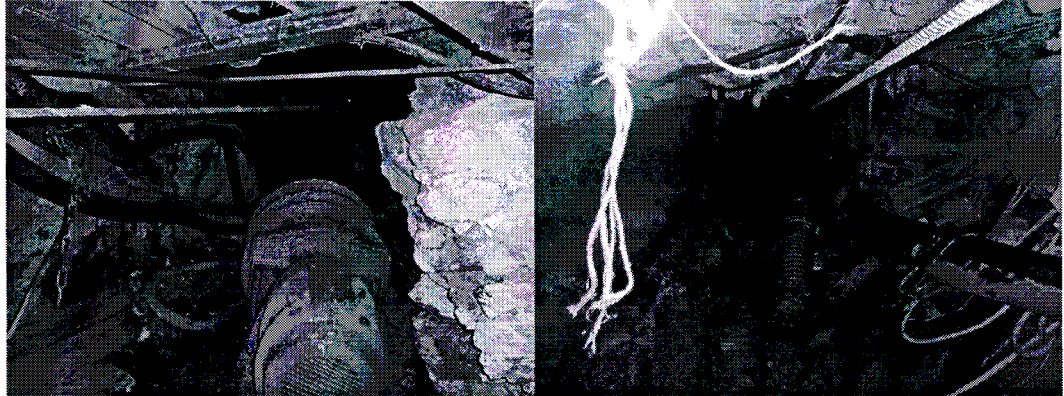
*Figure 3-25 Photos showing conditions of panel*

All the equipment was taken from East Driefontein 1 Shaft to Novatek Systems for repairs. After repairs and enhancements to the system had been carried out, the units were taken to East Driefontein 4 Shaft so that the underground tests could be continued. It was decided by CSIR: Mining Technology team members that the beams should be taken in disassembled. This would make the transportation process easier since taking the units in disassembled reduces the system weight by approximately 50% and provides more manoeuvrability for carrying the individual beams. Also, the separated beams could be carried by two individuals instead of four, thus reducing the manual effort involved in the lifting process.

The equipment was taken to East Driefontein 4 Shaft in the first week of December 2004. The transportation process proved to be a difficult one because of the conditions of the travelling ways to the test area. Low access ways and sloping ground, coupled with the weight of system components, posed the main sources of difficulty for moving equipment.

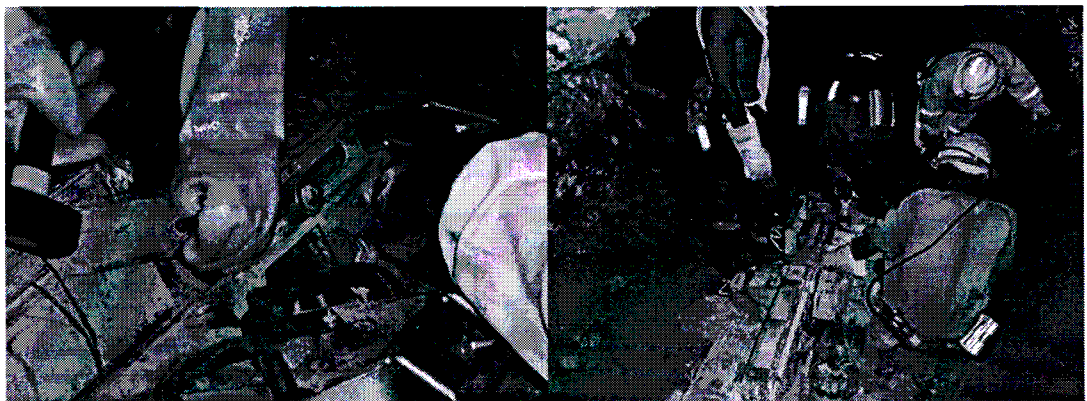
Figure 3-26 shows photographic examples of the travelling ways that imposed difficulties on the transportation process. The photo on the left shows an inclined travelling way. The photo on the right shows an access way that entailed a person crawling through a passage of approximately

30 m to get to the other end. Future development stages for the walking beam system should include methods for easing the transportation process through mine access ways such as these.



*Figure 3-26 Travelling ways*

The individual beams were assembled to each other before they were carried into the face area. The weight of the beams made their positioning and aligning difficult. The beams were placed on wood off-cuts in order to align them to each other, as alignment on uneven ground was difficult. However, once aligned, the actual securing of the beams proved to be an easy process as the crank-groove interface involves assembling a modular set of components. Figure 3-27 shows photos of the positioning and assembly of the beams.



*Figure 3-27 Assembly of the individual beams*

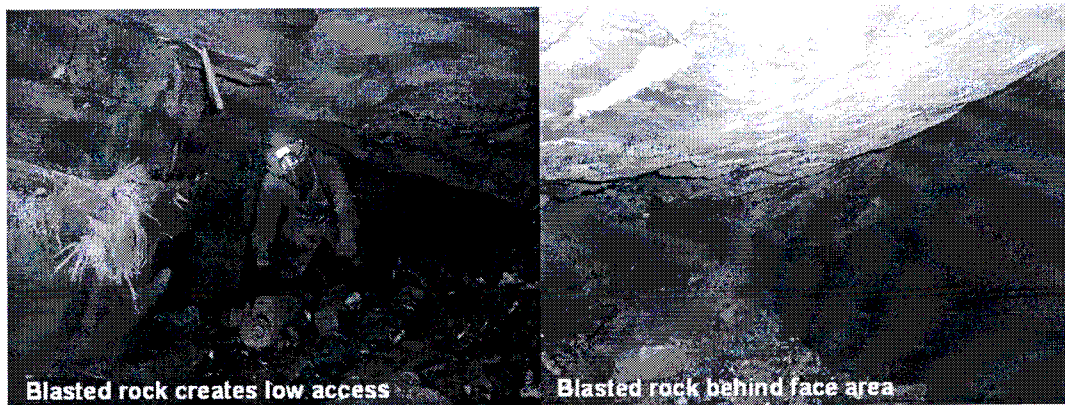
After the beams had been assembled, they were carried into the test panel. Appropriate positions were identified in the face area for installing the units. The units had to be positioned between

the pack supports so that they would not be in the path of the scraper during cleaning operations. Also, the area between the pack supports was out of the way of the winches that are generally used to transport material up and down the panel.

The installation process was hampered by the following factors:

- Blasted rock between the pack supports had to be cleaned before installation could take place;
- The accumulation of blasted rock with the progression of the mining cycle made it difficult to establish a time window for the positioning and installation of the units;
- The accumulation of blasted rock seriously hampered transportation of equipment to positions for installation in the face area because of the reduction in size of the access ways to the face area; and
- The availability of air and water was poor on many occasions as drilling operators were forced to monopolise their usage to meet their production targets.

As a result of these factors, the project encountered numerous time delays with the installation of the units. Figure 3-28 shows the narrow access ways created by blasted rock and the condition of the area behind the face after a blast.

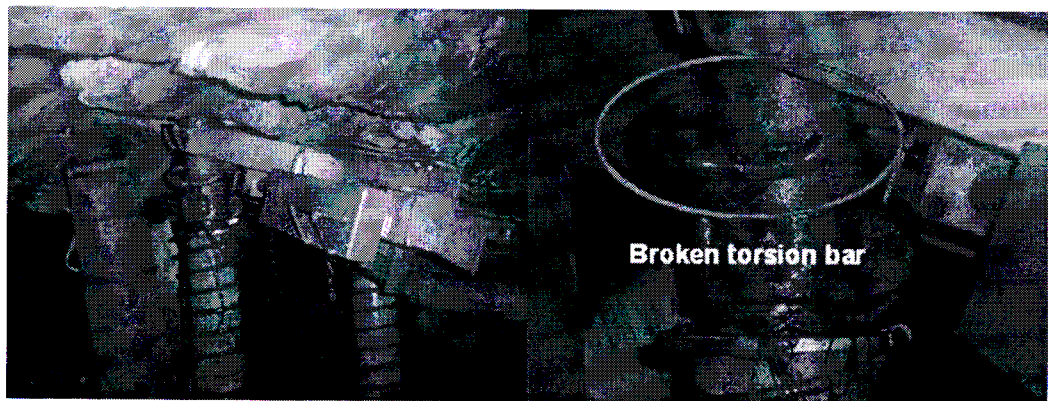


*Figure 3-28 Condition of test panel after blasting operations*

After much effort, one unit was installed between the first row of packs, approximately 3 m behind the face. The beam assembly was placed in position and the hydraulic props were attached to each beam. Thereafter, the system was lifted and held in position. While it was being

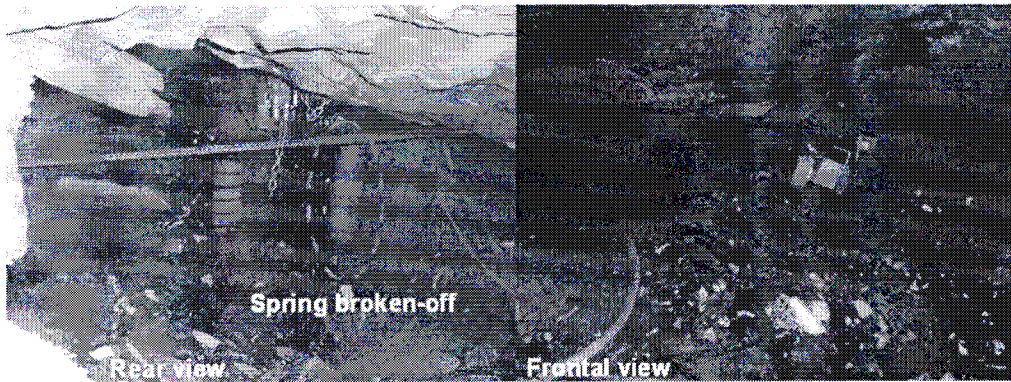
held in position, the hydraulic props were pressurised until the unit securely installed, with the beams adequately contacting the hangingwall.

During the pressurisation of the props and the subsequent setting of the beams against the hangingwall, one of the torsion bars (the bar that secures the prop to the beam) had broken as a result of the unevenness in the hangingwall. The severity of the hangingwall undulations had resulted in the beams twisting considerably during pressurisation against the hangingwall. This resulted in high stress concentrations being induced in the torsion bar and these stresses eventually exceeded the failure stress of the material. Figure 3-29 shows the beam contact and the broken torsion bar.



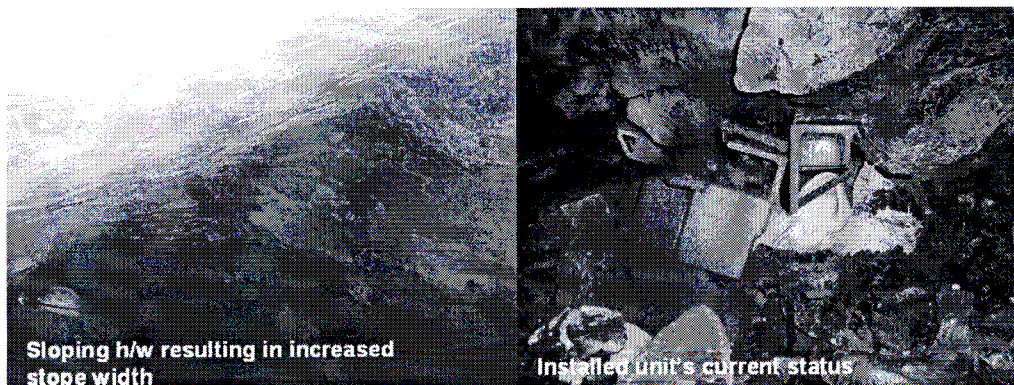
*Figure 3-29 Photos showing installed unit and broken torsion bar*

The unit was blasted on, and after the blast it was inspected for damage. The exposed hydraulic prop retraction spring had snapped, but the unit sustained no other serious damage. The unit was buried in blasted rock, and advancing it forward would have to entail cleaning the blasted rock around the unit after each blast, which is a timely and difficult process. Figure 3-30 shows the system after it had been blasted on. Note that the hydraulic prop spring is missing from the prop circled in a broken line; this is because it had snapped off its connection on impact.



*Figure 3-30 Rear and front view photos of unit after blast*

The remaining units could not be installed as a result of the problematic factors discussed above. In addition to the above problems, the stope width of the test panel had increased to approximately 2.1m, which is beyond the machine's stope width operating range. This was an unforeseen problem as the issue of reef width widening had been discussed with mine personnel before proceeding with the underground tests in this panel. Figure 3-31 shows the sloping hangingwall and the increased stope width of the test panel. The status of the installed unit after a succession of blasts is also shown in the figure.



*Figure 3-31 Photos showing increased stope width and buried unit*

At this point it was concluded that no further testing was required as a good overview of the machine's operational abilities and limitations was clearly derived from the underground tests that were performed thus far. Chapter 4 summarises these outcomes together with recommended initiatives for further development of the 'walking beam'. Further development work, however, could not continue due to budgetary constraints. The results of the surface and underground trials of the system were presented and accepted by SIMRAC.

## 4 CONCLUSIONS AND RECOMMENDATIONS

Following the SIMRAC (Safety in Mines Research Advisory Committee) project GAP 708, it was concluded that a system using remotely advanced headboards was the most viable concept. In this project the remotely advanced headboard system was developed from its conceptual stage to the prototype level, to enable testing of the system in an underground environment. In its prototype stage, the system was named the 'walking beam stope support system'.

The walking beam went through rigorous stages of design modifications and surface and underground trials that clearly highlighted the system's abilities and its operational shortcomings. From the surface trials, the functionality of the system was demonstrated in terms of the actual dynamic operation and its support capabilities. However, the underground trials showed up the following operational shortcomings of the system:

- The weight of the system's components has the following negative implications:
  - The transportation of the system is difficult, especially in areas of low access and sloping ground.
  - Assembly of the headboards and hydraulic props is a time-consuming process.
  - The initial installation of the system to set the beams against the hangingwall is very labour-intensive.
- The headboards are not adaptable to very uneven hangingwall conditions. If the headboards do not contact with the hangingwall sufficiently, the system becomes unstable.
- The hydraulic control system operates on air and water, which means that the system's application underground relies largely on the availability of air and water supply conditions underground.
- Hydraulic props do not have a wide range of stope width applications, rendering the walking beam inflexible with regard to accommodating widening stope widths.
- Integration of the system with mining processes proved to be difficult as the probability of finding a suitable time window to position the components, assemble the props and install the system was very low.
- Removing rock around the system after blasting operations is a time-consuming and labour-intensive process.

The need for designing and developing new underground stope support systems was emphasised through the initiative of this design project. The operational shortcomings mentioned, were foreseen to some extent during the design phase, however, these could only be clarified by means of testing the prototype system in the harsh conditions of the mining environment, and implementing the necessary or manageable changes as testing progressed.

The complex and sometimes chaotic nature of mining operations, activities, rock engineering and seismological criteria and anomalies, coupled with the extremely adverse conditions of South African gold mines posed major challenges for the execution of the underground trials of the prototype support system. This was proven by the number of design modifications and repairs that had to be performed on the system. The walking beam's mechanical design did provide for the critical support requirements of extensive areal coverage and, rockfall and rockburst protection. The learning's from the system component design and the shortcomings that were derived from underground testing provides fundamental value to the rock engineering and underground support design practitioner.

The development process for the prototype support system entailed a large amount of planning activities especially for the underground testing phases of the system. This was an essential part of the project due to the inherent difficulties and challenges associated with implementing new technologies in the South African gold mining sector. As highlighted in the introductory chapter, the implementation planning process for equipment is a critical step, as any deficiency in this area would be the primary cause for system failure or rejection by the mining industry. The sole purpose of designing and developing an alternative stope support system was to alleviate the problem of rock-related injuries and fatalities in the face area, however, this process had to begin by obtaining buy-in from the mining industry.

The process of creating an understanding for the purpose behind the design project and the requirements for testing the system underground followed a top-down approach. All mining personnel involved in the testing process from high-level management to the lower level personnel were clearly briefed and acknowledged of what was entailed in the development process. This led to the successful initiation of the underground tests at the respective mines.

From the experience of performing the underground testing of the walking beam units it was established that even though a lot of effort was put into the planning and execution of the underground trials, there were a number of unforeseen circumstances that presented themselves. A particular example of this was the increased stoping width scenario where the stope width

exceeded the operating stope width range of the machine. This happened on three occasions and led to large periodic delays on the project. The extent to which a stope widens is a difficult parameter to predict as a number of factors/variables are involved. For instance, the drilling and blasting method that was used could be one of the factors that could attribute to a widened stope, and encompassed in this method itself, one has to also analyse other parameters such as the blast hole configuration, drilling accuracy etc.

Another example of an unforeseen issue was the logistical issues associated with the shaft conveyance system that transports equipment to the respective underground levels. On occasion, the usage of the shaft system was not scheduled and priority usage had to be given for certain incidental circumstances. In essence, even if one builds in a sufficient amount of contingency provision into the project plan, which is a given for any design project, contingency planning for the implementation of new systems and technologies in the local mining sector, especially when the system is in its prototype design phase, is a complex issue.

To conclude, in its present state, the walking beam stope support system requires further development and hence is not suitable for usage on the mines. Integrating the support system with other essential in-stope equipment may justify the cost of development that is required to provide a more directly implementable system. The following measures are recommended for future development initiatives for the walking beam system:

- The system needs to be made lighter to ease the transportation, assembly and installation processes.
- The system requires incorporated mechanisms to enable it to adapt to changing hangingwall and higher stope width conditions.
- Methods for making the system more flexible with regard to air and water supply conditions should be investigated.
- Barricades or alternative shielding mechanisms must be investigated to ease the cleaning process, and also to provide additional shielding for components.
- Double-acting hydraulic prop support systems could be modified or designed to integrate with the walking beam headboard components.
- The system could be made more multi-functional and not solely a support unit, e.g. by integrating a drilling system into the walking beam system design.

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**APPENDIX A: WALKING BEAM STOPE SUPPORT SYSTEM:  
RISK ASSESSMENT**

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Walking Beam Support System:  
Risk Assessment**

**14 June 2004**

**REPORT Number JR-004-06-2004**

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## **A1 Introduction**

This report details the results of a risk assessment carried out on the walking beam support system currently being developed by CSIR: Mining Technology and Novatek Systems, an industrial manufacturing partner.

A previous assessment of the system was carried out in April 2002 (Ref: Snowden Mining Industry Consultants JR-011-04-2002). Modifications arising from the recommendations of that report and the development that has taken place in the intervening period made it necessary to revisit the assessment and update it. This assessment was carried out on 10 June 2004 and evaluated the risk associated with the transport, the in-stope assembly, and the installation of the system. The operating cycle, the interaction with normal mining activities and the disassembly process before relocation were also assessed. The focus of the assessment has been on identifying any areas of risk to the safety of personnel using the equipment in the underground situation.

The objective of this assessment is to identify and evaluate risk and to make recommendations for the mitigation of risk to personnel using the walking beam support system. The assessment also fulfils the requirements of the Mine Health and Safety Act, Section 21, in respect of the duty of any supplier of equipment to the mines to ensure that “the article is safe and without risk to health and safety when used properly”.

The risk assessment was carried out in a facilitated workshop with project team members in attendance. Actual units were inspected *in loco* at the test rig at the CSIR: Mining Technology facility.

## **A2 Summary of findings**

The walking beam support unit is a temporary face-support system for use in multiple units underground in conventional breast stoping applications. It is undergoing trials and is therefore in the development phase. The system is the subject of a SIMRAC research project which is being conducted by CSIR: Mining Technology in collaboration with the unit’s manufacturer (Novatek).

The main sources of risk associated with the handling, transport, assembly, installation and use of the unit can be categorised into four areas:

1. Manhandling of heavy props, headboard units and other components
2. Local falls of ground induced by installation or operation of the system
3. Accidental/wrong operation of air and hydraulic controls and hoses
4. Ergonomic impacts of additional equipment in the confined space of a working face.

High-risk issues are identified in the following situations:

- Removing the hoist prop causes injury by falls of ground (FOG).
- Pressurising the props of the unit causes injury by FOG in bad ground.
- Wrong Camlok lever orientation results in injury to the operator from FOG when the Camlok is released.
- FOG from the area vacated by the walking beam unit injures the operator.
- The system causes obstruction in the stope, resulting in operators having to move in the face or back areas.
- Depressurising of props leads to FOG and injury to the operator.
- Final removal of the walking beam units without proper equipment causes injury from toppling of units.

Risk exposure in the transport and use of the walking beam support system will be significant for four groups of personnel:

- Stores and material handlers
- Transport personnel
- Operators
- General stope personnel.

Adequate controls are embodied in two areas:

1. Fit-for-purpose design, with collaboration between scientific/engineering design and the testing establishment (CSIR) and the industrial manufacturer (Novatek)
2. Recommended transport, assembly, installation and operational procedures.

Shortcomings in controls exist in the areas of the eventual supplier and mine/user responsibility.

In the case of the supplier, these areas are:

- Handling of the heavy unit components
- Unlabelled control levers
- The hoist prop, which has no safety device to prevent dislodgement during the installation process.

Areas of concern that should be addressed by the user are:

- Provision of proper mine handling systems for offloading, storage and transport
- Training of operators
- Provision of appropriate personal protective equipment (PPE) to operators and handlers
- Integrating the support system with current support strategy
- Maintenance of standards for the making safe of the hangingwall prior to and during operation of the units.

### **A3 General recommendations**

The walking beam unit is in the development stage and CSIR: Mining Technology should consider further action to reduce risk in the following areas:

- Investigating suitable packaging for minimising damage or injury from poor handling
- Investigating the installation of proper handling points/lugs on the headboards
- Investigating preventative measures for inadvertent operation of control levers, such as the labelling of all controls
- Investigating the provision and use of a safety chain to secure the hoist prop.

The responsibility of the user should include ensuring the following actions are considered for implementation:

- The mine should use trained personnel and fit-for-purpose equipment for handling of walking beam components.
- Proper PPE (e.g. gloves) should be provided to all operators of the support unit.
- Operators should be trained according to the supplier's recommended procedures.
- The walking beam system should be integrated with current mine support strategy.
- Mine standards should be applied in making safe prior to installation.
- Proper supervision and safety inspections should be carried out.

## **A4 Risk assessment methodology**

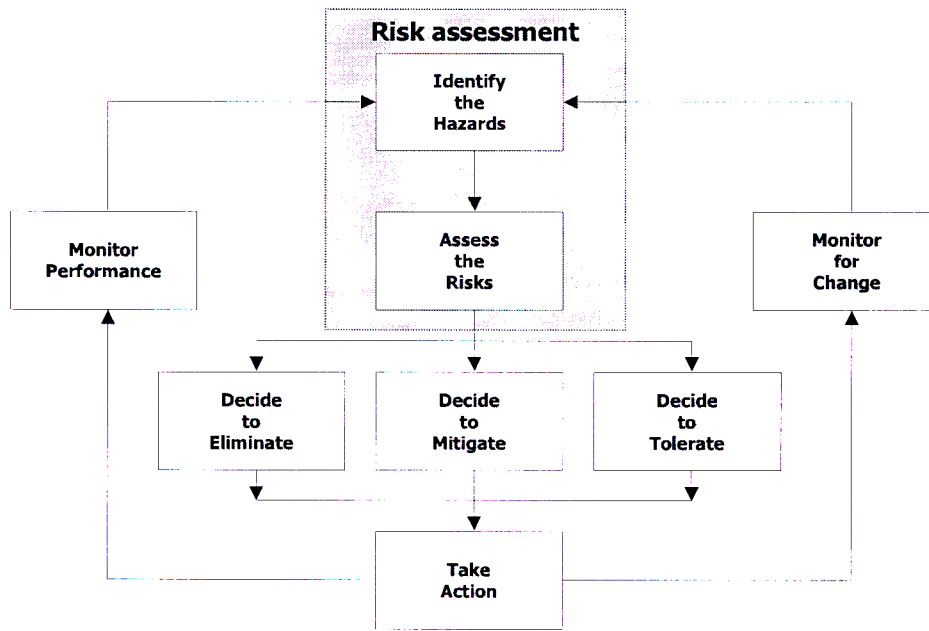
### **A4.1 Approach**

The risk analysis follows an accepted method of process risk analysis. All risk analyses follow a general scheme, which can be summarised by the following list of instructions:

- Describe the system under analysis (including equipment, personnel, procedures, work environment, management and supervisory systems, etc.).
- Identify loss scenarios (i.e. sequences of events leading up to potential or actual losses, such as incidents or accidents) in the form of hazards, potential productivity interruptions, asset damage events, environmental contamination issues, etc.
- Evaluate the risks of each loss scenario by determining the relative likelihood of each event, and the relative consequences of each event.
- Evaluate the currently planned controls, barriers and safeguards.
- Identify additional potential controls, barriers and safeguards.

### **A4.2 Risk management model**

The risk management model on which the assessment is based is shown in Figure A-1. The risk assessment is the first step, after which further action is taken, depending on whether the controls to be applied require elimination, mitigation or toleration of the risk.



*Figure A-1 Risk management model*

The risk assessment follows a standard method of a semi-quantitative measurement based on the concept of risk as a product of probability and consequence. The categories used in this assessment are tabulated below.

#### **A4.2.1 Probability categories**

Probability categories (A – E) were defined as shown in Table A-1.

Table A-1 Probability categories

Category	Probability
A	Virtual certainty/very common
B	Likely to happen
C	Could happen
D	Rare/unlikely to happen
E	Extremely unlikely/practically impossible

#### A4.2.2 Consequence categories

Table A-2 shows the definitions of the consequence categories (1 – 5).

Table A-2 Consequence categories

Category	Safety	Production
1	Fatality	Loss of face for one week or more
2	Serious injury	Loss of face for 4 – 5 days
3	Average lost time injury	Loss of face for 2 – 3 days
4	Minor injury	Loss of face for 1 day
5	Dressing station case	Temporary loss of face

#### A4.2.3 Risk categories (high/medium/low)

Risk categories were defined by combining the probability and consequence categories above according to a matrix of prioritised risk ranking (Figure A-2).

Figure A-2 Risk matrix

		Probability category					Legend
		A	B	C	D	E	
Consequence Category	1	1	2	4	7	11	1-6 High risk 7-10 High-medium risk 11-15 Medium risk 16-25 Low risk
	2	3	5	8	12	16	
	3	6	9	13	17	20	
	4	10	14	18	21	23	
	5	15	19	22	24	25	

A risk score of 1 denotes the highest (most significant) risk; a risk score of 25 denotes the lowest (least significant) risk. A high-risk rating includes the range 1 – 6; a high-medium risk the range 7 – 10; a medium risk 11 – 15 and a low risk 16 – 25.

Evaluation of risk has been carried out for both pre-control (primary risk) and post-control (residual risk) situations in order to highlight issues requiring further attention to mitigate the risk.

## A5 Participants

The risk assessment was carried out by the project team, facilitated by Snowden Mining Industry Consultants (Snowden). The members of the risk assessment team are recorded in Table A-3.

Table A-3 Risk assessment team

<b>Name</b>	<b>Designation</b>	<b>Company</b>
Mark Grave	Project Engineer	CSIR: Mining Technology
Dushendra Naidoo	Research Engineer/Project Leader	CSIR: Mining Technology
Tony Jager	RE Consultant	CSIR: Mining Technology
Alistair Forbes (Facilitator)	Consultant Mining Engineer – Risk Management	Snowden

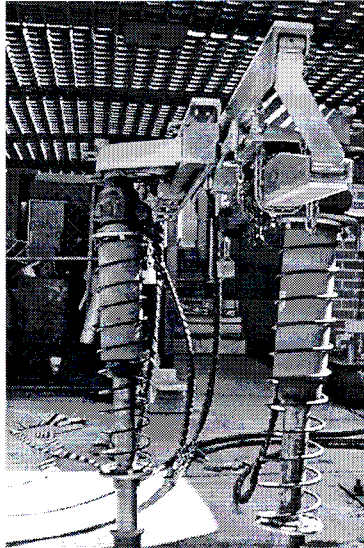
The assessment was carried out on 10 June 2004 at CSIR: Mining Technology, Johannesburg.

## A6 System description

The walking beam is a multi-component support unit, which is used as mobile support in the face area of working stopes. The support system involves the use of a number of units spread along the face of a conventional underground breast-stoping layout at intervals of approximately two metres.

The walking beam support unit (see Figure A-3) comprises two headboards that are mechanically linked, and each headboard is attached to a rapid-yielding hydraulic prop. The headboards contain rotational and linear actuating cylinders, which operate hydraulically. These cylinders enable the support system to be advanced forward via a rotational cranking motion, followed by a linear sliding advancement, with simultaneous extensions and retractions of the corresponding prop. The advance of the support unit can be varied to correspond to the advance achieved by a blast. Thus, the advance and setting of the units is carried out remotely with no physical effort required of production personnel.

The hydraulic control system is linked to a custom-designed pump, which requires an air pressure of about 4 bar or better for optimal operation.



*Figure A-3 General arrangement of the walking beam unit*

The system offers increased support in the area next to the face, and can be manoeuvred both forwards and backwards to accommodate the face scraper.

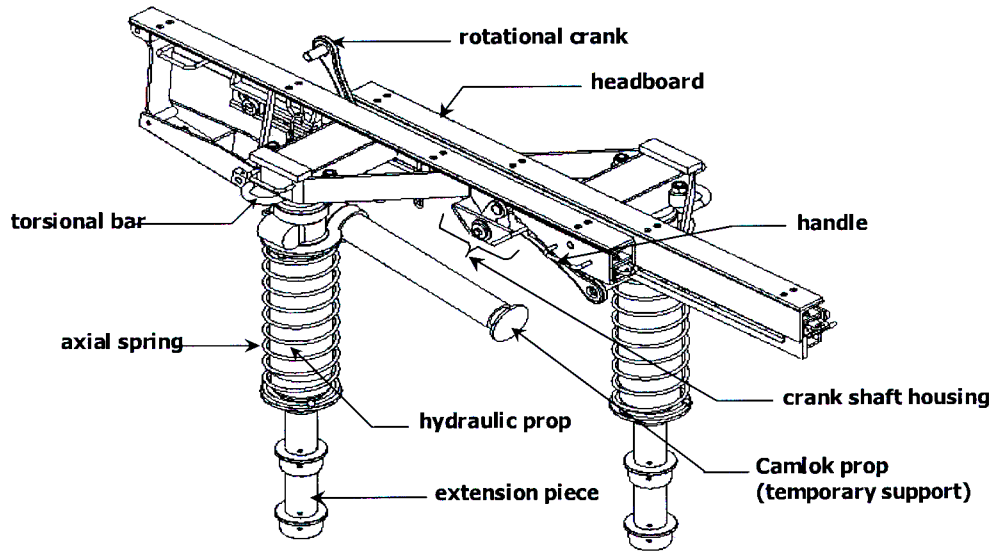
All support and auxiliary components for installation and operation of the units are shown in the accompanying component list (Table A-4). As the weights of some of the components are more than can be carried by one person, these are identified where relevant.

Table A-4 Component list for the walking beam system

Qty	Item	Approximate weight
1	Headboard assembly (including two headboards, hydraulic cylinders and associated fittings)	110 kg
2	Hydraulic props (main supports)	46 kg
1	Camlok prop	20 kg
1	Box of accessories	
1	Pump assembly	
1	Control box assembly	50 kg
8	Hoses	
1	Hoist/chain block in box	
2	Hydraulic installation props	
1	Box spares/tools	
1	Box prop extension pieces	25 kg

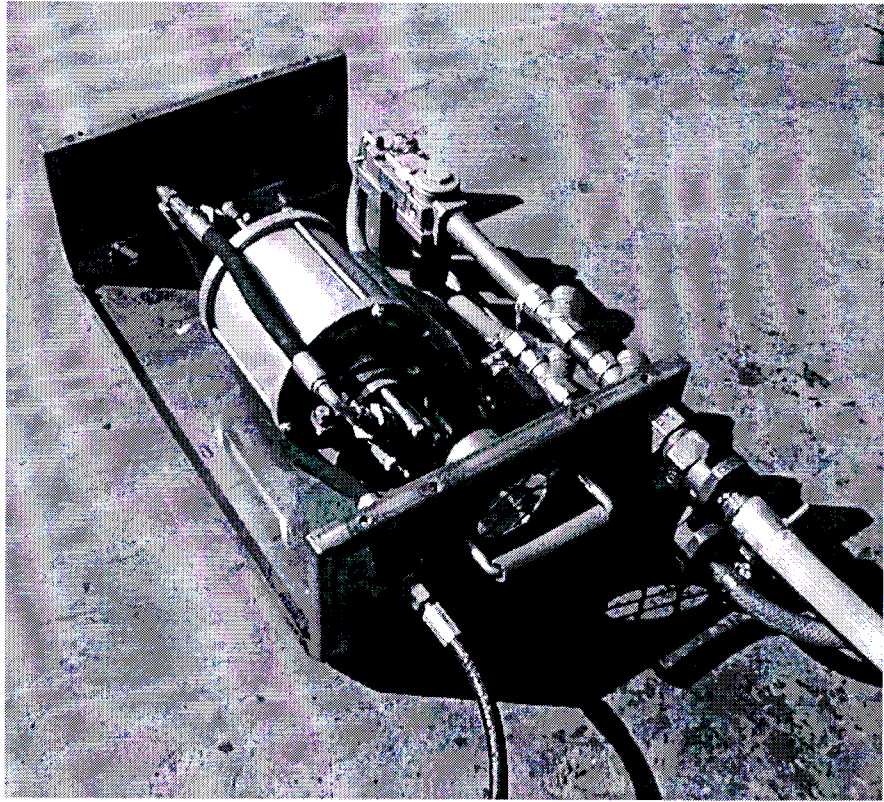
The diagrammatical representation in Figure A-4 shows the elements of the beam assembly. Each of two headboards is supported by a hydraulic prop. The headboards can be moved independently through the activation of the rotational crank by hydraulic cylinders.

A temporary support Camlok prop is attached to the front of the unit for further support between the unit and the rock face.

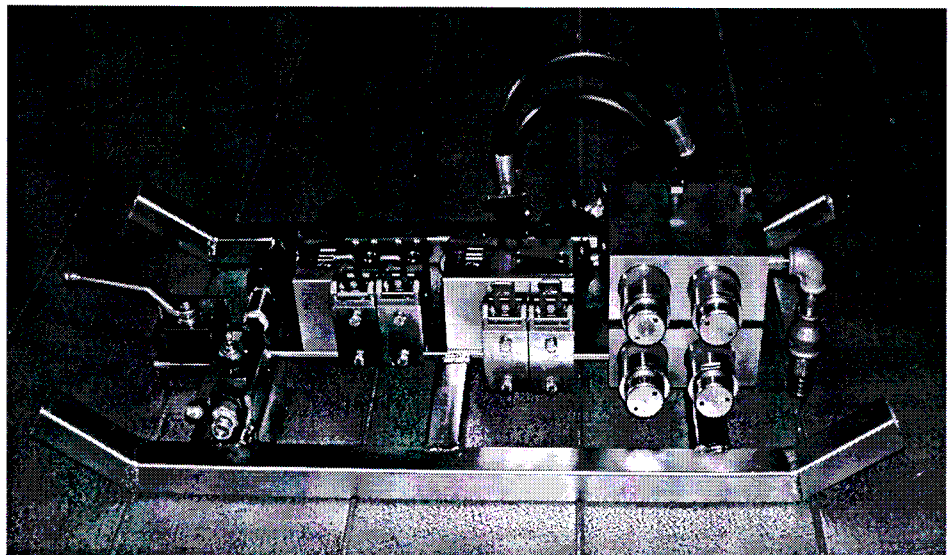


*Figure A-4 Main elements of the walking beam unit*

General arrangements of the pump and control box assemblies without the steel protective covers are shown in Figures A-5 and A-6.



*Figure A-5 General arrangement of pump assembly (without cover)*



*Figure A-6 General arrangement of control box assembly (cover removed for clarity)*

The risk assessment follows a process-based approach, for which the process flow diagram shown in Figure A-6 has been developed. For clarity, each activity in the process diagram is referenced with the section numbering used in the results table (Table A-5).

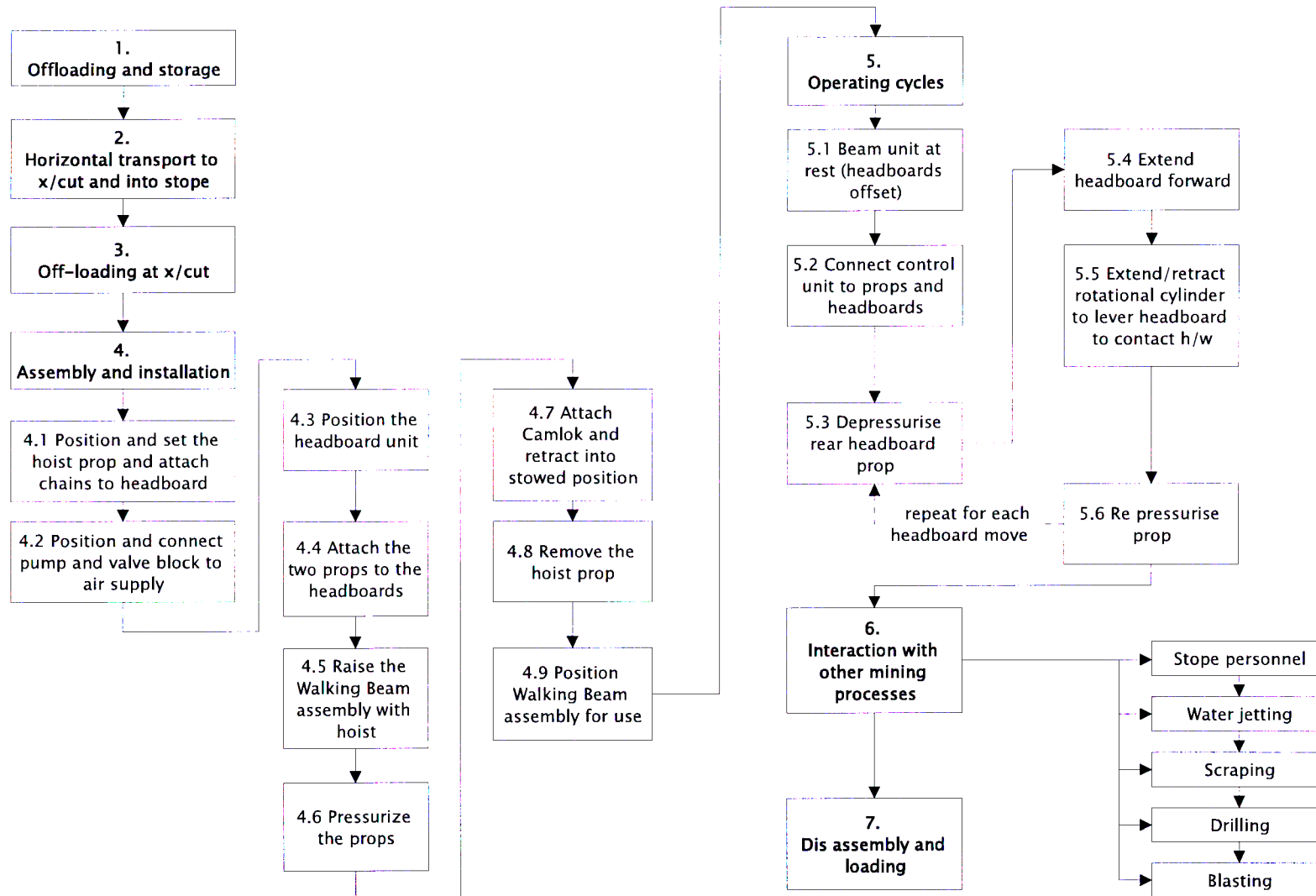


Figure A-7 Process flow diagram of the use of the walking beam support unit

## A7 Results

### A7.1 Table of results

The results of the assessment workshop are given in Table A-5 below. The headings to the table are defined as follows:

1. **Section** – Refers to the process activity as defined in the system description.
2. **Loss event/scenario** – Describes the event or scenario, which leads to loss, damage or injury to people.
3. **Effect** – Is the most likely result of the loss event described.
4. **Pre-control risk rating** – (based on the Probability (P) of an event occurring and the consequence (C) of the loss event as defined by the categories in Section A4.2). The Risk Rating (R) is a value obtained from the risk matrix as shown in Figure A-2, and indicates the level of primary risk.
5. **Current controls** – Defines which controls are in place to mitigate the risk.
6. **Post control risk rating** – As (4) above, but is an evaluation of the risk after the application of controls, showing the level of residual risk.
7. **Recommendations** – For further action to reduce the risk to acceptable levels based on the views of the risk assessment team.

Table A-5 Risk assessment results table – walking beam

	Section	Loss Event/Scenario	Effect	Pre-controls			Current controls	Post controls			Recommendations
				P	C	R		P	C	R	
1	Offloading and storage	a. Poor handling causes units to fall from transporter during unloading process.	Damage to units. Potential for injury to person standing next to the transporter.	C	2	8		C	2	8	Mine should use trained personnel and fit-for-purpose equipment for handling of walking beam components. Investigate suitable packaging to minimise damage from poor handling.
		b. Prop rolls off transporter onto nearby person.	Injury to person when struck by heavy prop.	D	2	12		D	2	12	Mine should use trained personnel and fit-for-purpose equipment for handling of walking beam components. Investigate suitable packaging to minimise damage from poor handling

			Pre-controls			Post controls					
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations	
2		c. Hand of person caught by scissor action of headboard unit.	Injury to hand	D	2	12	Grab handles are provided.	D	4	21	Use of PPE. Mine should use trained personnel and fit-for-purpose equipment for handling of walking beam components. Investigate suitable packaging to minimise damage from poor handling.
		d. Imbalance of headboard unit causes rotation and falls from grip onto person.	Injury to handler	C	3	13		C	3	13	Investigate proper handling points/lugs on headboards. Use PPE.
	<b>Transport to cross-cut (x/cut)</b>	a. Mishandling during loading onto scotchcar; items fall onto foot of person.	Injury to handler	D	3	17					

			Pre-controls			Post controls					
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations	
	b. Units fall from scotchcar during transit.	Damage to unit	D	4	21						
3	<b>Offloading at x/cut and manhandling into stope face</b>	a. Lifting of heavy unit by single person causes back injury.	Injury to handler	D	2	12	Handling procedures, including use of PPE Grab handles on beams	E	3	20	
		b. Mishandling during offloading by hand; unit falls onto leg/foot of person;	Injury to handler	D	3	17	Handling procedures, including use of PPE Grab handles on beams	E	3	20	
		c. Mishandling during transit in stope causes heavy units to fall against person.	Injury to person	C	3	13	Handling procedures, including use of PPE Grab handles on beams	D	4	21	
		d. People handling headboard slip and fall in confined space, resulting in injury from contact with the unit.	Injury to person	C	3	13	Handling procedures including use of PPE Grab handles on beams	D	4	21	

			Pre-controls			Post controls					
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations	
4		e. Weight of unit causes hand injuries during handling in stope.	Injury to person	C	4	18	Handling procedures, including use of PPE Grab handles on beams	C	5	22	
		f. Headboard unit falls back into gully while manhandling into face.	Damage to unit Potential injury to any person travelling in gully below unit	C	2	8	Handling procedures, including use of PPE, and people not to be down dip during manhandling Grab handles on beams	D	3	17	
	<b>Assembly and installation</b>										
	4.1 Position and set hoist prop and attach chains to headboard	a. Dirt gets into valves, resulting in malfunction.	Damage to unit Delays to production	C	4	18	Procedure for pre-use precautions	D	4	21	
		b. Air pressure supply too low, causing malfunction.	Delays to production	B	4	14	Pressure gauge on pump Operating procedure and minimum operating requirements for air pressure	B	5	19	

			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
4.2 Position and connect pump and valve block to air supply	a. Operator slips and falls while handling pump or valve units (50 kg).	Injury to operator	C	3	13	Handling procedures, including use of PPE Holding points on pump and valve units	C	4	18	
4.3 Position headboard unit on f/w	a. Beam slides down/dip on footwall, causing injury to person.	Injury to operator	C	3	13	Installation procedure	E	5	25	
	b. Beam positioned down/dip of installed unit, which prevents proper erection.	Delays to production Use of existing unit to anchor hoist chain	C	3	13	Installation procedure	D	4	21	
4.4 Attach props to headboard units	a. Mishandling of heavy prop unit, resulting in injury to operator.	Injury to operator	C	2	8	Installation procedure	D	3	17	
	b. Props topple and fall onto operator. (props weigh 46 kg)	Injury to operator	C	2	8	Installation procedure	D	3	17	

			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
4.5 Raise Walking Beam assembly with the hoist	c. Operator catches fingers in torsion bars while connecting props.	Hand injury	C	3	13	Installation procedures, including use of PPE	D	4	21	
	d. First prop topples down dip onto person holding second prop.	Injury to operator	C	3	13	Installation procedures, including use of PPE	D	3	17	
	e. Filler valve on prop aligned incorrectly (towards face).	Damage from blasting Malfunction of unit and potential loss of production	C	4	18	Installation procedures	D	5	24	
	a. Hoist prop becomes dislodged, resulting in assembly toppling down dip.	Potential for injury to people Damage to unit	C	2	8	Installation procedures	D	2	12	Investigate provision and use of safety chain to secure hoist prop. Operators should be trained in the correct procedure.
	b. Head slides towards hoist along footwall.	W/b unit slides out of position	B	5	19	Installation procedure				



			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
4.7 Attach Camlok and retract into stowed position	c. Prop is underpressurised.	Props fail to set properly; beam unit falls over	C	3	13	Installation procedure	D	4	21	Operators should be trained in the correct procedure.
	d. Fall of ground dislodged by beam during pressurisation.	Injury to operator	C	1	4	Installation procedure	D	2	12	Application of mine standards in making safe prior to installation.
	a. Operator catches fingers in pinch points while inserting pin.	Hand injury	C	4	18	Use of gloves stipulated in recommended operating procedures	C	5	22	
	b. Retaining pin/chain not secured properly, resulting in Camlok falling out onto operator.	Potential for injury to operator	D	3	17	Initial use will be by trained operator Installation procedure				

			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
4.8 Remove hoist prop	a. FOG (fall of ground) whilst removing prop, resulting in injury to operator.	Injury to operator	C	2	8	De-installation procedure for remote removal Mine standards for removal of temporary support Installed adjacent to permanent support	D	5	24	
	b. Hoist prop falls over and rolls down slope, injuring person.	Injury to stope personnel down dip who are struck by rolling prop	C	3	13	De-installation procedure, requiring securing of prop to prevent rolling down dip on depressurisation	D	4	21	
4.9 Position walking beam unit for use, using control panel	a. Both props released at same time, causing beam unit to topple over.	Damage to equipment as it falls over	B	4	14	Initial use will be by trained operator Operating procedure Use of trained operator	D	4	21	Label controls. Investigate addition of lockout mechanism to prevent this situation.
	b. Hoses incorrectly attached to props and headboards.	Controls operate wrong unit, resulting in the unit toppling over onto operator	C	2	8	Initial use will be by trained operator Operating procedure Hose attachments are marked	E	2	16	Investigate mechanical impedance to prevent misconnection of hoses.

			Pre-controls						Post controls		
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations	
	c. Hose staples not inserted (all hoses) on beam and control unit	Hoses come loose, injuring operator with high-pressure whiplash.	C	3	13	Initial use will be by trained operator Operating procedure (spares to be kept at all times) Use of trained operator	E	3	20		
	d. Inadvertent falling on or accidental operation of controls	Unit can topple over as it is depressurised. Potential for FOG and injury of people.	D	3	17	Initial use will be by trained operator A shroud is fitted over the controls	D	4	21	Investigate preventative measures for inadvertent operation of control levers.	
	e. Hose connecting control valve to the hydraulic pump not screwed tight/correctly	Hose comes loose, injuring people close by with whiplash effect.	C	3	13	Initial use will be by trained operator Operating procedure Use of trained operator	D	3	17		

			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
	f. Operator positioned too close to or down dip of walking beam unit during operation.	Potential for catching hand in between moving parts. Any malfunction may cause the unit to topple onto operator.	C	2	8	Initial use will be by trained operator Operating procedure Use of trained operator	D	3	17	
	g. Operator positioned too far to observe beam contact situation at h/w.	Headboard's contact with h/w badly sited, resulting in rotating and toppling of unit.	C	3	13	Initial use will be by trained operator Operating procedure Use of trained operator	D	3	17	
	h. Loose rocks in the prop springs prevent proper operation of the props.	Slow deployment of unit Production delays Damage to prop	B	5	19	Operating procedure, cleaning of props before use Use of trained operator	D	5	24	

			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
	i. Cavity in h/w prevents proper pad contact, causing rotation of beam unit and point loading.	Beam orientation becomes misaligned. Beam unit may become dislodged and topple over.	B	3	9	Use of extension pieces on pads Operator trained to identify hazard	C	4	18	
	j. Brow in h/w greater than 40 cm prevents advance of walking beam.	Unit unable to negotiate brow; requires resetting of unit.	C	4	18		C	4	18	Develop procedure/ methodology for negotiating brows
	k. Persons hold onto walking beam during operation, resulting in hand injuries.	Injury to hand/arm	C	3	13	Operating procedure, including hazard awareness training of stope personnel	D	4	21	
	l. Prop loses pressure, causing unit to topple when second prop is depressurised.	Loss of support, with potential for h/w collapse	C	2	8	Operating procedure (check for water/hydraulic leaks and pressure in each prop before depressurisation) Use of trained operator	E	2	16	

			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
	m. Camlok release lever is installed with incorrect orientation.	Potential injury to operator from FOG on release of Camlok	C	1	4	Initial use will be by trained operator Installation procedure Operator training	D	1	7	Carry out mine supervision and safety inspections.
	n. FOG in area vacated as walking Beam advances.	Potential for injury to operator	C	1	4	Installation procedure – permanent support is installed behind the unit prior to unit advancing.	D	1	7	Integrate walking beam with current mine support strategy. Ensure that permanent support is installed.
5	<b>Operating cycle</b>									
	5.1 At rest	a. One prop loses pressure.	C	5	12	Observation for leaks Training of operator to detect leaks	D	5	24	
		b. Both props lose pressure.	E	2	16	Observation for leaks Training of operator to detect leaks	E	2	16	
5.2 Connect control unit to props and headboards	a. Wrong hoses connected.	Unexpected lowering of units	C	5	12	Training of operator Operating procedure	D	5	24	



				Pre-controls			Post controls			
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
6.1 Personnel	a. Personnel contact with beam units during transit in stope, resulting in injury.	Injury from contact with steel frame	B	4	14	PPE issued by mine Operating procedure, including hazard awareness training of stope personnel	D	5	24	
6.2 Water jetting	a. Water spray on beam assembly induces damage to hydraulic cylinders.	Cylinders malfunction and require replacement. Beam loses support and falls over.	D	4	21	Fit-for-purpose design (corrosion-resistant materials) Examination and maintenance procedures	D	5	24	
	b. Water spray forces fines into moving parts on the beam assembly, resulting in premature failure of hydraulics.	Cylinders malfunction and require replacement.	C	4	18	Fit-for-purpose design (corrosion-resistant materials) Replace with conventional support. Maintenance procedures	D	5	24	

			Pre-controls			Post controls				
Section	Loss Event/Scenario	Effect	P	C	R	Current controls	P	C	R	Recommendations
6.3 Scraping	a. Scraper contacts beam assembly and dislodges support.	Loss of support resulting in FOG Production delay as unit falls into scraper path	C	3	13	Integration of mining cycle with walking beam Trained operator aligns unit appropriately	D	5	24	
6.4 Drilling	a. Beam assembly stands in rockdrill operator's way during face drilling, resulting in misaligned holes.	Reduced blasting efficiency Production delays	C	3	13	Integration of mining cycle with walking beam	D	5	24	
6.5 Blasting	a. Blast dislodges multiple beam assemblies, resulting in FOG in face area.	Loss of support and potential for FOG Production delays	C	2	8	Fit-for-purpose design – blast-resistant Bar down and install temporary support	D	4	21	
7	Disassembly of unit for relocation	a. FOG when props are depressurised for removal.	C	1	4	Operating procedure, withdrawal to supported area before removal Trained operators	D	1	7	
		b. Unit is removed without proper equipment.	C	1	4	De-installation procedure Trained operators	D	3	17	

## **A7.2 High primary risk issues**

### **A7.2.1 Offload/Storage/Transport**

High and high-medium primary risk issues, i.e. those that have the potential to result in serious injury to either operators or stope personnel prior to the application of controls, are highlighted by activity as follows:

- Offload/storage/transport
- Poor offloading causes heavy headboard assembly or props to fall onto person (high-medium)
- Headboard assembly or props fall on operator during manhandling in stope (high-medium).

### **A7.2.2 Assembly and installation**

- Removing hoist prop causes injury by fall of ground (FOG) (high).
- Pressurising props of unit causes injury by FOG in bad ground conditions (high).
- Wrong Camlok lever orientation results in injury to operator from FOG when Camlok is released (high).
- FOG from area vacated by walking beam unit injures operator (high).
- Dislodging of the hoist prop causes unit to topple onto operator or person in close proximity (high-medium).
- Mishandling of unit during installation, topples onto operator (high-medium).

### **A7.2.3 Operating cycle**

- Wrong or inadequate operation of control box levers leads to unit depressurising in both props, causing toppling over onto operator (high-medium).
- Operator positioned too close to or downdip of unit is injured by unit malfunction (high-medium).
- Uneven hangingwall causes unit to be dislodged and topple onto operator (high-medium).
- Prop loses pressure (malfunctions) and unit topples during operation (high-medium).

#### **A7.2.4 Interaction with other mining processes**

- Props are affected by prolonged water jetting and malfunction, causing depressurisation and toppling onto operator (high-medium).

#### **A7.2.4 Disassembly**

- Depressurising of props leads to FOG and injury to operator (high).
- Removal of walking beam units without proper equipment causes injury from toppling of units (high).

### **A7.3 Hazards**

The operating hazards associated with the transport, assembly and operation of the walking beam units include the mass of the unit and its major constituents, falls of ground, and the hazards associated with compressed air pressure and hydraulic pressure.

The main sources of risk can be categorised into four areas:

1. Manhandling of heavy props, headboard units and other components
2. Local FOG induced by installation or operation of the system
3. Wrong operation of air and hydraulic controls and hoses
4. Ergonomic impacts of additional equipment in the confined space of a working face.

#### **A7.3.1 Mass of the units**

The mass of the unit is more than one man is reasonably able to lift or carry. The headboard unit weighs 110 kg, the two props 46 kg each and various items of support spares are of similar weight. Physical manhandling of the units has to be undertaken to get the equipment to the working face. Since the mean spacing between the units in the installed position is approximately 2 m, a fully equipped face will require up to 15 units. These will have to be manhandled into the stope and installed by hand. This physically onerous task raises the potential for mishandling of heavy items and consequential injury to people in the stope. This may also be aggravated by dip, where use of the system in dips of up to 25° is envisaged, and equipment rolling or sliding down dip out of control can cause serious injury.

### **A7.3.2 Falls of ground**

The walking beam system is a temporary support system and is subject to the requirements and risk of the setting and removal of temporary support in an underground context. The hazard of falls of ground during the setting or removal phases exists. The proximity of operators during the setting and removal phases is necessary, and this results in a high exposure to the hazard by operators. There are several instances where exposure is highest:

- During the setting and removal of a hoist prop (used for hoisting up each walking beam assembly on installation);
- During the installation of the walking beam unit, as it is first pressurised against the hangingwall
- During the setting up of the unit, with the Camlok at the front of the unit
- During the operating cycle of the walking beam unit, as it moves forward and back, exposing previously supported hangingwall and leaving it without support
- During the operating cycle of the walking beam unit, as it depressurises as a result of malfunction of either of the props
- During the disassembly of the units for resiting.

Normal underground practices for the control of falls of ground will apply, but the correct and thorough application of these must be ensured with the walking beam system.

### **A7.3.3 Air and hydraulic pressure**

The hydraulic pump is powered by compressed air and makes use of the same mine supply as the rockdrills at the stope face. The hazards of handling air at these pressures will be the same as for rockdrill operators, especially with regard to loose hoses under pressure and the direct application of compressed air to limbs and body. These hazards have not been highlighted in this assessment because they are generic to compressed air rather than to the walking beam. The effects of hydraulic pressure are relevant to the operation of the support unit as the props are operated by a hydraulic pump attached by hoses to a control box. The lack of hydraulic pressure in the props at the appropriate moment in the assembly or operating cycle is likely to result in the greatest risk to the operators of the unit. This can be caused by either a malfunction in the hydraulic circuits, which results in pressure drop in the prop hydraulics and failure of the support unit, or inadvertent/poor operation of the controls to the props. The prototype controls are as yet unlabelled for correct identification.

#### **A7.3.4 Ergonomic impacts of additional equipment in the working face**

The impact of the support system on other mining activities within the stope may result in a change in conditions, for which the impacts are not immediately known. Displacement of working space for rockdrill operators, travelling of stope personnel along the faces and casual contact with the units are examples of areas where potential for injury can exist.

Once the system has been tested in a working condition, these features may be resolved through on-site practical solutions worked out by the stope staff.

#### **A7.4 Risk exposure**

Risk exposure in the transport and use of the walking beam support system will be significant for four groups of personnel:

1. Stores and material handlers
2. Transport personnel
3. Operators
4. General stope personnel.

The exposure to risk is greatest for the operators of the support system. These personnel will be exposed to the underground environment where the hazards of falls of ground and physical manhandling of the heavy gear in reef dip conditions constitute the principal risks.

Stores and material handlers and transport personnel will be exposed to the hazards of high mass and poor handleability of the various units that make up the walking beam assembly.

General stope personnel will be moving in close proximity to the units and will be exposed to hazards of reduced working space, displacement to other travelling ways, or casual contact during critical setting phases of the units.

#### **A7.5 Assessment of controls**

The supplier of the units has ensured that adequate controls are embodied in:

- Fit-for-purpose design, with collaboration between scientific/engineering design and the testing establishment (CSIR) and industrial manufacturer (Novatek)

- Recommended transport, assembly, installation and operational procedures.

The effect of these controls is to reduce the levels of primary risk to low or medium levels in the majority of cases. Risks that remain at a high or high-medium level (high residual risk) are described in the paragraphs that follow.

**Offloading/storage**

Poor handling at the offloading or storage stages when the units are delivered to the mine results in injury to people. The units are not pre-packed or protected by any packing and, because of the complex assembly of headboards, cylinders, brackets and fittings are awkward to handle, either mechanically or manually. Appropriate packaging of the units could add protection during the handling process and also facilitate the handling process and reduce the risk of error.

**Installation in stope**

The hoist prop becomes dislodged while the walking beam unit is being raised, resulting in the unit toppling down onto other stope personnel. There is no safety device for restraining a hoist prop should it become dislodged for some reason (e.g. bad ground conditions) during the installation of the walking beam unit. The provision and use of a safety restraining device for the hoist prop should resolve this issue.

Pressurisation of the support unit when it is being installed causes a fall of ground. Adequate training in the 'making safe' process prior to the installation procedure should alleviate this issue.

**Positioning of the walking beam unit**

Wrong operation of the controls leads to toppling of the support unit as it becomes depressurised. The control unit levers are not labelled. This situation should be remedied with the use of proper labelling.

Camlok release lever is installed with release lever in wrong direction. The operator should be trained in the correct use of the temporary support.

A fall of ground could occur in the area vacated by the walking beam as it is advanced towards the face. This is the area immediately behind the unit. The mine support strategy should be aware of this issue and integrate measures with its current system.

**Disassembly of the walking beam unit**

Falls of ground could occur when props are depressurised. The correct placement of the operator under supported ground during the disassembly process should address the issue. The remote positioning of the operator is facilitated by the long hose lengths of the control system.

**A8 Recommendations**

Recommendations for reducing risk are split into two areas of responsibility: those of CSIR: Mining Technology and those of the user or mine.

**A8.1 CSIR: Mining Technology**

The walking beam unit is in the development stage and to reduce risk CSIR: Mining Technology should:

- Investigate suitable packaging to minimise damage or injury from poor handling.
- Investigate installation of proper handling points/lugs on the headboards.
- Investigate preventative measures for wrong operation of control levers, such as the labelling of all controls.
- Investigate the provision and use of a safety chain/device to secure the hoist prop during the raising of the unit.

**A8.2 User/mine**

The responsibility of the user/mine should include:

- Use only trained personnel and fit-for-purpose equipment for the handling of walking beam components.
- Provide proper PPE (e.g. gloves) to all operators of the support unit.
- Ensure training of operators according to supplier's recommended procedures.
- Integrate the walking beam system with current mine support strategy.
- Ensure the application of mine standards in making safe prior to installation.
- Carry out proper supervision and safety inspections.

## **APPENDIX B: ROCK ENGINEERING SPECIFICATIONS FOR THE PROPOSED STOPE SUPPORT SYSTEM**

This is a complete reproduction of the report by Daehnke, A. et al., 2000, pp 22-30, with minor amendments and changes by the author.

**Full Reference: Daehnke, A., Roberts, M.K.C, Van Zyl, M., Acheampong, E., Harper, G., Ottermann, R., Burger, D., Von Wielligh, A., 2000, The design and development of an effective support system for tabular stopes in gold and platinum mines. SIMRAC Final Project Report (GAP 708), CSIR: Division of Mining Technology and University of Pretoria.**

## **B Rock engineering specifications for the proposed stope support system**

### **B1 Introduction**

The open stoping mining method used to mine the tabular orebodies in South Africa daily exposes new areas of discontinuous hangingwall, which then have to be made safe and supported. For this reason stope support systems are generally designed to support the fractured rock peripheral to the excavations. Cook (1960) states that stope support effectiveness depends on the extent to which the fractured rock, especially in the hangingwall, can be kept in place to reduce the damage and production delays resulting from rockfalls and rockbursts. The primary function, therefore, of a stope face support system is to maintain the integrity of the fractured hangingwall beam and thus prevent falls of loose rock for a specific period.

Support is achieved by applying sufficient force to the immediate hangingwall to generate frictional forces between individual segments of the hangingwall beam, by restricting bed separation and by directly supporting any loosened blocks. The nature and extent of the required forces to be generated by the support units and system depend largely on the local ground conditions, and on the type of support used. The low closure rate in shallow stoping conditions requires support that can exert a resisting force on the hangingwall and footwall at small deformations. In deep stoping conditions, the support is required to exert clamping forces on the immediate fractured hangingwall, before loosening of any inherently unstable blocks can take place. It is therefore generally required that, irrespective of mining depth, initial forces be generated rapidly by support systems. This opinion is shared by a number of researchers (Gay and Jager, 1987; Roberts *et al.*, 1993). Furthermore, the support system in the face area should be able to provide areal coverage to prevent falls of loose rock. This is particularly important where discontinuities delineating unstable key blocks are unfavourably orientated.

Support should also be installed as close to the face as is practically possible, to prevent deterioration of the most recently exposed hangingwall and fall of ground (FOG) accidents in this highly populated area of the stope.

## B2 Support requirements for quasi-static conditions (rockfall conditions)

Under quasi-static conditions, stope support is primarily required to prevent rockfalls in the working area between the stope face and the back area. The prevention of these rockfalls relies on the generation of support resistance to stabilise the hangingwall keyblocks.

The determination of the required support resistance criterion was undertaken by back-analyses of falls of ground (Roberts *et al.*, 1995). The analyses indicate that 50 % of falls are less than 0,45 m thick, 85 % are less than 1,0 m, and 95 % are less than 2,0 m in thickness. The support resistance is typically calculated on the basis of tributary area theory, and has become the design methodology most commonly used by South African gold and platinum mines. Here, a given weight of rock, determined by an area in the plane of the reef and the height of possible fall, is divided between a fixed number of support elements according to the attributable area. Thus, the support resistance (generally expressed in kN/m<sup>2</sup>) is directly proportional to the height of fall (b) and inversely proportional to the attributed area (A).

$$\frac{F}{A} = \rho gb, \quad [\text{B1}]$$

where: F = load carried by support unit  
 A = tributary area  
 ρ = rock density (taken typically as 2 700 kg/m<sup>3</sup>)  
 g = acceleration due to gravity  
 b = height of fall (or fallout thickness)

In a further work, Roberts *et al.* (1995) made use of a comprehensive accident database recording all rock-related fatalities on the gold mines since 1990. Cumulative percentage fallout heights were determined and a criterion set, such that the support system caters for 95 % of all rockfalls. The criterion has been updated (Daehnke *et al.*, 1998) to include more recent fatality data, and Table B-1 shows the fallout thickness for the 95 % frequency level, i.e. 95 % of all falls were the indicated thickness or less. The table also shows the associated support resistance criteria calculated using equation [B1].

Table B-1 Fallout thickness for the various reefs at 95 % frequency level

Reef type	Fallout thickness	Support resistance
Carbon Leader Reef	1,0 m	26,5 kN/m <sup>2</sup>
VCR	1,2 m	31,8 kN/m <sup>2</sup>
Vaal Reef	1,2 m	31,8 kN/m <sup>2</sup>
Basal Reef	1,8 m	47,7 kN/m <sup>2</sup>

On the basis of the requirements above for a particular reef type, the support system selected would then be evaluated to ensure suitability in terms of closure acting on the support, loading requirements and spacing, to effectively cater for rockfall conditions and, more specifically, this would then be done for individual ground control districts.

Considering the scope of this project, it would not have been practical to design a support system for each of the reefs referred to above. Therefore, a representative figure of 40 kN/m<sup>2</sup> was used as the support resistance criterion for this project, and strength requirements for elements and components of the support systems were based on this figure. Where higher support resistances were required locally, the density of support units was increased.

The procedure for the evaluation of a support system for rockfall conditions is presented in Figure B-1. The flow chart shows that the support system is evaluated once the support resistance criterion has been set. The evaluation is based on the tributary area theory, which is applied to determine whether the load-bearing requirements of the support system are met.

The support resistance requirements may be met by a particular support system at a given spacing (Steps 1 to 6 in Figure B-1), but this system may not be able to prevent failures associated with rock mass instabilities between adjacent support units. Daehnke *et al.* (2000) identified two failure mechanisms, namely instabilities due to beam buckling, and shear failure due to slip at the abutments. An evaluation of a possible occurrence of either of these types of failure mechanism is carried out in Steps 7 to 9 in Figure B-1.

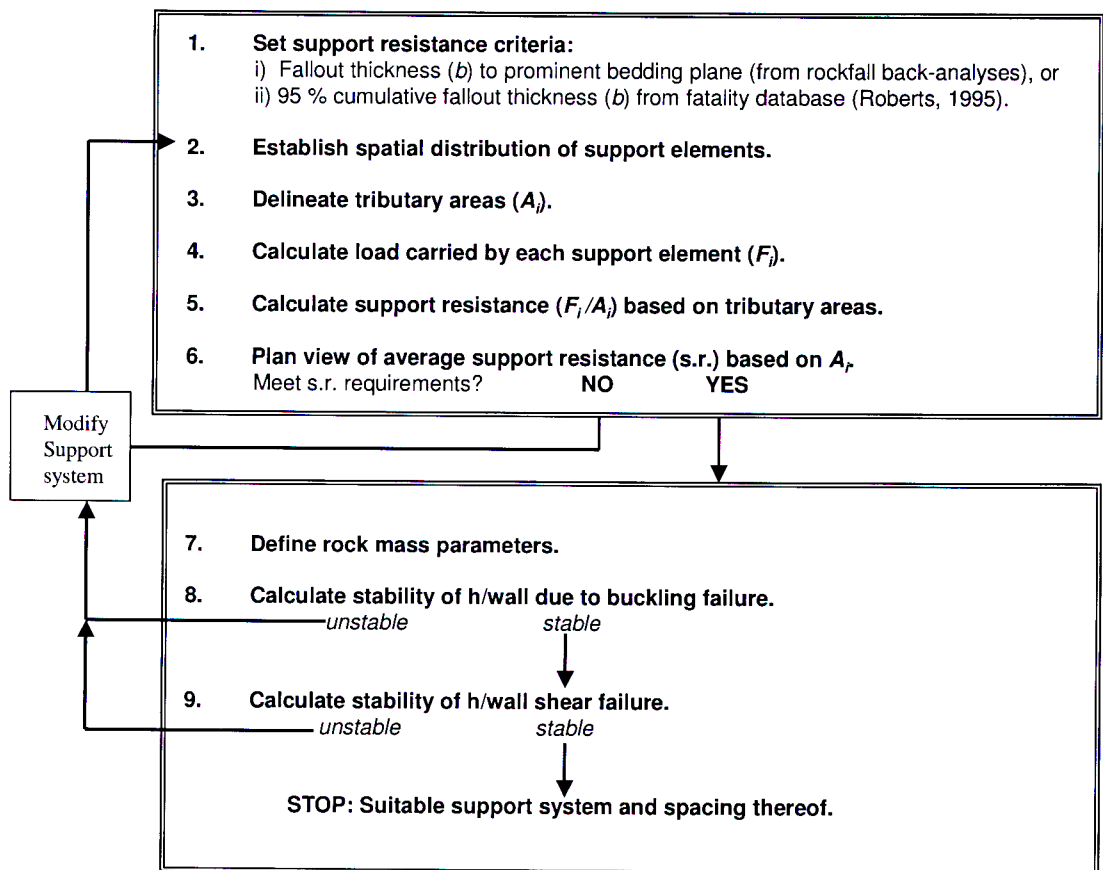


Figure B-1 Rockfall support design procedure (modified after Daehnke *et al.*, 2000)

### B3 Support requirements for dynamic conditions (rockburst conditions)

Roberts *et al.* (1993) identified the capacity of a support system to do work and absorb energy as the most appropriate criterion for effective support design under dynamic conditions. Until recently, a general energy-absorption criterion has been applied in stopes subject to seismicity and rockbursts, which requires that the support system should be capable of absorbing 60 kJ of energy per square metre of hangingwall. As a consequence, the support system should have a yielding capability. The basis of this criterion was a support resistance of 200 kN/m<sup>2</sup>, required to arrest the hangingwall displaced through 0,3 m at an initial velocity of 3,0 m/s during a rockburst and in the process absorb 60 kJ/m<sup>2</sup> (Roberts *et al.*, 1993). Therefore, the energy-absorption capacity of a stope support system needs to be evaluated against the energy absorption requirement of 60 kJ/m<sup>2</sup> (COMRO, 1988).

As in the case of the rockfall criterion, the energy absorption requirements for specific ground control districts have recently been modified. Using block ejection thickness for the different reefs, the minimum energy-absorption requirement that a support system should provide to

stabilise the stope hangingwall in 95 % of cases has been suggested by Roberts *et al.* (1995). The ejection velocity is assumed to be 3 m/s and it is further stipulated that, during the dynamic event, the hangingwall should be arrested through a displacement of no more than 0,2 m, typical of the yieldability of hydraulic props and yielding timber props.

Therefore,

$$E = \frac{1}{2}mv^2 + mgh \quad [B2]$$

where: E = energy that needs to be absorbed by a support unit  
 m = rock density x attributable area x ejection thickness  
 v = 3,0 m/s  
 h = 0,2 m  
 g = 9,8 m/s<sup>2</sup>

The energy-absorption criterion is detailed for specific reefs in Table B-2.

Table B-2 Ejection thickness for the various reefs at 95 % frequency level and the associated energy-absorption criteria

Reef type	Ejection thickness	Energy absorption
Carbon Leader Reef	2,2 m	38,4 kJ/m <sup>2</sup>
VCR	1,8 m	31,4 kJ/m <sup>2</sup>
Vaal Reef	1,2 m	20,9 kJ/m <sup>2</sup>
Basal Reef	2,6 m	45,4 kJ/m <sup>2</sup>

For the purposes of this project, it is impractical to design a support system for each of the reef types above. For this reason, a general energy-absorption criterion of 40 kJ/m<sup>2</sup> has been considered appropriate for this project.

To evaluate a support system against the criterion, Daehnke *et al.* (1998) proposed the following methodology.

The hangingwall is assumed to have an initial velocity of 3 m/s. However, the displacement is determined from the energy-absorption capabilities of the support unit (Figure B-2 and equation [B3]). Thus, the total hangingwall displacement, up to the point in time when dynamic movement ceases, is greater for a support system providing less support resistance, in comparison with a high load support system that will arrest the hangingwall within a shorter distance. In the first case, the hangingwall deceleration is reduced, but the potential energy component that needs to be absorbed by the support system is increased. In the second case, the hangingwall deceleration is higher, with the potential energy component being decreased.

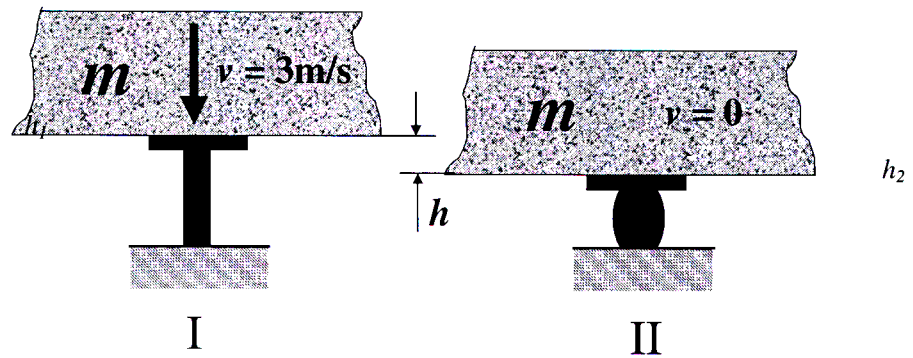


Figure B-2 Conceptual model of dynamic hangingwall displacement and associated energy-absorption requirements of the support system (after Daehnke et al., 1998)

$$\int_{h_1}^{h_2} F(x) dx = \frac{1}{2}mv^2 + mg(h_2 - h_1) , \text{ where } h = h_2 - h_1 . \quad [\text{B3}]$$

To illustrate the loading requirements of a support unit during a dynamic event, assume that a support unit with force-deformation characteristics as shown in Figure B-2 is installed underground. Due to pre-stressing and stope convergence, the unit is quasi-statically deformed up to point  $h_1$ , after which a rockburst occurs and the unit is rapidly compressed up to point  $h_2$ . The total energy required to arrest the hangingwall is represented by the hashed area under the graph in Figure B-3. For the support system to meet the rockburst loading requirements, the following criteria apply:

- The total dynamic hangingwall displacement ( $h$ ) should not exceed 200 mm. It is postulated that if the hangingwall is dynamically displaced in excess of 200 mm, the differential downward displacement between the face and support units, as well as between different support unit types of varying stiffness, could compromise the

post-rockburst hangingwall integrity, leading to an irregularly deformed hangingwall with reduced frictional constraints and lowered structural strength.

- It is assumed that the support system should be able to accommodate 150 mm of stope closure during normal operations in the stope face area. This is the upper limit of closure that could be expected some 5 m behind the face at a closure rate of 30 mm/m of face advance. Should a rockburst occur, the support system would be required to yield a further 200 mm (see Figure B-4). The total yield requirement of the support system would then be 350 mm. A maximum value of  $h_2 = 0,35$  m is considered realistic and suitable for current support system design.
- The stope width minus  $h_2$  should exceed 0,55 m to ensure sufficient post-rockburst stope width to allow movement of, and prevent serious injury to, mine personnel.
- To ensure post-rockburst stability, the load carried by the support units after the rockburst, i.e.  $F(h_2)$ , should exceed the deadweight of the potentially loose rock.

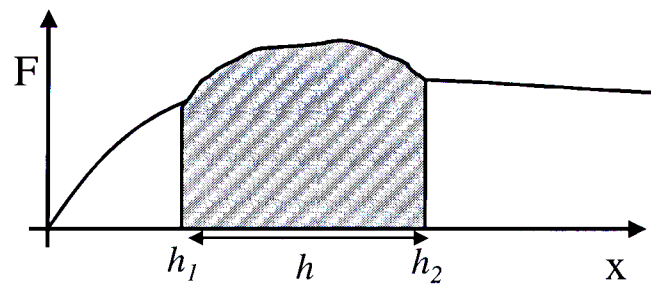
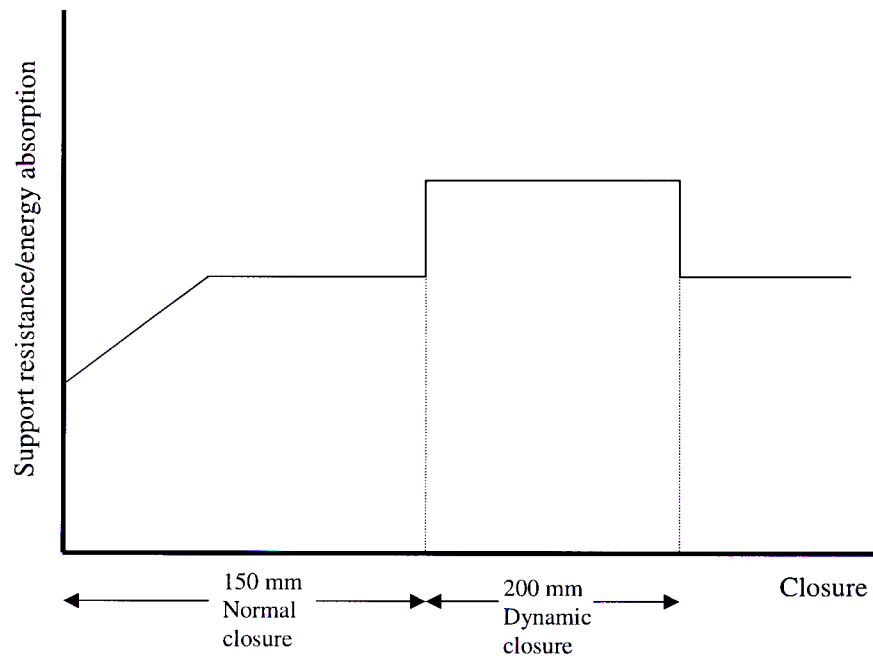


Figure B-3 Quasi-static and dynamic force-deformation behaviour of a support unit prior and during a rockburst (after Daehnke et al., 1998)



*Figure B-4 Support resistance/energy absorption versus closure*

The rockburst support design procedure follows a scheme similar to the rockfall support design methodology, and the assumptions made for rockfalls also hold for the rockburst case. The main difference between the two design procedures is that, in the rockburst case, the effective hangingwall weight (Daehnke *et al.*, 1998) is used, as opposed to the rockfall case.

Figure B-5 gives the salient features of the rockburst support design procedure.

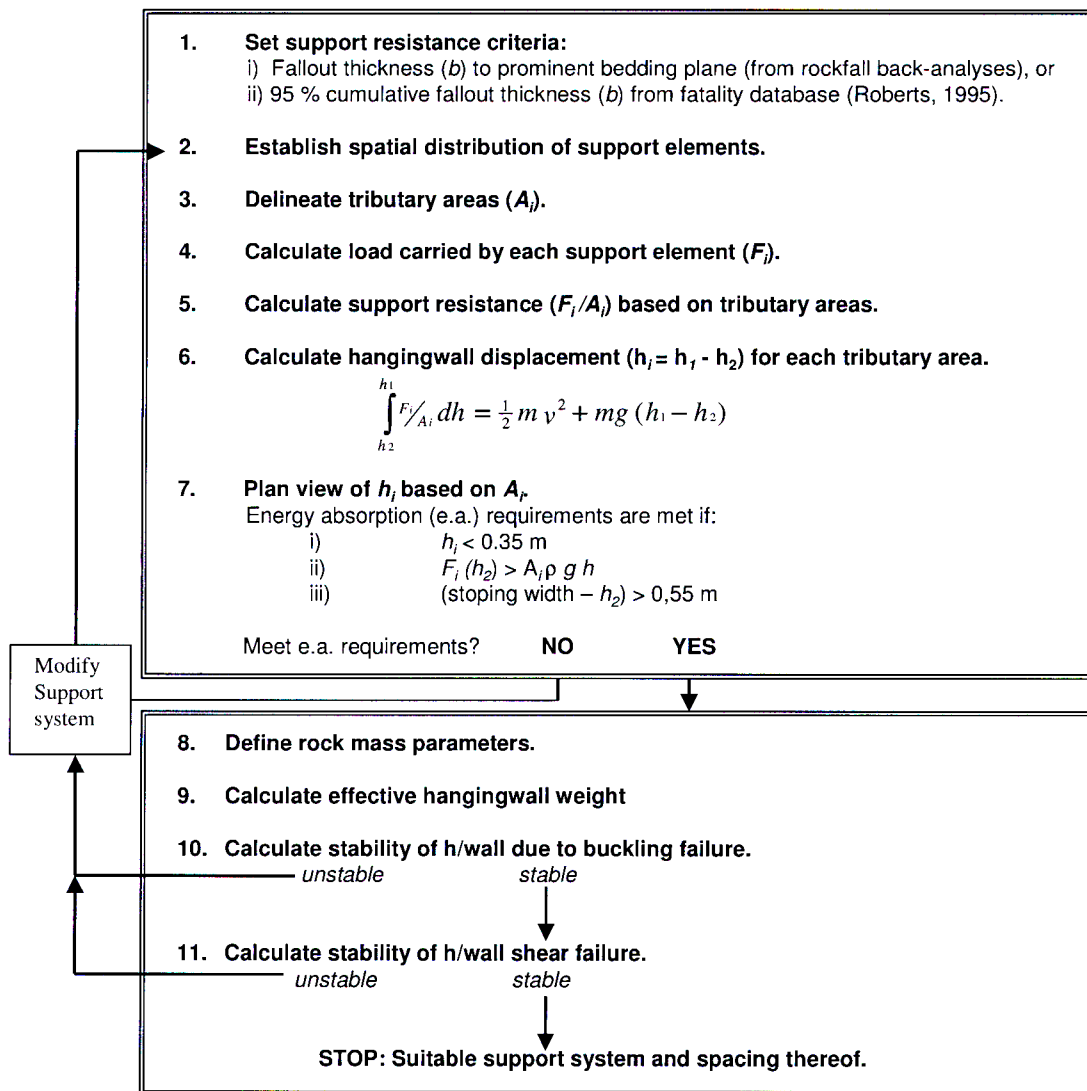


Figure B-5 Flow chart showing features of the rockburst support design methodology (modified after Daehnke et al., 1998)

#### B4 Areal coverage

Support units are designed to withstand such forces and deformations that the combined capacity of a certain number of support units is sufficient to stabilise a particular area of stope hangingwall. Analysis of data on accidents that took place in the gold mining industry between 1990 and 1996 indicates, however, that, for the support units associated with rockburst fatalities, a total of 207 failed and 841 were ineffective (i.e. could not stabilise the hangingwall). Similarly, a total of 54 support units failed and 247 units were ineffective during rockfalls. This

means that the number of support units that failed as a result of excess loading is relatively small. Individual support units can, therefore, be assumed to be sufficiently strong and/or yieldable to accommodate the majority of rockfalls and rockbursts. In severe rockburst conditions, however, it has been found that a high percentage of support units had failed (Roberts, 1999).

It can be concluded that the main cause of support failure may not be attributable to the inadequate structural strength of support units. The most obvious causes for failure of the support system appear to be lack of areal coverage of unstable fragments of rock, and support units that are too widely spaced.

The current support design methodology does not take into account areal coverage requirements. The ideal face area support system is one that will provide or incorporate adequate areal coverage. The face area support systems currently in use employ mostly headboards and similar forms of load spreaders as a form of areal coverage. They are often applied in such a way that the direction of these devices is perpendicular to the typical mining-induced face-parallel fractures and, therefore, the efficiency of such spreaders seems to be maximised. These headboards, however, do not span the entire distance between support units and therefore their effectiveness decreases with increasing hangingwall friability. In many cases, the load-carrying capacity and performance under dynamic loading of these headboards can also be questioned.

Any support system should, therefore, incorporate mechanisms that will provide high levels of areal coverage in highly discontinuous hangingwall stopes. Lesser coverage is acceptable in stopes with relatively widely spaced discontinuities.

## **B5 Spacing between support units**

In designing stope support layouts, the substantial benefit of reducing the distance between the stope face and the closest line of support is generally recognised. In a working stope, it is necessary to provide space to accommodate both the blasted rock, and the rock handling and cleaning arrangements, particularly in the cases where the face is cleaned by scraping methods. When the face is prepared for drilling, manoeuvrability is essential for the drilling equipment. Ortlepp and Stacey (1995) report that, because of these unavoidable needs, the unsupported span is usually greater than that dictated by safety requirements. Currently, temporary support is used to reduce this unsupported span. Too frequently, however, compromises are made, usually

in the interests of maintaining the production rate and reducing costs, sometimes at the risk of not ensuring safety.

The influence of support units on adjacent support units and the extent of the zones of influence of individual support units have recently been researched (Daehnke *et al.*, 1999). This work has indicated that, in general, a dip span of not greater than 1,5 times the strike span is appropriate.

The objective of this project, therefore, is to provide improved areal coverage in the immediate vicinity of the stope face during the full mining cycle.

### **B6 Support-hangingwall contact stresses**

The stress at the contact between the support and the hangingwall or footwall should not exceed 30 MPa. Above this stress, hangingwall and footwall punching can become a problem.

### **B7 Summary**

A summary of the rock engineering specifications required of improved support systems is presented in Table B-3 below.

Table B-3 Summary of rock engineering requirements of a support system

Parameter	Requirement
Support resistance (rockfall situation)	40 kN/m <sup>2</sup>
Energy absorption (rockburst conditions)	40 kJ/ m <sup>2</sup>
Velocity of dynamic closure	3 m/s
Yieldability	Total = 350 mm (normal stope closure = 150 mm and dynamic closure = 200 mm)
Post-rockburst stope width	≥ 0,55 m
Unsupported spans	The unsupported span in the dip direction should not exceed 1,5 times the unsupported span in the strike direction.
Support contact stress	≤ 30 MPa

## B8 References

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**APPENDIX C: OPERATING SPECIFICATIONS FOR THE  
PROPOSED STOPE SUPPORT SYSTEM**

This is a complete reproduction of the report by Daehnke, A. et al., 2000, pp 31-35, with minor amendments and changes by the author.

**Full Reference: Daehnke, A., Roberts, M.K.C, Van Zyl, M., Acheampong, E., Harper, G., Ottermann, R., Burger, D., Von Wielligh, A., 2000, The design and development of an effective support system for tabular stopes in gold and platinum mines. SIMRAC Final Project Report (GAP 708), CSIR: Division of Mining Technology and University of Pretoria.**

## **C Operating specifications for the proposed stope support system**

### **C1 Introduction**

The inability of traditional stope support technology to minimise the damage caused by rockfalls and rockbursts to within acceptable limits has led to the call for new stope support innovations and improvements to existing systems over the years. This continuous search for the ultimate support technology that addresses support limitations has led to the over 130 different types of stope support systems that are in use at present. In the design of these products, the emphasis is placed on the technical and engineering capabilities with, in some cases, very little attention being paid to the operational requirements. Without giving sufficient attention to the added operational aspects of support practices, it will be impossible to ensure the successful implementation of any rock engineering support strategy in the stope face area.

In the discussion that follows, the operating specifications that need to be satisfied by any support system are presented. The specifications are based primarily on information gained from underground visits, discussions with production and rock mechanics personnel, and knowledge gained from the SIMRAC projects GAP 606 and GAP 613.

### **C2 Safety of workers**

The ability of a support system to ensure the safety of personnel at all times is perhaps the most important requirement of any support system. With the current stope support technology, the cleaning shift enters the panel after the blast and is more vulnerable to rockfall and rockburst incidences, due to increased unsupported spans. (Almost 50 % of all stope fatalities involve people whose activity at the time of the incident is related to cleaning or making safe (Jager, 2000.)) The situation is exacerbated when removable face area support units are used. The removable face support system protects personnel during drilling and charging-up activities; this support system is, however, not present during barring, making safe and cleaning of the stope face. These comments apply equally to elongate face support systems as some of these support units can be dislodged by the blast, and the unsupported span at the face is increased by the face advance of the blast.

Another important requirement of the stope support system is its ability to minimise damage caused by face bursts. The inability to address face bursts is a serious shortcoming of current

face area support technology. To this end, the system should, where necessary, have a component that protects workers from the lateral ejection of rock from the face.

### **C3 Integration into the production cycle**

A support system is just one of a number of integral aspects associated with stoping. The others include:

- Drilling and blasting
- Cleaning
- Movement of personnel in the stope area.

A truly effective support system should not compromise, and should not be compromised by, any of the areas of activity mentioned above.

The integration of the support system with mining operations is essential close to the stope face. Currently, however, the conflicting demands of the support requirements in the stope face and the cleaning of the blasted ore have led to compromises in both areas. Ideally, the support system should either remain in place during the blast, so that both the cleaning and support requirements can be met, or it should be such that in the post-blast situation, it will ensure immediate support of the hangingwall.

The support system should not interfere with the drilling operation or compromise the blasting pattern. Currently, occasions do arise when either the support is moved to a position where it does not interfere with the drilling process, or, alternatively, the drilling process is compromised by changing the drill angle or shifting the collaring position. In the first case, the action compromises the support system, while in the latter it compromises the blasting pattern, which may lead to further damage to the hangingwall.

Of great importance is the requirement that the support system should not be compromised by the cleaning process. With current support technologies, damage to support units by the scraper scoop or rope is common, and, in most cases, results in the downgrading of the support quality and slows down the cleaning process.

During stoping operations, the movement of personnel in the immediate stope face area is inevitable, especially during activities such as making safe, installation of support, drilling, charging and cleaning. The system should therefore not impede the movement of workers in the face area, otherwise time spent on stoping activities will increase, resulting in reduced efficiency.

Blasting is the primary rockbreaking method used to mine the reef in South African gold and platinum mine stopes. The blast ejects rock from the face at high velocity. As a consequence, face area support units can be dislocated, skewed or blasted out by the fly-rock. Hence, where the support system is of the non-removable type, it should be able to withstand the effects of the blast, without compromising its ability to provide adequate support.

#### **C4 Handling**

The current stope support technology relies heavily on human effort to transport and install it. This has been shown by Daehnke *et al.* (2000) to be a major operational constraint, considering the limited time available for a production shift. Blasts have been lost, and continue to be lost, as a result of difficulties in handling support accessories in stopes. Alternatively, blasts occur with the support not to standard, and safety is compromised in this way. In earlier work done by the Chamber of Mines Research Organisation (COMRO) (Van Rensburg *et al.*, 1991) to determine the physiological demands associated with handling of hydraulic props underground, it was shown that a reduced hydraulic prop mass results in significant reductions in physical effort and installation time. To assess the physical effort associated with the installation of each prop, the time taken, the number of heartbeats and the oxygen consumption of the workers were measured. It was found that the trend for each set of measurements was remarkably similar, adding up to a measure of 'physical effort'. A prop mass of 32 kg was suggested as the maximum that could be effectively handled without over-extending an individual in the unfavourable physical conditions in stopes. Other considerations, such as size, provision of accessories to enhance handling, etc., could, however, increase this maximum mass.

In this project, therefore, a mass of 30 kg is the recommended maximum for any support system component that has to be regularly lifted or dragged by one person. Also, the design of all units and components should take into account ergonomics so that the system will be readily accepted by the operators and easily implemented. For example, heavy components should be provided with handles and designed for ease of handling, with particular attention to prevention of accidental injury to users.

The support system should be protected against premature functional deterioration from impact during transportation and handling, from blast damage, corrosion and the ingress of abrasive quartzite dust and grit.

## **C5 Support installation and removal**

Daehnke *et al.* (2000) report that poor support installation practices at the stope face area should be addressed if the current high fatality rates are to be reduced. Amongst the reasons enumerated as the fundamental causes of poor support practices are: unavailability of labour, shift time constraints, production pressures, and a lack of worker knowledge and motivation. This list emphasises the importance of the role of human behaviour in the successful implementation of support strategies aimed at reducing the high accident statistics. The design of a new support system should, therefore, strive to eliminate these factors as far as possible by making the system simple and easy to use.

A related issue is the concern expressed by several mines about the increased reluctance of workers to use supports of a type that have to be installed and removed later, only to be installed again after the blast. This applies to the mechanical props that are widely used as temporary face supports, as well as to rapid-yielding hydraulic props. There are two main concerns: the first is the safety of the process, and the second is the perceived additional work involved. (Note: in the period between 1990 and 1997, 120 workers were killed on gold mines while installing or removing supports of all kinds (Glisson, 1998)). The most cost-effective and human-effort-efficient support systems are, however, of the reusable type. Any system based on this concept has to ensure that the people lowering and advancing the units will be under a supported hangingwall (i.e. removing the unit remotely).

Any new stope face area support system should, therefore, reduce the extent of human involvement considerably. This will mean the incorporation of some degree of mechanisation or automation into the installation process. Apart from the benefit of properly installed support, such a system will also help to minimise the injuries and accidents associated with the installation of some of the current support types. Alternatively, the support system could be such that less human effort and time are required for installation. The ease with which the system is installed is, therefore, of importance.

## **C6 Resistance to blast damage**

Bakker (1995) states that the nature of mine operations and standards means that workers are exposed to an inadequately supported hangingwall during a considerable period of a shift. This contributes significantly to the high accident fatality rate in the gold mines. This statement was echoed by Daehnke *et al.* (2000) who asserted that the creation of potentially unstable, unsupported spans near the stope face is a consequence of current stope support technology, and that the shift that normally enters the panel after the blast is most vulnerable to instability of the hangingwall owing to the large unsupported spans. To reduce this span, and thus minimise the incidence of rockfalls and rockbursts, blast-on-type supports are increasingly being used. In some instances, however, the blast-out rate has been high (> 10 %) and where rockbolting is used, loss of rockbolt tension is common. The tendency for non-removable support units to tilt or be damaged as a result of the blast is also high. The proper pre-stressing of support units can, however, reduce this problem considerably.

## **C7 Reef geometry**

Reef geometry encompasses reef thickness, reef dip, faults and reef rolls.

### **C7.1 Reef thickness**

The height of mining (stope width) is greatly influenced by reef thickness, although, in most instances, hangingwall conditions and mining practices could alter the stope width. From Table C-1 it can be seen that the majority of stope widths are in the range from 1,2 to 1,5 m, with minimums and maximums of approximately 0,8 m and 3,5 m respectively. Stope widths in excess of 2 m are not common, comprising approximately 12 % of gold mining production.

A critical requirement that needs to be addressed by any new support system is its ability to be easily installed and to function as designed under sudden variations in stope widths, which often occur. In very high stope widths, the practical use of hydraulic props is limited by their mass as handling is difficult and individual props become a safety hazard by toppling before installation. In stope widths of more than 2 m, the buckling of hydraulic props and elongates becomes a problem, particularly if dynamic loading occurs. In narrow stope widths (< 1 m), the handling of heavy support units is difficult, worker movement is restricted, and the ease with which support units are installed is reduced.

Any support system should, therefore, be designed to operate effectively between mining heights of 0,9 m and 2,5 m, or at least include a method for coping with abnormal conditions.

Table C-1 Summary of reef geometry parameters (after Daehnke *et al.*, 1998)

Reef type	Average stope width (m)	Average dip (degrees)
Merensky	1,15	13
UG2	1,25	14
Basal	1,46	25
VCR	1,54	24
Vaal	1,24	14
Kimberly	1,24	25
Carbon Leader	1,25	22
Beatrix	1,37	9
Main	1,81	18
UE1A	3,50	0
Composite	1,09	20
Leader	1,53	14
Elsburg	2,0	24
Kalkoenskrans	1,40	18
C	1,36	21
A	1,65	7
B	1,24	8
Deelkraal	1,17	30

South	0,80	25
Black	1,04	20
Kloof	1,57	30

### **C7.2 Reef dip**

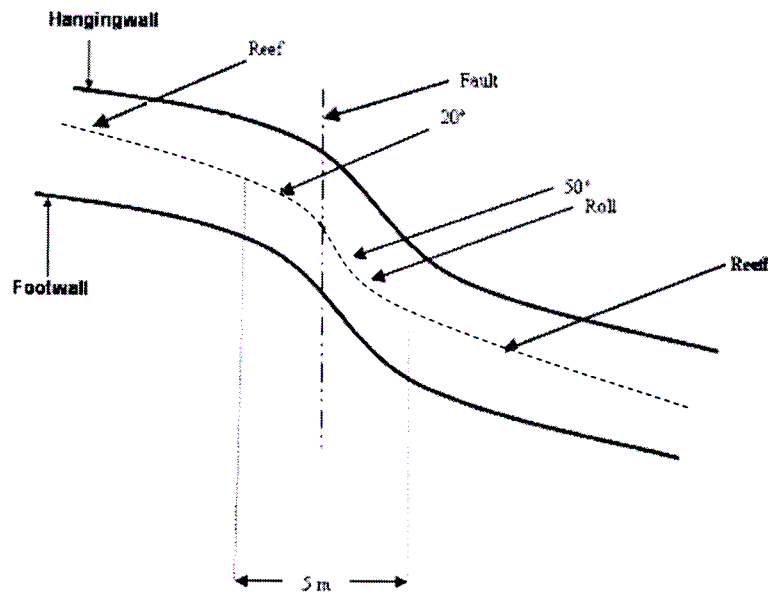
As the dip of the reef increases, so does the difficulty in handling and installing support. A consequence of this is that support units are often not installed properly, are unstable and tend to topple or buckle prematurely. Off-centre loading on units is also a common consequence of changes in reef dip. The tendency for workers to hold onto supports when ascending and descending the panel is high, thus increasing the fall-out rate of supports and the risk of injury to workers.

In flat-dipping stopes, the transportation of support units along the face becomes difficult, and this is aggravated if the stope width is narrow. Transportation difficulties mean that more time is spent moving support units and less time is available for the correct installation of a sufficient number of support units. The advantage of reusable units is obvious.

As can be seen from Table C-1, the average dip of all reefs is approximately 17 degrees (minimum of 0 degrees and maximum of 30 degrees). Any ideal support system should, therefore, be designed to operate optimally up to a dip of 30 degrees, but should also be effective in terms of handling and performance in situations where the dip changes.

### **C7.3 Faults and reef rolls**

Due to the fact that the dip of an orebody can change abruptly, any viable support system should be able to accommodate this change (see Figure C-1). A support system should be able to negotiate any change in dip. The implementation of a new support system should ensure that the re-establishment of the excavation on the orebody, when a reef roll is encountered, can be done safely and economically. Furthermore, the ease of moving and installing supports and, most importantly, the support performance should not be compromised by the frequent unevenness of the footwall and hangingwall.



*Figure C-1 Strike section of a stope showing the geometry a support system may be required to cope with in the case of a reef roll or fault*

## **C8 Production system**

### **C8.1 Flow of ventilation**

The installation of a support system requires workers to construct or install supports to a prescribed pattern (unless the system is automated). The successful installation of the support system and the execution of other tasks in the stope face area are influenced by, among other factors, the levels of temperature and humidity. Any new support system should, therefore, not obstruct the flow of ventilation along the working faces. Neither should its use also generate more humid conditions in the stope face area; e.g. the excessive use of water is to be avoided (unless chilled water is used).

The support system should in no way increase the likelihood of dust entering the ventilation flow, and susceptible parts should be sealed against the ingress of abrasive dust and grit particles.

The material from which the support units are made should be fire resistant and not promote the spread of any fire, nor should it release any toxic fumes if it is exposed to fire.

## C8.2 Productivity

Attempts to increase productivity have resulted in the focus shifting to a reduction in total face length mined and a corresponding increase in face advance. Many mines, for example, now plan to blast each stope panel every day. This means that the productivity (operational efficiency) of each mining activity will increase commensurately. The rate at which support should be installed also increases with the face advance rate.

Table C-2 gives a summary of mining data collected for 1997. As can be seen from the table, the average cycle time was a blast every two to three days. These are broad average figures, however, and many stopes are blasted more frequently (daily).

Table C-2 Summary of average production statistics (after Daehnke *et al.*, 1998)

Reef type	Average face advance (m/month)	Approximate number of days/blast	Depth (metres)
Merensky	11,0	2	769
UG2	12,0	2	1 003
Basal	7,4	3	1 745
VCR	7,9	3	1 137
Vaal	8,1	3	1 943
Kimberley	9,7	2,5	1 316
Carbon Leader	7,8	3	2 094
Beatrix	13,1	2	885
Main	5,3	5	1 206
UE1A	5,0	5	800
Composite	6,8	4	3 090

Leader	8,0	3	1 376
Elsburg	9,7	2,5	1 753
Kalkoenskrans	6,7	4	2 000
C	9,6	2,5	1 950
A	7,4	3	1 416
B	11,8	2	1 750
Deelkraal	8,0	3	1 500
South	6,0	4	1 500
Black	7,0	3,5	588
Kloof	8,0	3	2 500

Any new support system should be designed for the highest productivity scenario, in which the time required for support installation fits into a production cycle of a safe blast every day.

Should a new mining method be introduced, or a modification of existing methods be envisaged as a result of the introduction of a new support system, the current extraction ratios should be maintained or be improved upon. The current planned extraction ratios in the industry are between 75 % and 85 %, depending on the mining method. In practice, however, ratios of 65 % to 70 % are being achieved (Vieira, 2000). Vieira further states that, should any new mining method be proposed, it should allow the safe and efficient extraction of reef in situations where unpredictable yet common geological structures, such as faults and dykes, are encountered.

## **C9 Summary**

A summary of the operating specifications required of any new support system aimed at reducing the high accident statistics to more acceptable limits in South African gold and platinum mines is presented in Table C-3.

Table C-3 Summary of operating specifications of a support system

Parameter	Requirement
Safety	Afford adequate protection for workers during all stoping operations, i.e. barring, making safe, marking, drilling, charging and cleaning operations, with the objective of reducing the FOG or rockburst fatality rate by at least 50 %.
Integration into mining cycle	Should not interfere with or compromise stope face area activities.
Handling	Maximum mass of support components that need to be handled regularly should not exceed 30 kg.  System should be easy and safe to handle.
Installation & removal	Some degree of mechanisation is required.  Should be easy and safe to install/remove.
Blast damage	System should be able to withstand the effects of blasting.
Mining height	System should be easily installed and function effectively in stope widths between 0,9 m to 2,5 m or, if limited to lower widths, have a facility for accommodating the abnormal higher widths.
Reef dip	The system should be able to accommodate dips of up to 30°.
Faults and reef rolls	The system should be able to negotiate an additional change in dip of 20° over 5 m. It should also allow for safe and efficient undercutting operations.
Production system	The support installation rate should fit into the production rate.  The system should not obstruct the flow of ventilation.
Maintenance	Minimised maintenance requirements to follow a simple standard procedure. Replacement parts should be easy to install.

Finally, the system should be subjected to a formal risk assessment so that possible hazards can be identified and risks quantified. The risks need to be considered and minimised during the large-scale design of any new, improved support system. Training manuals should highlight ways of overcoming the hazards.

## **C10     References**

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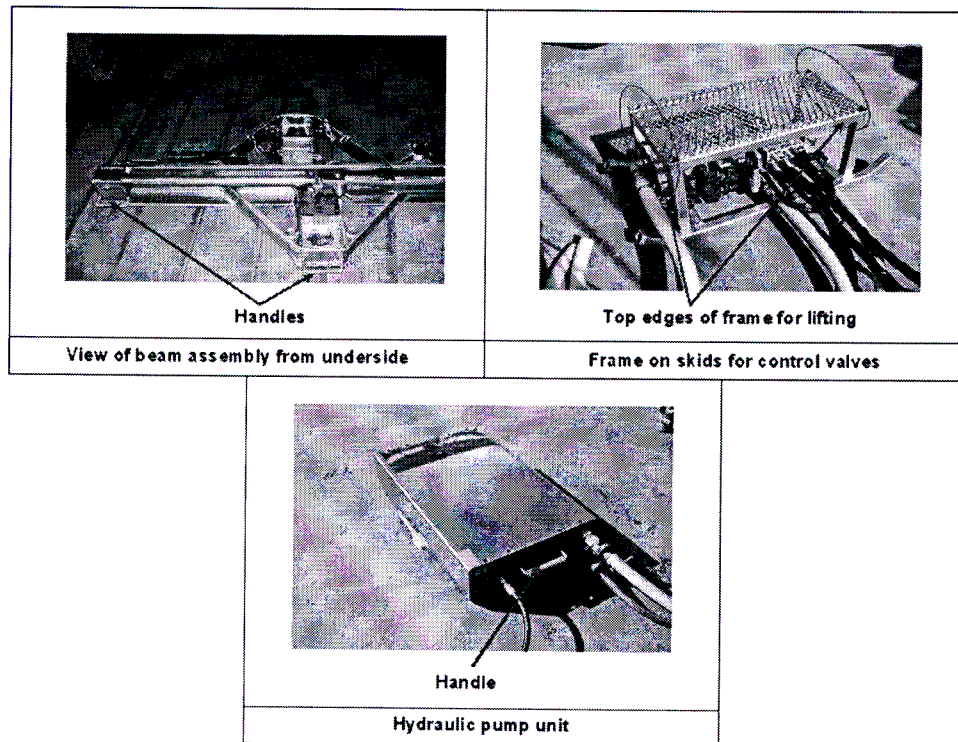
**Vieira, F.M.C.C. 2000.** Personal Communication. CSIR: Division of Mining Technology, Carlow Road, Melville, Johannesburg, South Africa.

## **APPENDIX D: OPERATIONAL PROCEDURES FOR THE WALKING BEAM STOPE SUPPORT SYSTEM**

### **D1 Handling procedures**

Onloading/offloading storage and transportation

- The mine should use only personnel who have been trained to use the walking beam system by the appropriate individuals (system designer or manufacturer). Fit-for-purpose equipment must be used for handling of the walking beam components.
- When packed or stored, all items/components must be arranged in a manner that allows good accessibility for the equipment that is used for onloading or offloading them. Also, heavy items, especially the headboard assemblies, must not be stacked on top of each other so that potential falling accidents and damage to the equipment can be prevented.
- During lifting and moving activities, the appropriate lifting points on the equipment must be used. On the headboard assemblies, pump and toolboxes there are grab handles for lifting. On the control box the edge of the top surface of the box should be used for lifting. Some of these lifting points are indicated in Figure D-1 below. The control valves are mounted in a skid frame and this can be dragged to positions along the footwall during system operation.



*Figure D-1 Group of photos showing lifting points on components of significant mass for assembly, installation, operation and system removal*

**Beam orientation**

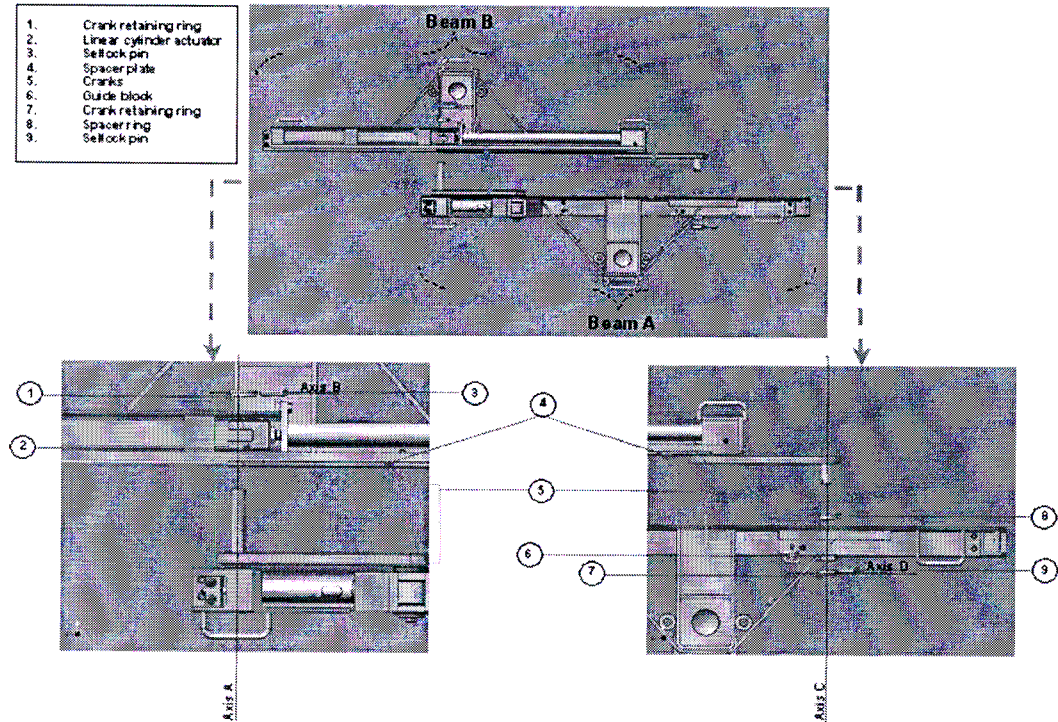
In the rest of this report the beam with the rotational cylinder will be referred to as beam A and the beam with the linear actuating cylinder as beam B.

**D2 Assembly**

The system can be transported underground in one of two configurations:

1. The headboards can be pre-assembled on surface, without the hydraulic props attached to each headboard. The assembled headboards and hydraulic props can then be transported separately. The hydraulic props can be attached to the beams once the beam assembly has been placed in position for operation.
2. The individual beams can be transported to site and thereafter assembled/linked to each other, with the hydraulic props detached and transported separately. The hydraulic props can be attached to the beams once the beam assembly has been placed in position for operation.

## D2.1 Assembly of the individual beams



*Figure D-2 The components for the assembly of the individual beams and their assembly axes (plan view)*

Figure D-2 details the layout of the beams for assembly. The primary components involved in the assembly are numbered from 1 to 9. The components to which these numbers are assigned are as follows:

- 1 – crank retaining ring;
- 2 – linear cylinder actuator;
- 3 – sellock pin;
- 4 – spacer plate;
- 5 – cranks;
- 6 – guide block;
- 7 – crank retaining ring;
- 8 – spacer ring; and
- 9 – sellock pin.

Four axes of assembly are denoted in Figure D-2 as A, B, C and D.

The beams are assembled as follows:

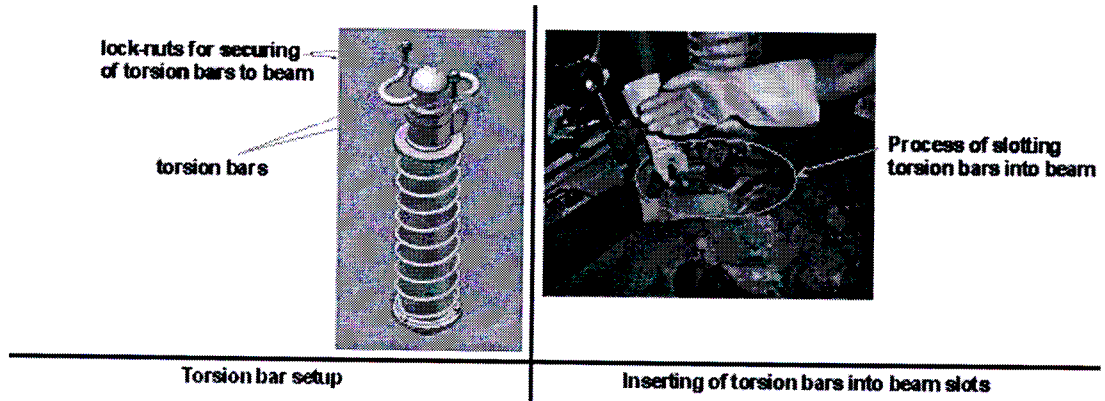
- Insert the cranks simultaneously into the respective components along axes A and C.
- Along axis A, insert the crank through the linear cylinder actuator (2) and insert the crank retaining ring (1) into the crank, flush against the beam. Knock the sellock pin (3) into position along axis B through the crank retaining ring to secure the crank in position.
- Similarly, along axis C, slot the crank into the guide plate. Prior to slotting the crank, insert the spacer ring (8) on the crank.
- Insert the crank with the spacer ring into the guide plate until the spacer ring is flush against the guide plate.
- Insert the crank retaining ring (7) into the crank
- Secure the crank to the beam by knocking the sellock pin along axis D through the crank retaining ring.

## **D2.2 Attaching the hydraulic props to the beam assembly**

The hydraulic props are attached to the beam assembly by means of torsion bars. The following procedures should be adhered to for attaching the hydraulic props to the beams (refer to Figure D-3):

- Lay the beam assembly on the footwall such that the underside of the assembly faces upwards. Refer to the handling procedures in Section D-1, as the correct procedures must be adhered to when handling heavy components.
- Insert two torsion bars into the slot located on the head of one of the hydraulic props. The torsion bars must be inserted from each side of the slot such that the horizontal sections of the bars meet each other halfway within the slot.
- Position the hydraulic prop in the beam's spherical recess, while simultaneously guiding the threaded section of the torsion bar into the slots located on the beam.
- While the hydraulic prop is held in this position, tilt the beam assembly to its side and wedge it in this tilted position relative to the footwall.
- Secure the torsion bars with lock nuts, but be sure not to over-tighten the locknuts as this will lead to improper setting of the hydraulic prop.

- Repeat the above steps for assembly of the other hydraulic prop.



*Figure D-3 Hydraulic prop assembly*

### **D3 System set-up and installation**

#### **D3.1 Stopping criteria for application of the system**

- The ideal stope width operating range for the machine is between 1.1 m and 1.4 m.
- Air and water supplies are required for the system's operation.
- The minimum air pressure required is approximately 4 bar.

#### **D3.2 Installation procedures**

- When many units are being installed in a panel, start installing from the downdip side of the panel and progress with installing units updip.
- The units should be installed in positions whereby they do not hamper mining activities.

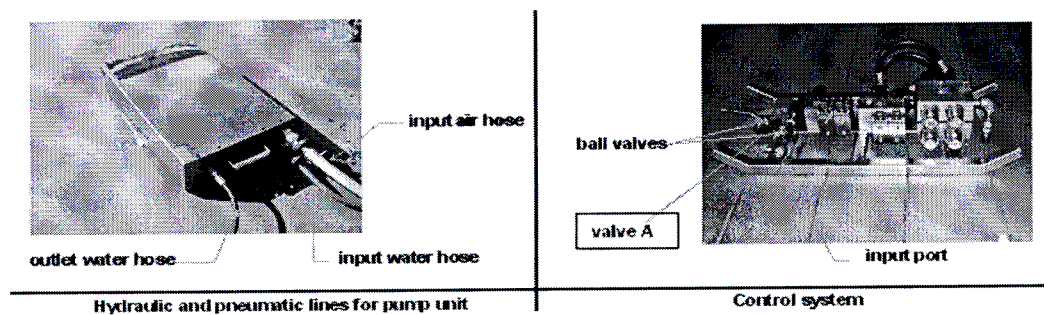
##### **D3.2.1 Setting up the control system**

The sequence for setting up the control system is as follows:

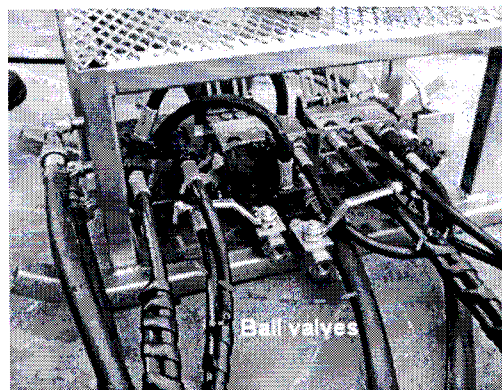
- Switch off the air and water supplies;
- Connect the air and water supply hoses to the air and water ports of the pump unit;

- Connect the outlet hose from the pump to the input port of the hydraulic control box;
- Close all the ball valves on the control box;
- Ensure that all the staples on the control valves are secure; and
- Switch on the air and water supplies.

In Figure D-4 the valve marked A is a shut-off valve that serves to integrate or isolate the hydraulic fluid with the control system. The ball valve adjacent to valve A as depicted in the figure serves to relieve the pressure in the control system. The ball valves in Figure 2-6 serve to relieve the pressure on the hydraulic lines to the beam cylinders.



*Figure D-4 Photos of hydraulic pump and control system*



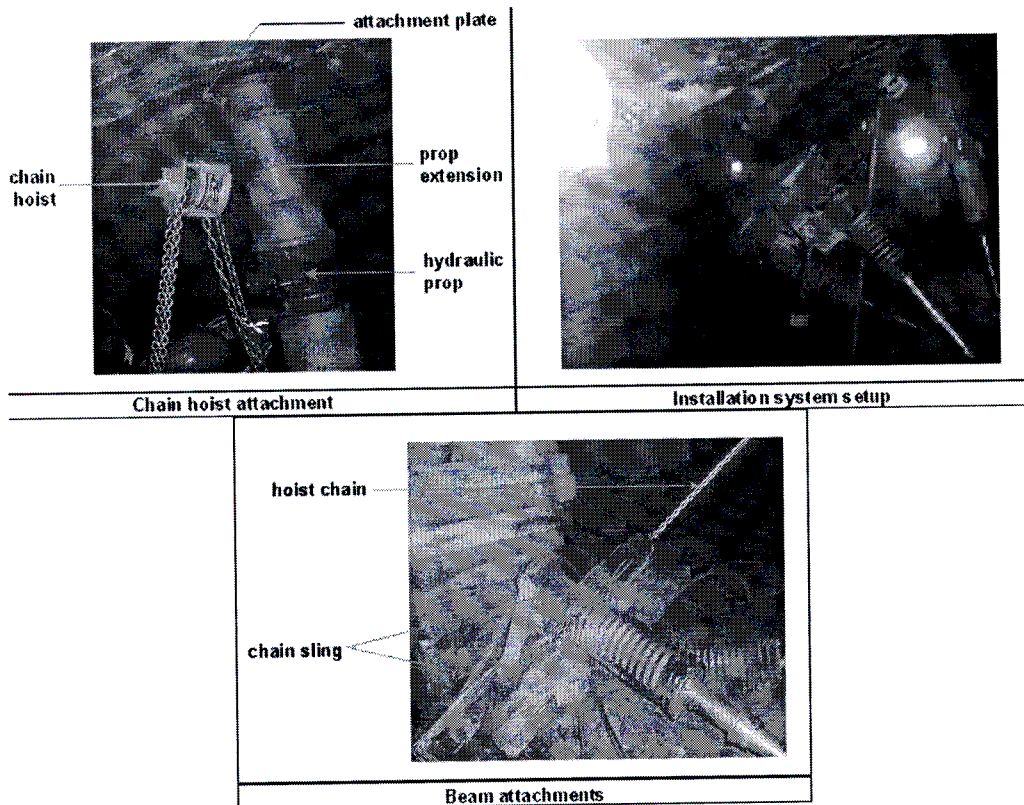
*Figure D-5 Ball valves for pressure relief on hydraulic lines*

### **D3.2.2 Setting up the installation aid**

Refer to Figure D-6. The sequence for setting up the installation aid is the following:

- Place the hoist attachment plate on top of the hoist prop;
- Pressurise the hoist prop against the hangingwall, on the updip side of the walking beam;
- Use a safety chain/device to secure the hoist prop;
- Attach the chain hoist (1/2 ton or greater capacity) to the hook that is positioned on the plate;
- Attach the chain sling to the hoist and to the attachment points located on the beam;
- Adjust the slack on each of the chains that hook onto the beam; and
- Begin the lifting process by operating the chain hoist.

**Note: The walking beam stope support system is a development model and the installation system described here is part of this development initiative. The development process by which this procedure was designed is described in Section 3.5.1.4 and Figure 3-19. Further development is still possible and the procedure is therefore not a final solution for the efficient installation of the system.**



*Figure D-7 Photos showing installation system and auxiliary components*

Once the walking beam has been lifted into position, connect the filling valve marked A to the prop attached to beam A and the filling valve marked B to the prop attached to beam B. The markings A and B are engraved on the respective filling valves. Make sure that when the custom-designed valve is inserted into the prop-filling valve, the latch on the custom-designed valve is properly set in position. This latch secures the valve interface. Refer to Figure D-8.

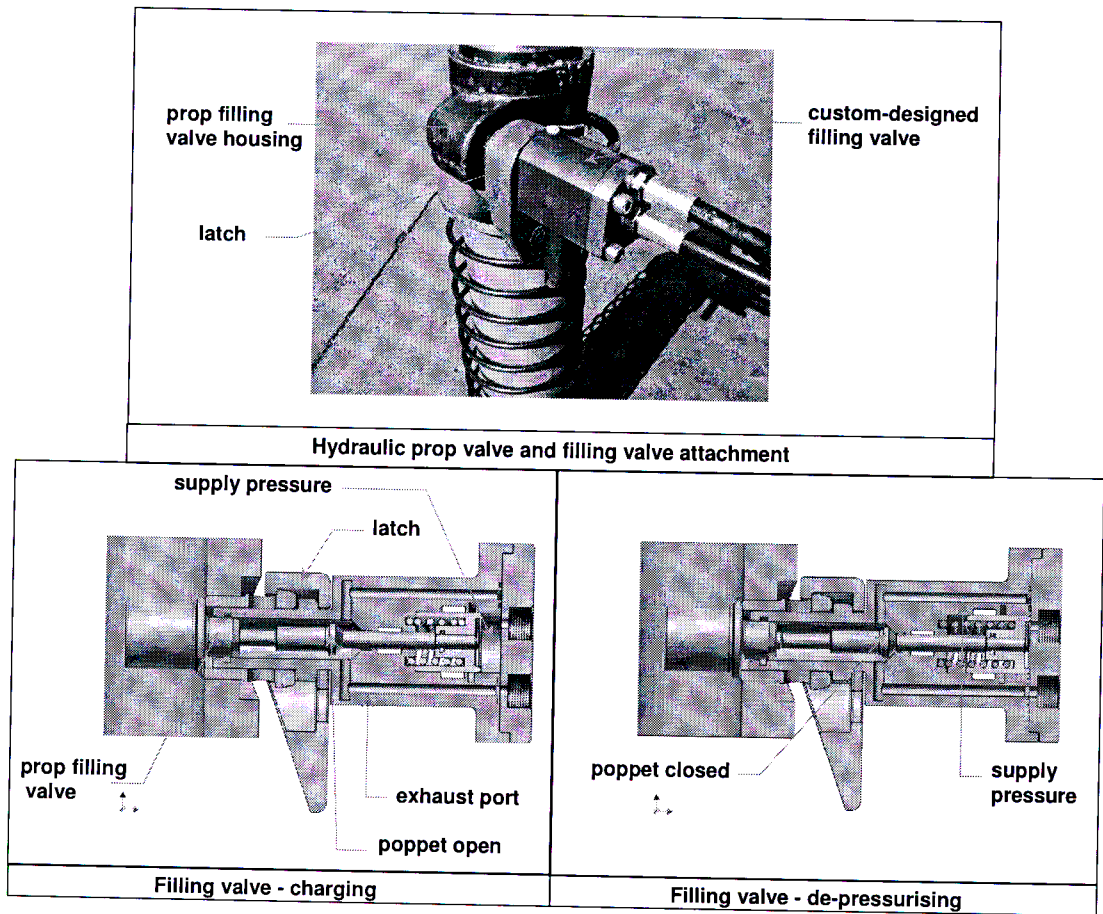


Figure D-8 Hydraulic prop and filling valve interface

Open the ball valve located on the input line of the control system. Refer to Figure D-4. This valve is denoted in the figure as valve A.

Pressurise each of the props, a little at a time, until the beams make contact with the hangingwall. Ensure that the headboard pads make proper contact with the hangingwall. See Section D-4 for procedures involving operation of the control valves. This process would be described in the operator training programme. The unit is now installed and ready for operation.

## D4 Operation

The walking beam system is a remotely operated machine and the operator must be positioned at a safe distance behind the machine during its operational control. No one must be in contact with or in unsafe proximity to the machine during the advancement/operational process.

### D4.1 Connecting hydraulic hoses for control of beam cylinders

Connect the hose adaptor block marked A (engraved on the component) to the input ports on beam A and connect the block marked B (engraved on the component) to the input ports on beam B. Refer to Figure D-9 for a description of the hydraulic ports. The hose adaptor blocks slot on the respective hydraulic ports located on each beam.

Secure the connections of these hose adaptor blocks with staple lock pins.

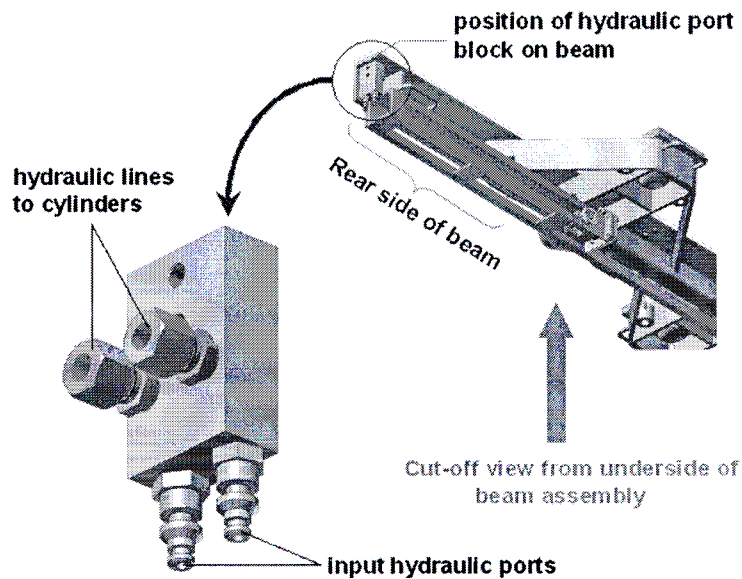


Figure D-9 Hydraulic lines for operation of beam cylinders

#### **D4.2 Sequence of steps for system operation**

The sequence of steps set out below is to be followed:

- Leave valve A (Figure D-4) open, but close all other ball valves;
- Lower beam B;
- Advance beam B forward;
- Adjust the rotational cylinder to allow initial contact of the beam with the hangingwall;
- Pressurise the beam against the hangingwall and ensure proper contact between the headboard pad and the hangingwall; and
- Similarly, move beam A forward and set it against the hangingwall.

The above steps represent a single cycle of motion of the machine. For the next cycle, the process is repeated.

During the advancement of the individual beams, the respective beam clearances between the beam and the hangingwall (for irregular hangingwall conditions) may have to be adjusted, depending on the hangingwall conditions. In such cases, the angular positioning of the beams relative to the hangingwall can be adjusted by operating the rotational cylinder.

The control diagram in Figure D-10 illustrates the controls for operating the machine and the corresponding control action for activating the different operational steps.

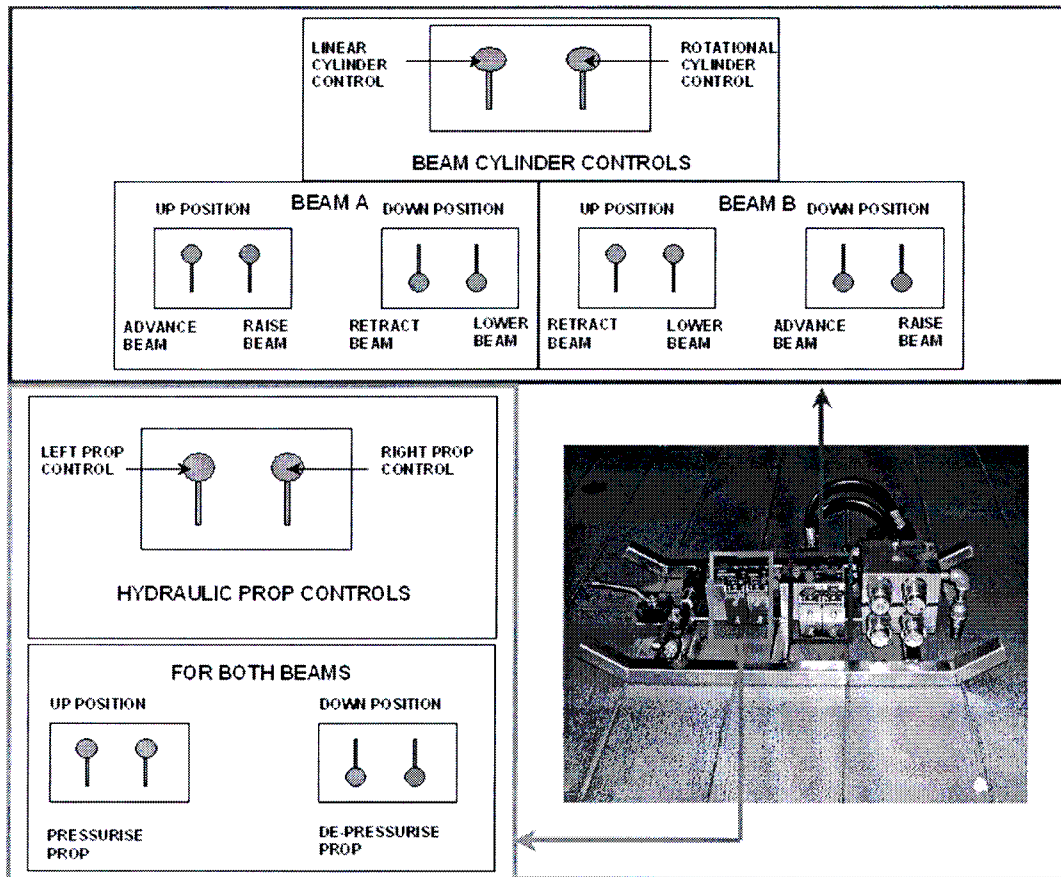


Figure D-10 Control diagram

## D5 System removal

The steps set out below should be followed:

- Install the hoist system as described in Section D3.2.2;
- Attach a chain sling to the hoist and to the attachment points located on the beam;
- Adjust the slack on each of the chains that hook onto the beam;
- Depressurise each of the props, a little at a time, until the beams are no longer in contact with the hanging wall;
- Gradually lower the machine by operating the chain hoist;
- Once the machine is lying on the footwall, remove the chain hoist from the installation prop hook connection;

- Remove the installation prop by using the standard hydraulic prop-release mechanism to depressurise the prop;
- Disassemble the hydraulic props from the beams by removing the nuts that secure the torsion bar attachments to the beams;
- Disassemble the beams (refer to Section D2.1). Remove the sellock pins by knocking them out of the crank retaining rings, and separate the beams by unslotting the cranks from their respective grooves;
- Remove all the components from the area and load them into a scotchcar for transportation to the shaft, from where they can be taken to the surface; and
- Install permanent support (according to mine support standards) in the position/area from which the walking beam system was removed.

## APPENDIX E DESIGN CALCULATION FOR THE SPLINED SHAFT DIAMETER

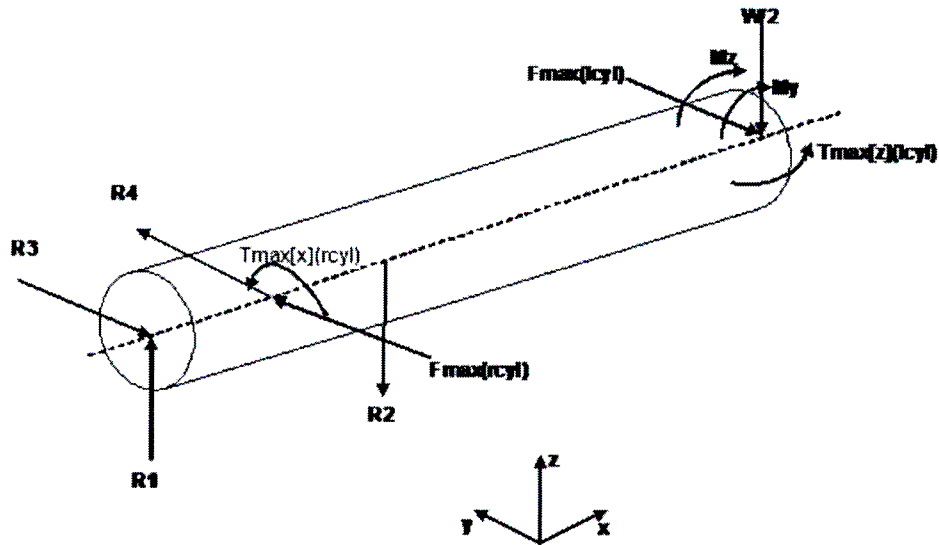


Figure E-1 The derived force diagram for the splined shaft

$R_1, R_2, R_3$  and  $R_4$  = reactional forces that are induced in the splined shaft.

$F_{max}(rcyl)$  = maximum force that is applied by the rotational cylinder to the actual spline.

$T_{max}[x](rcyl)$  = maximum torque that is applied by the rotational cylinder via the internally splined crank mechanism and oriented about the x-axis.

$F_{max}(lcyl)$  = maximum force exerted by the linear actuating cylinder

$T_{max}[z](lcyl)$  = maximum torque applied by the linear cylinder via the crank system about the z-axis.

$M_z$  = bending moment that is applied to the shaft about the z-axis due to the actuation of the linear cylinder

$M_y$  = bending moment that is applied to the shaft with respect to the y-axis and is a function of the weight  $W$ .

The rotational cylinder had to be capable of lifting each beam assembly, a hydraulic prop, and the Camlok prop. From the CAD model, the following masses were determined:

Approximate maximum mass of beam assembly = 60 kg

Hydraulic prop mass = 40 kg

Camlok prop mass (headboard + prop) = 10+25 = 35 kg

Total mass = 135 kg as a worst-case scenario. An additional 15 kg was added to this as contingency. Hence, total mass requirement for rotational cylinder = 150 kg, and therefore weight  $W = 1500 \text{ N}$ .

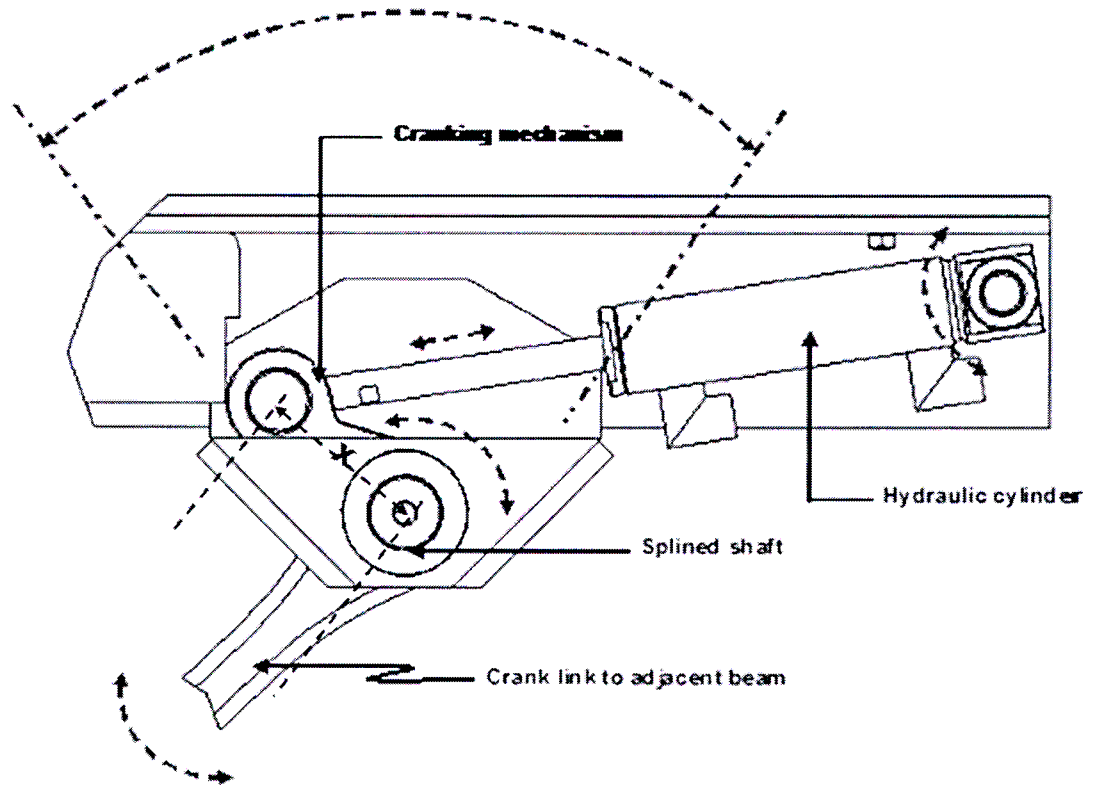


Figure E-2 Rotational cylinder components and dynamics

If the distance marked X (Figure E-2), is set as 70 mm and the crank length is set to 250 mm, in order to lift a load of 1500 N, a force of 5000 N is required. This force was found from the moment calculation below:

$$W \times 250 = F_{\max}(\text{rcyl}) \times 70$$

$$\text{Therefore, } F_{\max}(\text{rcyl}) = (1500 \times 250)/70 = 5357 \text{ N (say 5000 N)}$$

$$\text{For a safety factor of 1.5, } F_{\max}(\text{rcyl}) = 5000 \times 1.5 = 7500 \text{ N}$$

The rotational cylinder's rod diameter was specified as 20 mm with a crown diameter of 30 mm. The operating pressure for the rotational cylinder is 20 MPa. The maximum force that can be applied with these specifications is calculated:

$$F = P \times A$$

$$= 20e6 \times ((\pi \times 0.03^2)/4)$$

$$= 14137 \text{ N (say 14000 N)}$$

A force of 14000 N is sufficient for meeting the calculated requirement of 7500 N for the rotational cylinder ie.  $F_{\max}(\text{rcyl}) = 14000 \text{ N}$

The force exerted by the linear cylinder is equated to the weight requirement of 1500 N, which is an over-estimation but valuable and sufficient for a trial calculation.

Now,  $T_{\max}[x](\text{rcyl})$  and  $T_{\max}[z](\text{l cyl})$  can be calculated:

$$T_{\max}[x](\text{rcyl}) = F_{\max}(\text{rcyl}) \times X$$

$$= 14000 \times 0.07$$

$$= 980 \text{ Nm}$$

$$T_{\max}[z](\text{l cyl}) = F_{\max}(\text{l cyl}) \times (\cos 45^\circ \times \text{crank length}), \text{ assuming the crank is oriented at } 45^\circ$$

$$= 1500 \times 0.707 \times 0.25$$

$$= 265.13 \text{ Nm}$$

$M_y$  is the product of the half of the weight  $W$  and the centre of mass (C.O.M.) for the beam assembly. The centre of mass is determined from the 3D CAD model (Figure E-3)

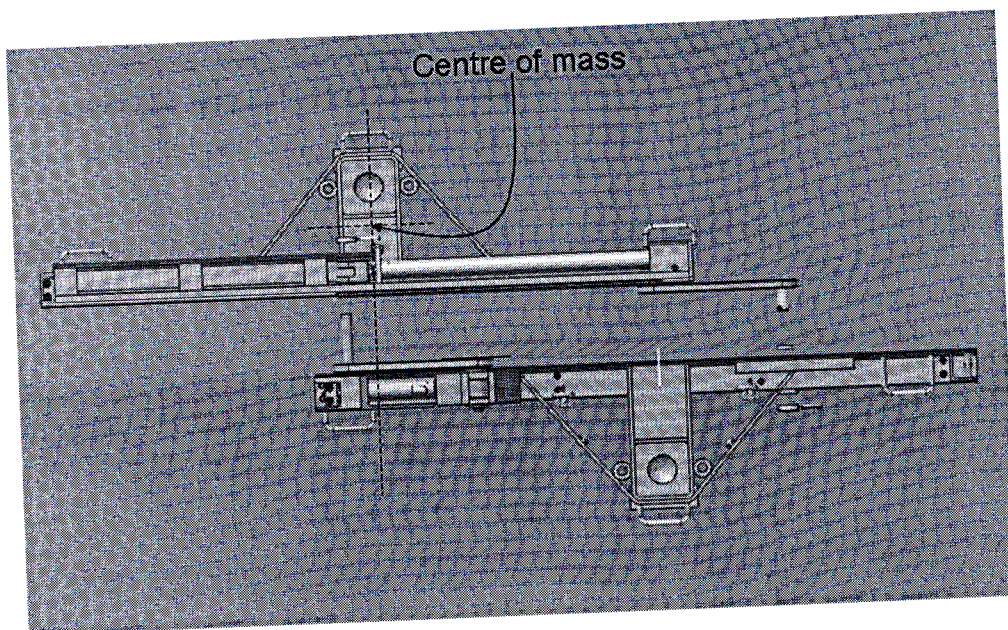


Figure E-3 Centre of mass for beam assembly

$$\begin{aligned}
 M_y &= W/2 \times \text{C.O.M.} \\
 &= (1500/2) \times 0.215 \\
 &= 161.25 \text{ Nm}
 \end{aligned}$$

The positions of the shaft bearings and dimensions thereof were determined from the 3D model generated using AutoDesk Inventor (CAD software). These positions represent the reactional force positions on the splined shaft and are indicated in Figure E-4 (58 mm and 29 mm respectively). Figure E-3 shows the x-z and x-y planar views of the splined shaft respectively.

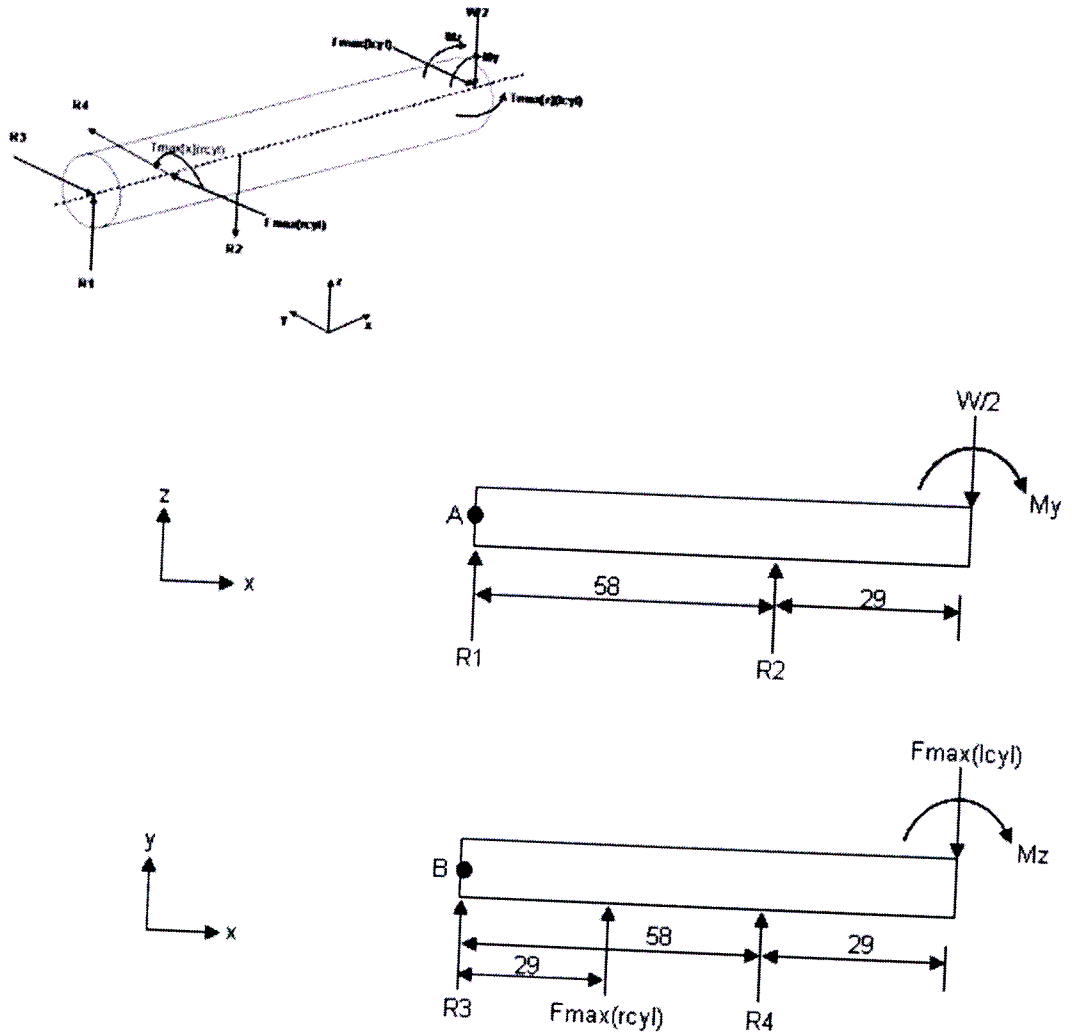


Figure E-4 Planar views of the splined shaft showing forces and moments

With reference to Figure E-4, the equilibrium calculations are as follows:  
 In the x-z plane, taking moments about point A,

$$(-R_2 \times 0.058) + (W/2 \times 0.087) + M_y = 0$$

$$-0.058 R_2 + 65.25 + 161.25 = 0$$

$$R_2 = 3905 \text{ N}$$

$$\sum F_z = 0: R_1 + R_2 = W/2$$

$$R_1 = -3155 \text{ N (ie. 3155 N downwards)}$$

In the x-y plane, taking moments about point B,

$$(-R_4 \times 0.058) - (F_{\max}(\text{rcyl}) \times 0.029) + (F_{\max}(\text{lcyl}) \times 0.087) + M_z = 0$$

$$-0.058 R_4 - (14000 \times 0.029) + (1500 \times 0.087) + (1500 \times 0.071) = 0$$

Note:  $M_z = F_{\max}(\text{lcyl}) \times (\text{Centre of mass for crank linkage})$

$$= 1500 \times (0.071: \text{from CAD model})$$

$$R_4 = (406 - 130.5 - 106.5)/0.058 = -2914 \text{ N}$$

$$\sum F = 0: R_3 + R_4 + F_{\max}(\text{rcyl}) - F_{\max}(\text{lcyl}) = 0$$

$$R_3 = -9586 \text{ N}$$

The maximum bending stress is calculated as follows:

$$\sigma_b = (M \times y)/I,$$

where  $M$  = Maximum bending moment;

$$y = d/2 \text{ (d = splined shaft diameter);}$$

$$I = \text{moment of inertia} = (\pi \times d^4) / 64 \text{ for a cylindrical bar}$$

The maximum bending moment was determined from the vector sum of the bending moments in the x-z and x-y planes (Figure E-5).

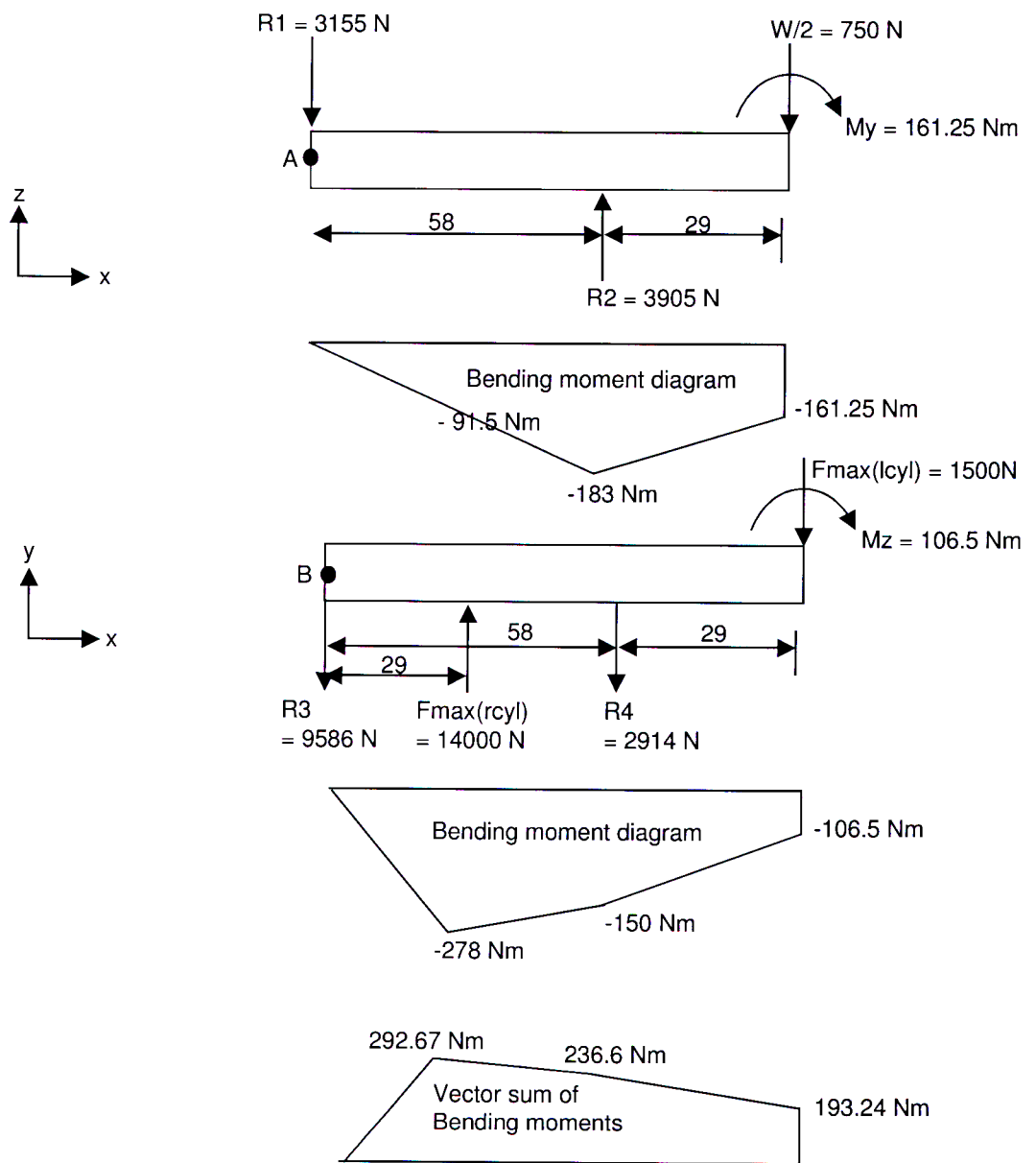


Figure E-5 Bending moment diagrams

Now,  $\sigma_b = (M \times y)/I = (292.67 \times d/2)/((\pi \times d^4) / 64) = 2981/d^3$

The torsional stress is calculated as follows:

$\tau_t = (T \times r)/J$ , where

T = total torque applied;

r = radius = (d/2); and

J = torsional modulus =  $(\pi \times d^4)/32$

$$\tau_t = (Tr)/J = ((980 + 265.13) \times d/2) / ((\pi \times d^4)/32) = 6341.4 / d^3$$

The shear stress is calculated as follows,

$$\tau_s = (4 \times V) / (3 \times A) \text{ where,}$$

V = Maximum shear force; and

A = shaft cross-sectional area

The maximum shear force was determined from the vector sum of the shear forces in the x-z and x-y planes (Figure E-6).

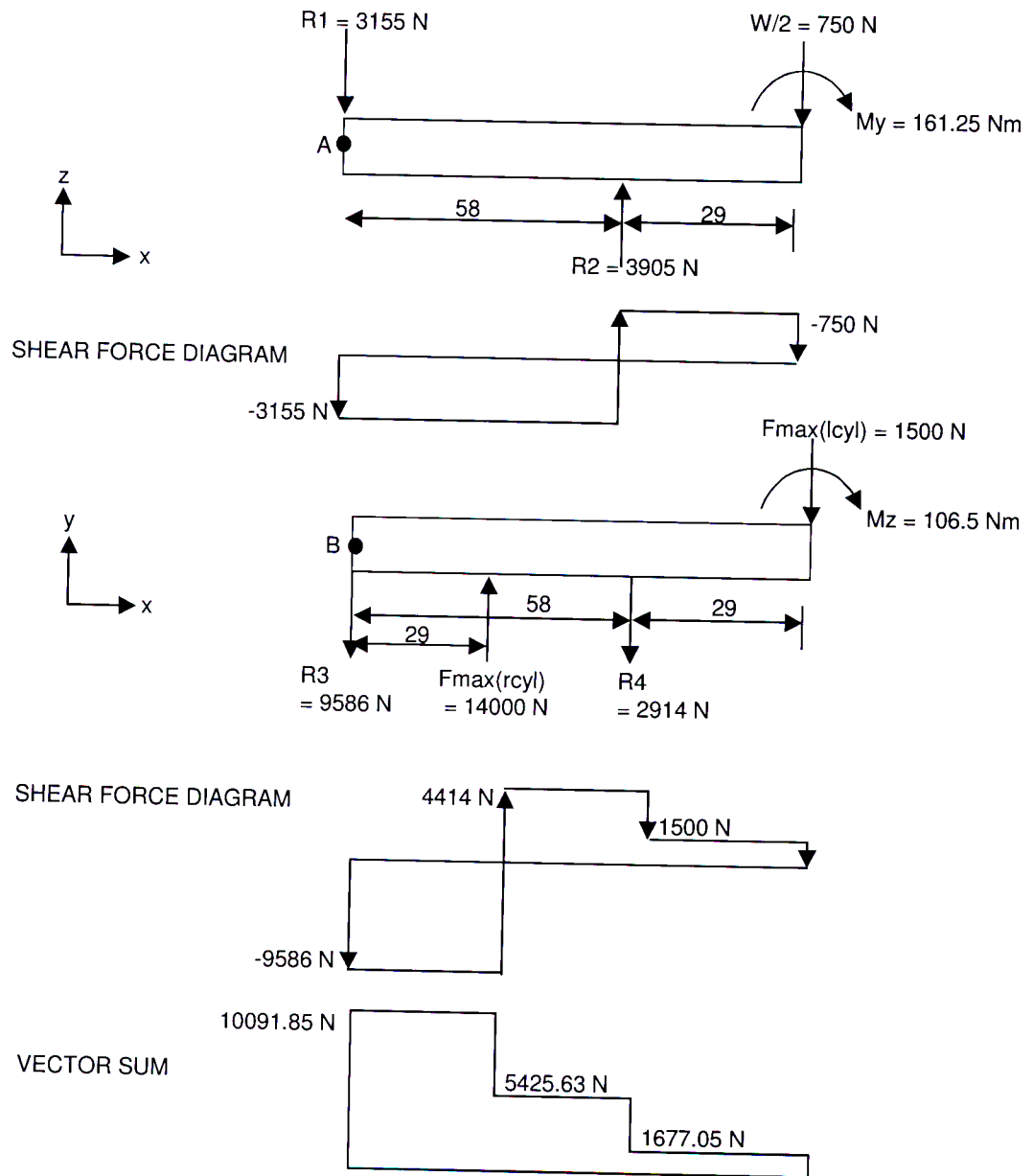


Figure E-6 Shear force diagrams

Now,  $\tau_s = (4 \times V)/(3 \times A) = (4 \times 10091.85)/(3 \times ((\pi \times d^2)/4)) = 17133.9/d^2$

Using Von Mises Stress Criterion,

$\sigma_e = ((\sigma_b^2 + 3((\tau_r + \tau_s)^2))^{1/2} = S_y/n$  where,

$S_y$  = material yield strength = 200Mpa; and

$n$  = safety factor = 2

therefore,  $(200e6)/2 = ((2981/d^3)^2 + 3 \times (6341.4/d^3 + 17133.9/d^2))^{1/2}$

Solving for  $d$  results in a value of  $\sim 49.3$  mm.