BEST PRACTICE FOR PERSONNEL, MATERIAL AND ROCK TRANSPORTATION IN ULTRA DEEP LEVEL GOLD MINES

In fulfillment of the Doctoral degree

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ABSTRACT

Ultra deep mining presents many challenges to the mining engineer, one of which is the logistics to support mining operations quickly and efficiently. Typically, Witwatersrand gold mines operate at depths in excess of 2000 m with stoping taking place to 3500 m and investigations underway to mine to a depth of 5000 m. As mining progresses deeper and further from the shaft, the role of logistics becomes increasingly important if production targets are to be achieved. Access to the workings is often via sub vertical and even tertiary subvertical shaft systems with working faces as far as five kilometers from the shaft. It is inevitable therefore, that distance will negatively impact the working time available at the stope face, material transportation and distribution, as well as the removal of broken ore. Possible solutions to these logistical problems may be found in the use of different transportation systems or by applying sound design and operational principles to transportation systems, both in the horizontal and in-stope areas.

This thesis investigates the challenges of logistics for ultra deep level gold mining in the Witwaterstrand basin for mining layouts planning to mine between 3000 m and 5000 m underground with typical horizontal distances of over 3000 m. The transportation needs analysis recognised that vertical transportation is a well-managed and organised system and is mainly the same for both shallow and deep level operations. As a result of this, the thesis only focuses on the logistical issues of the horizontal and in-stope processes.

The literature review indicates that the majority of work previously conducted on transportation focused around the area of horizontal transportation with limited inputs to in-stope transportation systems. The review concludes that the traditional locomotive transportation system is the most applicable mode of horizontal transportation. Thus, special emphasis is given to trackbound transportation.

An integrated approach is taken towards mine transportation advocating that underground logistics be considered as equally important as any other discipline, i.e. rock engineering, ventilation, etc. In addition, the transportation process should consider each area equally important. All to often, the transportation of rock is considered of paramount importance over the transportation of personnel and material. Thus, the planning any transportation system should incorporate personnel, material and rock. To enable this, scheduling, communication and control are important with special attention required for transfer points in the transportation system.

As each site has its own particular requirement, thus the final transportation systems must be drawn up based on the specific requirements of each mine. A guideline is proposed for the design of ultra deep level underground transport systems for personnel, material and rock transportation. Thus, providing mining engineers with sufficient information and data to select an appropriate transportation system to meet specific mine requirements. The thesis highlights areas requiring consideration by mine engineers when designing a transportation system from shaft to the working face.

Glossary of Terms

ADT	Articulated Dump Truck
DME	Department of Minerals and Energy
COMRO	Chamber of Mines Research Organisation
CSDP	Closely Spaced Dip Pillars
GHz	Gigahertz
h	Height
HPE	Hydro Power Equipment
hr	hour
Hz	hertz
Kg	Kilogram
Km	Kilometre
kW	Kilowatt
l	Length
LDV	Light Duty Vehicle
LHD	Load Haul Dump vehicle
LSP	Longwall mining with Strike Pillars
m	metre
MCF	Mine Call Factor
Mhz	Megahertz
mm	millimetre
MRAC	Mining Regulatory Advisory Committee
PGM	Platinum group metals
R.A.W.	Return Air Way
RFID	Radio Frequency Identification system
s	Seconds
SDD	Sequential Down Dip
SGM	Sequential Grid Mining
t	ton
t-m/hr	tons – metre per hour
T/w	Travelling way
w	Width
°	Degree
%	Percentage

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1. Introduction

Transportation is a significant factor in the ultra deep mining environment, due to the complex nature of mining, for example, increasing working distances from the shaft, an increase in support requirements, face advance and tonnage produced. Transportation systems must become more efficient by utilising the appropriate technologies to ensure that personnel and material arrive at the workforce at the appropriate time and that ore is removed timeously to support the planned production rates. Thus, good management coupled with the increased use of communication, automation and remote control systems are required to increase transportation efficiencies.

Future transportation systems for ultra deep level mining in the Witwatersrand basin will require personnel, material and rock to be transported quickly and efficiently, thus maximising the time spent at the working face. Currently, workers travel on the levels predominantly by foot, or in locomotive drawn carriages, and by foot in the reef horizon. Material must be supplied to the working face in sufficient quantities and when required, while rock needs to be removed from the stope in order that personnel and material can easily reach the working face.

Transportation from surface to the desired level is, for the most part, well managed and efficient. It is the opinion of most consulting engineers and mine managers that very little requires changing in the area vertical transportation other than improved shift control practices (Diering, 1999). Diering states, "elements of the twenty-first century mine have always been present in vertical transportation, where standards, use of technology, understanding of the capability and use of monitoring equipment is all in place." Therefore, this thesis investigates the area of underground transportation for ultra deep level gold mines of the Witwatersrand in terms of horizontal and in-stope transportation.

1.1 Motivation

Typically the cost of transportation in mining is in the order of 15% to 20% of the total operational costs. While transportation may play a major role in mining, it is generally given fairly low management attention. This coupled with the fact that currently, ultra deep level mines require a large workforce, substantial amounts of material and high volumes of rock; has lead to transportations systems that are outdated. Therefore, a culture must be created for transport systems to operate efficiently. This requires a mindset change of current managers, engineers and line supervisors. However, just having the correct mindset is not enough, as the actual transportation systems must be efficiently planned and equipped with the appropriate machinery and technology. This thesis captures the current practices applied to the transportation of personnel, material, and rock from the shaft station to the working face. This document will also serve as a design guideline for mine engineers in the selection of transportation systems applicable to ultra deep gold mine environments.

1.2 Methodology

The primary objective of this thesis is to assess the ability of current transportation systems to meet the requirements of ultra deep mining operations. Further objectives are to provide best practice guidelines for the transportation of personnel, material, and rock in ultra deep mines. Undertaking the following tasks performed the thesis:

- A literature review was conducted to investigate past and current transportation applications that are available in the industry.
- Identifying, describing and characterising all relevant horizontal and in-stope transportation systems that are in use in South African Gold Mines and evaluate the systems.
- Formulating guidelines and recommendations for horizontal and in-stope transportation systems in ultra deep mines.

2. Background

Narrow vein gold mining in the Witwaterstrand basin is typically conducted at stoping widths between 0,8 m and 1,8 m (Figure 2-1). Mining advances along strike at an average face advance of 6 m to 10 m per month with 1 m drill holes (slashing holes) being blasted over panel lengths ranging from 25 m to 50 m. The panels are blasted in a 1:3, 1:2, or 2:3 cycle (1 blast every 3 days, etc.) and the blasted rock from the stope panel is pulled down dip by a scraper and tipped into a strike gully (transport drift). From here, the rock is scraped to a central orepass. Support is made up of timber packs and pre-stressed mine poles with blast barricades positioned along dip to contain the blasted rock to the immediate face area. The blasted ore is transported to the strike gully (transportation drift) via water jet assisted face scraping. The ore is then moved along the strike gully to an orepass located in the centre gully. Workers and material utilise the same strike gully to gain access to the stope face.

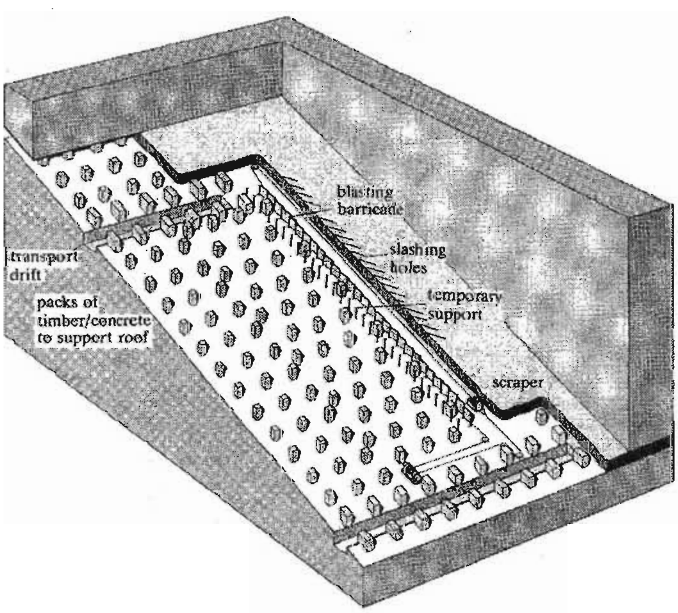


Figure 2-1: Typical stope layout (Brady and Brown)

On the level, trains are used to remove the rock from the orepass into hoppers. The train then proceeds to the shaft station where the hoppers tip the ore into the shaft orepass system. The ore is then transferred to shaft bottom where it is transported to surface via skips operating in the shaft. These same haulages are used to transport personnel and material to the working places and therefore must "compete" to utilise the haulage.

Due to the large costs (R4 billion rand) associated with ultra deep shaft structures, the number of shafts that can be utilised to support mining operations are limited. Future operations are more likely to be supported by a single "super" shaft complex rather than several smaller shafts, which is often the case for shallow and medium deep mines. In most scenarios, ultra deep mining will be an extension of current mining operations and will place increasing demands on the existing transportation systems as mining progress down dip and further from the shaft. For example, as higher support densities are required to reduce the risks associated with seismicity the number of support units will increase. Similarly, rock handling requirements will increase as tunnel dimensions become larger to meet ventilation demands. Indications (MacNulty, 2000) are that an ultra deep mine would require a significant increase in production to ensure profitability under such circumstances. It is estimated that a production profile of 45,000 centares per month is required to support a new ultra deep level gold mine. Thus, transportation requirements must improve to meet these demands.

2.1 Transportation needs analysis

For ultra deep mining to depths of 3000 m to 5000 m, mine planning requirements may vary from those used at present. Therefore, to ascertain the requirements for good transportation practices in ultra deep level gold mines, a need analysis was conducted through discussions with mining engineers and mine planners from those mines currently operating at deep levels or planning to operate at ultra deep levels. The mining houses whose transportation requirements were investigated are namely; AngloGold, ARM, Durban Roodepoort Deep, Gold Fields Limited, and Harmony. The major requirements regarding the effective future operation of transport systems are summarised as follows:

- The selected transportation system must be capable of moving personnel, material, and rock quickly and efficiently.
- Transportation systems will require a process orientated management structure and high levels of communication for efficient operation.
- Locomotives will be used for transport in ultra deep level mines; however trains with higher payloads may be required.

- Train speeds of 16 km/hr may be required for main haulages and speeds as high as 30 km/hr or 40 km/hr may be necessary for specialised operations.
- Tramming distances in the order of 4 km to 5 km are common and distances as far as 20 km could be required in the future.
- Haulages will need to be placed under the control of dedicated transportation supervisors.
- Current track installation and maintenance skills are insufficient and will need to improve.
- One of the primary restrictions of mining at ultra deep levels is the ventilation requirement. In order to maintain adequate environmental conditions to acceptable levels large quantities of air will need to be circulated, thus haulages will be large, having cross sectional areas in the order of 20 m^2.
- Future mining will most likely be concentrated, with face advances in the order of 15 m/month.
- Where concentrated mining takes place, alternative transportation systems may need to be implemented to handle the high tonnage associated with concentrated mining.
- Support requirements, thus material handling demands will increase to maintain the integrity of the haulages and stopes.
- Where these distances exceed 1,5 km to 2 km, mechanised personnel transportation systems are a necessity. For shorter distances, access to the working place should be by foot.

3. Literature Review

A comprehensive literature review was conducted to investigate those transportation systems applicable to ultra deep level mining operations for both horizontal and in-stope areas. Information was obtained from the CSIR:Mining Technology Library, Anglo American library, the University of the Witwatersrand, Natal University and current journal articles. This included manual and database searches for relevant documents. The following sections highlight the important issues facing transportation for mines planning to operate at ultra depths (3000 m to 5000 m).

3.1 General transportation issues

3.1.1 Supply chain

Typically the cost of material used in mining is in the order of 30% of the total operational costs. In some operations the cost (purchasing, handling, storing and transport) are higher than labour and all other overheads. This is coupled with the fact that while logistics may play a major role in mining, it is often given lo management attention. Hence supply chain management is a process requiring good organisation and control along with the correct information and systems.

Graulich (1999) reported on a study conducted by Anderson Consulting on supply chain management in the South African and Australian mining industry. The study identified the ability to plan and source materials, the distribution of goods and services as being crucial to efficient supply chain management. One of the main problems highlighted was that the responsibility of supply chain management was not at a senior level, but rather placed with middle management.

Supporting the importance of supply chain management was a comment raised by Sanders (1978), who stated that considerable reserves could be realised if the overall underground transport system was optimised. Sanders advocated the use of modeling and simulation of the mining layout and transport systems to optimise transportation.

Northard (1976) recommended that transportation objectives be established based on available resources. The need to reduce the number of workers deployed must be recognised, along with higher vehicle speeds and the minimising of walking distances through the implementation of personnel riding. Efficient transport systems would contribute to the achievement of higher performance and efficiency by maximising capacity and minimising delays to production, operational and maintenance manpower, and costs. Furthermore, before considering new opportunities an examination of the traditional systems in operations may reveal situations from which more immediate benefit could be forthcoming. For example, by reducing transportation costs and improving efficiencies previously unpay areas could become payable reserves.

3.1.2 Management structure

Sargeant (1997) reported on the improvements in mine transportation after re-designing management structures at Randfontein Estates Gold Mine. In the past, all workers had to walk to their sections taking up to 55 minutes. The improved transportation system increases productivity and time spent on the working face while reducing time spent on travelling to and from the working place.

The restructuring highlighted the following improvements:

- Increased transportation team discipline and morale by implementing appropriate remuneration systems.
- Miners travelling with their crew.
- A revised shaft schedule with personnel carriage times coinciding with cage times.
- No tramming operations during the transportation of the workforce.
- A marked increase in safety due to fewer tramming incidents and accidents.
- Control of transportation systems through surface and underground control rooms.

Hattingh (1997) reported on the changes made to the operational management structures at Vaal Reefs Gold Mine. The new management structure addressed the problems associated with the traditional structure; namely, poor safety, low productivity rates and high transportation costs. The main problems were caused because responsibility areas were not clearly defined, which resulted in conflict, poor communication and undefined areas of responsibility in terms of safety and costs between the various departments. The new structure clearly defined the responsibilities of shaft hoisting, tramming and mining. This resulted in improved track installation and maintenance, reduced engineering and transportation costs, rolling stock capital expenditure and derailments. It also improved safety, tramming speeds and the general underground environment due to good house keeping.

Maslen (1997) reported on the management of transportation systems in South African gold mines. Problems with the management structure contributed to the poor track conditions, communication, control and training. To rectify this, a centralised basis of transportation management, with dedicated managers for vertical and horizontal transport was recommended.

Watson (1979) found that whenever possible, personnel riding and material systems should be arranged to operate in separate tunnels. A good signalling system was found to be essential for high-speed operations and block signalling offered the most satisfactory solution for multiple trains. Speeds of up to 40 km/hr could be achieved in safety and comfort if installation and maintenance were in line with good engineering standards and practices. Good standards of track laying and maintenance were essential for high-speed transport. Underground tunnels posed special problems with their difficulties of stability and limited clearance.

Wilson (2002) proposed that material transportation be dedicated to day shift and rock transportation be conducted over the remaining two shifts. Personnel transportation would take place at the start and end of each shift, thereby, haulages and cross cuts would be dedicated to one form of transportation at any given time.

3.1.3 Communication and tracking

Pickering and Cousins (1997) commented on the complex nature of underground transportation of personnel, material and rock. They identified the need for qualified personnel, suitable equipment, appropriate standards and procedures, and advocated the use of information systems to gather and analyse data to improve transportation scheduling and management. The basic underground tracking system and devices were described and a system using Radio Frequency Identification (RFID) and Remote Intelligent Communications (RIC) were recommended. Identification tags placed on assets or personnel were also advocated.

Becker (1993) reported on Walter Becker's tracking system. The system comprised of active electronic tags and tag readers. The tags operated in the UHF band, transmitting a unique ID code at random intervals. The readers monitored the tag frequency continuously, reporting the presence of any particular tag to a PC-based monitor program, via a leaky feeder network. This program logged the tag ID, reader ID and reader-derived timestamp for every report into a database. From this point, the database could be queried in any form to extract tracking information according to the application requirement.

The Walter Becker tracking system comprised:

- Vehicle and personnel tags.
- Tag readers.
- The communications backbone.
- The communications backbone to LAN data bridge.
- A computer running the tracking front-end processor.
- A database.
- Various software tools to provide a view of the tracking data and the ability to manage the system.

Hindle (1999) wrote about 'MultiCom', the advanced leaky feeder system that El-Equip had introduced. This system incorporated automatic gain control eliminating the need of amplifiers located at the end of log amplifier chains. This improved the performance of systems with more than 30 amplifiers. Hindle also commented the new FLEXCOM RF system from Mine Radio Systems. These offered up to 32 voice/data/control channels and up to 16 video channels, which could be operated simultaneously. The system operated on a single leaky feeder cable, which was extended at 350 m intervals by installing additional amplifiers.

(Woof, 1999) reported that a Sweden firm, Tunnel Radio had released its "ULTRACOMM" underground communication systems. The system operated in either the 150 MHz or 500 MHz radio bands. The 500 MHz system achieved up to 750 m of "line of sight operation", allowing for broader mine coverage with fewer components. Signals could pass through certain metal obstructions and could reflect off walls for around corner operations. The 150 MHz system was a high power network aimed at operations where lateral coverage needs are not stringent.

3.1.4 System automation

The automation of underground transport systems has been under consideration for some time and there has been a growing trend to automate mining processes. Numerous automated rock transportation systems have been in use at mines in Australia, Europe, North and South America, and the United Kingdom. At Ruhrkohle, Germany, the system comprised of diesel or trolley locomotives handling up to forty 25-ton bottom discharge hoppers per train. These trains travelled at speeds of up to 40 km/hr. A feasibility study indicated that locomotive systems would have a capital cost advantages over belts systems for distances of 4 km upwards and an operating cost advantage for distances greater than 2,5 km (Baker, 1981).

Potts (1999) elaborated on the advances in remote LHD and locomotive automation in Europe and South America. At the Chilean state-owned Codelco's El Teniente copper mine, a LHD in one of the mine's sections was remotely controlled from a control room 1,5 km away. The mine found that the speed of the remote control operation was around 20% lower than with traditional hands-on control. However, taking into consideration labour force productivity in the long term, a substantial increase in efficiency was envisaged, as one remote operator should be able to operate two or more machines simultaneously.

Woof (1999) documented the advanced simulation of processing plants, shaft systems, rock crushes at a joint project between Modular Mining and Komatsu in Australia on surface haul truck automation. The system was controlling a small fleet of automated trucks on the same haul roads as conventional machines. Tunnel Radio of Sweden has been testing LHD automation by integrating LHD systems with the mine management system called "Tram II". The system had been tested at LKAB's iron ore mine in Kiruna, Sweden, and provided an accurate position display, communication and operation report to and from mobile machines. The hardware included a rough duty mobile mounted PC and position locator specially designed for mining.

Craig *et al.* (1999) discussed aspects of civil rail tunnels highlighting the need for train control systems to prevent rail collisions through the use of communication systems and the use of visual displays of the train's positions between drivers and the control room.

Stocks (1992) postulated that automated systems could significantly change the economics of underground truck haulage as an automated system eliminates the use of operators. Companies such as INCO were developing surface to underground systems to operate mobile and stationary equipment, and expected to have extensive automation and centralised control of equipment.

3.2 Horizontal transportation

3.2.1 Trackbound transport

Current gold mines utilise trackbound transportation as the preferred option and it is envisaged to remain as such for ultra deep level mining operations. However, a possible variation to current systems could take place, as described by Bennets (1994). One possible solution is for personnel, material and rock transport to operate along a ring track system joined by a trackless system moving out from the ring to the stoping and development ends. Bennets quoted M. Bailey from the University of Witwatersrand, as stating that heavier gauge rail systems running up to 50 km/hr would be possible on main routes. Also, transportation of material could be integrated into the system for handling stores, e.g. monorails that could guarantee delivery to the workplace within 30 minutes.

Bailely (quoted in Bennets, 1994) listed advantages of rail as:

- Increased discipline due to fixed routes.
- High-mass capacities.
- Less maintenance and better energy efficiencies.

Disadvantages as:

- Problem of encountering single point failure.
- Difficulty in negotiating steep grades.
- Expensive infrastructure.

Maslen and Cousins (1999) identified further advantages of track-bound transport as:

- Flexibility – being able to transport personnel, material and rock.
- Low power to load ratio.
- Minimum haulage dimensions required.
- Lower heat output than trackless vehicles.
- No pollution if battery powered or trolley locomotives used.
- Readily available – local manufacturing predominates.
- Defined travel path – can be fully automated.

Trackbound transport is advantageous in that it can be sent to all points in the mine required and can be adapted within a wide range to the demand, thus ensuring a high degree of flexibility (Sanders, 1978).

The speed in which underground trains travel at is an issue often raised by mine engineers. Baker (1981) noted that speeds on tracks should be assessed in relative terms, 40 km/hr is high-speed on a good track, whereas 6 km would be fast on poor track and at adverse gradients.

Personnel riding at high average speed for the whole journey from the mine entry to the face was identified as desirable in order to maximise productive time. Typical rail transport systems had main haulage trains that wait until all the workers for a number of faces or headings have arrived before setting off on a journey. This would be interrupted at various points to allow smaller groups of workers to continue their journey to individual workplaces by secondary transport systems or by foot (Bristow, 1979).

Maslen (1999) stated that locomotives were classified according to their size, since ultimately it was the weight of the locomotive that will constrain its maximum tractive and braking effort. Two power systems were available, diesel and electric. Electric power is derived from either an overhead trolley line or a battery unit. Boast (1986) claimed that electrically powered equipment was preferred due to it being pollution free, quiet, easy to clean, reliable, easy to maintain, and often having a longer life span than diesel.

In terms of the underground trackbound transportation, Maslen observed that many studies had confirmed that poor safety and productivity could be attributed to three main areas:

- Track condition.
- Locomotive and rolling stock condition.
- Overall management system.

Maslen further noted that the South African mining industry's utilisation of the horizontal infrastructure averaged 28%. This poor utilisation was due to ineffective management structures, poor track conditions and a failure to operate transportation as a process. Problem areas requiring attention within any proposed solution needed to consider that:

- Transportation should be a value-adding process.
- Transportation required a focused and dedicated management system to ensure performance.
- Poor quality track increases maintenance and general operating costs.

Maslen (1999) also added that the condition of locomotives underground were extremely poor. Key factors responsible for this included:

- Operational environment.
- Standard of maintenance programs.
- Quality of operators and the extensive time spent underground before necessary engine overhauls.

Du Plessis *et al.* (1999) advocated the use of trackbound trains pulled by locomotives as the most suitable method for horizontal rock transportation for deep mines.

Ling (1999) described a train system pulled by an underground rope haulage system at a coal mine in Nova Scotia, Canada. The system was commissioned in October 1996. The tracks extended from surface at an inclination of 7,5° for 7 km. The rope was 15 km long and weighed 63 tons. The system was used to transport personnel and material, and could transport up to 208 workers at a speed of up to 15 km/hr. Disadvantages of the system included long delays during rope splicing operations; damage to the rope occurring at turns in the haulage and the inability of aligning the rope properly.

3.2.1.1 Battery locomotives

Baker (1981) identified that battery electric power was simple, reliable and operated over moderate distances. In addition, battery locomotives were easier to maintain than diesel locomotives. Battery supply accounted for 40% of underground locomotive transport in UK coal mines and operated up to distances of 6 km and at maximum speeds of 14 km/hr.

Boast (1986) described a typical steel wheeled 15-ton, 67 kW locomotive, high-speed, battery powered locomotive. This unit was built for high-speed personnel riding and ore haulage with an average travelling distance of up to 565 to 710 km per week. Personnel were transported at speeds of 24-32 km/hr and materials at speeds of 16 km/hr.

3.2.1.2 Trolley wire locomotives

Figure 3-1: 20-ton trolley locomotive (Goodman, 1999)

Trolley locomotives have a reputation for efficient operation, low maintenance, long life and being most suitable for large installations where high power and continuous operation are required. Initially costs are high, primarily due to the cost of fixed installations, i. e. transformers, rectifiers, overhead trolley-wire, etc., but operating costs are low. Northard (1976) commented that to achieve high-speeds over long distances, the use of electricity to supply the locomotive power was preferred.

At Gedling colliery, reported Boasts (1986) the train consisted of two, 13-ton locomotives (112,5 kW). Speeds of up to 40 km/hr on gradients of 1 in 22 were reached. The overhead line voltage was 500V using the rail as a return conductor. A 3,3 kV AC supply was rectified and transformed to 500V DC to feed the line. Two pantographs were used to minimise sparking and an overspeed device operated from magnets buried beneath the track.

Baker (1981) reported that trolley wire locomotives were advantageous due to their tractive effort, capability to negotiate steep gradients, and ability to reach great speeds. As a result, trolley locomotives are suitable for high-speed underground transport applications. The systems however, require large cross section roadways to safely accommodate wires and pantograph assemblies. At Mount Isa, Australia, 600-ton ore trains were hauled at 40 km/hr by trolley locomotives. Backup power systems were identified as being important in the case of power failures. In the USA, combination battery and trolley wire locomotives were often employed.

3.2.1.3 Diesel locomotives

It has been established that rail transport in the mining industry has moved towards using electrical instead of diesel powered locomotives, due to costs, easy use, etc., (Boast, 1986). Diesel locomotives, however, still have their place and there are many duties for which they are the only suitable vehicles. They are also the only locomotive type that can be fully flameproofed, and are predominately clean burning with advance flame trapping and cooling systems. Boast reported of a diesel haulage system in the Philippines at Philex Consolidated Copper Mine, which hauled 1200 tons of ore per trip at speeds of up to 20 km/hr. The system used four 25-ton diesel locomotives spaced down the train, which communicated via single wire operating lights in each cabin. The system operated continuously, having a single shift per week for maintenance, cleaning of the roadway, etc.

3.2.1.4 Non-conventional locomotives

Zhuwakinyu (2003) describes fuel cell technology as combining hydrogen and oxygen using platinum or another PGM as a catalyst to produce energy in a chemical reaction. Unlike a battery, a fuel cell does not run down or require charging, and will continue to produce energy as long as hydrogen fuel is supplied.

Zhuwakinyu quotes Kevin Moxham, Lonmin Platinum's group mechanisation and automation manager as stating "one pillar of (Lonmin's) mechanisation and automation strategy is the use of alternative power sources, including fuel cells. Trials on a locomotive are planned to begin next year (2004), and the idea will be to learn and expose ourselves to this new technology."

Baker (1981) commented that compressed air locomotives were inefficient and had limited range of operation. A West German company developed a hydrogen-powered vehicle and claimed that the technique could be applied to mine vehicles in the future. From a mining point of view, hydrogen power has the advantage of leaving no pollutants in the exhaust gas.

Captive rail systems were used extensively in secondary transportation of personnel and materials in coal mines and could negotiate gradients of up to 1 in 3, as at Selby Mine in the UK (Baker, 1981). These systems used tyres made of polyurethane, which enabled them to negotiate gradients of 1 in 7 provided the length of the steep sections were limited. Tyre wear was reported to be a major problem but was reduced when large driving wheels were utilised.

3.2.1.5 Technical improvements to tracks and transport systems

In 2000, the Department of Minerals and Energy (DME) assembled a guideline for the compilation of a mandatory code of practice for underground railbound transportation equipment. The guideline was created to address the safety concerns of underground transportation. A tripartite task group was established under the auspices of the Mining Regulatory Advisory Committee (MRAC) to revise the existing DME guideline for underground railbound transport. This document enabled mining houses to compile manuals for the construction and maintenance of underground rail track.

Baker (1981) reported that efficient braking was essential for the safe operation of all underground rail transport systems, particularly for high-speed locomotives. All braking systems should be fail-safe with braking systems utilising pneumatic, hydraulic, electrical or spring powered brakes. In addition, considerable interest was being shown in brakes that apply directly to the track rather than to the wheels of the locomotive or rolling stock.

Boast (1986) noted that there had been improvements in the UK mining rail transportation. These included fail safe brakes, improved suspension systems, finger tip controls, fire hazard reduction and dual lined compartments.

Battery improvements included cells made from flame retardant, easy to clean polypropylene material and joints that prevented acid leakage. Developments to sodium/sulphur batteries had led to considerable improvements in the ratios of energy to weight volume. Compared with a lead/acid battery, the same electrical energy could be drawn from a sodium/sulphur battery with one seventh of the mass and one third of the volume. However, it was noted a major drawback with this type of battery was that the cells had to be maintained at over 300°C so as to ensure that the products of the chemical reaction in the cell remained liquid at all states of charge (Baker, 1981).

3.2.1.6 Track construction

Track construction plays an important role in terms of the duty of the haulage, the speed of the locomotion, and the overall operating costs of trackbound transportation Maslen (1997) commented that main haulage construction should be the responsibility of highly trained teams and specifically managed. An example of this was evident if one viewed the track installation between Harmony's Saaiplaas' No. 4 and No. 5 shafts (Masimong gold mine). The rail construction was overseen and managed by personnel who were trained by Spoornet. The track had been kept in excellent condition for a number of years, while servicing a high-speed trolley locomotive transport system.

The Chamber of Mines (1984) stated that track structure could be divided into the following components:

- Bottom formation.
- Ballast.
- Sleepers and sleeper fastenings.
- Rail and fishplates.

Each was interdependent of the other and, like a chain; the overall strength of the system was that of the weakest link.

Maslen (1999) described a conventional track as consisting of two rails supported by sleepers that had been placed on top of a solid footwall foundation. The tracks were made from 9m lengths of rail, which were joined together by welding or fishplates. Other track types included concrete tracks and tubular tracks.

Concrete supported tracks involve the placement of rails within a concrete structure that maintains alignment (vertical and horizontal) and support. This type of track construction is more costly than conventional ballasted track, but has lower maintenance requirements with several advantages. Maslen found that concrete tracks are well suited to environments where an excessive amount of water is present and which could negatively affect a ballasted track. Concrete tracks require minimal maintenance and is further benefited by the placement structure, as the underground environment is extremely corrosive and placing the tracks in concrete reduced corrosion of the tracks.

Tubular tracks provide continuous longitudinal support underneath the rail flange and has a drain section installed between the rails. However, tubular and concrete tracks have no elastic adjustment medium (ballast), which is a necessity in areas that are subject to footwall movement.

Grinaker Duraset claimed to have developed a sleeper that could increase rail efficiencies and lowered costs (Bennets, 1994). The Duraset sleeper features a built in cant of 1 in 20 utilising self-locking fastenings (Pandrol clips) with clamping force of 0,5 tons on a 30 Kg rail. The clips are mounted on top of the rail to minimise contact with corrosive mine water. Duraset also manufactured a sleeper that is positioned longitudinally along the track. This optimises the stress concentration in the sleepers, reducing the number of sleepers required and improving rail sleeper contact.

Ebersohn and Visser (1990) discussed the problems related to rail transport in a typical deep (2000 m – 3000 m) level gold mine and provided details of proper track construction methods. It was pointed out that the cost of tramming was directly related to the quality of the design and construction of the track structure. The authors concluded that the correct design and construction of a proper support and drainage system would ensure a more reliable track. It was stressed that the transport systems at a mine should be considered as an entity, which could be planned, financed, constructed and operated to the benefit of the whole mine.

3.2.1.7 Track and rolling stock maintenance

Bennets (1994) quoted Murray Franz, the director of the Underground Rail Association (URA) of RSA in 1991, as stating that, due to the harsh conditions underground, it is uneconomical to constantly maintain the tracks in first class condition. Generally, there is very little time available in the work cycle for such maintenance. Thus, installation of this class of track must be done correctly during the initial installation. Life expectancies of locomotives and mine cars, of which about 10% are replaced annually, is about seven years, while wheels and bearings have a life as short as six months. Track conditions are unacceptable at many mines, mainly, because of uncontrolled water flow (which affects the sub-grade ballast), poor track standards (particularly at joints which cannot accommodate the loads imposed on them) and general wear and tear.

Murray stated that track standards could be divided into three classes.

- Class A, complying with high standards in geometry, is designed to accommodate the loads imposed and has welded joints, as well as a proper maintenance programme.
- Class B is constructed according to high standards but with ordinary fishplate joints, and is generally poorly maintained.
- Class C does not comply with any specific standard and has poor geometry and sub-standard joints.

A class "A" track need not be maintained at 100% standard but it must not be allowed to fall below 80% in standard. Sub-standard joints were a major cause of track degeneration. They reduced tramming speeds significantly, created unsafe working conditions and were the cause of numerous derailments. High-quality fishplates, correctly matched to the rail type, should be selected and correctly torqued fishplate bolts used and maintained regularly. High duty tracks should be welded to eliminate the need for joint maintenance.

3.2.2 Monorails

Jagger (1999) reported that diesel driven monorail systems were used extensively in the coal mines of Europe for underground material transport Diesel monorails have been readily applied in flammable gas environments due to their flameproof nature and electrically powered monorail systems have been specially developed for application in hard rock mines. Possible future uses of the system could include remote control of the units combined with video cameras, the development of various mining utility vehicles adapted for use with the monorail, and the use of the monorail in stoping operations.

Bennets (1994) reported on the monorail tested at President Steyn Gold Mine. The unit had been used for material transportation and development operations. Advantages of the machine included high transporting and loading efficiency. The system was ideal for use in areas with small cross sections and was suitable for negotiating dips and slopes up to 45° and turns 4,0 m in horizontal radius.

The monorail track was suspended from chains securely anchored and grouted into the roof, using 25 mm diameter, 2,5 m long expansion bolts. Track sections allowed for horizontal and vertical movement, permitting the adaptation of the rail track to roof contours over extended distances.

Becker (1993) detailed the monorail specifications as used at President Steyn Gold Mine. The unit was modified for development purposes, so the rail sections used were shorter and heavier then normal.

- Length of rail: 1980 mm.
- Weight of rail: 300 kg.
- Roofbolts per rail: 2.
- Installation time per rail: 1h 30 min.
- Max speed: 7,2 km/hr.

Gradients up to 12° are suitable for the friction wheel drive however, for inclinations greater than this, the friction wheel experiences excessive wear and the rack and pinion drive unit should be considered. The rail could withstand vertical and horizontal deflections of up to 3° and up to 30 m of rail could be installed daily. Installation costs were estimated by Becker (1999) to be:

- For gradients up to 12°: ± R800/m (friction drive).
- For higher gradients: ± R1600/m (rack and pinion drive).

Baker (1981) stated that the capital and operating costs of a monorail system were competitive with conventional haulage systems, but they had lower capacities for bulk material transportation. They were most frequently used for the transport of personnel and materials.

Guse and Weibezahn (1997) discussed the application and advantages of the monorail system in hard rock mines. The electrical system required substations every 800 m. Inadequacies with the current system for material transport were highlighted and it was suggested that inventory levels for underground could be reduced if a 'just in time' delivery system was utilised. Monorails could easily be integrated with existing transport systems. The monorail modules were less than 2,5 m in length and 2,5 tons in weight, which allowed them to fit into shaft cages.

Simplified maintenance on the monorail was achieved by making most of the parts easily accessible and concentrating the 'high-tech' components in exchangeable modules. The monorail system was designed to transport up to 120 workers at speeds of up to 4 m/s.

3.2.3 Conveyors

Conveyors are becoming popular in South African mines and offer the flexibility of combining personnel and rock transportation. A diagram showing a personnel/rock conveyor is shown in Figure 3-2. Presently, such belts are in operation at Target Gold Mine, East Rand Proprietary Mines Gold Mine, Tshikondeni Coal Mine, Lavino Chrome Mine, and the Bafokeng-Rasimone Platinum Mine.

Belt conveying is considered an efficient means to transport rock over short to medium distances and can operate at steeper gradients than trackbound transport. Conveyors are flexible in their capabilities for receiving rock from one or more locations and are generally considered safe and reliable with a wide range of capacities.

MacNulty (1999) reported that, in normal operation, belts could run at grades of up to 22°, and at maximum speeds of 2,5 m/s (9 km/hr) for personnel transportation and 5,0 m/s for rock transportation. In comparison, the angle of limitation for tracked and trackless vehicles is approximately 2° and 12° respectively with greater transporting speeds. It was commented that conveyors have the disadvantage of not being able to transport material and required additional systems such as a monorail or trackless transport to move material.

MacNulty listed support methods for conveyors as:

- Attached to the sidewall with knee brackets.
- Supported on the footwall using stands.
- Supported from the hangingwall using roofbolts and chains.
- Combinations of the above.

It was concluded that generally, roof supported conveyors suffered less damage from mining vehicles, but the conveyors were more difficult to maintain.

Conveyors generate considerable quantities of heat and this heat is dissipated into the atmosphere. Baker (1981) estimated the heat output as 30% higher than that from a locomotive system. Associated with the heat is the risk of fire, in 1981, 15% of all underground coal mine fires in the USA were a result of ignition of belt conveyors. For underground conveyors, their fire resistance characteristics are given preference to their cut, gorge and abrasive resistance.

Advantages quoted by Baker (1981) of conveyor systems over conventional locomotive/rope haulage systems included:

- Little additional equipment was needed where conveyors are used for rock transport.
- Conveyor personnel riding could be used where the installation of other means of personnel riding was prohibited.
- Personnel were transported individually and the hazards associated with crowding workers at riding stations were avoided.

MacNulty (1999) listed some advantages of conveyor belt transport as being:

- Specific energy consumption less than other haulage systems.
- Operating cost and skilled labour costs reduced.
- System could be automated.

  Disadvantages included:

- High initial capital investment.
- Lack of ability to negotiate curves and inclinations greater than the angle of repose of the material being conveyed.
- Poor flexibility (generally applicable to permanent installations).
- Sensitivity to large particle size and abrasive material.
- Breakdown results in a 'standstill' of the conveyor system.
- Directional changes require tipping and loading facilities.
- Fire risk

Hughes (1997) reported on the design considerations for the installation of the decline personnel/rock conveyors installed at Target Gold Mine. Due to the nature of the orebody mining is concentrated far from the shaft. This led to the decision to use a conveyor for the transportation of ore; later it was decided to use the conveyor for personnel transportation as well. The initial design was for an 1800 m conveyor at an angle of 8,5°, running at a speed of 2,5 m/s. Adjacent to the conveyor, it was decided to install a monorail system for bulk material transportation. Track and trackless transport was considered, but the monorail system was chosen due to the steep inclination of the haulage and the probable damage to the conveyor installation from trackless vehicles. Provisions implemented to facilitate personnel riding on the conveyor belt included:

- Boarding and alighting platforms.
- Additional clearance between top and bottom belts.
- Brakes to prevent runaway belts.
- A personnel barrier above the tail pulley.
- Additional slipping of the hangingwall and sidewall to provide adequate clearance for riders.
- Belt slip, rip and misalignment detectors.
- Chute blockage trips.
- Additional lights, alarms, signals and notices at boarding and alighting platforms.

Particular attention was also paid to:

- The belts acceptance by the workforce.
- Training facilities.
- Misuse of the system.
- Riding on the bottom belt.
- Maintenance and supervision.

The implementation of the "man riding" belts has contributed to the quick access of personnel to the work face and to the productivity of the mine. It is believed that a significant cost saving was realised compared to the use of a dedicated personnel carrier or chairlifts.

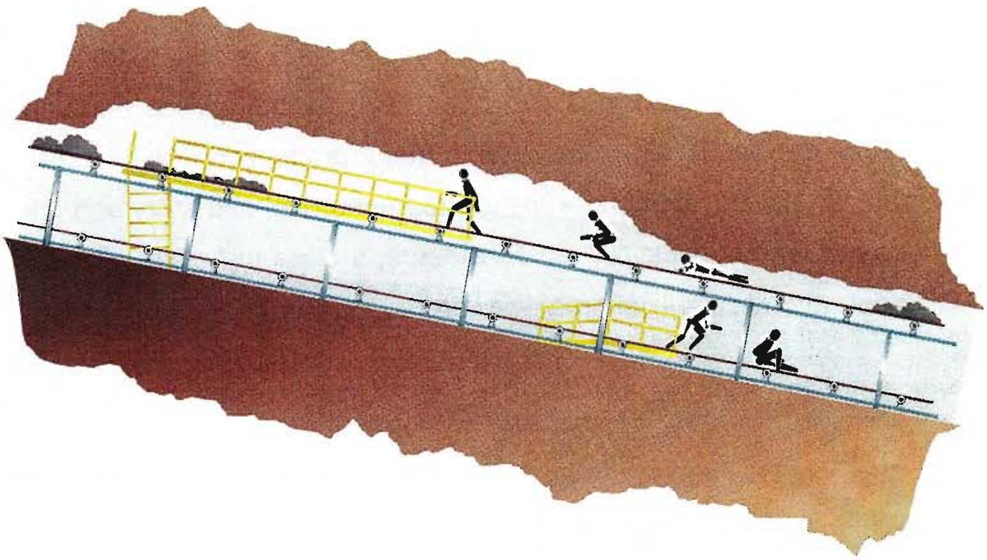


Figure 3-2: Typical boarding and alighting arrangement (Hughes, 1999).

The specifications of Target's personnel and rock belt included:

Table 3-1: Belt conveyor specifications (Hughes, 1999).

DESCRIPTION	DATA
Length; Lift; Gradient; Speed	1 800 m; 266 m; 8.5°; 2.5 m/s
Belt width; Type	900 mm; ST1250 F (fire retardant)
Material duty	Waste rock and gold ore at 225 tons/hr (3.75 tons/min)
Men duty (top and bottom belts).	30 persons/min (2.25 tons/min based on 75 kg/person at 5 m intervals)
Motor and gearbox	Two 160 kW, 525 V AC induction motors, through 26:1 reducers
Drive configuration	Tandem drives, shaft mounted on return pulley.
Brakes	Two electro-hydraulic brakes with 800 mm diameter disc
Pulleys	Lagged, crowned at edges
Electrical control	Twin 525 V variable speed thyristor convertor drives with vector controlled IGBT's
Take-up and tensioning	Sinking: Winch take-up at top end Permanent: Gravity tower take-up at tail end
Idler configuration, top	3 roll, 35^{o} trough, series 25, 127 mm diameter, spaced at 2 m
Idler configuration, bottom	2 roll, 10^{o} 'V' return, 127 mm diameter, spaced at 2 m

Leonard (1997) commented on the problems experienced with a conveyor belt installed for East Rand Proprietary Mines. The mine had installed a maximum tensile strength fabric belt on the rock-carrying conveyor instead of a steel core belt. It was believed that conditions in the decline during development would not be suitable for steel core splicing. However, the splices made in the fabric belt broke frequently, which resulted in major production delays. The proper splicing techniques only became evident after numerous laboratory and on-site tests. The lesson learned was: "When considering a class of belting off the standard for an application in a critical area --beware!"

3.2.4 Chairlifts

Chairlifts (Figure 3-3) allow for efficient and cost effective transportation of the workforce over short fixed distances, i.e. down inclined shafts. The riding chairs are either permanently attached, or simply hooked on and off a cable, which is guided by regularly spaced roller stations. The cable forms a loop with a drive station at one end and a return station with a tensioning tower at the other end. Embarking and disembarking stations are situated at the drive and return station and at specific points along the route.

The literature review provided little information in regard to chairlifts. Information is limited to discussions held equipment suppliers, namely DBT/Scharf and Walter Becker.

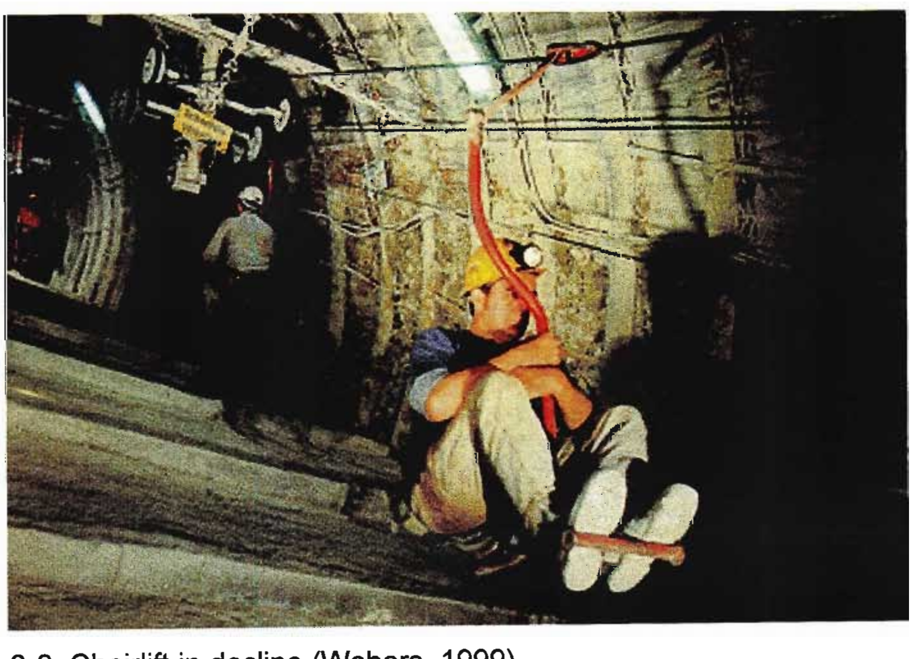


Figure 3-3: Chairlift in decline (Webers, 1999)

Technical specifications for chairlifts are as follows:

Table 3-2: Chairlift specifications.

Description	Data
Manufacturing and installation costs	R1200/m
Speeds	Permanent chairs 1.5 m/s Temporary chairs 2.5 m/s
Transportation capacity	900 workers/hr
Distance (max)	5 Km
Gradients	Permanent chairs 0 – 45° Temporary chairs 0 – 18°
Horizontal curves	Permanent chairs – no curves allowed Temporary chairs 0 – 120° turns

The chairlift is limited to personnel transportation and is usually associated with areas serviced by inclined shafts that transport material and rock.

3.2.5 Trackless vehicles

Dreyer (2001) reported that the Kiruna Electric Truck (35 and 50 ton capacities) is used to transport uncrushed ore from the mining levels up an incline ramp to a surface tip. The truck operates off a three phase overhead line but is battery powered at loading and tipping stations with the pantograph engaging/disengaging automatically under instruction of transponders installed at specific points. Two variable speed 690 V AC motors drive the front and rear axles respectively. High reliability results from the integration of well-proven systems. Parking/emergency brakes are spring applied, hydraulically released.

The major advantages experienced by Kiruna with this technology over diesel driven trucks are:

- Higher payload-speed combinations achievable.
- Reduced investment in ventilation as no emissions are produced.
- Reduced operating costs for same hauling capacity.
- Long lifetime and minimum maintenance required.
- Improved working environment due to less noise and no exhaust fumes.

Bennetts (1994) quoted M. Bailey from the University of Witwatersrand who noted trackless system advantages as having high flexibility, with the ability to access difficult areas, being ideal for short distances and having a large variety of specialised equipment. Bennets also quoted G. Suelhoff of IMG Engineering in Germany who stated that "the introduction of trackless vehicles to all areas of mining had considerably boosted production on a world-wide scale, had cut accident rates and had demonstrated outstanding rationalisation potential in the infrastructure concerned with handling of personnel and materials". Suelhoff commented that operating costs and purchase price must be realistic and that component supply and ease of maintenance were important factors. He further stated that safety considerations were important, i.e. twin tyres should not be used on large high-speed vehicles due to of the increased accident risk. Disadvantages included lack of discipline, high-energy consumption, high maintenance costs compared to rail, limited axle capacity and difficult traffic management.

Baker (1981) noted that the replacement of worn tyres represented a considerable proportion of the running cost of trackless vehicles. Good roadway floor surfaces were key to the successful operation of trackless vehicles. Uneven, soft and muddy floors presented severe hazards. Good drainage could improve surface conditions but some kind of roadway surfacing may be necessary. Many substances were being tested to provide such surfacing; the French coal industry had developed a surface of gravel and ash bound together with a bituminous emulsion. Experiments on cement and natural floor material were also conducted.

Bristow (1979) found that trackless vehicles for the transport of personnel and materials underground offered a number of potential advantages over more conventional tracked methods of transport such as railways and monorails. Considerations were manoeuvrability and passing ability and the potential reduction in initial capital investment from absence of track laying costs. Gradient was a factor that had always limited rail transport underground necessitating rack systems or trapped rail captivation for steep sections.

Bristow noted that there were constraints on the use of trackless vehicles. Poor riding at moderate speeds, stability and control problems at high-speeds mitigate against the use of these vehicles for long journeys. Low payloads per vehicle and relative high-energy consumption made railway trains attractive by comparison.

Floor maintenance problems were critical, particularly at junctions and turns. For personnel riding and material transport at moderate speeds, better floors were required for reasonable ride comfort. At higher speeds, very good floors were necessary for stability and to ensure adequate guidance control to be exercised. In addition, good surfaces were essential if vehicle maintenance costs were to be kept low.

Bristow commented on two of the obvious limitations of most trackless vehicles. In order to provide transportation throughout the shift, one vehicle would probably remain idle during most of the shift resulting in a low utilisation. The second was the low efficiency of trackless vehicles when compared with rail vehicles. Railbound transport has a lower rolling resistance, and therefore a more efficient system in terms of energy consumption.

3.3 In-stope transportation

In-stope transportation has been traditionally limited to cleaning issues focused on improving face advance and productivity. Work conducted by the Chamber of Mines Research Organisation (COMRO) in the 1970s and 1980s focused on stope face conveying and water jetting. Face conveying was found to be applicable with mechanised mining and the reciprocating flight conveyor was implemented with the impact hammer mining trials conducted at Doornfontein and Kloof Gold Mines (Wilson *et al.*, 2000). In the 1980s, stope face cleaning assisted by water jetting was determined to increase cleaning rates (Morris, Bryne, Solomon and Holland, 1988) and since this finding has been commonly utilised to assist face cleaning. During this same period, the COMRO trialed and tested the use of continuous scrapers in the strike gully (Pickering, Morris, and Bryne, 1988). To date, the continuous scraper has found limited use in the South African gold mining industry.

Research into the area of in-stope logistics ceased with the restructuring of the COMRO. The interest into stope logistic only became revitalised with the development of the Deepmine Collaborative Research Programme where research was conducted on in-stope processes. This was followed by investigations into man transportation (Rupprecht and Williams, 2000(a)) and in-stope material handling (Wilson *et al.*, 2000). In 2000, novel cleaning methods were used to evaluate a stoping layout that removed stope orepasses from the mining layout (Rupprecht *et al.*, 2001).

The Futuremine Collaborative Research Programme continued to explore the in-stope process by researching cleaning methods (Rupprecht, 2002) and reviewing monowinches and monorails (Rupprecht and Leong, 2002).

Jochen (2002) reported that there was considerable interest in remotely controlled winches during the 1980's. Within the industry, the development thrust was driven mainly by Anglo American Technical Development Services. With the changes that took place in the industry during the 1990's, interest in remote controlled winches diminished. Recently, interest in remote controlled winches has been revived with the interest being extended to automatic controlled winches.

4. Horizontal transportation

Horizontal transportation commences from the shaft station and ends at the stope cross cut. Traditionally, horizontal transportation is inefficient in comparison to vertical transportation efficiencies and accounts for a high proportion of mine accidents. This area can improve significantly by applying appropriate management and scheduling and management systems. This chapter reviews the technologies applicable to horizontal transportation and their ability to meet the qualitative requirements for ultra deep mining.

4.1 General transportation issues

Transportation must be seen as a process, which includes the transportation of personnel and material, and which is not focused only on rock. Mine planners must consider logistics as important as any other discipline. For the most part, rock transportation is of paramount importance and the transportation of personnel and material a secondary consideration. Current technology is sufficient to support underground transportation systems, however this technology must be appropriately applied. New technology is often discarded as it initially fails to deliver its proposed benefits. Yet, on closer examination it is evident that such technology often fails due to human interferences, i.e. inadequate machine operating and maintenance skills, the unavailability of spare parts and insufficient time being given for the workforce to acquaint themselves with the technology.

A culture to move personnel, material and rock safely, quickly and efficiently must be created and this requires a mindset change of current managers, engineers and line supervisors. However, just having the right mindset is not enough as the actual layouts must be efficiently planned and equipped with the appropriate technology. Some layouts are poorly designed for transportation. In addition, the movement of the workforce must be linked to the shaft schedule with trains departing at set times to set destinations with management control systems in place that will allow the system to work.

4.1.1 Mine planning and simulation

Transportation is an expensive process, thus, there are compelling reasons to ensure that the time and effort spent on mine logistics is kept to a minimum. Few mines have detailed layouts indicating the transportation flow rates required for personnel, material and rock with layouts often having many legs making the distances to be travelled long and arduous. This is further complicated in that transportation's supervision, communication and maintenance systems are difficult to manage. The first priority must be to have an efficient transport system for the transportation of personnel, material or rock. When considering a mine design, the available face time, delivery of material and rock removal must be regarded as an essential factors as they directly affect the overall productivity of the mining method.

4.1.1.1 Mine Planning

Strategic planning is required to reduce the transportation constraints. This can be achieved through properly designed transportation systems in the main and secondary haulages. In order to achieve this, layouts should be evaluated in terms of:

- Speed.
- Capacity.
- Ease of installation and maintenance.
- Manpower requirements to install and maintain the transportation system.
- Cost and time to establish the transportation system.
- Maturity and availability of equipment.

Mine planning criteria include:

- Duty requirements of haulage (personnel, material, and rock).
- Gradient.
- Maximum speed.
- Type of loading and tipping arrangements.
- Track curvature.
- Life of haulage.
- Track conditions and geometry.
- Interference with other trains.
- Braking consideration.
- Geotechnical input.

The planning of curves is important to mine planners, as the radii of curve should account for the desired speed and size of the trains. Small radii curves do not accommodate large personnel carriages or high-speed trains. The main strike haulage must be straight; its position should be given preference over shorter cross cuts, thus the mine planner must beware of geotechnical inputs when positioning haulages. This geological information is important; as ideally, the mine planner should have geological information detailing the magnitude of geological disturbances, including the positioning of the reef horizon to minimise the turning of the main haulage for adjustments in reef position. Portions of the main haulage should be provided with turn outs so that high priority trains can have preference over low priority trains, without placing delays on the overall transportation system. At regular intervals, designated areas should be made in the haulage for personnel loading bays.

4.1.1.2 Simulation

The logistics involved in a large mining operation can be difficult to manage, schedule and plan. However, operations can be simulated; changes can then be made and evaluated to determine their impact on the real operation. Simulation tools have been used to determine transportation requirements. For example:

- The production capacity of the levels.
- The optimum allocation of locomotives per level.
- The infrastructure requirements for the required number of locomotives i.e. the number of tips, battery bays, personnel loading bays, and parking bays.
- Areas of constraints or bottlenecks.
- The benefits associated with the upgrading of tracks, installing double tracks, etc.

Some production problems faced in ultra deep level gold mines as a result of poor mine logistic planning include short face times, poor/low face advances, choked gullies, insufficient delivery of supplies and excessively long backlengths. Other problems include single track haulages with a high level of traffic, an inability to effectively control locomotive traffic through turnouts, and the absence of loops and double tracks around tipping points. In extreme cases, entire mining layouts have had to be changed, resulting in compatibility problems between the existing infrastructure and the new mining layout. Such errors jeopardise the feasibility and profitability of a mining operation at ultra depth.

Examples of simulation practices on mines:

- Mponeng Gold Mine has reviewed the movement of the development shift from the crush to the working place to identify the time required to transport the shift. This time study, combined with the simulation of the entire development process highlighted the importance of transportation in determining mining cycles.
- Kopanang Gold Mine and Great Noligwa Gold Mine have done simulation studies on major levels to optimise transportation and have implemented centralised transport dispatch systems on certain levels.
- Elandsrand Gold Mine has conducted extensive simulation studies were done to determine the optimum layout and transportation requirements for their deepening project.

Simulation studies are though generally not widely used on the mines as they are expensive and time consuming. However, advances in mine planning software and the integration of such software with simulation software, are making simulation studies more accessible to mines.

4.1.2 Transportation management structure

Management structures in which the responsibility of transportation management is shared between the mining and engineering disciplines (Figure 4-1) often result in poor control.

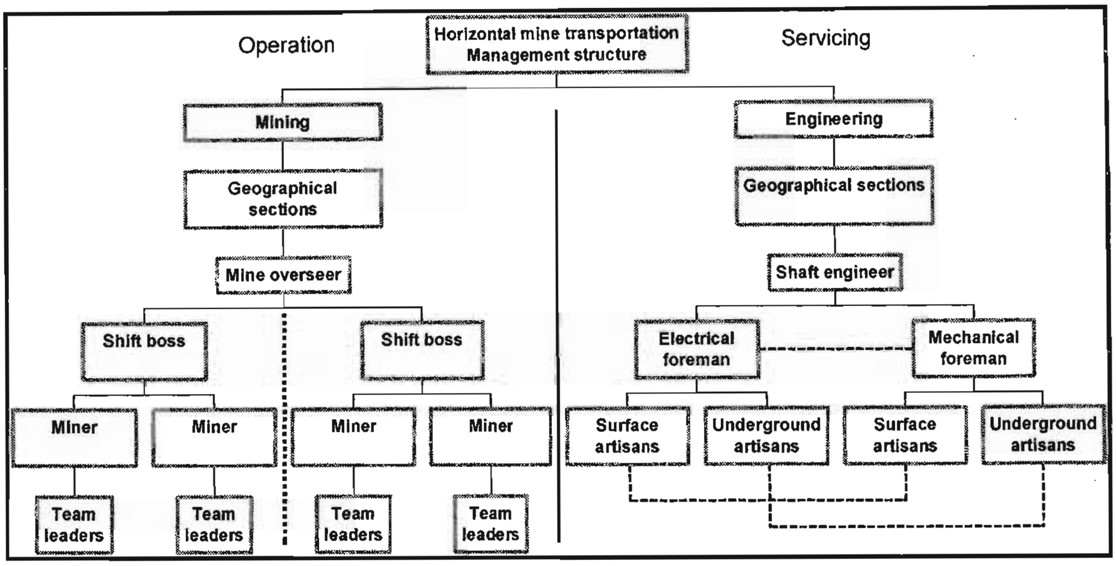


Figure 4-1: Traditional management structure (Maslen, 1997)

To alleviate these problems, a process orientated management structure is required to co-ordinate all factors associated with the transportation system, such as planning, management and maintenance. A processed orientated system (Figure 4-2) is controlled by one person rather than several, thus the transportation system is streamlined with duplicate equipment removed and levels of utilisation increased. Coupled with superior standards of track work, this management system has improved the utilisation and efficiencies of the transportation system and has reduced conflict between production and logistic personnel (Van Rensburg, 2000). This process orientated management structure has been put in place at a number of deep level mines, to name a few, Deelkraal, Driefontein, Elandsrand, Kloof, Mponeng, and Tau Tona Gold Mine.

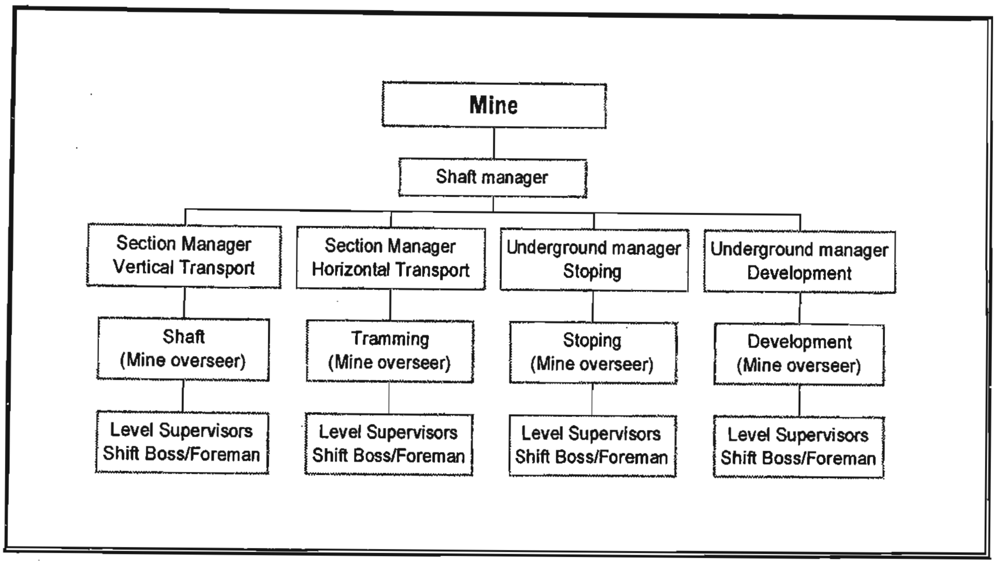


Figure 4-2: Process management structure (Maslen, 1997)

4.1.3 Communication

Radio communications in the underground environment have been under utilised for a long time. Communication reduces transportation requirements, as bottlenecks are prevented as drivers are informed whether to wait for on coming traffic or to proceed (Rupprecht *et al.*, 2002). However, mine layouts do not lend themselves to short wave radio communication systems, as these require direct line of sight to operate efficiently. The large rock mass and frequent bends in mine layouts prevent this; hence, most communication systems underground have been in the form of a closed telephone circuit system. Such a system is inflexible, as it does not allow for mobile communication systems. Locomotive drivers, shift bosses and other personnel wishing to communicate with other parts of the mine must go to a designated location where a telephone is installed. Locomotive drivers, particularly, are not in communication with each other, and, with the large number of trains in operation, delays in the tramming of personnel, materials and rock often occur due to trains meeting in the haulage travelling in opposite directions. Passing loops can be provided at regular intervals in the haulages to allow trains to pass, but, without an effective communication system, drivers have no way of knowing whether they need to stop and wait, or continue.

Future communication systems should be an integral part of the mining system making the tracking of personnel, equipment, and ore possible in most parts of the mine. Communication also enables interaction with the mines information system, for example, supervisors could carry hand held personal digital assistants (PDA's) and order stock or update plans immediately from underground. Future cap lamps should incorporate lighting with gas detection and tracking technology to actively interact with the communication system. It is estimated by the author that installing effective communication and asset management systems a productivity increase of up to 20% could be achieved.

4.1.3.1 Leaky feeder systems

Developed in the 1970s, the leaky feeder communication system is based on the use of coaxial cable that 'leaks' radio waves along its length such that a VHF or UHF radio signal can be transmitted up to 50 m away. The signal is fed in to the cable and propagates along it, continuously 'leaking' out a signal that other radios may receive. Similarly, transmitting radios can induce a signal in the leaky feeder cable. Amplifiers are inserted into the cable every 300 m to re-amplify the signals, which diminish due to their radiation along the length of the cable (Kononov, Lishman, and Abbot, 2002). Transmit and receive channels in the system occur at different frequencies.

The radio signals can be converted into electrical signals to communicate with monitoring equipment and on board computer systems. A typical cable installation can potentially handle 64 voice/data channels and 16 video channels. Unlike the hardwire (telephone) system, data can be transferred at any point along the cable making it possible to send and receive data from a moving train. Thus, this system is ideal for voice communication as hand held radios can be used to communicate to other radios in the system or to the telephone network.

Currently, leaky feeder systems are utilised in main haulages providing voice communication and low bandwidth data transmission along haulages. However, the possibility for further technological improvements exists and current trials are being conducted in terms of the ability to monitor personnel, material and ore movements; provide early evacuation warnings; enforce vehicle speed limits; control of ventilation doors and rail switches; and ensure the correct tipping of reef and waste.

The use of the leaky feeder system allows locomotive drivers to communicate with each other thereby improving tramming efficiencies. This type of communication system, while an improvement on the simple driver to driver network, still relies on driver input to determine the whereabouts and status of each train in the system.

A further improvement on this system then would be the introduction of a train monitoring system. These systems, known as "Supervisory Control And Data Acquisition" (SCADA), allow data about the train locations to be transmitted to the dispatcher's office via sensors located at strategic points. This allows the dispatcher to observe the status of any train in the transport network at any given time. Location details are passed to the dispatcher in real-time, allowing decisions to be made on data as it is received and modifications to the scheduling system can be made in order to operate around production problems such as chute blockages or derailments.

Once such a system is in operation, a further refinement is the introduction of a signalling system. Together with the SCADA system, the controller is able to communicate to the locomotives by means of signals, thus relieving the need for verbal communication with the drivers. The communication network and signalling system could be integrated into a complete traffic control system, allowing for the prioritisation of trains within the network.

4.1.4 Traffic control systems

Some means of traffic control is required for underground haulages. Many mines utilise a basic control system whereby the first train to enter a section has the right of way. Other mines utilise similar approaches where certain trains, i.e. trains with full hoppers or personnel carriages are given the right of way. However, as haulages become busier and multiple trains are in operation over a single track, a more sophisticated traffic control system is required. The use of radio communication between trains and a central control room is currently the most advanced form of traffic control underground. This system however, relies on the drivers adhering to the instructions given by the control room and thus the element of human error still exists.

The use of systems that monitor and control the ability of trains to enter designated areas by means of automatically stopping or derailing the locomotive removes the human element from the control system. These are ideal systems and are often used by surface operations to control trains. Robots issuing a red signal have switches positioned into turnouts so that the cleared track is given priority and any train ignoring the signal is switched off the main track.

4.1.4.1 24 Hour Clock

A generic, horizontal locomotive operation schedule, over 24 hours, for the transportation of personnel, material and rock is shown in Figure 4-3. The schedule shows: driver change, battery change and actual operation. As can be seen in the schedule, the personnel locomotive changes over to material upon completing the transportation of the workforce.

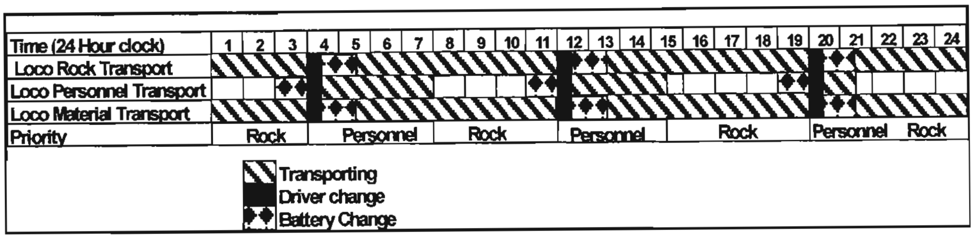


Figure 4-3: Horizontal locomotive operating schedule over 24 hours

The available time of the locomotive for transportation purposes is approximately 6 hours. This time is influenced by the time required to change the locomotive's battery and the time required for drivers to do locomotive pre and post shift inspections. Priority on the mode of transport during the shift depends on the particular mine's management philosophy. Generally, locomotives transporting rock are given higher priority, as indicated in Figure 4-3.

4.1.5 Automation

Ultimately, all transportation systems should move towards some form of automation. In the trackbound transportation structure, the use of a SCADA system in conjunction with a traffic control system allows the controller to observe each train in the system. With the automation of switches, the dispatcher has the ability to open or close switches as trains approach, and also to automatically divert trains into passing loops until a higher priority train has passed. Automation will reduce delays, specifically those caused by trains having to wait at switches or ventilation doors while they are manually opened or closed.

A further refinement could be the fitting of electric locomotives with speed control mechanisms that automatically slow the locomotives at dangerous track sections, such as curves, reducing the risk of derailment due to inappropriate speeds. Such a system could be used in conjunction with sensors on the bogeys of the rolling stock to automatically shut down a train in the event of a derailment. This prevents the derailed rolling stock from being dragged long distances, which, in turn, will reduce damage to the track structure. In such a system, a driver will still be necessary, but only to put the train in motion or to perform shunting operations. All other operations could be carried out by the traffic control system. Once the train is set in motion, the driver would have no further control except to stop the train in the event of an emergency.

Although the above examples are based on trackbound locomotives, automation could also be applied to other systems. Monorails could operate using the same SCADA and traffic control system as used by the track system. Many conveyor systems are already automated proving the benefits of automation, such as dispatch control and early warning systems.

4.1.5.1 Automated ventilation doors and switches

The use of remote control to operate ventilation doors should improve the flow of transportation, as trains would only be required to slow down while passing through the doors. Currently locomotives are required to stop before and after passing through the ventilation doors to manually operate them. The remote control devices could be attached to the drivers cap (Figure 4-4) or attached to the locomotive.

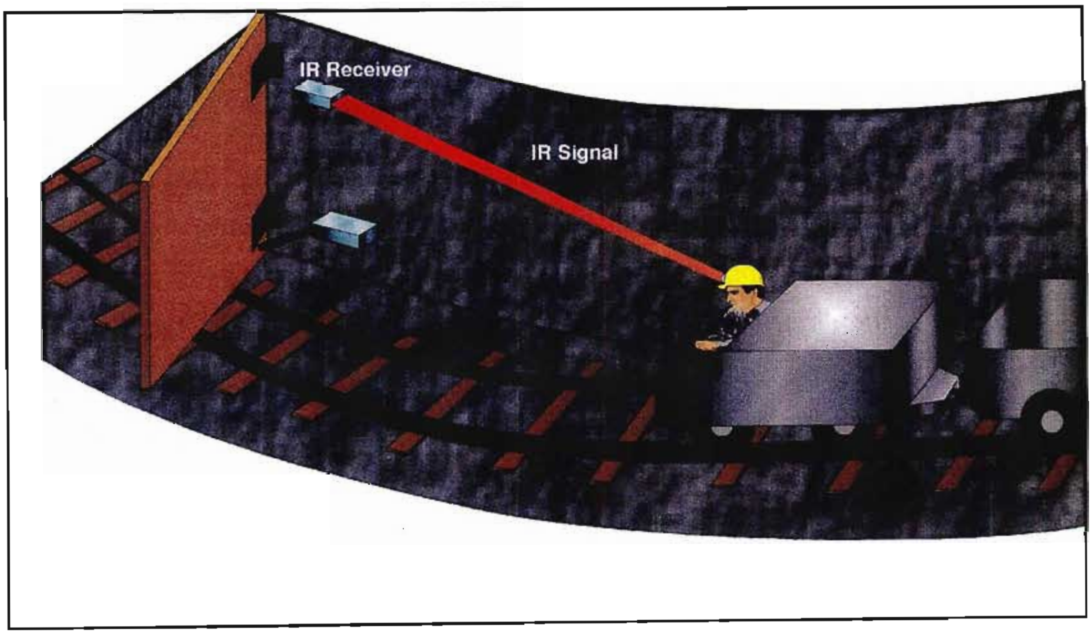


Figure 4-4: Remote controlled ventilation door (Kononov, 2000)

The use of remote control could improve safety, as drivers have been known to leave a slow moving train to open and close ventilation doors or to leave doors open during busy tramming periods. The use of remote control would also advocate the removal of stop block or skid sprags in the area of the ventilation doors.

Communication and automated systems will lead to significant changes to underground transportation and it is essential that any transportation system utilised in an ultra deep mining operation is equipped with at least a controller operated communication network. Therefore, an effective communication system with fully integrated communication networks, SCADA, signalling systems and traffic control is an absolute must.

The benefits of such a system would include:

- Maximum availability and utilisation of equipment.
- Minimum exposure of personnel to harsh environments.
- Reduced operating staff.
- Minimum abuse of equipment i.e. equipment programmed to operate within set limits.

Although the initial installation cost may be high, the running costs could be reduced considerably, ultimately having a greater effect on the profitability of the mine.

4.1.6 Illumination of haulages

Better lighting improves haulage conditions as problem areas can easily be identified. All junctions, ventilation doors, loading areas, and places where personnel congregate should be illuminated. Illumination systems should be systematically installed to a high standard on all levels. High-speed haulages should be well illuminated to improve visibility and account for the increased braking distances. In the case areas where illumination is not feasible, haulage sidewalls should be white washed to improve visibility.

4.1.7 Track work and track maintenance

In order to facilitate an efficient transportation system the track work of the haulage must be of a class suitable to match the tramming duty required. Track installation and maintenance is an important aspect of lateral transportation. One illustration of the importance of track work is Kopanang's experience where initial improvements to their 64 level tramming were not forthcoming after the initial installation of a dispatch system. Until the level's tracks were extensively upgraded up to 160 derailments (Figure 4-4) occurred monthly (Maree, 2000).

Figure 4-5: Hopper derailment

Track maintenance can either be carried out by mine personnel or outsourced to contractors. Mine personnel tend to be less qualified then contractors, as contractors employ experienced track maintenance workers. Benefits of using contractors include that a contractor can be called in when required and leave when the job is complete. Another option is to have tracks installed by contractors and then utilise mine personnel to perform maintenance. Whatever system is chosen, a system to quantify and qualify the work done must be in place. For example, Kopanang employs a quantity and quality surveyor to confirm work completed by the contractor before any payment is made.

By installing tracks initially to high a standard it is possible to minimise the amount of maintenance. For example, Moab constructs its haulages based on “A” class track installation with the surrounding walls supported with shotcrete, and drains installed with berm walls. Thus, the haulages are equipped in such a manner to support a track maintenance crew based on 1 worker for every 2 km of track. This exceeds the current industry ratios for haulage maintenance range from 1 worker per 500 m to 1000 m of track. In other instances, the ratios haven't been identified and maintenance is done on an ad hoc basis.

Daily inspections also play an important role in maintaining good track work. For example, Middleburg Colliery utilises one worker whose duty it is to walk the entire length of track checking for any problems and to grease and clean all switches. Other mines prefer supervisors to walk to the working place so that haulage conditions can be informally checked on a regular basis.

4.2 Personnel transportation

4.2.1 Principles of personnel transportation

South African gold mines differ on the management and operation of the lateral transportation system for the transportation of the workforce. Some mines such as Driefontein, Tau Tona, and Tau Lekoa cease all other transportation activities, i.e. rock and material transportation, while the workforce is being transported. During this period, the tramming locomotives can do pre shift checks and battery changes, thus no significant amount of time is lost for the locomotive fleet.

Mines, such as Kopanang and Kloof, share personnel transportation with rock and material transportation. However, they stress the importance of having procedures in place in order to minimise congestion on the level. For example, by establishing parking bays for personnel carriages and the changing transportation crews on a "hot seat" basis.

Other mines believe that walking distances of two to three kilometres are not excessive and provide transportation only for exceptional cases. Elandsrand provides personnel carriages for development crews but the majority of the shift still walks to the working place. Mponeng believes that personnel carriages should not be utilised, as they will interfere with rock tramming. Furthermore, the storage of the personnel carriages is also perceived as a liability if parking bays are not available.

Designs for personnel transportation start in earnest on the station. The plan must allow for the flow of workers to the loading bay where personnel carriages are used to transport the shift to their workplace. Loading bays should be provided at the appropriate positions and sufficient area must be provided for the workers to queue while waiting for transport.

4.2.2 Personnel loading bays

Properly planned and constructed personnel loading bays are important, as they increase the efficiency of the overall transportation system. Loading bays also become more important as working places get further from the shaft, thus necessitating the use of carriages.

4.2.2.1 Practices

There are two basic types of personnel loading bays. The first type is used for main levels planned to be in use for a significant period and tend to be high quality installation. This type of loading bay is capable of handling a large number of workers. The loading platform level is ergonomically designed so that the floor of the platform and carriage are level with one another and openings in the barriers matching the openings of the personnel carriages. Personnel loading bays for major levels tend to be high quality installations. Figure 4-6 shows loading bays at Great Noligwa Gold Mine.

Figure 4-6: Personnel loading bay at Great Noligwa Gold Mine

The second type of personnel loading system is more informal. In these circumstances a queuing system may be applied where workers are allowed entrance to the loading area once the personnel trains have arrived. In this case the loading of the trains is done informally with a predetermined number of workers allowed into the loading bay and the personnel carriages filled on a first come first serve basis. Personnel loading on the shaft stations is often conducted in designated area carriages. Off loading of the personnel carriers is done as the train stops at the start of the stope cross cut, the waiting place or a short section of double track in the cross cut.

4.2.3 Carriages

It has been identified that personnel carriages pulled by locomotives will be the preferred method of lateral transportation for the workforce (Rupprecht and Williams, 2001 (a)). The use of the carriage system only become necessary once travelling distances between the shaft and stope cross cuts exceeds 1,400 m to 1,800 m. Walking would be appropriate for shorter distances, as the period taken to spot and load the carriages would be longer than the actual time to walk to the working place. However, in certain situations it may be advantageous to restrict workers from travelling in the haulage, thus requiring the workforce to utilise a personnel carriage system. The transport system must ensure the continual flow of the workforce, as a crew should arrive as a unit and not in 'dribs and drabs'.

Personnel carriages are made to mine specifications and designs are available with a capacity from 12 to 120 workers. The use of a standing carriage offers greater capacity compared to seating carriages; a standing carriage with a capacity of 60 workers is the equivalent size of a seating carriage with a capacity of only 48 workers. The ergonomics and safety aspects of the design must be matched to the duty of the carriage. For long distance travel seating carriages are more comfortable then standing carriages.

An example of the capacity of a typical horizontal transportation system based on two 60 person carriage, a working place located 3 km from the shaft station with an average train speed of 10 km/hr. In this case, 120 workers could be transported per train per hour.

$$\begin{aligned}\text{Capacity of horizontal transportation} &= \text{Capacity of carriage x No. carriages X No. trips}\\ &= \text{60 persons x 2 carriages x 1 trip/hr}\\ &= \text{120 person per hour per train}\end{aligned}$$

Equation 4-1: Personnel capacity for horizontal transportation

Where:

$$\text{The time for one trips} = \frac{\text{distance}}{\text{speed}} + \text{shunting loco at loading} + \text{off loading point} + \frac{\text{distance}}{\text{speed}}$$

$$= \frac{3\text{km}}{10\text{km/hı}} + 5 \text{ minutes} + 5 \text{ minutes} + \frac{3\text{km}}{10\text{km/h}}$$

$$= 0.333 \text{ hours} + 0.083 + \text{o}.083 + 0.333$$

$$= 0.832 \text{ hours or } 50 \text{ minutes}$$

Equation 4-2: Time for one trip

and

$$\text{No. trips} = \frac{1 \text{ hour}}{50 \text{ minutes}} = 1.2 \text{ trips}$$

$$= 1.0 \text{ trips}$$

Equation 4-3: Number of trips per hour

To minimise time losses, the horizontal transportation schedule should tie into the shaft schedule. Ideally, the personnel train capacity should match the cage capacity (±120 workers per cage) and the train should depart immediately for the working places once the workforce has been loaded into the train. However, where a large number of personnel are to be transported to a working place far from the shaft it may be advantageous to utilise larger carriers and wait for the additional cage to fill the personnel carriages. Sufficient personnel trains should be available to transport the number of workers required for a mining area on a level. Typically, this would amount to approximately 225 to 300 workers requiring between two to three personnel trains depending upon the number and size of the personnel carriage.

Once the transportation of the workforce is completed, the locomotives can change batteries (if required) and proceed to carry out other duties such as the transportation of rock or material. While the locomotive is busy with other duties, the personnel carriages can be parked in dedicated parking bays until they are required at the end of shift, when the operation as described above is reversed.

Personnel transportation systems work well for the start of shift but are seldom as efficient at the end of the shift. Inefficiencies in the personnel transportation system, such as unreliable train schedules encourage workers to leave their workplaces early to ensure that they do not miss their cages. To overcome this problem personnel trains should run on strict and reliable schedules. Another factor is that workers finish their tasks at different times, i.e. the machine operators finish sooner then the charging up crew. This also results in personnel reporting to the shaft station at different times. Multi-skilling and new mining technology will ultimately overcome this problem by requiring smaller crews, which will make it easier to group the workers as a team throughout the shift. This in turn could facilitate the main haulages to be classified as a "no-go" zones during the shift, keeping the haulage free of unauthorized personnel.

4.2.3.1 Carrier Design

Personnel carriages should be equipped with:

- Reflectors at each end.
- A braking system acting on the wheels of the carrier that can be activated by the locomotive drive.
- Doors positioned on one side of the carriage, roller shutter or sliding, securable from the outside with an adequate entrance space.
- An emergency exit that can be opened by personnel within the carriage.
- Means for passengers to communicate with the driver or to stop the train in an emergency.

The design of the personnel carrier should ensure:

- Adequate ventilation, at least 20% of the side area should be of wire mesh.
- Standing height of at least 2,0 m with an aerial coverage of 0,12 m^2 per person.
- Capacity of the personnel carrier displayed on the outside of the carriage.

4.2.3.2 General transportation practices with carriages

Tau Tona Gold Mine utilises two 65-person carriages which travel up to 16 km/hr with personnel loading bays placed in the main cross cut of the longwall layout. During the transportation of the shift, all other transportation activities are halted.

At Kopanang Gold Mine, the majority of the workforce travel to their workplaces on foot. However, on the main production levels there are between three to six personnel carriers in use, each capable of seating 45 people. These trains are double headed, utilising a locomotive at each end of the train. This configuration reduces the transportation cycle time, as there is always a locomotive available to pull the carriages, thus no shunting of the locomotive is required. The personnel carriers operate from the station to positions and along the footwall drives but do not enter the cross cuts. Once these locomotives complete the transportation of the workforce, the locomotives exchange their batteries and then are utilised to transport explosives and material.

Although personnel trains can reach up to speeds of 16 km/hr, it is more important to focus on a well scheduled and managed system rather than a high-speed system. Kopanang has improved track work, procedures and communication to ensure the transportation system operates smoothly. A critical area identified by the mine is the end of shift procedure, where it has been identified that personnel carriers should be provided with parking bays and the change of the locomotive drivers should be done timeously preferably on the level and with the locomotives parked in the appropriate position.

Driefontein Gold Mine utilises carriages with capacities of 20 and 45 persons per carriages. Four personnel carriers per locomotive are utilised and up to two trips with two locomotives are required for busy levels. During the transportation of the shift, other tramming activities are stopped and the haulages are considered a "no-go" zone to pedestrians. At the end of the shift when personnel are leaving the work area late, it is permissible for the latecomers to travel in the haulages by foot. Informal loading bays are utilised on the station and at the waiting places. In order for the personnel carriers to be ready for the shift, the previous shift prepares the personnel train for the next shift. Once personnel transportation is complete, carriages are parked in redundant cross cuts or on the station in designated areas.

Utilising hoppers to transport both rock and personnel does away with the time lost to change from hoppers to personnel carriages as well as the problem of storing the carriages when not in use. Matjhabeng Gold Mine has experimented with the use of hoppers as a means to transport the workforce, utilising a stretcher as a floorboard and a net over the hopper as a protective cover. The hopper transportation system though requires quality track work to prevent hopper derailments and good worker discipline to maintain safety standards and for these reasons the hopper transportation system has not been put into practice.

Several mines are currently testing Anglogold's prototype of the "21'st century locomotive" where the train can be driven from either the locomotive directly or the caboose by remote control. Thus, the train can always be driven from the front end. This saves time in terms of shunting the locomotive. Another feature of the 21'st century locomotive is that the driver cabin and caboose are enclosed. Thus, there is a potential to utilise air conditioning for the train. This allows the train to potentially operate in return airways or in haulages with higher temperatures and by so doing, reduces the mines' cooling costs.

4.3 Material handling

4.3.1 Principles of material handling

Material handling is the movement and handling of goods from the point of supply to the point of use. The movement of materials can be completed as a single operation or can be part of a complex system in which the material is subjected to a number of operations or processes during its journey. The latter is the case for a typical mine.

Methods and equipment are available to handle the large variety of consumable materials and equipment used underground, through the systems of shafts, underground haulages and stopes. The important principles applicable to the handling of materials are described in the following sections. While some of these principles may appear obvious, it is the successful application and integration of these principles that will yield an overall efficient and effective system.

4.3.1.1 Fit for purpose design

Material handling systems should be specifically designed and specified to suit the needs of the final user and materials being transported. It should also be capable of handling all possible materials that must be transported. Hence, each application should be subject to site-specific evaluation and the equipment specified accordingly.

4.3.1.2 Standard loads and packaging

Handling of materials is greatly simplified if loads, or the majority of loads are standardised in terms of both size and packaging. Although mine material is diverse in both mass and size, the majority of loads handled can be standardised. Ideally packaging should suit final user requirements. For this reason all material handling and packaging systems should be designed from the stope panel backwards towards the surface supply point.

4.3.1.3 Handling of material

The handling of the materials should be efficient with the transferring of material kept to a minimum, as the transfer of the material requires additional energy and usually the use of labour. In general, the amount of handling is a function of the design of the material handling system, each part of which must be subject to value analysis. Where the transfer of material is necessary, it should be accomplished through horizontal movement on low friction systems or through overhead lifting devices.

The reduction in manual labour has a number of benefits with regard to cost and cycle time, as well as avoiding potential accidents. Where re-handling is necessary, this should be accomplished using appropriate equipment that requires minimum human effort and contact. The materials must be capable of being handled safely and in a form that will not be damaged during transportation.

Storage facilities require space and unnecessary stored materials can be considered wastage of capital earnings. Conversely, the lack of storage space can lead to congestion and a shortage of materials, which can cause production losses.

4.3.1.4 Organisation

The movement of the materials through the system must be completed in a manner that allows the supplies to reach their destination efficiently with the correct goods being delivered to the proper place on time. Scheduling is important, as material will normally be transported in the same haulage as rock and people. There are a number of computer based systems which can assist with the control and management of the system, however the capital cost of these systems is significant and the decision to use them should be based on a sound business case.

4.3.2 Shaft Station

Bank and shaft station designs should be closely matched and standardised. This greatly simplifies operation and enhances safety. The following measures should be considered for the effective and efficient handling of material cars on the shaft stations.

Double track should be provided immediately adjacent to the shaft in an amount that is not less than the product of the average length of a material car and the maximum number of cars, which are expected to be handled on one shift. For example, if cars are 3 m long, and 60 cars are handled on a shift, then 180 m of double track is necessary.

$$\begin{aligned} \text{Double track length} &= Car_{lebgth} \times Car_{number} \\ &= 3\text{ m} \quad \times 60 \text{ cars} \\ &= 180\text{ m} \end{aligned}$$

Equation 4-4: Length of double track on station

Some means of moving cars to and from the shaft must be provided. Capstan winches, purpose designed pushers and modified locomotives may be used for this purpose. The choice, design, configuration and operation of shaft safety stopping mechanisms must consider material transportation needs. This is especially the case with long material.

The provision of storage space must be done in such a manner that it does not impact on personnel or rock transportation operations. Material cars should not be stored in the tip cross cut, as this will hamper rock tramming activity.

4.3.3 Horizontal material transportation

Horizontal material transportation commences at the shaft and ends at the cross cuts or other working places such as development ends or service excavations. To ensure the maximum utilisation of the transport car and minimum congestion in the haulages and cross cuts, material cars should be off-loaded quickly with the cars being sent back to surface as soon as possible.

The design of any material transport system should start with the requirements in the stope panel and worked back to surface. Materials should be packaged on surface in a form suitable for handling. Support elements and other materials such as cement bags should be packaged to fit on the cars and in accordance with the requirements of the stoping panel. Items not suitable for direct packaging can be loaded onto pallets or containers that are also sized for the cars. Long items including pipes, rails and switches should be transported in specially designed cars or bogies.

Horizontal organisation should be arranged such that full and empty cars are effectively handled on one shift. This "material shift" should not coincide with main rock tramming activities. This will be possible to achieve provided sufficient storage space and appropriate unloading equipment is available at the end points (i.e. the cross cut).

4.3.4 Cross cut

The design of cross cuts is critical to the operation of the material transport system. Most cross cuts are required to handle rock loading, material and explosives as well as provide access to the stoping horizon. In addition, mining services such as electricity, water, compressed air and backfill converge at this point.

The provision of a properly designed material storage bay in the cross cut cannot be over emphasised. Good practice dictates that at least two days material storage capacity should be provided in cross cuts. This should not hamper other operations such as rock loading. The storage space can be provided on a separate spur line or in a prepared storage area. The latter minimises material car cycle time, but increases the frequency of re-handling.

Where monowinches or other overhead material transportation systems are used, these should be positioned to minimise the amount of subsequent re-handling required from the storage area.

4.4 Rock transportation

4.4.1 Principle of ore transportation

The transportation of ore plays a vital role in the mining process. The failure to remove the ore can create bottlenecks in the process, ultimately causing production delays and losses. Similarly, over design of the system results in poor utilisation of capital and resources. Ideally, ore transportation should be a just in time approach whereby the number of trains match the requirements of the production sections.

Ore transportation is predominantly trackbound, however a few mines do utilise conveyor belts and trackless equipment and one mine utilises the monorail in a specialised application. Trackbound operations are generally spread throughout the mine with trains removing ore from the stope orepass and transport it to the shaft orepass system via 30-ton trains. A train comprises of 6 to 10 hoppers filled to a capacity of approximately 3 tons per hopper.

4.4.2 Ore tramming control and automation

In the past, underground tramming operations fell under the control of the responsibility of the shift supervisor who managed not only the mining operations but also the tramming of ore. The outcome of this was that every section was equipped with its own fleet of locomotives, hoppers and material cars in order to satisfy the needs of the section. This lead to tramming systems that was inefficient and ineffective. The large number of locomotives caused severe congestion in the haulages, as trains competed with each other for access to the tramming routes. Often, locomotives stood idle due to the congestion emitting fumes and heat into the surrounding environment. Utilisation of locomotive fleets was normally very low when compared to other systems in the mine particularly hoisting installations.

Tramming operations are now operated in a process oriented manner as discussed in chapter 4.1.2. The locomotive fleet is reduced with the operations falling under the control of supervisors of the horizontal transportation network. In the past, poor maintenance led to more locomotives being unavailable, which in turn meant adding more locomotives to the fleet. This aggravated the situation and began a vicious circle. The new management structure reduces the amount of maintenance required which improves conditions as more productive time can be spent on preventative maintenance rather than time spent on breakdowns. The new structure also ensures that all equipment is used effectively to maximise utilisation, thereby reducing costs and preventing tramming related production losses.

A team capable of handing all spectrums of the transport system including management, operation, construction, maintenance and control is required. This team must be capable of working independently from the production sections while being able to interface fully with these sections.

It is almost impossible to be able to control anything successfully without being able to measure its performance. There are a number of areas in which measurements could be made to provide useful information and to assist with control of the transportation of ore. In order to break away from traditional methods it will be necessary to remove the interface of the miner. To do this it must be possible to measure the rock available in the mining sections. Some other useful measurements would be the location of locomotives, hoppers and other cars and the monitoring of the flow of ore through the rock handling system of the mine. If sufficient information is available, this information could be used to control and plan tramming operations and could be used to assist with automation through a centralised control system.

4.4.3 Orepass storage measurements

Quantities of rock stored in any orepass or silo can be estimated from the top of the structure. For automatic level measurements, it would be necessary to use devices using ultrasonic or laser measurement techniques. The most promising of these are laser measuring devices which use a laser head to transmit a beam of light on to the rock surface. The time taken for the beam of light to be reflected back from the surface can be used to calculate distance and hence give an estimation of capacity. One drawback of the laser measuring system is that it can only work in direct line of sight and is unable to measure around bends or in constantly dusty atmospheres.

4.4.4 Position monitoring

Monitoring of locomotives, hoppers is possible using electronic tags. A typical system would use an active or passive tag fitted to the equipment. When the tag passes a detector placed in a haulage the detector will register the presence of the tag and its associated equipment classification. Although monitoring is not continuous, direction and speed of travel can be estimated from its progress through a series of detectors.

4.4.4.1 Ore tracking

Traditionally, ore transportation has been monitored using metal numbered washers. The washers are thrown into the blasted ore as it is cleaned from the stope and later recovered via a magnet located on surface. This method is rather crude as the recovery rate is very poor.

The tracking of ore utilising modern radio frequency identification (RIFD) technology is currently being tested by a number of mines. Instead of washers, a encapsulated RFID tag is distributed manually within the ore located on the stope face. The tag is then monitored as it is moved through the mine transportation system thereby being utilised as a corrective monitoring tool. The system has the potential to prevent gold-bearing ore from being transported to the wrong rockpass system (i.e. the waste pass). The system can also be used to identify bottlenecks in the ore transportation system as reader will be able to identify when the tag has passed certain locations thereby provide real-time information on ore transportation.

4.4.5 Belt weighers

Belt weighers are used to give a measure of rock passing along the conveyor belt. They consist of a set of idlers connected to load cells and calibrated for the specific belt. A number of configurations are available, the more complex generally regarded as being the most accurate. Even the simplest weighers are capable of accuracies of within 5% when properly calibrated and maintained.

4.5 Horizontal transportation audit

The following systems, or combination of thereof, have been identified as having the potential to fulfill the requirements of a horizontal transportation system for an ultra deep mine environment. The systems are:

- Trackbound transportation.
- Conveyors.
- Monorails.
- Chairlifts.
- Trackless mining vehicles.

4.5.1 Audit of trackbound transportation

4.5.1.1 Overview

Trackbound transportation is the predominant method of underground horizontal transportation. This is because it offers several advantages over other transportation system, such as the ability to transport personnel, materials and rock using the same basic infrastructure, a low power to payload weight ratio, relatively small heat output and a defined path of travel. Furthermore, trackbound transportation is widely used and well understood in South Africa with the necessary equipment and spares readily available.

Trackbound transportation makes use of a self-propelled locomotive, which hauls rolling stock such as personnel carriages, material cars, and hoppers. This transport method allows the locomotive to be uncoupled from one transportation mode and change to another. This equipment operates on a defined track structure consisting of rails, sleepers and ballast, providing an aligned and supported roadway. While the trackbound system generally operates on the level, the use of incline hoists or locomotives with toothed wheels and track can allow this type of transportation to operate on inclines of up to 14°, or steeper if necessary.

4.5.1.2 Capacity

The capacity of the trackbound system is a function of the type of locomotives utilised, the size and number of cars/hoppers pulled, the allowable tramming speed, and the transport distance. The actual capacity will be measured in terms of personnel, material cars or tons per hour. Therefore, in order to maximise capacity it is important to increase the load transported per cycle and reduce cycle time by increasing tramming speeds, but more importantly, addressing the loading and unloading elements of the cycle and creating a continuous transportation process.

Where possible, fewer and larger trains should be used in order to reduce congestion on the tracks. For example, a single personnel train may be capable of handling 95 workers an hour over a 3 km trip, however this could be increased to 190 workers if a larger train were to be utilised.

The capacity of material for a trackbound system is a function of the size and number of cars used, the time taken to load and off load the cars, time spent in storage, distance trammed and the speed of the tram. While the number and size of cars is theoretically limited to the locomotive used, the overriding restriction is the size of cage in the shaft and the size of the horizontal excavation.

In the case of rock tramming, larger trains should be capable of hauling over 50 tons per trip. This will ensure greater levels of efficiency and reduce the number of units in use underground. However, it will only be possible to consider larger trains if the condition of the track structure is addressed. Furthermore, by divorcing the loading and unloading functions from the actual tramming operation it could be possible to increase the tram rate as one locomotive could effectively operate two spans of hoppers. The speed of the train is another important point as it influences cycle times. High-speed locomotives are one way to improve tramming capacity and it is feasible for trains to tram at speeds of 30 to 40 km/hr within dedicated tramming haulages. However, the most important aspect to increasing capacity is the overall continuity of the process, whereby each mode of the transportation system is conducted efficiently. Thereby creating a continuous process as the train is constantly moving other than when loading or tipping.

4.5.1.3 Limits on length

There are no theoretical limits with respect to length of a rail installation although cycle times and capital expenditure increase with distance. Diesel, and battery locomotives can operate over long distances, only requiring refueling or recharging, which limits the range. However, with well-planned layouts this need not be a limiting factor.

Train lengths are limited however by the type and size of locomotives used. Larger locomotives are able to pull longer or larger trains. In practice the coupling up of more than one locomotive in a train is uncommon. Using two or three locomotives in tandem would however allow increased train lengths, which could be an advantage on longer haulages.

4.5.1.4 Environmental compatibility

The impact of the trackbound system on the underground environment will differ according to the type of locomotives used. Diesel locomotives will have the greatest impact on the environment in terms of noise, heat and exhaust gas emissions. The diesel engines could also be considered a fire hazard, however the use of flameproofing does reduces this risk. Scrubbing systems also greatly reduce the noxious content of the exhaust gasses, thereby reducing the impact of exhaust gas emissions on the environment. Even so, the use of diesel locomotives underground requires adequate ventilation.

Battery and trolley locomotives have the least impact on the environment, as they are relatively quiet and do not produce any exhaust gasses. Heat emissions from these locomotives are also very low. In the case of electric trolley locomotives, the danger of sparking at the pick-ups must be considered. In fiery mines this could constitute a fire risk and must be taken into account when the decision is made to install a trolley system.

4.5.1.5 Powering systems

Locomotives are classified according to their size. Two basic power systems are available, namely diesel and electric. Electric power is derived from either an overhead trolley line or a battery unit. A wide range of locomotive sizes is available, from small 5-ton units to 18-ton locomotives, although larger and smaller units are available. In general, most gold mines utilise locomotives in the range of 5 to 10 tons. The ultimate constraint on size is the maximum permissible axle load, which will be set according to the standard of the track. The size and power of the locomotive will determine the maximum haul weight. The various types of locomotives currently in use are described below.

Diesel powered hydrostatic

Diesel hydrostatic locomotives have been in operation both underground and on surface for many years and vary in size from a small 3½-ton unit up to a 20-ton unit. Figure 4-7 shows an example of a 9-ton locomotive for 914 gauge track.

The most popular type of diesel engine used is the Deutz Air Cooled range from a 3 cylinder, 30 kW unit to a V6 six cylinder, 120 kW unit. Speeds vary from 8 km/hr to the maximum 16 km/hr allowed by law for underground use. Speeds in excess of 16 km/hr can only be implemented with special permission from the Government Mining Engineer.

A standard diesel hydrostatic locomotive comprises a diesel engine and a hydrostatic transmission, consisting of a variable displacement pump and a fixed displacement radial piston type hydraulic motor. The transmission of power from the engine to the wheels is achieved by means of the hydraulic pump, which is mounted directly on to the engine and connected hydraulically to the motor mounted centrally in the chassis. Heavy-duty drive chains connect the motor output to the axles giving four-wheel-traction in either direction. Suspension comprises two heavy-duty coil springs mounted over each axle box giving the total deflection required for the various masses of the locomotives.

The exhaust system incorporates a spark arrestor and a fume diluter replacing the old water scrubber box system. The locomotive is provided with a hydrostatic dynamic service brake, which is an inherent feature of the hydrostatic drive and is achieved by returning the direction control lever to the neutral position and slowly moving it in the opposite direction. Braking is achieved on all four wheels. Hydraulic fail-safe spring boosters can be fitted providing braking on all four wheels in the event of system pressure loss or if a hose bursts. On larger size locomotives (10/12 and 15-ton trains) air brakes can also be utilised.

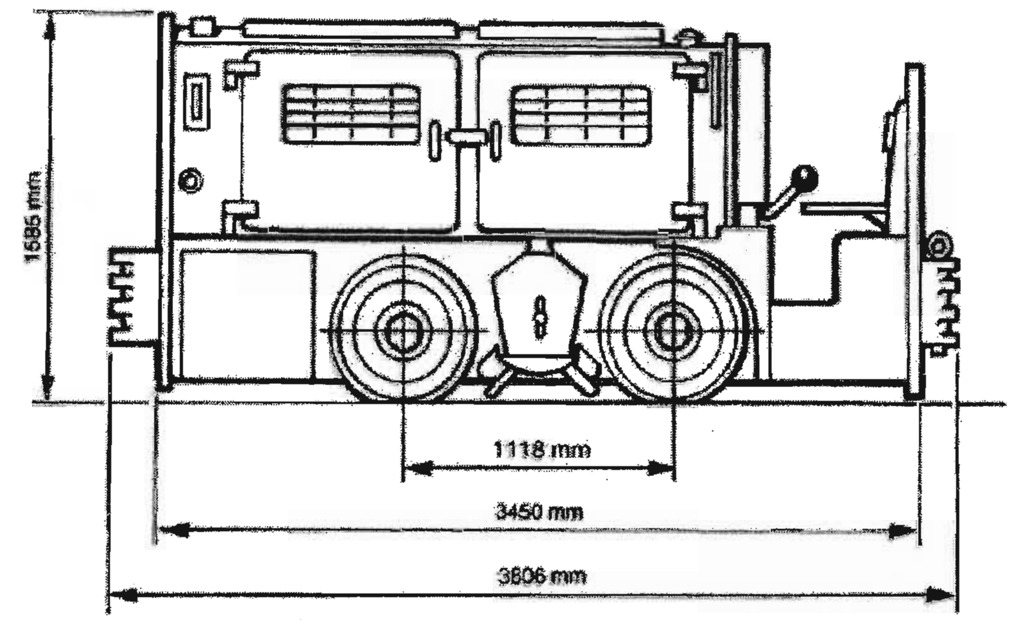


Figure 4-7: Diesel hydrostatic 9-ton locomotive
(Clayton Equipment Ltd, 1999)

Battery powered locomotives

More than 75% of all locomotives used in the South African gold mines are battery powered with the most popular sizes being the 5-ton and the 8/10-ton units. The 5-ton loco comprises a single axle mounted DC traction motor, whilst the 8/10-ton unit is fitted with two axle mounted DC traction motors. Battery powered locomotives have a payload capacity of 1,75 to 25 tons operating at maximum speeds between 12 to 16 km/hr. The battery voltage on the 5-ton unit is 84 volt from a 42-cell battery of 350 to 500 Amp hour capacities. On the 8/10-ton locomotive a 120 voltage is obtained from a 60-cell battery of 450 to 600 Amp hour capacity. Battery powered locomotive are very flexible in operation, however, due to the battery storage capacity the operating distances are generally limited from 3 km to 5 km. Battery units are the most popular type of traction power being used by the mining industry.

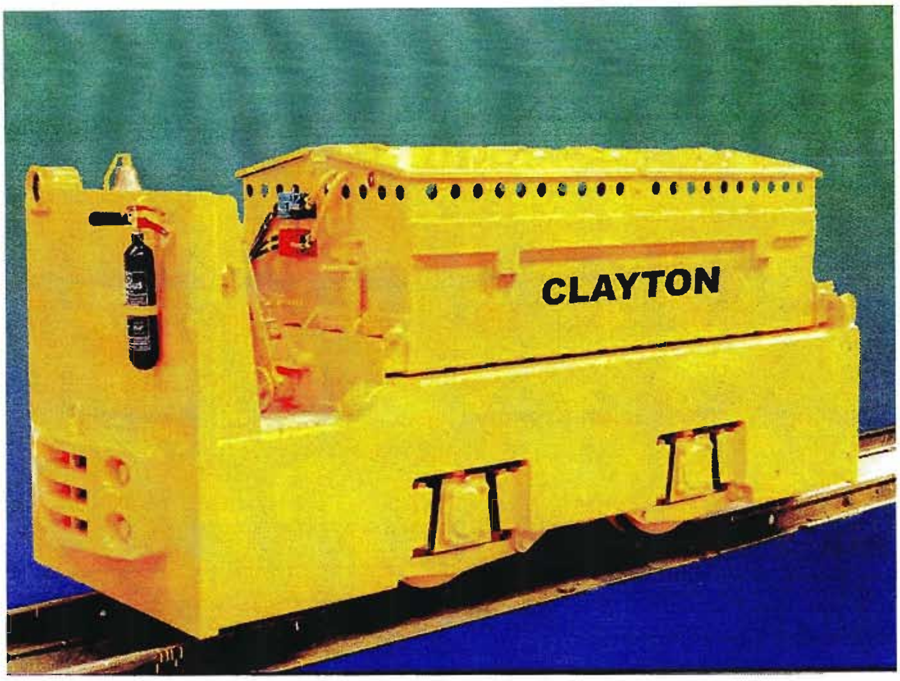


Figure 4-8: Battery locomotive (Clayton Equipment Ltd, 1999)

The most common type of battery locomotive is the standard on-board battery type (Figure 4-8). However, battery locomotives are also available with the battery on a separate tender (Figure 4-9). These locomotives have the same specification as the on-board type, but are of a more robust construction, as the mass of the chassis needs to be heavier to compensate for the loss of the on-board battery.

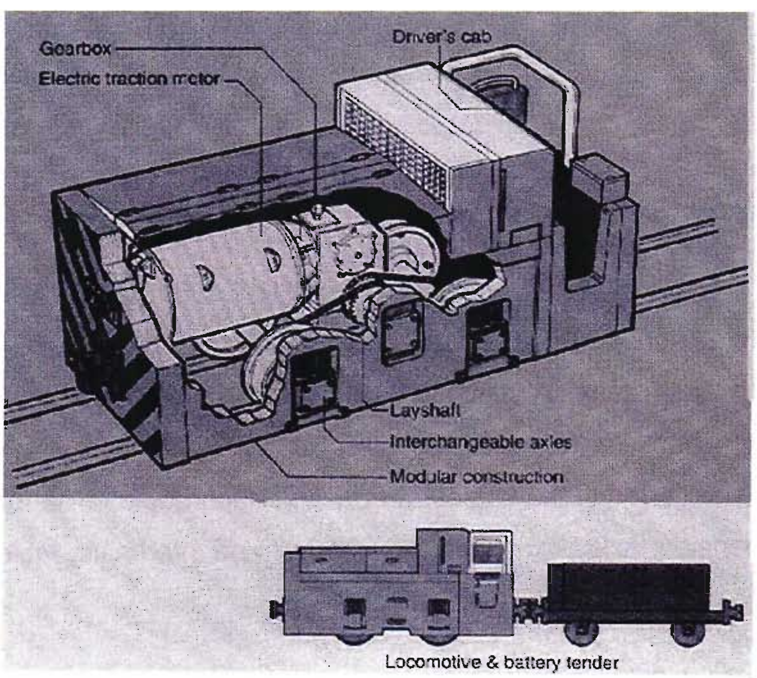


Figure 4-9: Battery locomotive with battery tender (Maslen, 1999)

Overhead trolley line – 500V AC power

Due to the cost of installation and the units themselves, trolley locomotives are mainly utilised on longer main haulage systems. The unit is inflexible as it can only operate where an overhead power supply exists. These units are mainly manufactured in excess of 12 tons. The most popular size being the 15-ton (Figure 4-10) and the 22-ton to 27-ton bogie type. They are very robust in construction, as a great deal of the ultimate mass must be built into the chassis. The 15-ton unit is fitted with 2 X 90 kW low speed axle mounted DC traction motors.

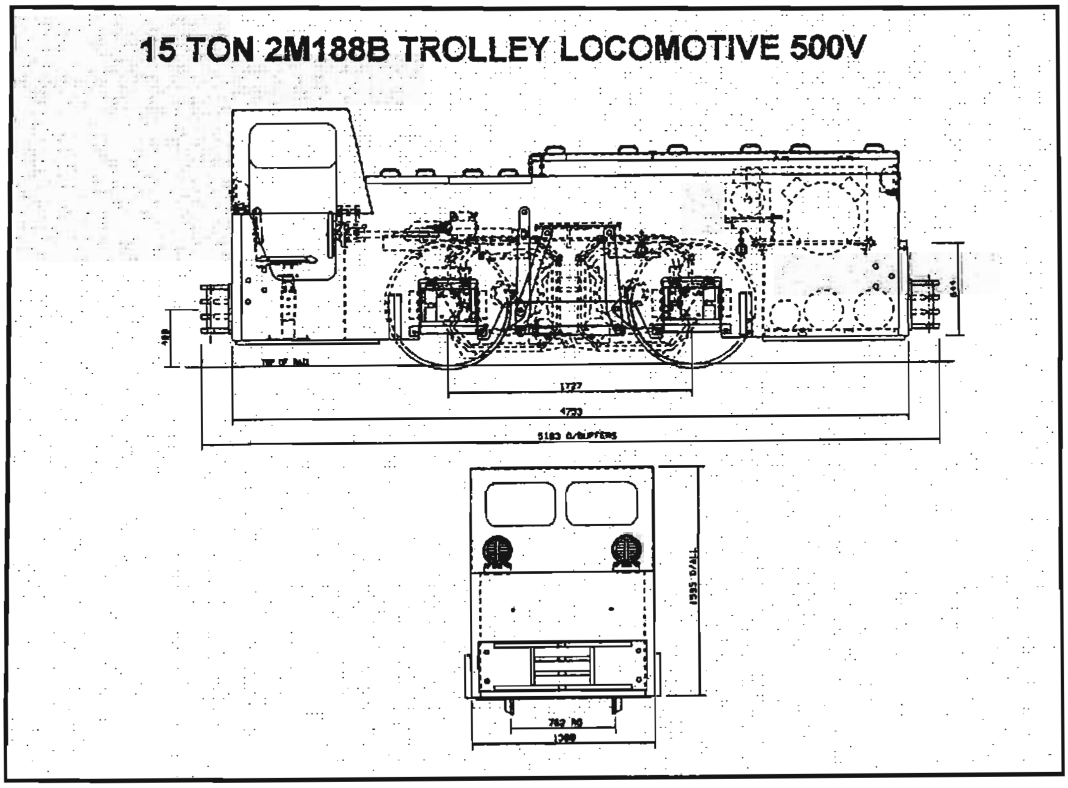


Figure 4-10: 15-ton 500 V trolley locomotive (Maslen, 1999)

The larger 22-ton to 27-ton, 500V AC trolley locomotive is fitted with two bogies, one in front and one at the rear. A centrally positioned driver's compartment lies between the two bogies. The four axle mounted 90 kW DC thyristor controlled traction motors are positioned in the bogies, with two in each bogie. The motors, wheels, axles, suspension, axle boxes, etc. are identical and interchangeable with the 15-ton units.

Combined battery and overhead trolley

The combined trolley and battery locomotive allows a trolley locomotive to operate away from the overhead line for short durations. The 12-ton to 15-ton locomotive is a 550V AC pantograph – battery operated type capable of reaching speeds in the order of 25 km/hr. The unit is fitted with an on-board 60 cell (120V) 600 Amp-hour rated battery. Power from the 550V AC overhead supply line is fed to two x 30 kW DC traction motors with braking provided by a fail-safe pneumatically operated braking system.

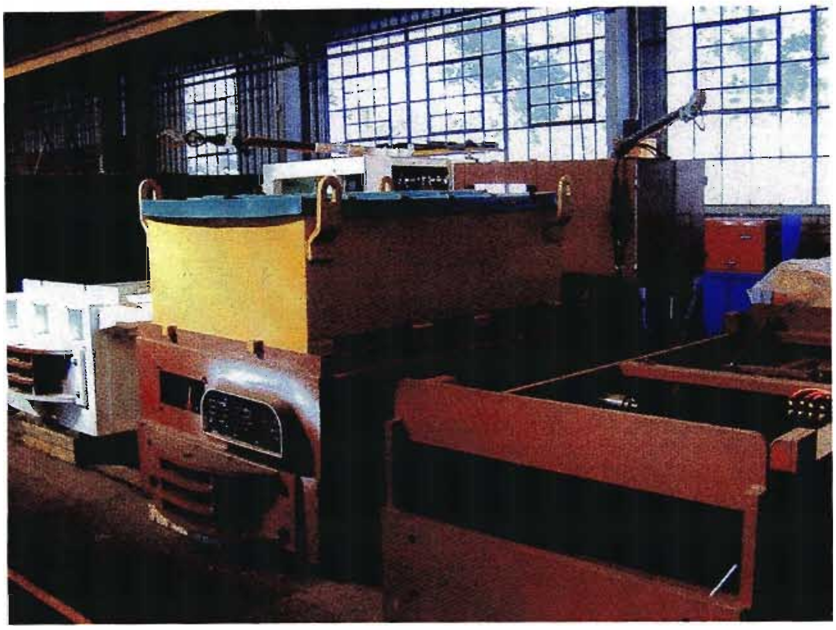


Figure 4-11: Goodman 8 ton hybrid locomotive (Dreyer, 2001)

4.5.1.6 Boarding and alighting

For personnel riding purposes, no special boarding and alighting arrangements are required. The trackbound system is flexible enough in that the train can stop at any point to allow passengers to board or disembark, the only proviso being that adequate room is available to allow passengers to do so in comfort and safety.

Standard practice is that an area is made available at the shaft stations for embarking and disembarking, but generally at the stopes, no special arrangements are in place. Personnel trains stop at the cross cuts, allowing workers to disembark at a point near their work place. Some mines have boarding points near the stopes in the main haulages, but these are little more than demarcated areas with benches placed against the sidewall to allow for workers to sit away from the track while waiting for the next carriage.

4.5.1.7 Feed and discharge arrangements

Historically, loading and discharging arrangements have been responsible for excessive cycle times and both activities are inundated with problems. Common loading or tipping times are commonly in the order of 30 minutes per activity. This can be contributed to many problems, from excessive water and mud in the cross cut to single tracks on the station or poor maintained loading chutes and tipping ramps.

Similarly, the handling of material cars should be equally efficient so that a minimal amount of the working shift is spent on material handling. Turn around times for material cars can be over 48 hours as cars are off loaded one at a time and then only as required.

4.5.1.8 Installation considerations

The installation of the track is normally undertaken by the tunnel development crew. These crews are not trained in track construction techniques and often the tracks are installed in environmental conditions not conducive to strenuous and physically demanding work. Poor management and maintenance of the track structure leads to further deterioration, resulting in safety hazards, high maintenance costs on the rolling stock, and tramming inefficiencies, all of which have serious implications on the mine's profitability. Poor track construction combined with poor management and maintenance, leads to a poor track structure. In order to overcome this problem some mines have begun to use trained and dedicated track construction crews. These crews follow behind the development crews and lay track in sections in the order of 50 m to 60 m to a standard adequate for the envisaged duty. For underground trackbound transportation to operate efficiently, the track structure needs to be constructed to a standard adequate for the intended application. On the following pages, Table 4-1 depicts the classification criteria for the class of track and Table 4-2 depicts the maximum permissible deviation.

Table 4-1: The classification criteria for the class of track (DME, 1999)

ITEM	CLASS OF TRACK				
TRACK RATING	1	2	3	4	5
TONNAGE PER MONTH (tons) (x1000)	>125	125- 50	50-20	20-7,2	<7,2
PERSONNEL	YES	YES	YES	N/r	N/r
ROCK	YES	YES	YES	YES	YES
EXPLOSIVES	YES	YES	YES	YES	N/r
MATERIAL STANDARD VEHICLE	YES	YES	YES	YES	YES
MATERIAL ABNORMAL LENGTH	YES	YES	YES	YES	N/r
MATERIAL ABNORMAL MASS	YES	YES	YES	N/a	N/r
MAX AXLE LOAD (tons)	15	10	7,5	5	5
MAX SPEED (km/hr)	45	24	16	10	5
PLANNED OPERATION LIFE (years)	25	20	10	5	3
RAILS (kg/m)	40/30	30	30/22	22	22/15
SLEEPERS (axel load tons)	10+	10	7	5	5
SLEEPER TYPE (Graded ballast, nominal 38,5 mm)	C/S/W	C/S/W	C/S/W	C/S/W	C/S/W
SLEEPER TYPE (Non graded ballast, max. 0-75 mm)	N/a	N/a,	N/a	C/S/W	C/S/W
MIN. CURVE RADII (m)(Restricted speed)	75	50	30	20	15
MIN. CURVE RADII (m) (Unrestricted speed)	250	150	90	60	30
JOINTS WELDED	YES	YES	YES	N/r	
JOINTS FISH PLATED	N/a	YES	YES	YES	YES
PEDESTRIAN TRAFFIC (Under restriction)	N/a	N/a	YES	YES	YES
SIGNALLING (Traffic control)	YES	YES	N/r	N/r	N/r

Note: Yes= recommended N/r =no recommendation N/a = not advised

Sleeper type: C=concrete S=steel W= wood

Table 4-2: The maximum deviations in track specifications (DME, 1999)

ITEM	CLASS OF TRACK					ADDITIONAL INFORMATION
	1	2	3	4	5	
From design level (mm)	±5	±7	±10	±20	±30	(Over a length of 5 m)
Cross slack (mm)	3	3	5	8	15	(Over a length of <2 m)
Straightness (mm)	5	7	10	20	30	(Over length of 5 m) measured on gauge face
Gauge (nominal)	+5 -2	+5. -2.	+10 -3	+ 15* -3	+25 -5	
Gauge (widening)	+3 -2	+3 -2	+5 -2	+5 -2	N/r	
Sleeper spacing (nominal)	± 20	± 20	± 50	± 50	± 75	
Steeper spacing (joint)	± 10	± 10	± 20	± 20	± 50	
Circular curves	± 5	± 5	± 10	± 15	± 25	(Over length of 5 m)
Super elevation on straight (mm)	5	5	8	10	20	(Over length of > 2 m.)
Height differential at joint (mm)	0	<1	<2	<5	<10	
Lateral differential at joint (mm)	0	<2	< 3	< 3	< 5	
Joint gap (mm)	0	<2	<6	<6	<10	

4.5.1.9 Maintenance

Track maintenance is a process whereby an existing standard or class of track is maintained to that level throughout its life (Table 4-1). Track reinstatement, unlike track maintenance, is able to increase the standard or class of track and is undertaken when a track has deteriorated to a point beyond which it is not maintainable. Poor track standards can be avoided by ensuring that the initial track installation is correctly installed, free of water, and maintaining both rolling stock and track work based on a structured maintenance programme. The provision of adequate technical skills and abilities is also a critical component of track maintenance and reinstatement. Maintenance should be of a continual, cyclic nature that aims at prevention rather than cure by constantly measuring and monitoring.

Both track maintenance and track reinstatement should address the following key areas:

- Material condition.
- Track joints.
- Turnouts.
- Drainage installations.
- Vertical and horizontal alignment.
- Clearances and restrictions.
- Track signage.
- Communications.
- Inspection and reporting procedures.

In many mines, no systematic track maintenance programmes are in place and a rail "emergency" crew is utilised to repair damage caused by derailments, but with little consideration for the actual track condition. These crews do little more than replace broken sleepers. Often the track formation and ballast is ignored to the point where the ballast is no longer functioning properly or doesn't exist at all. Other poor practices commonly seen are water that is allowed to run between the tracks, thus deteriorating the quality of the track. Another is that material is allowed to accumulate between the tracks to the extent that the wheels of the train run on the dirt rather than the rail. Fishplate joints are seldom tightened, and often broken rails and battered ends are left in place.

4.5.1.10 Compatibility with other systems

The trackbound transportation system is able to handle all the requirements of underground horizontal transportation i.e. personnel, material and rock. However, when required trackbound transportation can also be used in conjunction with other systems. Monorails are able to be led over the track structure and are able to pick up loads from the rail cars. It is possible for palletised loads to be taken down the shaft on a flat bed material car and then be picked up by a monorail for further transportation (Figure 4-12).

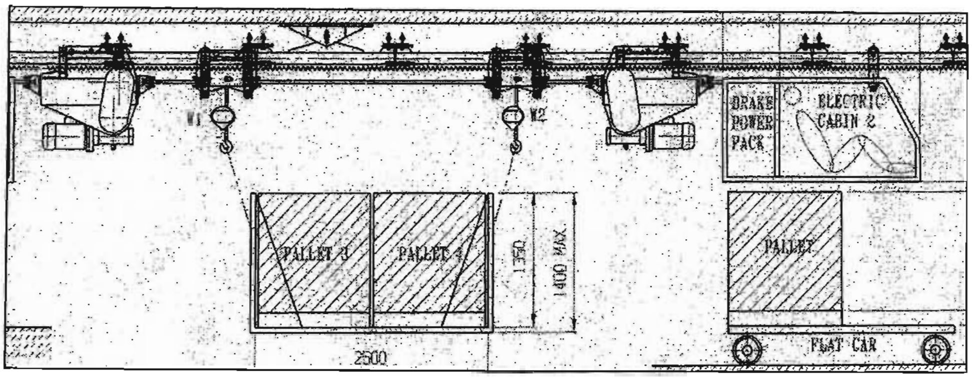


Figure 4-12: Monorail operating above material car (Deale, 2002)

Additional flexibility for trackbound transportation can be attained through using a combination of cars. Personnel carriages can be coupled together with material cars, allowing for the transport of both workers and materials on the same trains. Similarly, the guard carriage of trains can be used to transport small numbers of workers. This would allow those personnel requiring transport back to the shaft during the shift to do so without requiring special trains or additional trips.

<u>Trap rails</u>

Variations on the standard trackbound systems have been developed in the European coal mining industry. These variations use a similar set-up to rail transportation in that the wheels of the rolling stock are positively guided by the track structure. These systems are termed "trapped rail" systems (Figure 4-13).

These systems can be either self propelled, or propelled by means of rope guidance and drive. The trapped rail system is advantageous in that they are able to negotiate sharp curves, both vertically and horizontally, as well as three-dimensional curves.

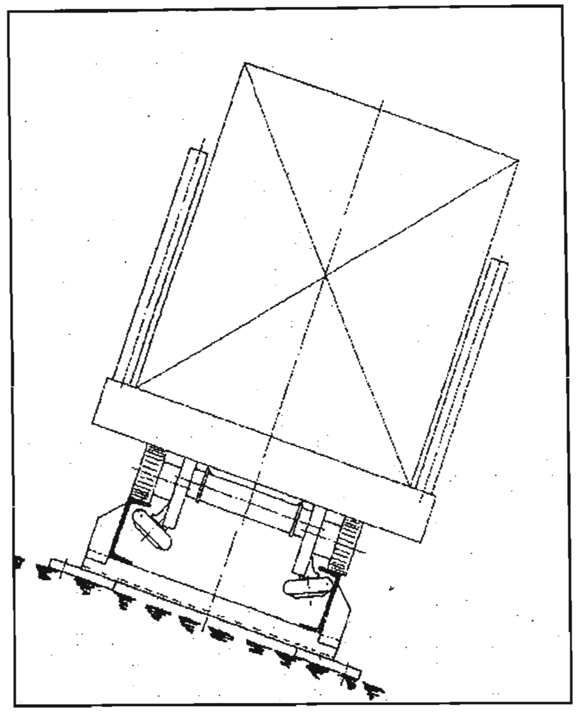


Figure 4-13: Trapped rail system (Webers, 1999)

Rail Hugger

The rail hugger system uses a conventional track structure, except that at curves additional channel section rails are added to effectively trap the rail bogeys preventing derailments. This system is suited for use on tracks where rope guidance systems are predominant, but can be adapted for self-propelled vehicles. On high-speed haulages, such a system may be advantageous in preventing derailments.

Continuous Rope Haulage

A further system that has been used in hard rock mines is the continuous rope haulage system, whereby locomotives are replaced by a continuous rope haulage running within the track structure. Use of the continuous rope haulage, as a system in its own right is suited to the movement of rock and material over defined travelling paths, as other transportation systems are more efficient for branch type of transportation networks.

4.5.1.11 Personnel carriages

A personnel carriage is a rail-bound vehicle utilised for the transportation of workers from the shaft to the working place where distances are too great for walking. Personnel carriages are available in a variety of sizes, ranging from the small fixed axle type with a capacity of 12 workers, up to large, double bogey versions that are capable of carrying in excess of 40 workers (Figure 4-14 and Figure 4-15).

Several factors dictate the size and type of personnel carriages that can be used:

- Maximum permissible axle load in terms of track standard.
- Cage or shaft compartment size.
- Haulage dimensions.
- Pipes and other obstructions within the haulages.
- Ventilation doors.
- Curve radii.

Smaller carriages with overall lengths between the buffers of less than 3 m can easily be transported down the shaft inside the cages. This makes it relatively easy to move these from level to level as mining progresses. Larger carriages however need to be slung under the cages. These types of carriages are therefore better suited to levels with a projected long life.

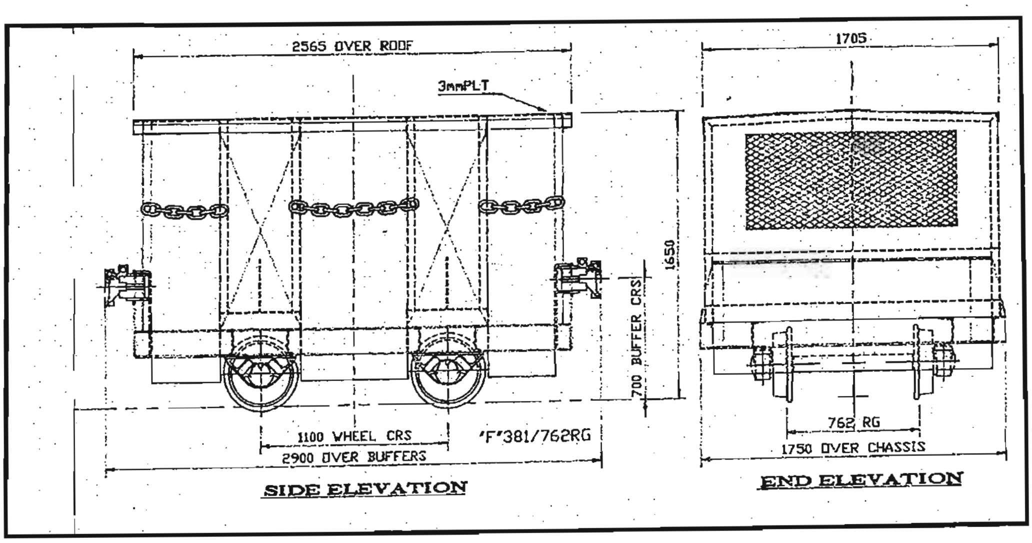


Figure 4-14: 12 person carriage (Galison, 2003)

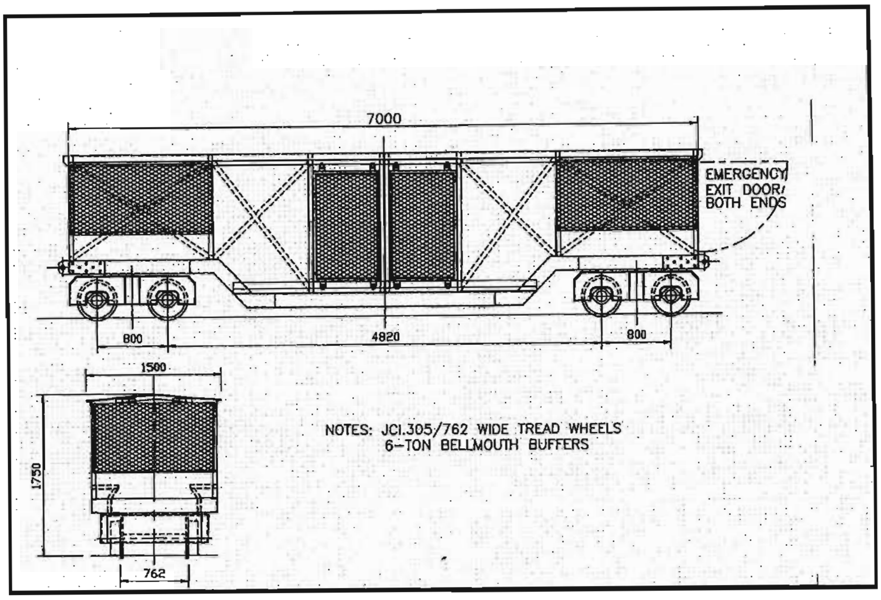


Figure 4-15: 40 person carriage (Galison, 2003)

It must be noted that the development and design of personnel carriages does not cater for speeds exceeding ± 20 km/hr. The non precision type wheels (cast), suspension systems and the draw gear, normally link and pin connecting type, have been developed to cater for the normal operating and running conditions with maximum speeds not exceeding 16 km/hr. Serious consideration will have to be given to the above if higher speeds are contemplated.

4.5.1.12 Material cars

Material cars are used to transport general mine supplies to the work place. Two types of material cars are utilised, material cars with sidings (Figure 4-16) and flat material cars (Figure 4-17). The material car with sidings is used to transport palletised material but due to the sidings it is also able to transport loose material, such as cement, paint, scraper ropes, snatch blocks, etc. The flat car is generally utilised for the transport of timber and it has an added benefit that it can be stacked one or two cars on top of another thereby reducing shaft time when hoisting empty material cars. A similar method of improving shaft utilisation is the use of collapsible pallets, which are shown in Figure 4-18.

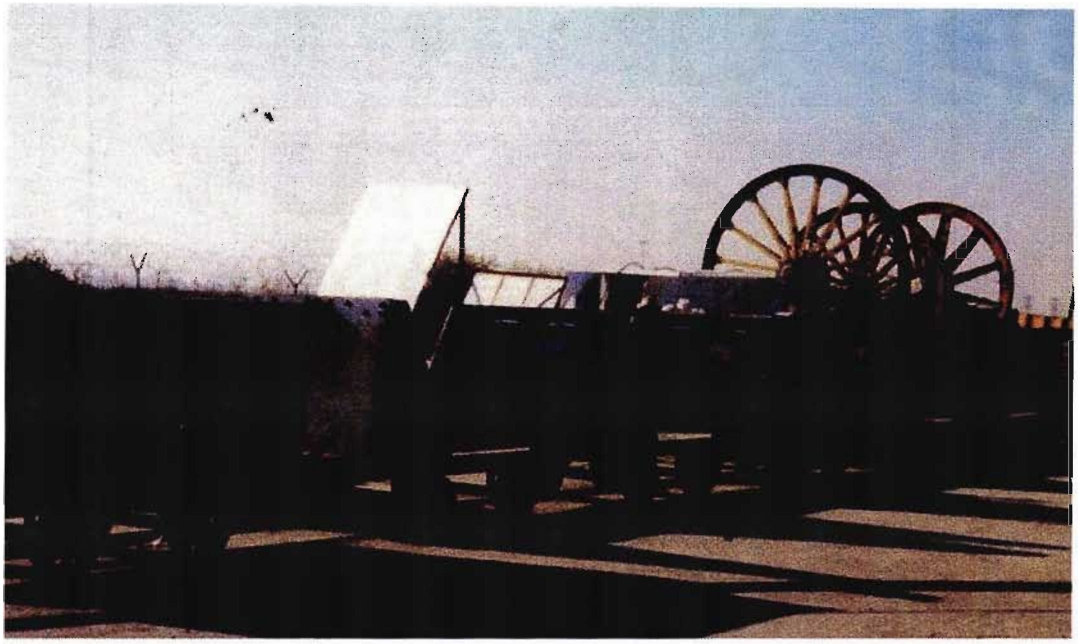


Figure 4-16: Material car with sidings

Figure 4-17: Flat timber car

Figure 4-18: Palletised material carriers stacked on flat material car

4.5.1.13 Hoppers types and sizes

Various hoppers are differentiated by the mechanism used to discharge the rock. Hoppers are available in a wide range of sizes, from a carrying capacity of just 1 ton up to 30 ton. The majority of the hoppers in use underground are of 6-ton payload size of either a tippler or bottom discharge type. In some instances, a 13/16-ton bottom discharge hopper is utilised.

Several factors dictate the size of hoppers that can be used:

- Maximum permissible axle load in terms of track standard.
- Maximum dimensions of equipment that can be slung down the shaft.
- Maximum dimension of the excavation of the haulage.
- Pipes and other obstructions within the haulages.
- Ventilation doors.
- Maximum hopper dimensions that will facilitate a loader within development ends.

Once again the need for high quality track conditions is emphasised, since this will determine the size of hopper that can be used. Outlined below are examples of the many types of hoppers available in South African gold mines.

Side discharge hoppers

Side discharge hoppers are capable of tipping large rocks. Due to the tipping mechanism, the hoppers are not influenced by the gauge of the track. There are two basic types of side discharging hoppers, the Granby (Figure 4-18) and the Gable (Figure 4-19). Both hopper types require a high draw bar pull force to operate through the tips and have an extensive number of moving parts that require continuous maintenance. The weight of the hopper is disproportionate to the payload and the hoppers are prone to leakage at the door. The hoppers have a high capital cost but have a long life with moderate operating costs. Common sizes are between 3,5 to 10 ton hoppers. From the 1950's to 1970's the side discharge hopper was very popular, but has since been replaced mainly by the Tippler type hopper.

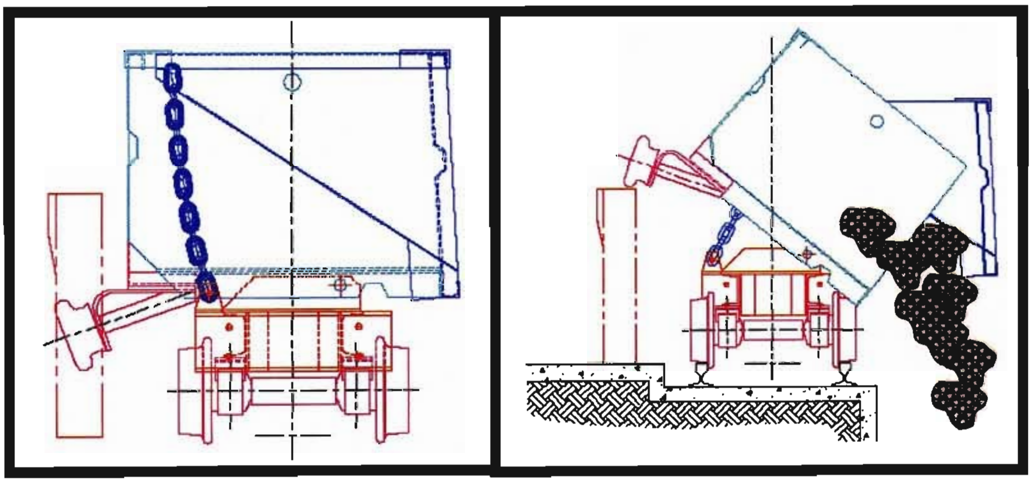


Figure 4-19: Granby hopper (Galison, 2003)

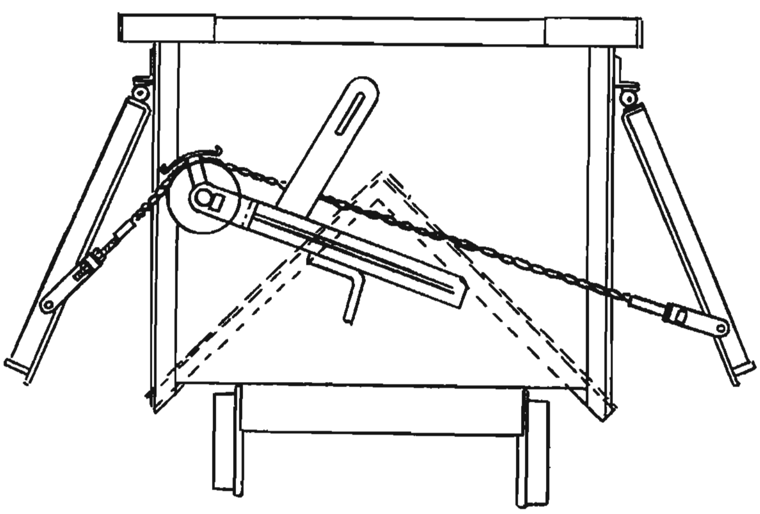


Figure 4-20: Gable hopper (DuPlessis *et al.*, 1999)

Bottom discharge hoppers

There are numerous types of bottom discharge hoppers used in the gold mines the most common of which are discussed below:

- Rockflow.
- Drop bottom.
- Swing door.
- Double radial door.
- Lifting body.
- Tippler.
- Roll over type.

Rockflow hoppers

The rockflow hopper discharges its payload as the body of the hopper is supported on rollers on the ramp and the base of the hopper drops away (Figure 4-20). A low draw bar pull is required to go through the tip, as there are a relatively small number of moving parts. The tip excavation is large, complex and expensive, however little maintenance is required due to the minimal amount of moving parts on the tip. The rockflow hopper is suitable for large rocks and sticky ore and has a high discharge rate (i.e. 180 tons in 20 seconds). These hoppers are ideal for high volume tramming levels.

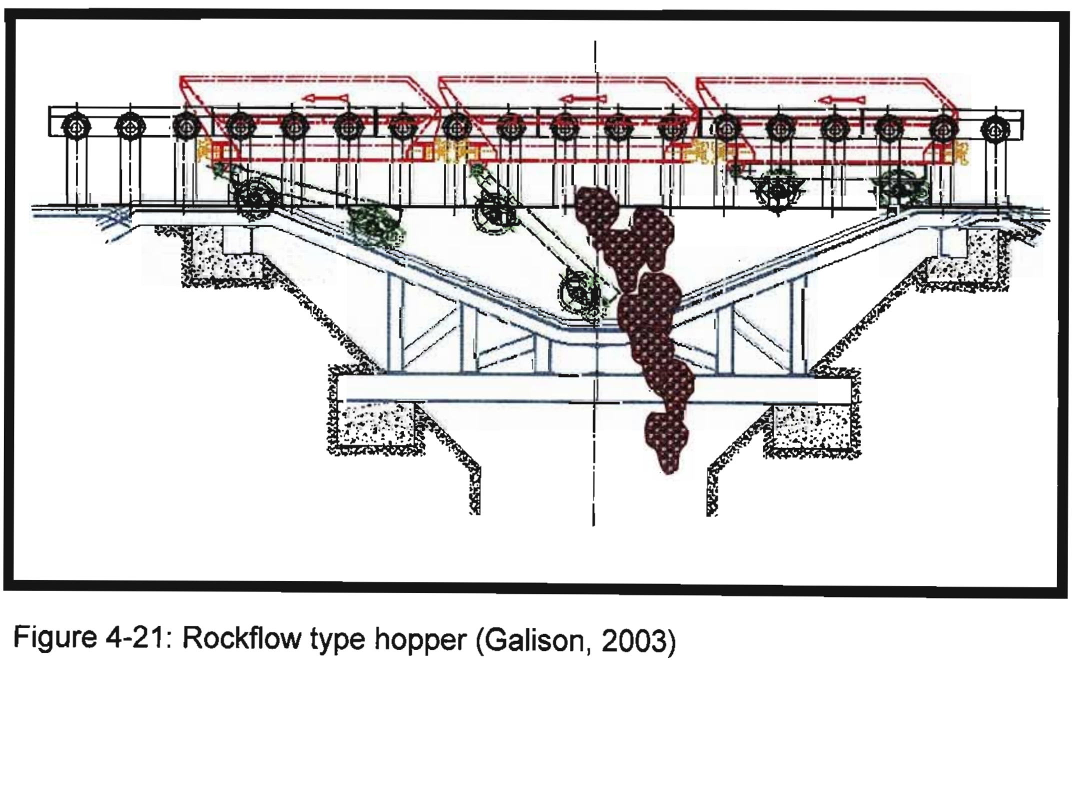


Figure 4-21: Rockflow type hopper (Galison, 2003)

Drop bottom hopper

The drop bottom hopper (Figure 4-22) utilises a door, or doors, depending upon the size of the hopper, which drop away as the hopper enters the tip. This hopper can be operated manually, thereby reducing cost for tipping arrangements. The hopper has a good volumetric efficiency and is quick and easy to tip. Because the rock discharges between the wheels, the gauge of the tracks influences tipping. The door itself can be a source of leakage and has the potential to open during tramming operations, thus good maintenance is required

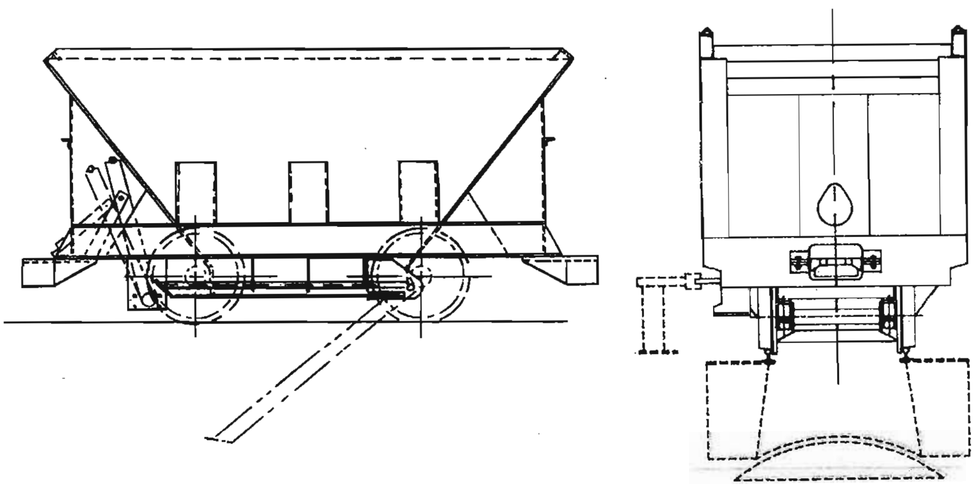


Figure 4-22: Drop bottom type hopper (DuPlessis *et al.*, 1999)

Swing door hopper

This hopper (Figure 4-23) has a door, which swings to one side of the hopper creating a large opening for the rock to discharge. The hopper is relatively leak-free but requires ongoing maintenance and requires a relatively high draw bar pull force to operate the hopper in the tip ramp.

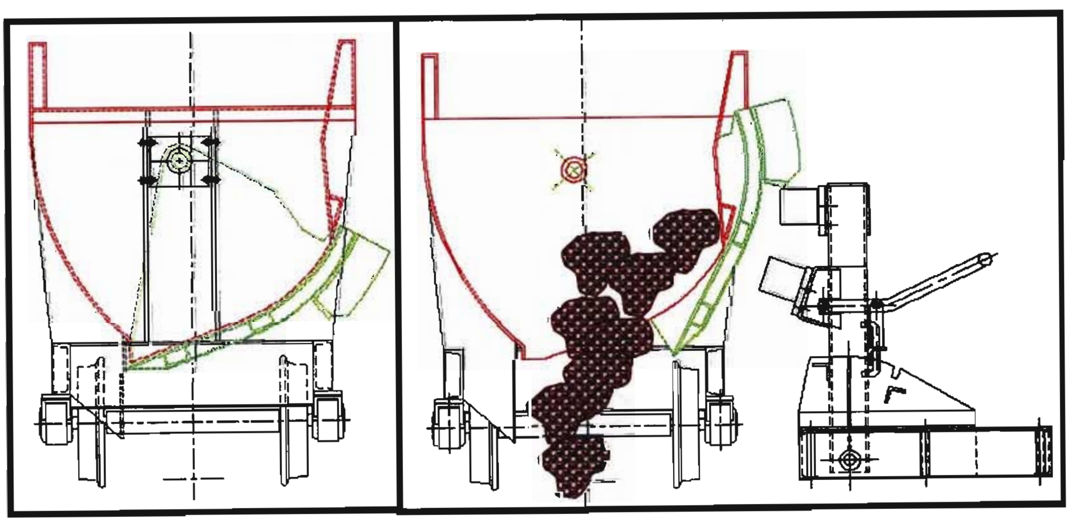


Figure 4-23: Swing door hopper (Galison, 2003)

Double radial door hopper

This is similar to a swing door hopper but has a double door opening (Figure 4-24); requiring twice the number of dolly wheels per hopper and double the number of ramps in the tip, thereby promoting higher maintenance costs. The hinges are below the doors and operate by dolly wheels that are forced downwards through the tips. This is a very robust hopper with good cleaning properties, as the rock tends to clean the hopper as it discharges. Similar to the drop bottom hopper the gauge of the track influences the size of the ore that can be discharged. Tips are relatively inexpensive and trains require a low draw bar pull to move through the tipping ramp.

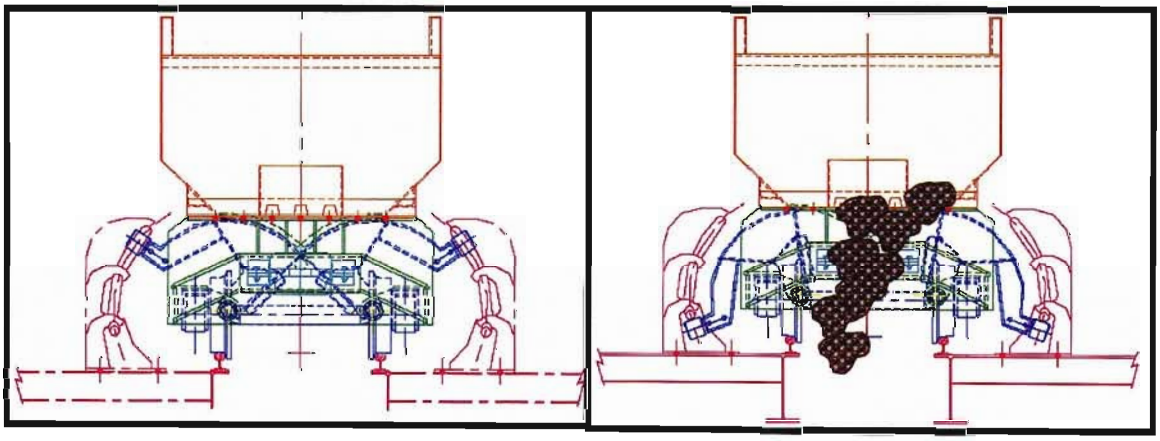


Figure 4-24: Double radial door hopper (Galison, 2003)

Lifting body hopper

The lifting body hopper (Figure 4-25) has double doors at the bottom of the hopper, longitudinal to the direction of the track that are opened by lifting the entire body off the chassis by means of dolly wheels. A large area is made available for discharge, thereby facilitating rapid tipping. A relatively low draw bar pull is required through the tips. This hopper is very expensive and has an unusually large number of hinges and links, which may require added maintenance.

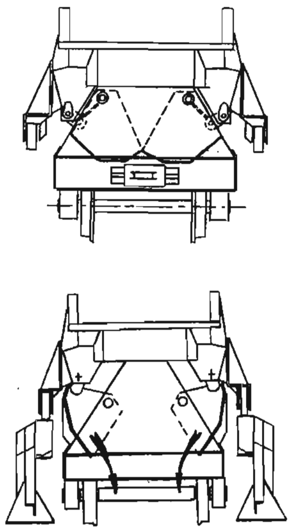


Figure 4-25: Lifting body type hopper (DuPlessis *et al.*, 1999)

Tippler type hopper

This is the most common type of hopper (Figure 4-26) in use underground as it requires little maintenance and the tipping system is cheap and easy to install with few moving parts. The body pivots around its own centre of gravity allowing an opening in the body, normally closed by a fixed door on the body. This hopper is low cost, extremely robust, and has only one moving part. There are no dolly wheels on the hopper itself since these are on the ramp, thereby reducing maintenance. Leakage is a problem and the limited size hopper discharge area makes it unsuitable for the tipping of large rocks.

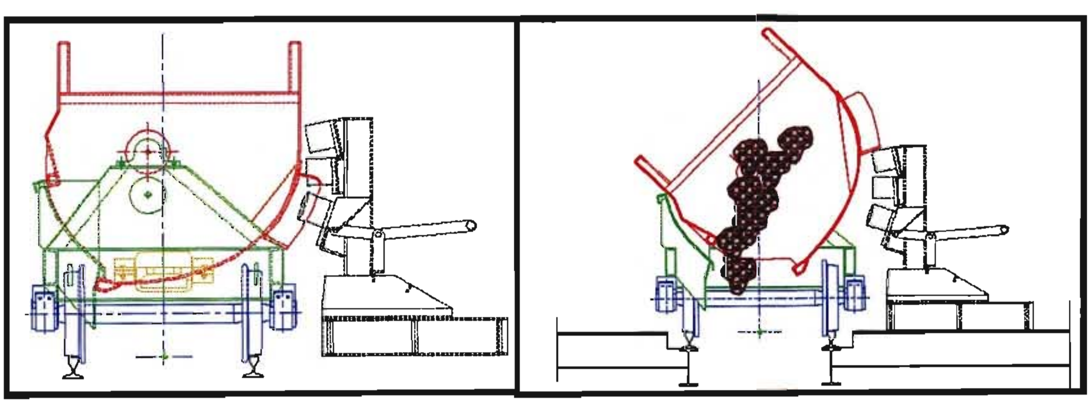


Figure 4-26: Tippler type hopper tipping (Galison, 2003)

Roll over type hopper

The roll over hopper (Figure 4-26) offers many advantages over the other hoppers mentioned above. It is an enclosed unit and is therefore leak and spill free. This hopper is generally used for higher payloads (15-tons) and operates on a bogie wheel sets situated on either end of the tub. Due to the size of the hopper, trackwork must be of a high quality requiring regular track maintenance. In addition, curve radii must be large enough to accommodate this size of hopper. The roll over type of hopper utilises a special ramp or slinging mechanism to tip the payload (Figure 4-27). This type of hopper is advantageous in that fines are not lost during transportation. The roll over method of discharge also reduces tipping time.

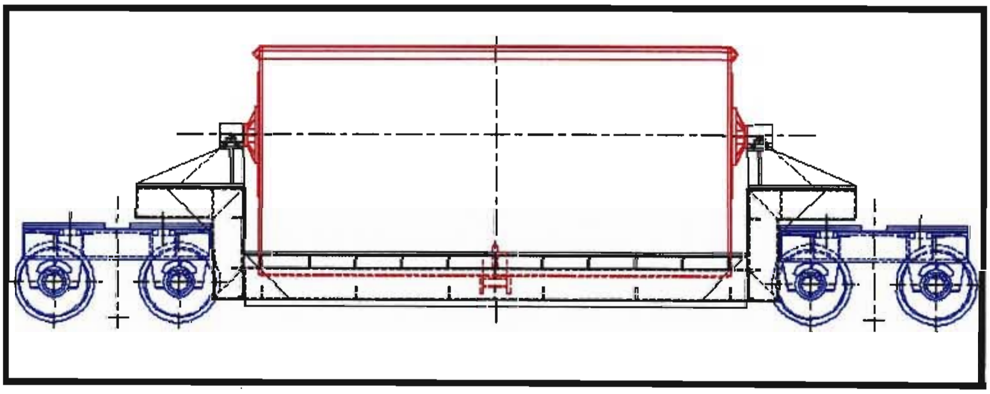


Figure 4-27: Roll over type hopper (Galison, 2003)

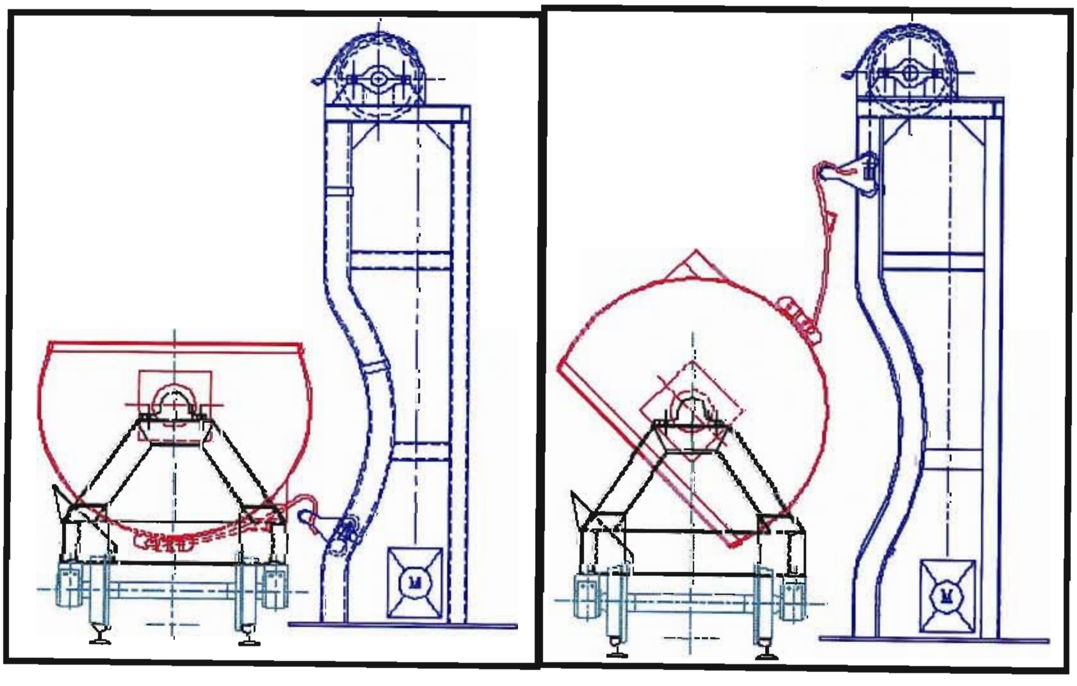


Figure 4-28: Roll over hopper (Galison, 2003)

4.5.2 Evaluation of trackbound transportation

4.5.2.1 Size

Current underground practice is that rail gauges vary from a narrow 610 mm gauge, to the standard surface gauge of 1065 mm. A narrower gauge of 457 mm is also available, however, this gauge is unsuitable for high-speed haulages.

The most common gauge in use is 762 mm. Should speeds higher than the current maximum of 16 km/hr are anticipated for ultra deep level mining, there is no reason why this gauge cannot be used. However, the wider 914 mm gauge track may be more suitable as large cross sectional tunnels will be required in ultra deep mines. The large gauges should therefore be adopted as the standard track gauge for ultra deep level mining as it provides greater stability and is more suitable for higher speeds than the 762 mm gauge.

The size of the rolling stock is also of importance. Clearances, both above and to the sides of the track, will dictate the size of the rolling stock to be adopted. Overhangs and wheelbases should be minimised to prevent the possibility of the carriages/cars/hoppers from fouling the sidewall on curves. Existing rolling stock can operate in haulages with a cross section of as little as 2 m by 2 m.

4.5.2.2 Flexibility

The trackbound system is flexible in that it can reach any part of the mine as long as there is a track structure to support it. The locomotive is ideal in that it can transport personnel, material and rock; the only other system capable of this flexibility is a trackless system.

Battery and diesel locomotives can operate independently of any external power source, and hybrid electric trolley/battery locomotives can operate within short distances beyond the extent of the overhead power supply. The obvious limitation on this form of transportation is the need for a track structure. A further limitation on conventional trackbound transportation is its inability to climb significant gradients. Conventional trackbound systems are limited to operations on the level or on minimal gradients (1:60). The use of rack and pinion drives can overcome this limitation, but the additional expense for toothed rail and special drive units must be taken into account.

Battery locomotives have a lower power output and a limited range in terms of battery life. Batteries are limited in their range or the number of trips between charges, which is estimated in terms of operating hours per charge. Backup batteries as well as battery charging bays are therefore a necessity for the efficient operation of battery powered locomotives. The logistics of changing batteries can be problematic when a large number of locomotives are operating out of one charging bay. Thus, serious consideration must be given to the scheduling and positioning of these charging bays.

Trackbound systems that do not make use of self-propelled drive units, such as endless rope haulages, are further limited in that they are unable to operate on a branched system. These types of systems are bound to a single defined track and are unable to be switched to other tracks due to their dependence on a rope for their locomotive power. These types of systems however do have an advantage over conventional self-propelled locomotives in that they are able to negotiate inclines by utilising trap rail technology.

4.5.2.3 Environmental compatibility

Diesel powered locomotives, due to their exhaust gas emissions and heat output, are not ideally suited to the expected high temperatures at ultra deep levels. Rock temperatures may be in excess of 70°C and emphasis must be on reducing the heat outputs of machinery. For this reason battery, overhead trolley locomotives, or hybrid battery/trolley locomotives should be adopted for ultra deep level mining.

Trolley lines carry a risk of arcing at the interfaces between the power supply and the power pick-ups of the locomotives, potentially resulting in an explosion. Ventilation of the haulages must therefore be considered a high priority in order to reduce the methane content (or other explosive gases) of the atmosphere to safe levels. With an overhead trolley line, a further danger exists in restricted heights, as persons or equipment may come into contact with the power supply lines.

4.5.2.4 Control and automation

Trackbound transportation is fully compatible with automated traffic control and monitoring systems. Sensors can be placed within the track structure to pick up the position, direction and speed of any train within the system. Data from these sensors can be relayed back to a central control system to enable real-time monitoring of the trackbound transportation system. One example of the advantages of using such a system is that trains can be slowed or stopped where necessary i.e. dangerous track conditions, curves and switches or to enable a higher priority trains to travel unhindered in the haulage.

The installation of electrically operated switches enables a central controller to switch a train into a cross cut or turnout as necessary. The automation of the switches will result in a time saving, as trains will not be required to stop in order for the locomotive driver to change the switch manually. Similarly, ventilation doors can also be automated in this manner.

The automation of a trackbound system requires a high maintenance effort on the track work, as well as the control system. Tracks must be kept clean and free of any rubble and maintenance teams will be required to inspect the track on a daily basis and replace any defective or broken parts immediately. Furthermore, transponders and locomotive control units will require regular inspections and testing to ensure that they remain operational.

4.5.2.5 Cost implications

Track installation costs are generally in the order of R600 to R950 per metre depending on the class and duty of the track. Track installation costs are in the order of R600 per metre for new track on ballast with fishplated joints. Welded joints increase the costs to approximately R650 per metre. On main haulages where a high track standards are required over a protracted period of 5 to 10 years, the higher cost applies. On a haulage with a shorter life, the lower track installation cost is applicable. In cross cuts, the installation costs can be further reduced, as the temporary nature of these tracks does not require a high track standard. Maintenance costs are duty dependent and are in the order of R3,00/ton (short life) to R9,00/ton (long life) with approximately 10% of the maintenance taking place in the stope cross cuts.

The following tables give an indication of locomotive and accessory costs, as well as the approximate costs of the personnel carriages.

Table 4-3: Diesel hydrostatic locomotives

TYPE	POWER (kW) AT 2150RPM	LOCO MASS	BUDGET PRICE (2000)
Deutz F3L912	29 kW	4500 kg	R250 000
Deutz F4L912	46 kW	6000 kg	R275 000
Deutz F4L912	46 kW	7500 kg	R350 000
Duetz F5L912	58 kW	9000 kg	R375 000
Deutz F6L912	70 kW	12000 kg	R425 000
Deutz F6L413FW	102 kW	13-15000 kg	R490 000
ADE 366N Flameproofed	77 kW	9-12000 kg	R500/550 000

Table 4-4: Battery powered on-board – axle mounted design

TYPE AXLE MOUNTED	POWER kW AT VOLTAGE	MASS	BUDGET PRICE (2000)
Single-Axle Mounted	15 kW at 84 V	5000 kg	R135 000 Including Controller
Twin-Axle Mounted	21 kW (42kW)	9-10000kg	R195 000 Including Controller

Table 4-5: Cost of auxiliary equipment

TYPE	ROLL OF RACKS	BATTERY CHARGER	LOCOMOTIVE MASS	BATTERY PACKS (2000)
Single Motor Axle Mounted 15 kW	R2500 2 REQD/Battery	R21 500 1 REQD/Battery 42 Cell 84V	5-6000kg	42 Cell-500Amp Hour. R29 000 2/3 Battery/Loco
Twin Motor Axle Mounted 21 kW Each	R3500 2 REQD/Battery	R22800 1 REQD/Battery 60 Cell 120V	9-10000kg	60 Cell 600Amp. Hour. R47 500 2/3 Battery/Loco

Table 4-6: Trolley electric 500V AC

TYPE	POWER RATIANG AT 500 VOLT AC	MASS	BUDGET PRICE (2000)
Twin – Axle Mounted	2X45 kW – 90 kW Max Speed 16 km/hr	15000 kg	R750 000
Twin – Axle Mounted	2X90 kW – 180 kW Max Speed 28 km/hr	15000 kg	R800 000
Bogie Type 2 Axles/Bogie	4X45 kW – 180 kW Max Speed 16 km/hr	22000 kg to 25000 kg	R1,700 000
Bogie Type 2 Axles/Bogie	4X90 kW – 360 kW Max Speed 28 km/hr	22000 kg to 5000 kg	R1,800 000

Table 4-7: Pantograph/battery – 550V AC

TYPE	POWER RATING	LOCO MASS	BUDGET PRICE (2000)
Twin Gearbox Twin Motor	2X28 kW 56 kW Max Speed 16 km/hr	12500 kg	R700 000
Twin Gearbox Twin Motor	2X28 kW 56 kW Max Speed 16 km/hr	15000 kg	R725 000
Twin Gearbox Twin Motor	2X42 kW 84 kW Max Speed 25 km/hr	15000 kg	R825 000

Table 4-8: Personnel riding carriages

TYPE	CAPACITY	BUDGET PRICE (2000)
Two Axle	20 Workers	R14 950
Four Axle Bo-Bo	40 Workers	R69 500

4.6 Monorail systems

4.6.1 Audit of monorail system

4.6.1.1 Overview

Two suppliers of monorail systems are currently operational in South Africa. These are Walter Becker and DBT/Scharf. The two suppliers essentially supply the same type of system with minor differences relating to the type of drive units supplied.

The monorail system itself, as currently applied in South Africa, consists of a single overhead I-beam of type I140E, suspended by means of chains or slings rockbolted to the hangingwall (Figure 4-29 and Figure 4-30). The rails are currently imported from Germany in standard lengths of 1,5 m and 3,0 m, with curved lengths and switches available. Dependent on the type of drive unit specified, conductor rails can be fixed directly to the rail section to allow electric traction to be available for electro-hydraulic and electric drive units.

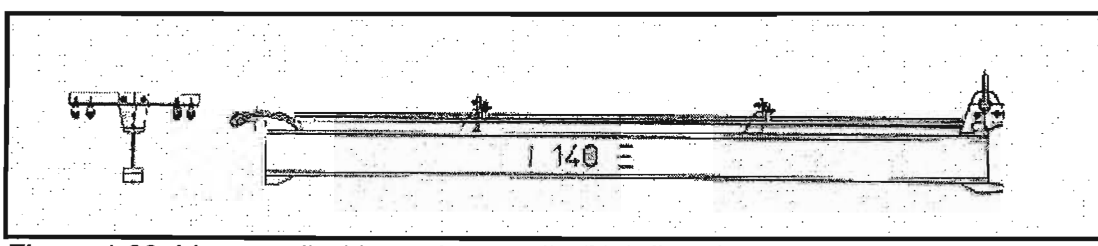


Figure 4-29: I-beam rail with conductor rails (Nehrling, 2002)

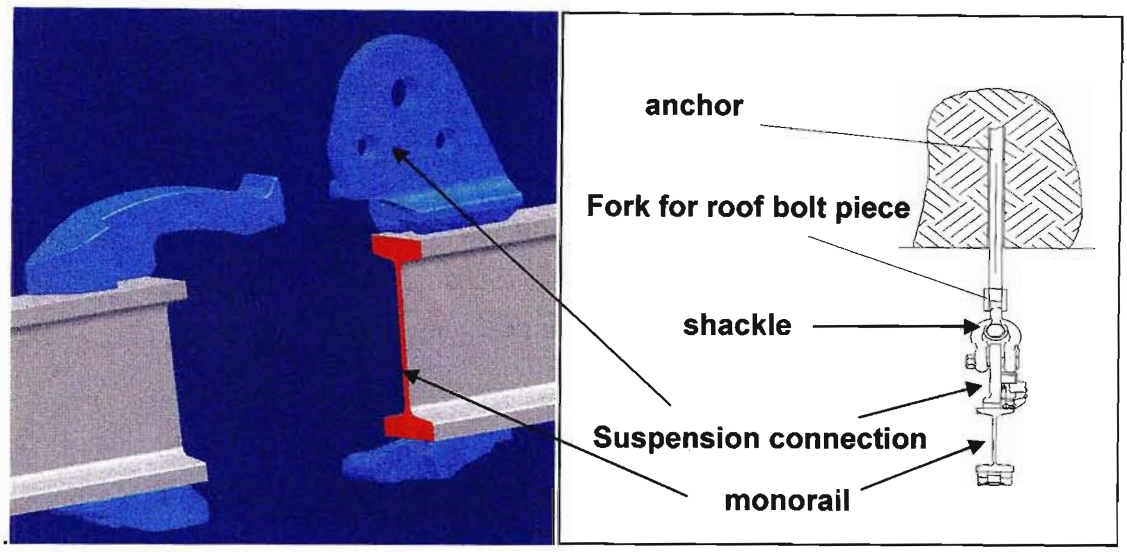


Figure 4-30: I-beam rail suspension (Nehrling, 2002)

4.6.1.2 Capacity

The capacity of the monorail system is dependent on the draw bar pull of the drive units, the load that they carry and the grade on which they operate. Dependent on the drive system chosen, drawbar pull can vary from as little as 25 kN up to 140 kN. With a standard friction drive unit, inclines of up to 20° can be negotiated, and up to 40° for a rack and pinion type drive (Figure 4-31). On inclines, the train lengths may be limited to a single load beam, but on the level, up to six load beams can be fitted. Even longer trains are possible with additional drive units.

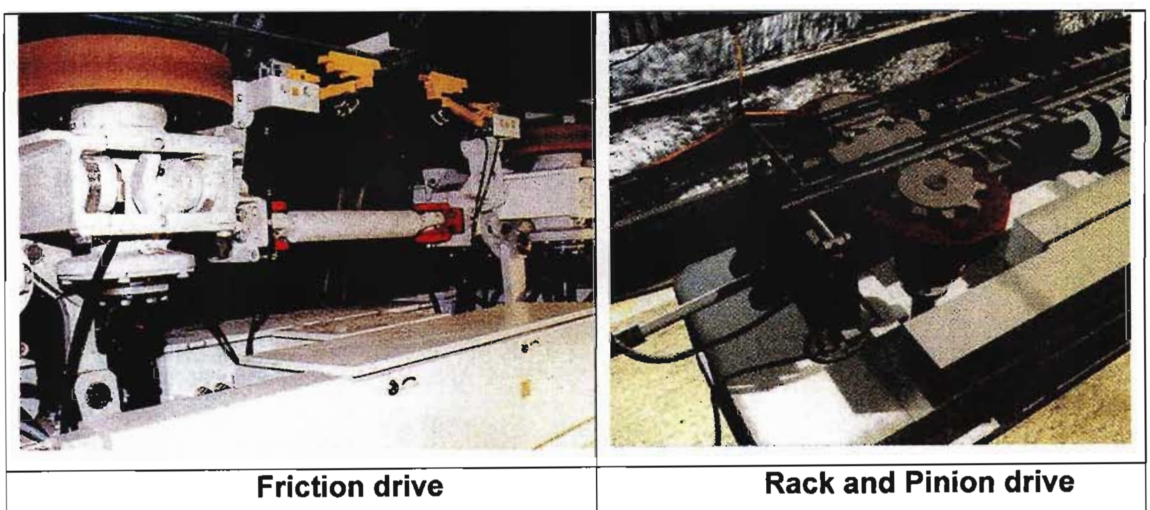


Figure 4-31: Friction and Rack and Pinion drives (Nehrling, 2002)

A typical diesel-hydraulic locomotive with four drive sets would have the following performance profile.

Drawbar pull		72 kN
Driving speed		1,5 m/s
Climbing ability	Friction drive	20°
	Rack and Pinion	40°
Capacity	Material and Rock	20 ton (single load)
	Personnel	up to 96 persons

Typical train sets (Figure 4-32) consist of two driver cabins, a single drive unit, with up to four drive sets, and three hoist carriages with two hoists per carriage. Up to six hoist carriages can be coupled to a train set, allowing some degree of flexibility in train configurations. The ability to couple and uncouple carriages is however restricted on the diesel-hydraulic and electro-hydraulic systems due to the hydraulic piping required for the drive sets. On battery or direct electric drive sets; this restriction does not apply, as no hydraulic system is present.

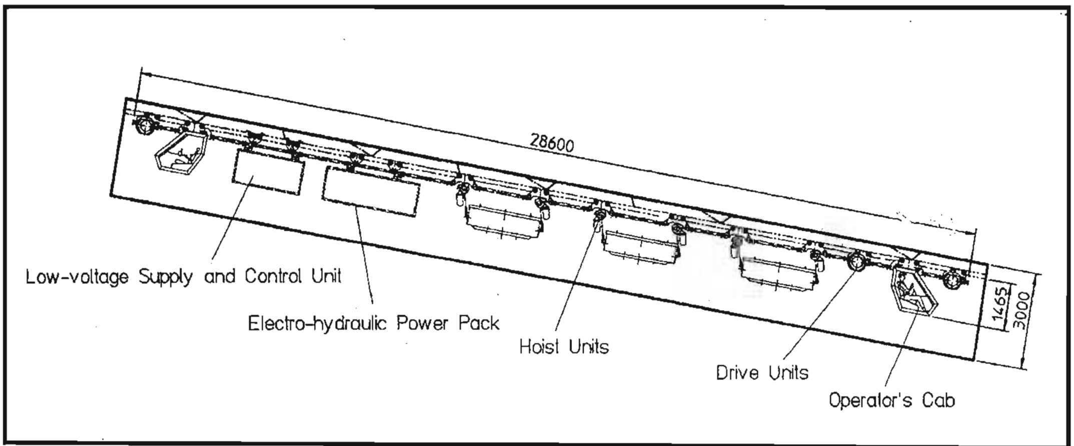


Figure 4-32: Typical monorail train set (Becker, 1999)

Through high construction and installation standards whereby vertical and horizontal "kinks" are eliminated speeds of up to 4 m/s (14,4 km/hr) can be obtained. However, common practice indicates speeds in the region of 1,5 m/s to 2 m/s are applicable.

4.6.1.3 Limits on length

The only limit on the length of the rail installation is associated with practical considerations such as travel time and communication problems. Diesel and battery locomotives can operate for as long as their respective ranges allow, while overhead traction systems can be extended as far as the electrical reticulation system, and should be well within the distances associated with underground mining.

4.6.1.4 Environmental compatibility

The electric motors have a minimal impact on the environment, with little or no contribution to noise and air pollution. However, where electro-hydraulic drive units are utilised there is the possibility of a rupture to a hydraulic line or poorly maintained connections, which may result in a spillage of hydraulic fluid.

Arcing at the electrical pick-ups represents a danger in areas with no through ventilation, although this has generally been eliminated due to the design of the overhead traction equipment. The conductors are partially enclosed by a non-conductive material, and the current collectors run inside this covering. A danger still exists at joints in the system, where poor maintenance could result in carbon build up at the joints and subsequent arcing. For this reason electric drive units are not utilised in places without through ventilation.

Diesel-hydraulic drive units effect the environment in that they contribute to both air and noise pollution. The diesel engines can also be considered a fire hazard, but flameproofing of the drive units greatly reduces this risk. The major environmental consideration is the diesel exhaust fumes from these units. A further consideration is that the diesel units contribute a significant amount of heat to the atmosphere.

Pneumatic systems are environmentally compliant, as they do not add any pollutants to the area.

4.6.1.5 Powering systems

A variety of powering systems are available. For long distance tramming, electro-hydraulic, diesel hydraulic, direct electric drive and battery drive units are available. In addition to this, pneumatic shunting units are available for short distance operations and rope driven units are currently being considered.

Electro-hydraulic drive units use electric power from conductor rails to drive hydraulic pumps. The hydraulic pumps supply power to a series of up to four drive sets. Diesel hydraulic drive units operate on the same principle, but with the diesel motor driving the hydraulic pumps.

Direct electric drive units are available making use of electric power from the conductor rails to operate the drive sets directly by means of electric motors and gears. Battery drive units utilise a battery to provide the power source. Where electric drive units are utilised, insulated conductor rails attached directly to the I-beam are installed for the full length. The conductor rails consist of a core conductor inside a shell of non-conductive material. The underside of the conductor rail is open to allow the power pick-up system to attach to the conductor rail. Similar to the requirements for conventional rail operations, the electrical supply is split up into different sections, each with its own transformer and electrical reticulation system. The nominal voltage supply can be either 525 volt or 1000 volt, 3 way AC power is supplied to the drive units by means of collectors connected to the conductor rail with the contact points concealed within the conductor rail. This type of system is not suitable for use in mines with a flammable gas (methane) contaminated environment as there is a risk of arcing occurring at the contact points, although many mines have had exemption granted by the Government Mining Engineer from specific requirements of the Minerals Act and Regulations.

Diesel engines as used in the diesel-hydraulic power units are fitted with exhaust gas scrubbing and cooling systems to improve the quality of the gas emissions leaving the engine. Diesel engines fitted with these systems are suitable for use in methane contaminated environments.

Pneumatic drive units are used primarily for "shunting" operations. That is where equipment and materials are moved short distances, such as from the cages to pick up points. This type of drive unit is not suitable for long haul operations as it requires a minimum (3,8 bar) air pressure to operate optimally and long hoses increase pressure losses.

Hydropowered drive units require high pressure water. This system is also restricted by the length of the hose and current designs require a driver to travel with the monorail.

Electrically powered systems with at least two drive units are currently favoured because they are not restricted by length. However, these units are not suitable for development or raising conditions as through ventilation is not present.

Drive wheels for all self propelled monorails are pressed against the web of the rail using hydraulic cylinders, providing the friction force for the drive and braking effort. Tyres are manufactured with a polymer rubber to improve the friction force. However, the force required for the tractive effort is very high which leads to a enormous wear rates.

For rack and pinion drive sets, a toothed rail is welded either to the underside of the rail, or to the top flange of the rail. The drive wheel is then pressed against the toothed section. Rack and pinion drive wheels and rails have little wear, but there is a price premium in that the toothed rail is significantly more expensive than the standard I-beam rails. Hybrid systems with both friction and rack and pinion drives are available, and these combine the climbing ability of rack and pinion with the higher speed capabilities of the friction drive.

The rope driven system operates on rails and is driven by a 55 kW winch. The system would be driven by a single continuous rope redirected by a return wheel at the end of the monorail's travel (Figure 4-33). The master trolley and the carrying trolleys are attached to the rope with clamps.

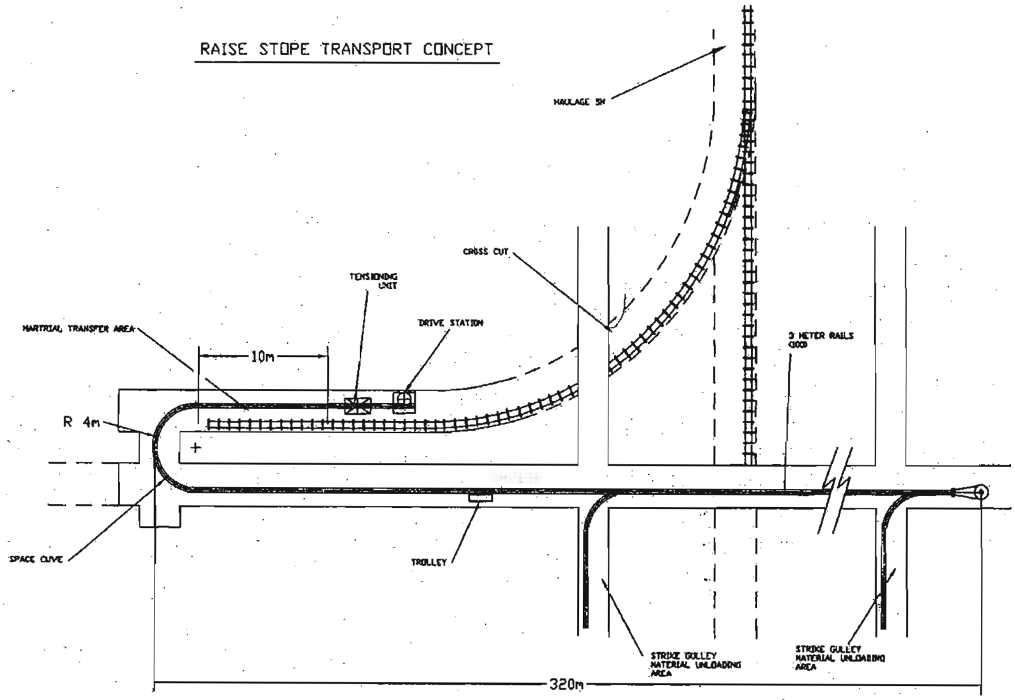


Figure 4-33: Rope driven monorail system (Nehrling, 2002)

Monorail load capacities and driving speed ranges at specific maximum dip angles for various powering systems are as presented in Table 4-9. Note that the specifications vary depending on the monorail powering systems and the number of drive units used. It is therefore important to ensure that the maximum dip angle is known, since the monorail drive units are specially designed for particular maximum dip angles.

Table 4-9: Specification for monorails powered by various means

Profile/ Parameter	Number of drive sets	Driving speed (m/s)	Inclination (degrees)	Maximum payload (kg)
Pneumatic	1	0.15 - 0.75	34°	1500 kg at 4 bar 2600 kg at 6 bar
Electric hydraulic	1	0.35 - 0.7	34°	3000
Hydropower	2	0,20	35°	1800
Diesel-hydraulic	2 - 4	1.5 – 2.0	27°	5600 – 8300
Electric	2	0.35 – 0.5	35°	3300
Rope driven	1	1, 0	40°	3000

4.6.1.6 Boarding and alighting

Due to the elevated nature of the monorail, special boarding and alighting platforms need to be constructed where it is not possible to lower the monorail to ground level. It is possible to make use of ladders for boarding and disembarking, but the safety aspect of this should be considered, especially as the personnel carriages would be prone to rocking from side to side owing to the nature of the single rail support structure. Where possible, the monorail system should be lowered to ground level to allow for boarding and disembarking to take place in such a way that persons can simply step into and out of the cabins. Some form of guide rails should also be provided to prevent the carriages from rocking from side to side.

4.6.1.7 Maintenance

The I-beams and conductor rails require a high maintenance effort in order to ensure that the system runs smoothly and efficiently. Misaligned rail sections can snag on the trolleys of the load carriages and with the high drawbar pull that the monorails are capable of can lead to the damage of the couplings and to rail sections being bent. Ground movement can affect the alignment of the monorails, and falls of ground could damage the rail sections and conductor rails. While wear on the drive and running surfaces of the rails is minimal, some of the local mines are experiencing problems with failures of the rail supports as pins and eyehooks are prone to malfunctions.

It is unlikely that a monorail will derail unless the rail sections have been badly damaged. At an incident at Impala Platinum Mines, a wheel set on one of the load carriages caught on the joint between two rail sections. The drawbar pull of the drive units was such that the drawbar of the carriage was torn from its mountings. The rail section was extensively bent, but no derailment occurred.

With more than one drive set per train, breakdowns of individual drive sets will not disable the train and the train will be able to be driven to the workshops for repairs. The hoist system allows for the materials being transported to be lowered to the footwall, and for a back-up train set to drive to the point of breakdown and pick up the waiting materials. In the case of a diesel drive unit failure; it will be necessary to tow the stricken unit to a workshop.

4.6.1.8 Installation considerations

Tunnel layouts should not have a major impact on the monorail system as underground monorails are generally suspended from the hangingwall and can be installed during the development stage. The rock bolts used to support the monorails can potentially form part of the roof support system. Extending the system is a matter of hanging the I-beam from the roof supports by means of adjustable chains, and with the use of two chains per support bolt, horizontal and vertical alignment is easily maintained even in the case of hangingwall movement. Misalignments due to sidewall heave, however, may require that new supports be provided.

4.6.1.9 Personnel carriages

The personnel carrying capabilities of the monorail are limited to the length of train sets and the type of personnel carriages available. Where monorails operate on an incline, a maximum of three personnel carriages is recommended. However, where the monorail operates on the level up to six carriages can be used per train set, although longer train sets are possible. Personnel carriages currently available in South Africa are limited to eight passengers (Figure 4-38), but 16 seat carriages can be supplied allowing up to 96 persons per train.

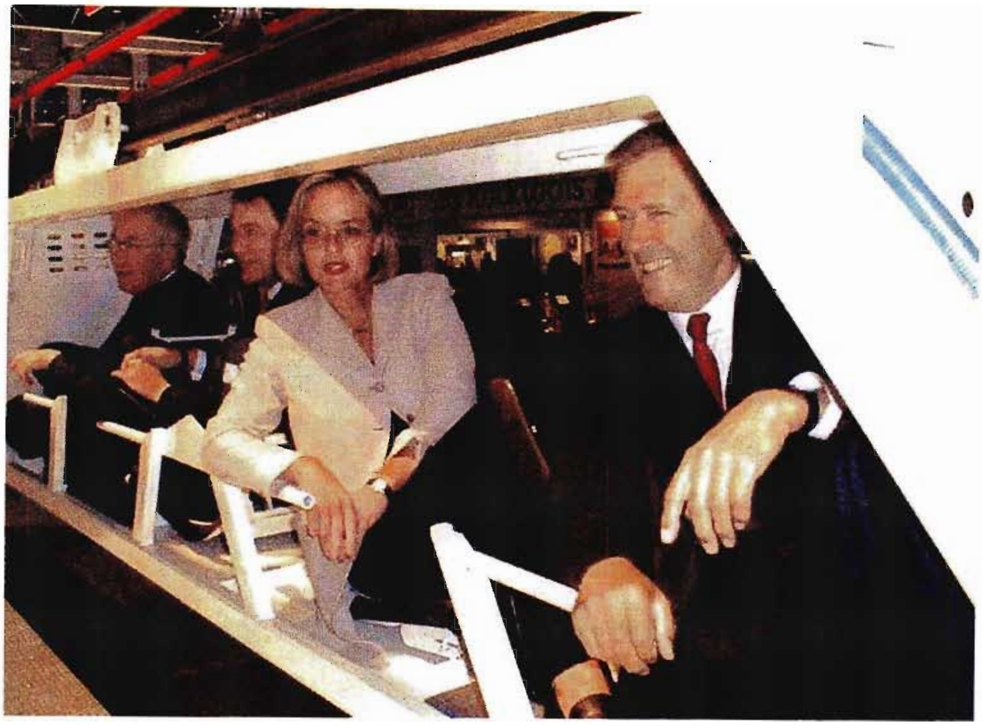


Figure 4-34: Personnel carrier (Nehrling, 2002)

4.6.1.10 Material handling

The principle of suspending material from a beam or rail attached to the hangingwall is one that is employed both underground and on surface. The primary role of the monorail is to move heavy material and up to three material carriages can be coupled to a train. Monorails can be installed in virtually any layout able to accommodate horizontal curves of 4 m radius in the horizontal plane and no less than 10 m on the vertical. Gradients of up to 20° can be negotiated with a friction drive, and up to 40° with a rack and pinion drive. The load carrying ability of the system however decreases with the degree of inclination to be negotiated. A drive set with a 10 ton load at 2,5 m/s capacity on the level, is reduced to a 5 ton load at 1,5 m/s on a 10° incline and a 2,5 ton load at 1 m/s on a 20° incline. Tolerances at joints range between 2,5° to 3° on the horizontal and 1,5° to 2° on the vertical axes. The ultimate performance and safety of the monorail depends on these tolerances not being exceeded. Therefore, installation of the support is critical.

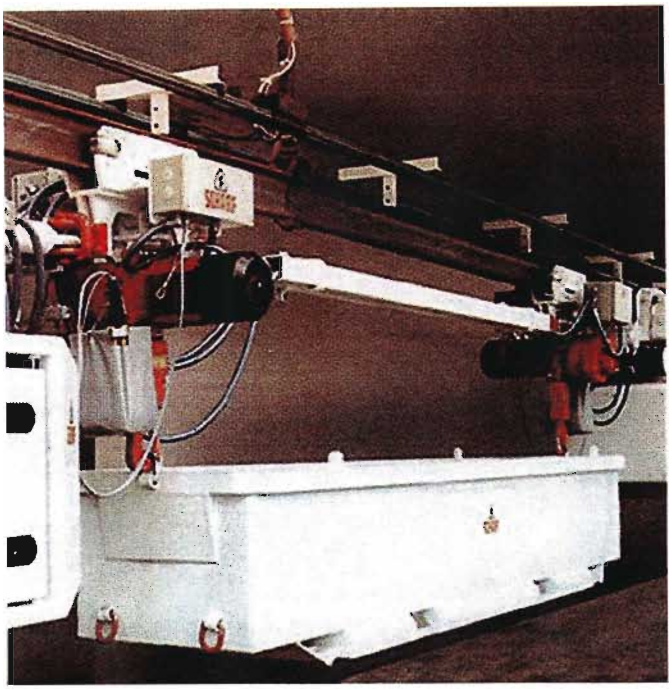


Figure 4-35: Material container (Nehrling, 2002)

4.6.1.11 Rock handling

The monorail is capable of transporting rock up to 20-ton payloads, however the system is not ideal for rock removal as it system is limited by the strength of the I-beam rails and associated suspending structure. The monorail system can operate as a combined loading and transporting unit, or only as a transportation unit. In the case of the combined unit, the rail mounted shovel loads into a 2,5 m³ container (Figure 4-36). As the loader is busy loading the containers sited near the blasted face, a second unit transports the filled containers to the rockpass (Figure 4-37) or transfer point, where they are tipped.

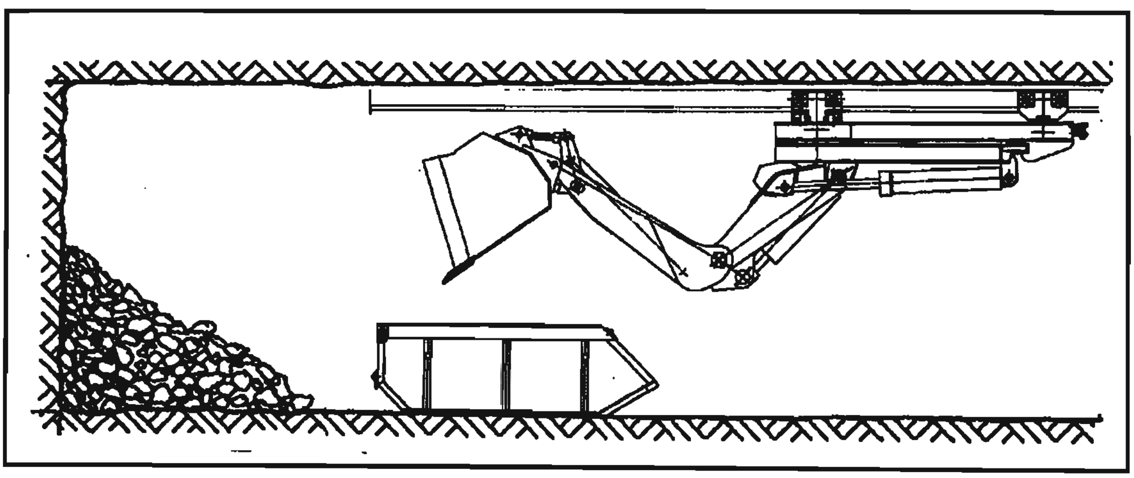


Figure 4-36: Loader cleaning an end into a container (Webers, 1999)

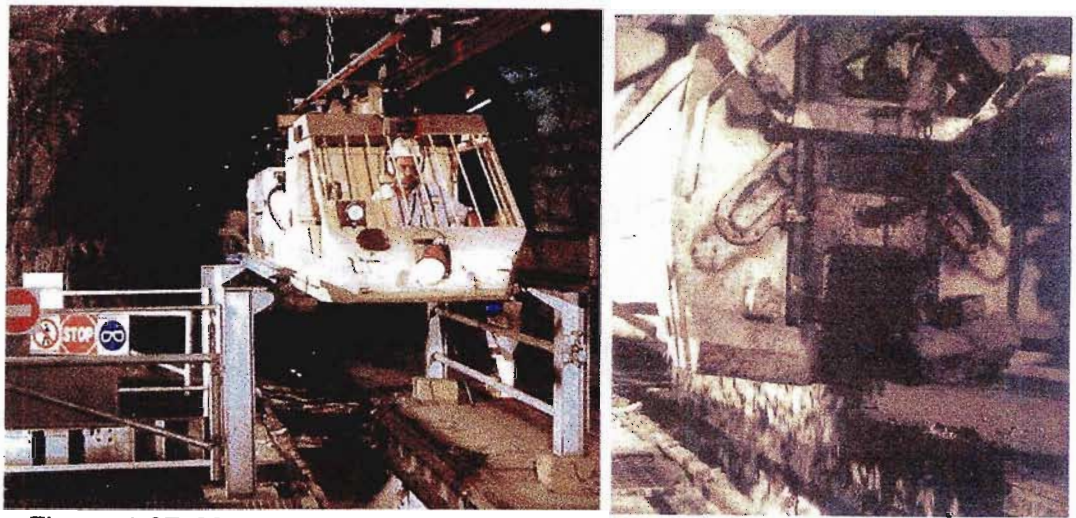


Figure 4-37: Monorail tipping ore into shaft rockpass (Nehrling, 2002)

4.6.2 Evaluation of monorail system

4.6.2.1 Size

The monorail does not take up any significant space laterally. Vertically, however, space is required to allow for the rail support structure and the I-beam. The beam used for the monorail is 140 mm high in section and with the addition of electrical power supply brackets extends this to 250 mm. A further 200 to 500 mm is necessary for the monorail support structure. Additional space is also required for the hydraulically operated switches at intersections, but this is localised at the switches.

Carriages and drive units are designed to fit into small tunnels. Standard size carriages hang 1,1 m to 1,3 m below the I-beam rail, and a further 200 mm to 300 mm clearance should be allowed below the carriages and drive units. Thus, the minimum height necessary for the installation of a monorail in a haulage will be 2,1 m. Smaller size carriages, drive and battery units are available as special orders, which allow monorails to be operated in tunnels with a height and width of as little as 1,3 m.

Due to the nature of the load beams, virtually any size and shape of load can be transported. However, when monorails operate in small cross sectional haulages, care must be taken to ensure that the material load can fit within the tunnel cross sections.

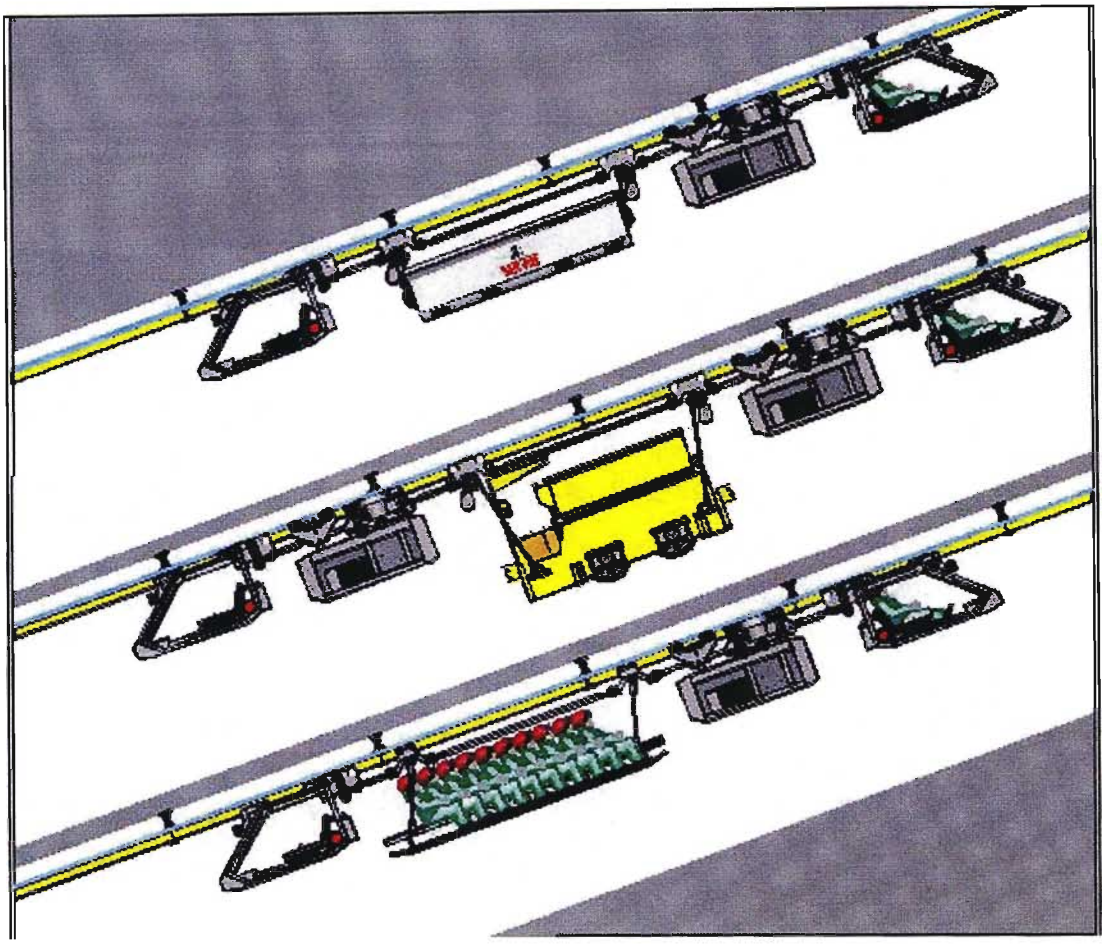


Figure 4-38: Different load configurations (Nehrling, 2002)

4.6.2.2 Flexibility

Monorails can reach virtually any part of the mine where the monorail track structure exists. An added advantage over conventional trackbound transportation is the ability of the monorail to climb gradients of up to 40°. Battery and diesel drive units can operate independently of any external power sources. However, electric powered drive units are bound to the part of the track structure that is fitted with an electric power supply. In the case of hyrdopower and pneumatic powered units, they are restricted by trailing hoses.

Due to their ability to climb inclines, their small drive units and load beams, monorails have the potential to operate in the stope. This is advantageous as material is not double handled and potentially material can be transported from the shaft station to the stope face without changing transportation systems. Monorails are also able to work in conjunction with conventional trackbound transportation as the monorail track is able to be led over the conventional track, enabling loads to be transferred from trackbound to monorail, and vice versa.

4.6.2.3 Environmental compatibility

The same considerations apply to the diesel, battery, and electric drive units for monorails as for conventional trackbound locomotives. Diesel powered drive units produce heat and noxious exhausts fumes and thus are not suited for ultra deep level mining conditions with high ambient rock temperatures. Electrically powered drive sets carry the risk of arcing at the power pick-up interfaces, with the corresponding risk of explosion in combustible gas (methane) contaminated mines. As with conventional trackbound transport, the advantages and disadvantages of each system need to be weighed up against the likely environmental conditions, with safety and expected duty being the deciding factors as to the type of drive units used.

4.6.2.4 Control and automation

The use of a leaky feeder radio transmission system allows two-way radios to operate. This facilitates communications between drivers and a central controller, thus the controller can advise drivers on operational issues. Transponders can be placed on the drive units and combined with sensors positioned on the monorail track to facilitate an integrated traffic control system.

As all switches on the monorail system can be operated electrically, hydraulically or pneumatically, it is easy to automate the switches. Thus, automatic switching systems can also be provided for the control of trains.

Automation of drive units can be attained through the installation of an additional control circuit in the electric power supply system for electrically operated drive units. Alternatively, a leaky feeder system can be used in conjunction with radio receivers in the driver cabins or drive units to allow remote operation of the trains.

4.6.2.5 Cost implications

The cost of a monorail train varies depending on the powering unit used and the components that make-up the train. Costs are site specific and should only be used as a guide.

The cost of the monorail drive units and other rolling stock is approximately R3 600 000 per electro-hydraulic train set, consisting of 2 driver cabins (front and rear), a 130 kW drive unit with three drive sets, and six hoists (three load beams).

Additional load beams consisting of two hoists amount to ±R150 000 with additional drive set costing R300 000.

An electric train consisting of two driver cabins, a 130 kW drive unit with four drive sets, and six load beams would cost approximately R4 800 000. This cost does not include additional items such as special material cars, personnel cars or hoppers, which will have to be supplied as a special order. Overall, costs are provided in Table 4-10 and Table 4-11.

Table 4-10: Monorail - cost of drive units (2003)

TYPE	POWER (kW) AT 2150RPM	ESTIMATE COST
Electro-hydraulic Power Pack	130 kW	R660 000
Diesel-hydraulic Power Pack	66 kW	R700 000
Power Supply and Control Unit	-	R350 000
Electric Direct Drive (EMTS) With Friction Drive	2 x 29 kW	R950 000

Table 4-11: Monorail - cost of other equipment (2003)

TYPE	ESTIMATE COST
Operator Cabins	R180 000
Lifting Beam with Two Hoists	R110 000
Hoist Unit	R40 000
8 Seater Personnel riding Cabin/Beam	R50 000
16 Seater Personnel riding Cabin/Beam	R76 000
Electric Switch (Turnout)	R95 000

Cost for other monorail units are:

- Pneumatically driven monorails cost under R500 000 (includes powering and drive units and a hoist carriage).
- Diesel powered trains cost up to R4 000 000 (for 2 driver cabins, 4 drive units, 1 diesel power pack and 4 lifting beams).
- Hydropowered R1 800 000 (including the power pack, drill rig, drive unit, man carriage, load trays, manifolds, hoses and rails).
- Rope driven monorails cost in the region of R550 000 for a typical raise, however this cost could be dramatically reduced if units are manufactured in South Africa.

The I140E I-beams cost R2000 per 3,0 m length inclusive of the suspension of the beam. Maintenance cost are in the range of 5% of the capital cost per year for the first five years, increasing to 10%, as the train gets older.

4.6.2.6 Compatibility with other systems

It is possible to use monorails in conjunction with trackbound systems as the monorail can operate directly over a railway line and, with the lifting hoists, be used to pick loads up off the trackbound material cars and transport them (see Figure 4-39). Load carriages with two lifting hoists can carry loads in a variety of configurations. Loads can be palletised and two pallets per load carriage can be carried, else bundles of pipes, roof support beams, etc. can be slung under the load beams. Special load carriages or containers can be fabricated for the conveyance of explosives and in the case of rock transportation; hoppers can be slung beneath the load carriages.

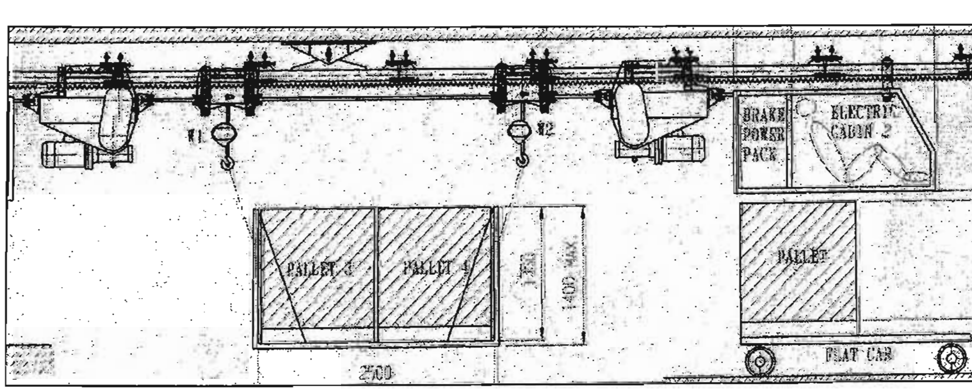


Figure 4-39: Monorail operating above material car (Deale, 2002)

4.7 Conveyor belts

4.7.1 Audit of conveyor belt system

4.7.1.1 Overview

Conveyors have long been considered one of the most efficient ways of transporting rock over short to medium distances. Belt conveyors operate continuously without loss of time for loading and unloading or empty return trips. Scheduling and dispatching are unnecessary as the material is loaded to and unloaded from belt conveyors automatically. Belt conveyors and chain conveyors are frequently used in mines and plants to handle rock. However, they are seldom used underground in South African gold mines other than in the shaft area.

A simple belt conveyor (Figure 4-40) typically incorporates an endless reinforced rubber belt guided on idlers, which are troughed on the carrying side. The belt is driven through a drive pulley situated at the head of the conveyor with a similar non-driven pulley would be located at the tail end. In order to provide sufficient belt tension for the drive unit, a take up pulley is situated on the return side of the belt and is tensioned using weights or by applied force.

Rock is fed onto the belt from a storage bin by means of a chute positioned close to the tail end and travels along the conveyor before being discharged at the head pulley. There are of course numerous variations on the basic layout including multiple loading and discharge points. Drive arrangements for large conveyors may have several drive pulleys to produce the belt tension required to move the belt and its load.

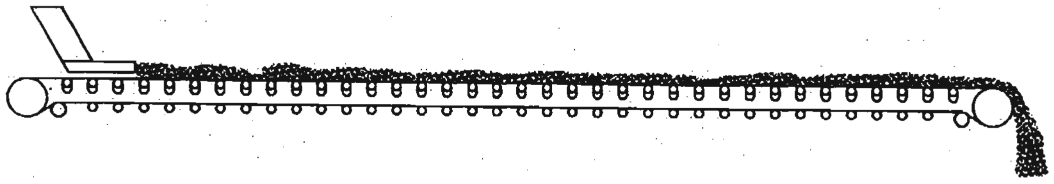


Figure 4-40: Simple belt conveyor (DuPlessis *et al.*, 1999)

Personnel

In Europe conveyor systems have also been used extensively for the transport of personnel, to the extent that this is now one of the most widely used methods of transporting the workforce. Speeds of up to 2,6 m/s are considered the norm, with speeds of 4 m/s to 5 m/s possible with the use of specialised boarding and alighting stations.

In South Africa there are currently very few personnel riding conveyors (Figure 4-41). Those that are in operation are restricted to a maximum speed of 2,5 m/s and the lengths of these conveyors are in the region of 1 km. Only two types of conveyer variations are suitable for personnel riding and these are:

- Conventional belt conveyor.
- Cable belt conveyor.

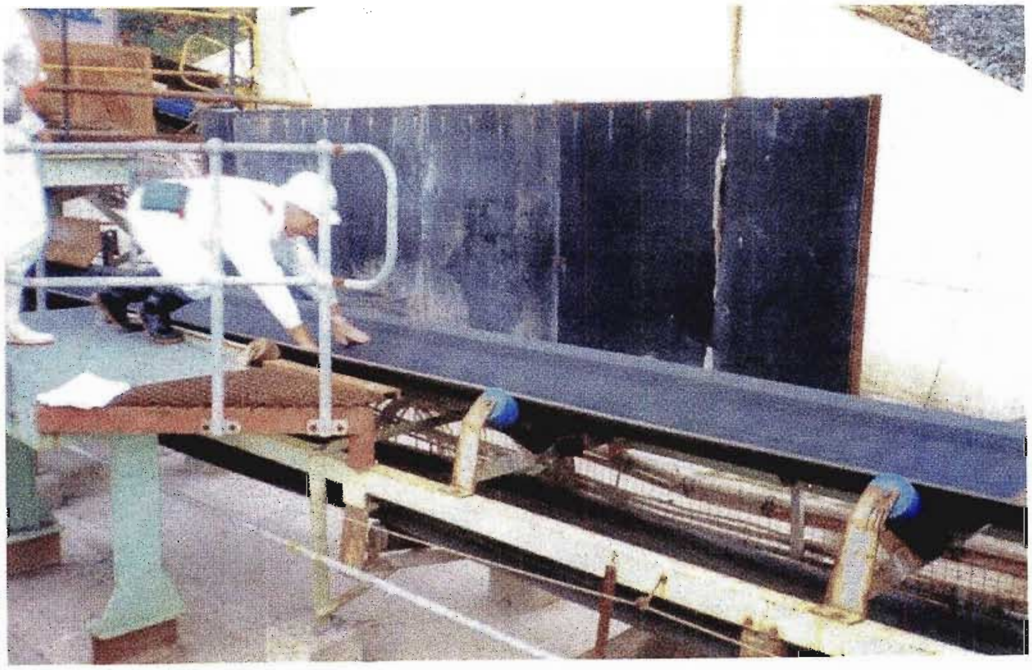


Figure 4-41: Worker boarding a personnel riding conveyor

Currently, only the conventional belt conveyor is being used in South African underground mines, hence it is the only conveyor considered. However, the boarding and alighting arrangements, as well as speeds and maintenance requirements, are similar for a cable belt conveyor. Where a personnel riding system is used, boarding and alighting platforms are constructed alongside the conveyor in order to allow the workers to embark or disembark.

Material transportation

Due to the limitations in carrying capacity, the use of conventional conveyor belts to transport material is limited to hand carried items only. Thus, the use of conveyors in mining operations would necessitate the need for an additional material transportation system, for example a trackless, trackbound, or monorail system.

Rock transportation

There are a number of variations within conveyor belt technology that are suitable for rock transportation. These are:

- Conventional belt conveyors
- Cable belt conveyors
- Enclosed conveyors
- Tube conveyors
- Vertical conveyors

4.7.1.2 Capacity

Personnel transportation

The carrying capacity of the conventional belt conveyor is a function of the belt speed and spacing between personnel. Theoretically, a conveyor with a belt speed of 2,5 m/s and a spacing of 5 m between persons, would have a carrying capacity of 1800 workers per hour (Equation 4-5).

$$\text{Personnel capacity of conveyor} = \frac{\text{Speed of conveyor}}{\text{Spacing distance between workers}} \times 3600 \, \frac{s - \min}{\min - hr}$$

$$= \frac{2.5m/\sec}{5m} \times 3600 \, \frac{\sec - \min}{\min - hr}$$

$$= 1800 \text{ workers}$$

Equation 4-5: Personnel capacity for conveyor

Rock transportation

The carrying capacity of a typical belt conveyor is a function of the following:

- Belt width.
- Belt speed.
- Troughing angle.
- Material properties.
- Belt strength.

Conveyors with relatively narrow belts are suitable for underground applications however; the width of the belt must be sized to carry the maximum load that would be deposited on any one portion along the belt length. A belt designed to carry an average of 80 t/h when loaded irregularly would need to be significantly wider than a belt loaded at a constant rate. Conveying speeds of 2,5 m/s are typical for this type of conveyor although modern installations can operate at far higher speeds. Conveyor belts are able to handle the tonnage requirements for a typical level. A small 500 mm wide belt would be more than sufficient when operating at a speed of 2 m/s. However, the actual belt width would probable be selected based on the lump size of the material. Conveyors are better suited for the transportation of large material quantities where the single conveyor would have many advantages over other system, which would require large numbers of operating units.

4.7.1.3 Limits on length

Conveyor lengths of several kilometres are possible; the limiting factor associated with conveyor length is the belt strength. There are currently no guidelines on material type in South Africa, but the trend locally is towards the use of steel cord belts. These are expensive to produce and their heavier weight requires larger drive units. The other limiting factor is that conveyors are unable to negotiate bends requiring an additional conveyor for every change of direction. For a typical level, this could account for at least two or three conveyors to serve a single mining area.

4.7.1.4 Environmental compatibility

Belt conveyors are environmentally more acceptable than other means of transporting ore, as electric or electro-hydraulic drives are utilised for powering conveyor systems, these have little impact on the environment except in the minimal production of heat. As the drive units are normally situated near the shaft station, the ventilation from the shaft will easily dissipate any heat generated by the motors. However, in the advent of a fire, the risk of smoke contamination from burning rubber is high and for this reason underground conveyor belts are required to be self-extinguishing.

While conveyors are not as sensitive to footwall movement as other systems, sidewall heave can have an impact on the operation of the conveyors, as they are not able to accommodate horizontal changes. Any support system for a conveyor then requires having some form of horizontal and vertical adjustment mechanism in order to allow for the rectification of any misalignments.

Crusher stations that are required to size the rock for conveying should be located close to the working areas. These crushers will produce heat and dust. The energy required to crush rock to 150 mm or less will not be significant and the heat and dust generated should be of little consequence if located in the return airway. However, open belt conveyors will allow dust to be blown from the rock being conveyed, polluting the atmosphere. The amount of dust liberated would be dependant upon the air velocity passing over the belt and the moisture content of the rock. In dedicated tunnels, this would not be a serious health hazard.

Spillage at loading points through poorly designed feeding system and chutes is common, as is spillage through overloading. Damaged or worn belts deposit rock along the length of the conveyor. Fines on the surface of the belt can also be deposited as the empty belt returns. Even the best cleaning systems have limited effectiveness and are difficult to maintain in practical situations.

In terms of ventilation requirements, conveyor installations are permanent obstructions in the tunnels and restrict the airflow along the full length of the belt. Therefore, the tunnel size should be increased to limit to account for the dropin pressure.

4.7.1.5 Powering systems

All large conveyors make use of electric or electro-hydraulic motors as a source of power, which is transmitted to the drive pulleys through a gearbox. Other than for small conveyors, a conveyor drive unit should utilise fluid couplings, electrical soft start or variable speed drives to apply the driving force smoothly to the belt, preventing slippage and over-tensioning. The drive and its associated electrical switchgear are positioned close to the head of the conveyor. For a system with multiple units, additional switchgear and cabling would need to be installed for each drive station.

4.7.1.6 Boarding and alighting

The maximum allowable gradient for personnel riding conveyors is 1:4 or 14°, except at boarding and alighting stations (Figure 4-42 and Figure 4-43), where the maximum gradient is not permitted to exceed 1:6 or 9.5°. A minimum belt width of 750 mm also applies.

Conveyors operating at a speed of 2,5 m/s are required to have boarding platforms that are not be less than 1,5 m in length, and must be between 450 mm and 900 mm in width. The length of alighting platforms must be at least four times the belt speed in metres, with a minimum of distance of 6 m with the same restriction widths applying to boarding platforms.

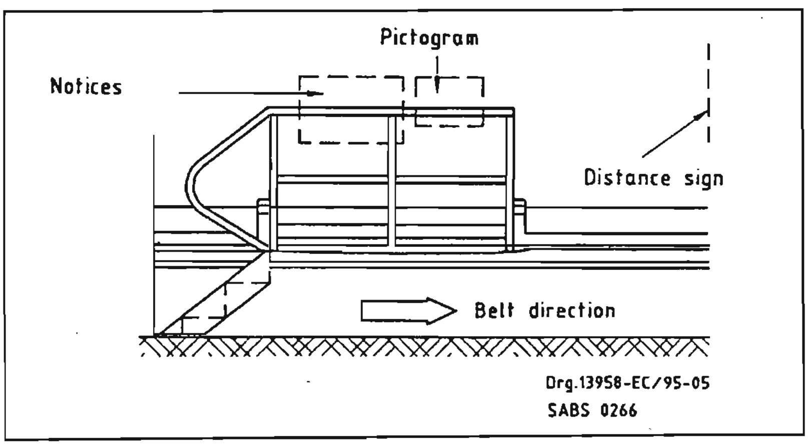


Figure 4-42: Boarding platform (Hughes, 1999)

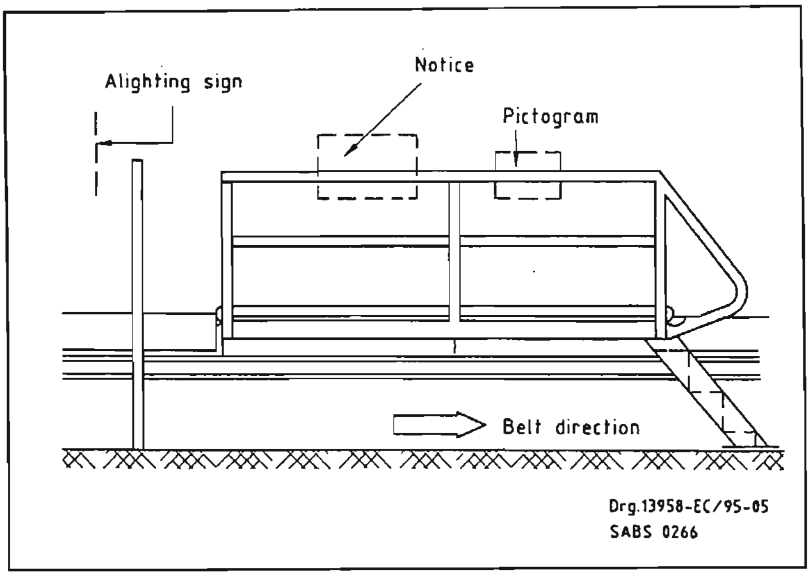


Figure 4-43: Alighting platform (Hughes, 1999)

Various safety devices need to be placed on the conveyors just beyond the alighting platforms. These safety devices are to prevent persons from being pulled into the drive or return units, or thrown into bins should they fail to alight from the conveyor. A safety gate with cut off switches is installed 2 m after the alighting platform to stop the belt. A further belt plough is positioned near the safe end of the conveyor to throw any persons off the belt should the safety gate malfunction (Figure 4-44).

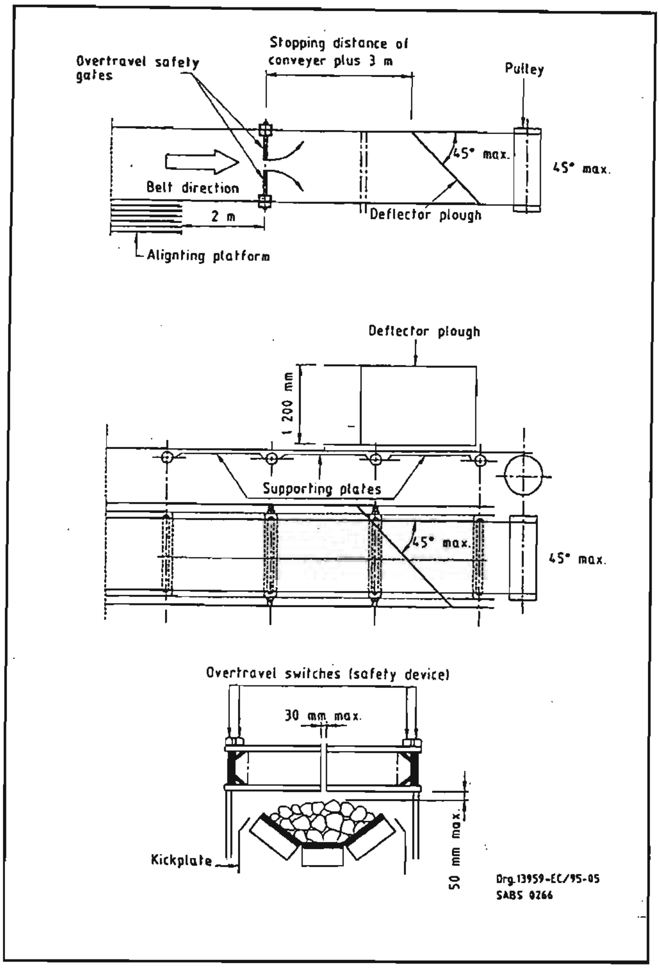


Figure 4-44: Safety devices (Hughes, 1999)

Minimum belt widths of 750 mm are required, for both the safety and comfort of the persons riding the conveyor. At boarding and alighting stations, headroom in the tunnels may need to be increased to allow for the minimum clearances required at loading stations (Figure 4-45). These minimum clearances are 1,4 m for boarding and 1,6 m for alighting. These clearances also have to be maintained 9 m beyond any boarding platform and 6 m before and 3 m after any alighting platform.

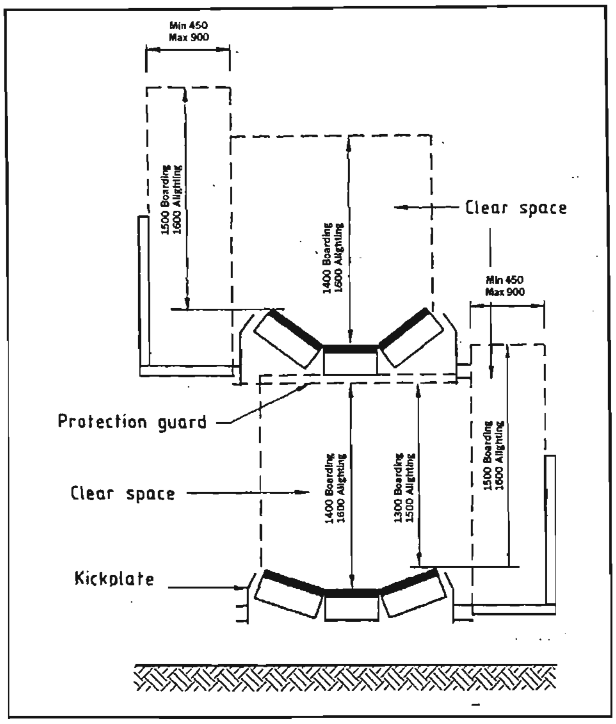


Figure 4-45: Boarding and alighting clearances (Hughes, 1999)

4.7.1.7 Feed and discharge arrangements

The successful operation of a belt conveyor requires: first, that the conveyor belt be loaded properly; second, that the material carried by the belt be discharged properly. Belt widths, along with tonnage requirements are influenced by the maximum lump size of the rock. Unless the mining methods can guarantee the maximum lump sizes in the rock, some form of primary crushing and screening will be required before the rock can be fed onto the belt. As the belt width is designed for the lump size of the rock, good sizing will be an important factor. In addition, to minimise the belt width, rock must be fed onto the belt in a regular flow to ensure an even distribution along the belt with the average tonnage as close to the design peak tonnage as possible.

The prime importance of feeding is to place the ore centrally on the belt in such a manner that the ore is discharged in the direction of belt travel as nearly as possible equal to the velocity of the belt itself. Thus, reducing belt wear, utilising minimal power to operate the belt, reduce spillage, degradation and dust. There are several types of feeder available, the most common being vibrating feeders using magnetic or 'brute force' motors. In other cases box chutes are manually operated (Figure 4-46), or operated by air cylinders. A suitable storage bin sized to provide sufficient buffer capacity is also required to ensure a constant feed.

Belts can be loaded at any point along the length of the conveyor and a belt conveyor requiring a number of loading points concurrently would need to be designed to accommodate the peak loading from all the loading points. Alternatively, belt weighers could be positioned before the loading points to determine the actual load on the belt, thus preventing the overloading of the belt.

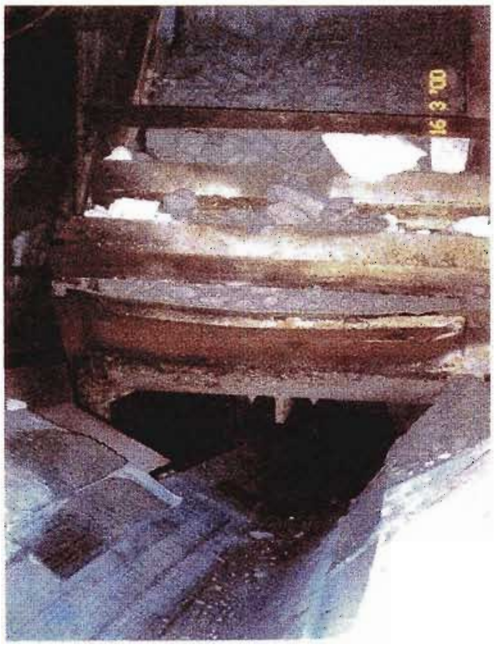


Figure 4-46: Manually operated loading box

Many of the problems associated with conveyors can be attributed to poor loading onto the belt. The design of chutes is often neglected and is seen simply as a means of getting rock onto the belt. Poor chute design can have a number of consequences including:

- Chute blockage.
- Generation of dust.
- High noise levels.
- Uneven belt loading.
- Spillage.
- Impact on belt causing damage and wear.
- Belt wear from side skirts.

Ore carried by a belt conveyor can be discharged from the belt in different ways to affect certain desired results. The simplest method of discharge from a conveyor belt is to let the ore pass over the end pulley and fall into a pile, however rock can also be discharged from any point along the conveyor using fixed or moveable trippers. By adding a suitable chute, the discharge can be directed to a pile, a bin, or another conveyor. The primary requirement of the discharge chute is to collect all the material discharged and cleaned from the belt. In the case of transferring rock from one belt to another, the chute design must integrate with the trajectory of the ore leaving the belt, so that the ore will fall on the bottom of the chute and not on the succeeding conveyor itself.

4.7.1.8 Installation considerations

Where personnel riding is allowed on the conveyor, it will be necessary for uninterrupted straight runs. Long radius curves are possible by tilting the idling rollers to angle the belt. Changes in direction are normally achieved by the introduction of an additional conveyor. Whereby rock is transferred from one conveyor to another travelling in a new direction.

Both fabricated fixed steel structures and modular systems comprising frame sets and idlers are suitable for use as conveyor supports. These can both be fixed to floor mounted foundations or suspended from the roof utilising rock anchors and chains. Support legs can be telescopic to allow adjustments to be made during installation or to compensate for ground movement. The use of chain support is more flexible allowing for easier adjustment, however, modular units are quick to install and are often manufactured with lightweight tubular sections. This is advantageous for maintenance purposes where damaged sections can be replaced easily and quickly. Some flexible structures using wire ropes for the horizontal members have also been used.

Belts are generally available in lengths of 250 m to 300 m depending on the thickness of the belt. Any conveyor will consist of a number of belts spliced to make up the conveyor length and are potentially weak areas as splices may fail. Thus, it is important that belt splicing procedures are carefully followed.

Distribution of the driving load along the length of the conveyor installation can reduce the tension in the belt. This can be achieved in a number of ways, for example instead of using a single conveyor; a series of conveyors could be installed. Each conveyor would discharge rock directly onto the tail end of the next conveyor in the series (Figure 4-47).

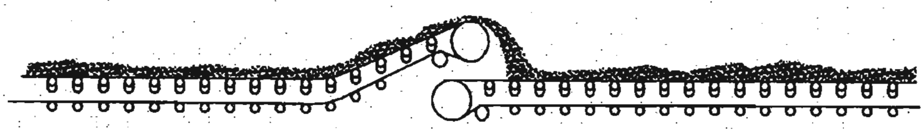


Figure 4-47: Conveyors in series (Du Plessis *et al.*, 1999)

Other methods of distributing the drive could involve the use of fixed tripper drives (Figure 4-48), which are positioned at intermediate points on the same conveyor, and which transmit their drive force to the conveyor by friction. Another method would be through the use of short narrow endless belts or rubber tyres transmitting the drive force to the top belt of the conveyor (Figure 4-49).

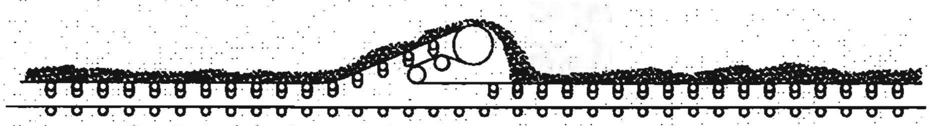


Figure 4-48: Tripper drive (Du Plessis *et al.*, 1999)

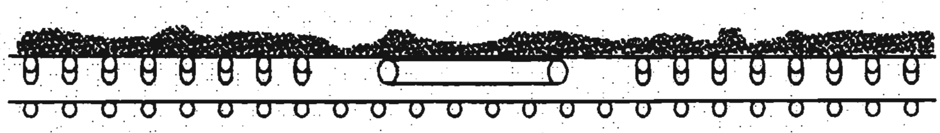


Figure 4-49: Friction drive (Du Plessis *et al.*, 1999)

As mentioned previously, one of the shortcomings of a conveyor is the need to provide a straight tunnel with little deviation in the horizontal direction. Any changes in horizontal alignment would require a new conveyor installation with drive, tail and tensioning units.

4.7.1.9 Maintenance

Spillage is one of the major problems associated with mine conveyor installations. Spillage can occur due to badly maintained belts, belt cleaning and belt loading systems. Belt splices can often be a weak point causing belt failure and spillage.

The large quantity of steelwork and equipment associated with conveyor installations, including support structures and idlers, need careful attention to ensure that the belt runs reliably and efficiently. Regular inspections of the belt are necessary in order to identify tears or damage to the belt timeously. Maintenance of the belt should be considered a high priority, with any damage repaired as soon as it is detected. Idlers, chutes and belting will require regular maintenance to ensure uninterrupted operation (excluding planned stoppages). A belt failure will completely stop ore transport operations for the sections served by the conveyor. Damage to the belt is often caused by foreign material in the rock, such as drill steels and pinchbars. Where underground crushing is used, foreign material should be removed from the rock during the screening process to prevent it from entering the system. Electro magnets and metal detectors have successfully been used to assist with the detection and removal of such objects.

Quartzitic rock associated with gold mining is abrasive and capable of causing severe wear to the belting material. Wear on the belt material itself must be reduced as far as possible by means of proper belt cover material selection. A number of steps can be taken to reduce wear including the correct feed chute design, good alignment of the troughing idlers, and non abrasive belt cleaning equipment. Personnel riding conveyors should also be thicker than rock conveyors. In order to make the ride as comfortable as possible for persons riding the belt, as well as to prevent breakages in the belt, this may have serious or even fatal consequences.

4.7.1.10 Compatibility with other systems

Personnel riding belt conveyor is ideally a dual transport medium, with both rock and workers being transported on the same belt. Restrictions are that riding on the top belt (return) is only permitted as long as the belt is not transporting rock and the rider is not within 5 m of any rock on the conveyor. Rock loading points must also be designed in such a way that they can be retracted when personnel riding is in progress.

Although it is possible to transport materials or equipment on a conveyor, the difficulties in loading and unloading and the limitations imposed by weight and size realistically dictates that another system for material transport be utilised. For example, Target Mine has utilised a tunnel width (6 m) wide enough to allow the movement of trackless vehicles, combining the system with a monorail for the movement of materials. At Bafokeng Rasimone Platinum Mine, a twin decline system is utilised, with personnel and rock being transported in the conveyor decline, and material transported in another other decline by means of an incline hoist and rail mounted material cars.

4.7.1.11 Characterisation of Cable Belt Conveyors

Cable belt conveyors are typically used where high tonnages are required for transportation over long distances. Cable belt conveyors differ considerably from conventional belt conveyors in that two endless steel rope cables carry the driving force and the belt, supported on the ropes, carries the rock load. Instead of idlers, the cable is carried and guided by small pulleys. The belt is located on the cables using a "shoe" moulded into the body of the belt. Figure 4-50 shows a section through a loaded belt section. The driving load is applied directly to the cables from the drive unit.

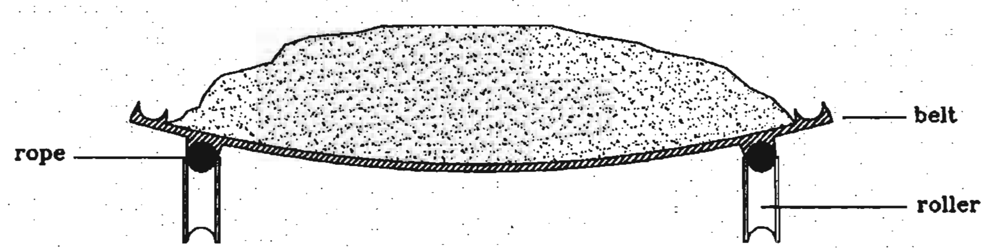


Figure 4-50: Cable belt conveyor (Du Plessis *et al.*, 1999)

Capacity

The functions relating to belt capacity are the same as those for a conventional conveyor. However, in the case of a cable belt conveyor the troughing angle is much smaller being formed by the weight of the load, or in some cases, with the addition of a pre-formed trough moulded during manufacture. Although the loaded capacity of the belt is lower than a conventional conveyor, the smooth operation reduces the flattening of the loaded profile that occurs with a conventional belt as the rock passes over the idlers. In addition, the return shoes at the edges of the belt help to prevent spillage. It must still be expected that this type of conveyor will have a lower capacity than a conventional conveyor with a similar belt width.

Limits on length

Cable belts have been installed with lengths of up to 30 kilometres using only a single drive unit. Underground installation consisting of a single conveyor run should present no problems.

Environmental compatibility

The same comments that apply to conventional conveyors can be made here, although the risk of spillage due to overloading or incorrect loading could be more serious with cable belt conveyors. A typical belt speed of 4 m/s is used but speeds of over 6 m/s are not uncommon.

Powering systems

Unlike the conventional conveyor, where the drive unit must be inline with the belt, the cable belt drive can be remote from the conveyor. The belt can be removed from the cable at the discharge point and re-located onto the returning (low tension side) cable. The driving (high tension) cable would continue to the drive unit, which could be located some distance away from discharge point. The running friction and hence, the power required will be lower than for a conventional conveyor.

Feed and discharge arrangements

Similar feed arrangements to the conventional belt conveyor would be required here although loading frames are used to centralise the rock on the conveyor. The discharge from the conveyor will be more complex as the belt must be lifted off the cable, passed through the discharge unit and returned to the cable.

Conveyor maintenance

The addition of the cable adds another item to the equipment maintenance requirements. The belt will not receive the same degree of damage associated with conventional belts, as there are no idlers in contact with the belt and the belt is not under tension. Splicing and repairs to the belt is easier as joints are not required to resist any significant tension.

Installation considerations

Gentle curves with radii of less than 200 m can be achieved by the use of special curving units. These units, which tilt the belt on additional rollers are more complex than the standard pulley units and are considerably more expensive.

Severe changes in direction can be achieved using a single belt. Angle stations discharge rock from the belt and re-load the rock after the belt has been re-directed. The cables are removed from the belts and, after passing through a series of sheaves and tensioning devices are re-directed. Although almost any change in direction can be achieved, a considerable sized station and excavation would be required to accomplish this.

4.7.1.12 Characterisation of Enclosed Belt Conveyors

Enclosed belt conveyors have introduced a significant transformation to the way in which conveyors operate. The conveyor belt is folded in two to form a bag enclosing the rock (Figure 4-51). Two profiles are attached to the edges of the belt, which fit together and are hung from rollers that guide and give support for the belt. Steel rope cords are moulded into the profiles to carry the loads required to move the conveyor.

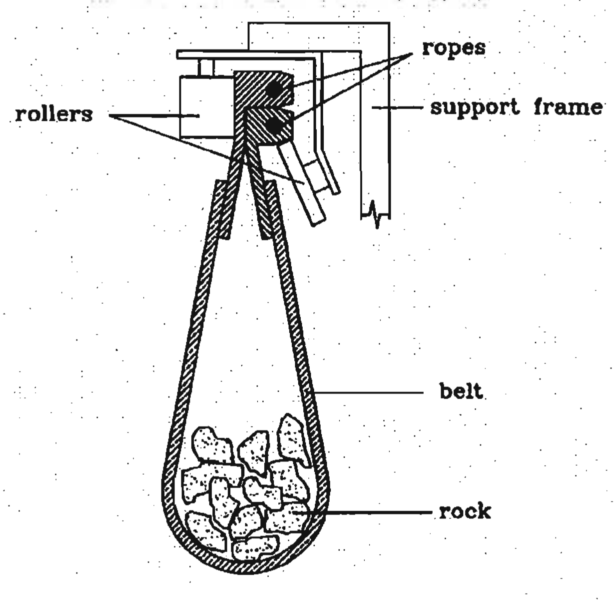


Figure 4-51: Enclosed belt conveyor (Du Plessis *et al.*, 1999)

<u>Capacity</u>

At present two belt sizes are available for this type of conveyor. The first is rated with a capacity of 17 to 170 m^3/hr while the other is rated at 170 to 680 m^3/hr. The maximum acceptable lump size is approximately 40 mm for the smaller belt and 70 mm for the larger. The enclosed pocket is quite small and larger sized lumps may cause problems when negotiating tight curves.

Limits on length

To date, conveyors with lengths of less than 1000 m have been installed. There does not appear to be any major reasons to prevent the installation of considerably longer belts. Additional drive units can be installed at various points along the conveyor to reduce tension in the belt with a number of drive units required even for a relatively short belt.

Environmental compatibility

Except during discharge, the belt is completely closed at all times eliminating spillage and the release of dust into the environment. However, primary and secondary crushing systems will probably be required and would generate additional heat and dust. An advantage of the enclosed belt conveyor is the small cross sectional area of the belt that will have a much lower restriction to the airflow as compared to conventional conveyor systems.

Powering systems

Friction drive units driven by electric motors are used to induce tension in the profiles at the top of the folded belt. Additional drives can be located at various points along the conveyor. Where the conveyor turns at an angle of 90 degrees or more, an additional drive unit could be installed. As with any multiple drive conveyor additional cabling, switchgear and control mechanisms would have to be installed for each drive.

Feed and discharge arrangements

Lump sizes specified for the belts currently in use are small and some form of secondary crushing may be necessary. Loading can take place at any point along the length of the conveyor. The belt is opened up at the start of the loading station using rollers. The loading station covers the opening completely and the belt is closed immediately on leaving the station preventing spillage. Similar precautions such as those detailed in the section for conventional conveyors need to be taken for multiple loading points. On discharge, the belt is opened out completely over a pulley and is closed again after discharging. Discharge stations can be installed at any point along the conveyor.

Conveyor maintenance

The reduction of spillage, belt damage and belt cleaning requirements are a major feature of this type of design. Damage of the belt due to foreign objects is unlikely to cause tearing of the belt or damage to the conveyor structure, although all necessary precautions should still be taken to prevent them from entering the belt.

Installation considerations

Unlike other conveyors, this system is able to negotiate very sharp turns. Changes in direction can be achieved with a bend radius of less than one metre. The envelope of the conveyor cross section is small and although the support structure is lightweight it does need to be fixed firmly and rigidly. Because of its inherent flexibility, the system should be reasonably tolerant of misalignment and small amounts of ground movement.

Transportation of long lengths of belt underground may be problematic as the combination of belt and cable make the belt bulky. The length of each belt section will probably not exceed 300 m for the smaller belt and 150 m for the larger belt. To join each belt length, the ropes can be spliced as with any wire rope and a rubber section added to bridge the gap.

Compatibility with other systems

This system is incapable of carrying personnel or equipment and therefore possibilities for an additional transporting system will be required in the return direction. It may, for example, be possible to transport backfill material back to the mining areas

4.7.1.13 Characterisation of Pipe Conveyors

As with enclosed conveyors, the pipe conveyor (Figure 4-52) is noticeably different in construction to a standard belt conveyor. The pipe conveyor has an open troughed section at the loading point. The belt is folded to form a pipe fully enclosing the rock being transported for the full length of the conveyor until it reaches the head of the conveyor where the tube is opened as it passes over the head pulley and discharges its load. Apart from the tubular cross section and idler arrangement, the conveyor is similar to a conventional conveyor with regard to the drive, take-up and tail arrangements.

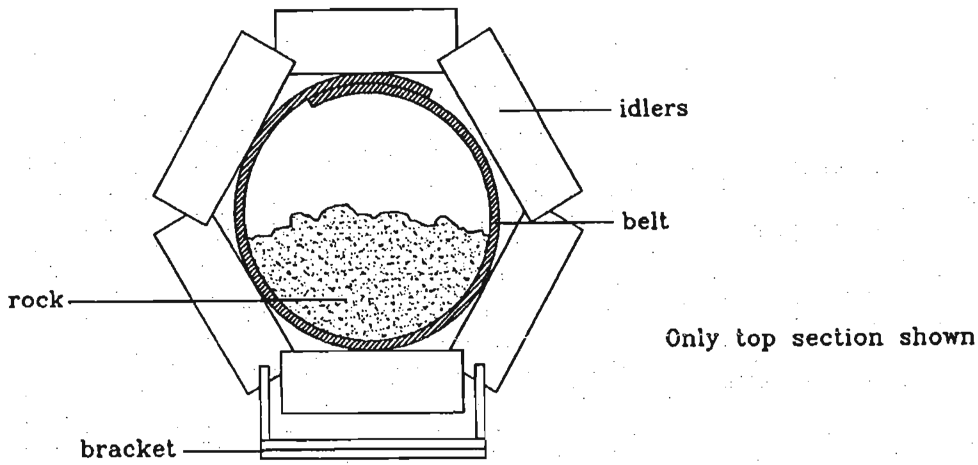


Figure 4-52: Tube conveyor (Du Plessis *et al.*, 1999)

Capacity

Belts of this type have been rated for duties up to 2 200 tons per hour with pipe diameters between 100 and 500 mm specified.

Limitations on length

This type of conveyor has similar properties to that of a conventional conveyor. The limitation on length will be primarily related to the belt strength. The required flexibility of the belt to allow the formation of the tube may not permit high strength belts to be used, however lengths of up to 5000 m have been specified by the manufactures. In addition, the nature of the design does not allow for intermediate drive units to be used hence the maximum length of a single conveyor will be limited by the tension produced at the drive unit.

Environmental compatibility

Except during loading and discharge the belt is completely closed at all times eliminating spillage and the release of dust into the environment. Spillage associated with belt cleaning is also eliminated as the dirty side of the belt is always on the inside.

Powering systems

The drive units for this type of conveyor will be similar to those of a conventional conveyor and the same principles apply.

Feed and discharge arrangements

The feed and discharge arrangements for this type of conveyor will be similar to those of a conventional conveyor with a maximum permitted lump size of 125 mm. This would require some primary crushing arrangement and in the case of the smaller diameter conveyor, secondary crushing may also be necessary.

Conveyor maintenance

The reduction of spillage, belt damage and belt cleaning requirements are major features of this type of design. Damage of the belt due to foreign objects could cause tearing of the belt or damage to the conveyor structure and thus all the necessary precautions would be required to prevent foreign objects from entering the belt.

Installation considerations

This type of conveyor system is able to negotiate reasonably sharp turns with radii of between 100 and 300 pipe diameters, depending upon the belt tension. Severe changes in direction could only be achieved using multiple conveyors.

The conveyor cross section is smaller than for a conventional and although the support structure is lightweight it requires to be fixed firmly and rigidly. Thus, adjustments for ground movement may be difficult to achieve.

Compatibility with other systems

This system is not capable of carrying personnel or equipment.

4.7.2 Evaluation of conveyor belt system

Conveyor systems of whatever sort, suffer from a number of disadvantages when considered for ultra deep mining. First, they require a well-sized and constant feed. This requirement is relatively easy to meet in coal mines, because coal is very much softer than gold mine ores, and it is relatively easy to install feeder-breakers at the feed ends of all section conveyors. There are also no heat or space impediments to installing this equipment. However, it is much more difficult to construct and maintain a large number of crusher stations in gold mines.

Second, conveyors require straight runs, or slow and constant curvatures. These are difficult to achieve in ultra deep gold mines, except over short distances or in areas where the rock mass is particularly stable, such as in shaft pillar areas. Third, conveyors are susceptible to high rates of mechanical wear, particularly when conveying very abrasive rock such as quartzite. Fourth, conveyor systems are prone to spillage, which would be very detrimental in ultra deep mines due to the number of working areas. Finally, conveyor belts, although flame resistant remain a major fire hazard.

Considering the above issues, it is unlikely that conveyor systems will be the most desirable method of transporting rock over long distances in ultra deep gold mines. This is further negated as additional transportation systems will be required for material and most likely, personnel.

4.7.2.1 Size

In the case of personnel riding belts, minimum dimensions are legislated for the widths and clearances of both the belts and the boarding and alighting platforms. A minimum belt width of 750 mm is allowed, a minimum clearance of 900 mm from the belt to any other belt or structure. Considering clearances and infrastructure requirements, a minimum tunnel height of 2,4 m and a minimum tunnel width of 1,8 m is required for the installation of a personnel riding belt. Where pump columns and other services are required to make use of the same tunnel as the personnel riding belt, the tunnel dimensions will have to be increased accordingly to ensure that the minimum clearances are maintained.

At boarding and alighting stations, minimum tunnel heights have to be increased to 4 m to allow for the legislated increased clearances of 1600 mm at the platforms. Minimum widths too would need to be increased to the order of 2,5 m to allow for the minimum required boarding and alighting platform widths.

4.7.2.2 Flexibility

Straight tunnel sections are required for the installation of conveyors. No bends within a single conveyor are possible although long radius curves can be accommodated. Any changes in horizontal alignment require the installation of a new section of belt.

Vertical changes in alignment can be accommodated, but the maximum recommended gradient for a conveyor is 15°. Personnel riding conveyors are primarily rock conveyors adapted to allow personnel riding. Therefore, they can be used both for the conveyance of personnel and rock. Materials on the other hand cannot be carried on the conveyor, thus an additional transportation system, such as rail or monorail would be required to convey material.

4.7.2.3 Environmental compatibility

The use of electric motors for the conveyor means that heat and pollution emissions are minimal. Electric motors are reasonably quiet and unless there is damage to the belt or rollers, belt operation is quiet except for a slight hum. Legislation requires that the belt material be flame resistant, so that in the event of a fire minimal toxic smoke is produced. As the rock being transported is wet, dust should not be an area of concern. In cases where the ore is dry, water sprays can be used to suppress the dust.

4.7.2.4 Control and automation

An operator at the drive unit can control a single conveyor system, however, for two or more conveyors, remote control operation is desirable. A central control station over the entire system can co-ordinate the start up and shut down procedures for the system. Feeding chutes can also be remotely controlled to ensure an even discharge of rock from secondary conveyors onto the main conveyor, thus preventing overloading and spillage.

Sensors can be installed onto the rollers or in the belt itself in order to detect the presence of rock or persons on the conveyor. Similarly, ID tags can be utilised in cap lamps to monitor the presence of workers. Linked to a control system, rock loading or personnel riding can be prioritised to ensure the smooth operation of the system.

At boarding stations, automatic-signalling systems can be installed in order to ensure the correct spacing between personnel riding on the conveyor. Emergency stop pull wires allow the safe stopping of the conveyor in the case of an emergency such as a person falling off the conveyor, or a tear in the belt.

4.7.2.5 Cost implications

An approximate installation cost for a single personnel riding belt of ± 1 800 m in length would be in the order of R9 000 000. This cost includes the cost of all steelwork, head drives, tail and take-up pulleys and all safety devices associated with personnel riding needs. This cost breaks down into a cost of R5 000 per running metre for the belt, supports, emergency stop and communication systems, and lighting and warning signs. The additional costs over a standard rock carrying belt due to personnel riding safety requirements, boarding and alighting platforms, control cubicle, brakes and safety devices are R2 000 000.

4.8 Chairlifts

4.8.1 Audit of chairlift system

4.8.1.1 Overview

Traditionally, chairlifts have been used for the transportation of workers along inclines, either directly from surface, or between levels. The chairlift system is solely intended for the transport of personnel (Figure 4-53), as materials and rock cannot be handled. Where materials and rock require handling parallel inclines are constructed.

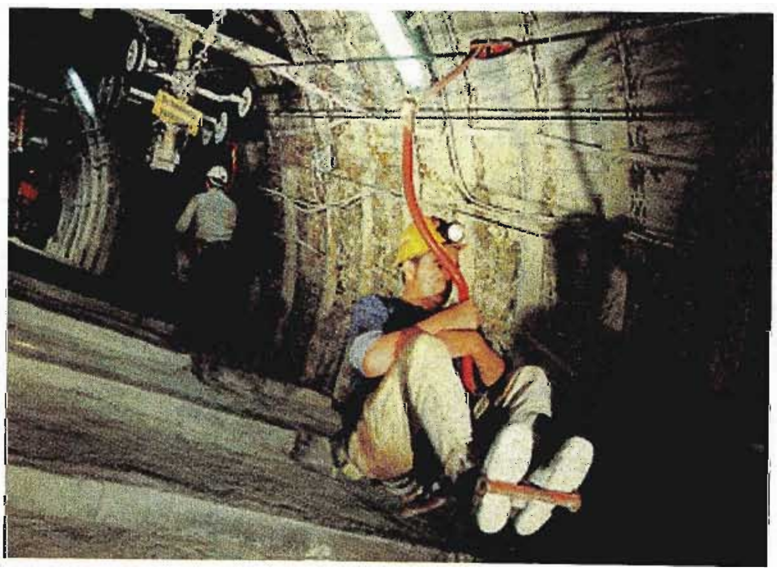


Figure 4-53: Chairlift in decline (Webers, 1999)

4.8.1.2 Capacity

The carrying capacity of the chairlift is a function of the rope speed, spacing of the chairs and the distance between supports. The distance between supports and the allowable tension on the rope determine the spacing allowed between chair, however a minimum distance of 5 m is required to allow for the safe boarding and alighting of personnel. Sagging between supports must be kept to a minimum, with 200 mm sag being acceptable.

Speeds of the chairlift are little more than walking speed, with 1,25 m/s being the norm. At this speed, a chairlift system with a chair spacing of 7,5 m would be able to transport 600 persons per hour. It is possible to increase the speed up to 2 m/s, which would increase the capacity to 960 persons per hour.

The use of detachable chairs could allow increased speeds of up to 5 m/s, but this is at the expense of flexibility and such systems prevent the use of intermediate boarding stations. Special boarding and alighting stations must be provided to allow for the chairs to match the rope speed.

4.8.1.3 Limits on length

Chairlifts with lengths of up to 3000 m have been installed and it is possible to install systems in excess of this length, particularly where detachable chairs are used. Drive units with power ratings of up to 132 kW producing pulling forces of 32 kN are available that can handle these lengths.

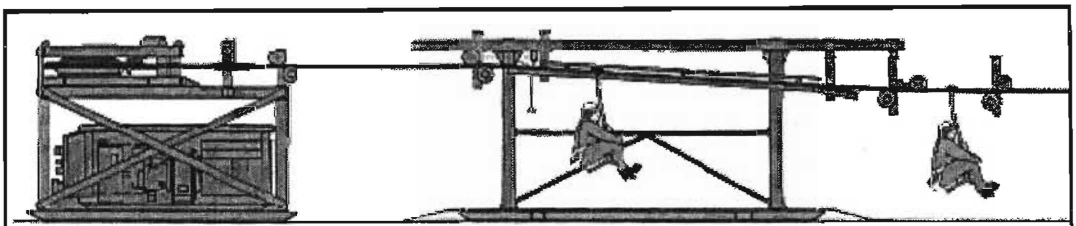
4.8.1.4 Environmental compatibility

Electric or electro-hydraulic drive motors are used to power the chairlift. These motors are efficient and produce no environmental contaminants. Chairlifts are sensitive to changes in horizontal and vertical alignment due to ground movement, but the impact of this can be reduced by the use of adjustable supports. Supports can be rigid supports either grouted into the footwall, or can be suspended from the hangingwall by means of slings or chains.

4.8.1.5 Powering systems

Electric or electro-hydraulic motors are the standard power sources for the drive units of the rope pulleys. Drive units are static motors usually placed directly in line with the chair lift system with power transmitted to the rope pulleys through a gearbox or hydraulic couplings. Tension is maintained on the rope pulleys by means of hydraulic pistons on the drive wheels, or via a tensioning tower at the return station.

Drive and return wheels correspond to the system gauge, which can range from 900 mm to 1835 mm. The drive units are placed at the head of the system in the case of an incline chairlift (Figure 4-54), or else at the shaft station on a level, where only one drive unit is used per system.



Drive station | Embarking/disembarking station

Figure 4-54: Drive source and embarking station (Nehrling, 2002)

4.8.1.6 Boarding and alighting

Chairlift systems with a maximum speed of up to 2 m/s do not require any special boarding or alighting arrangements. Persons simply walk behind the chair and climb on at the boarding stations, and similarly at the alighting stations they slip off the rear of the chair. Fixed chairs used on these systems have pre-set distances and gaps are maintained automatically. No attendant is necessary, but care must be taken when boarding not to produce any sideways motion in the chair, as this could cause the rope to slip off the intermediate support wheels and derail the system. Boarding stations are required to be level, and are typically 10 m to 20 m long.

Higher speed chairlifts make use of detachable chairs. At boarding stations, these chairs are placed on a special boarding rail, and released by means of a manual trigger. The chairs are then led down a set of small rollers to the end of the rail, by which time they have reached the operating speed of the system. The chairs are then held onto the rope by the friction between the rope and the support element. The manual trigger can only release the next chair when the previous chair has travelled a minimum distance of three times the travelling speed. At alighting stations, the reverse occurs, with the chairs being lifted off the rope onto a rail and slowed to a stop. The chairs are then simply lifted off the rail and placed in a depository.

As indicated in section 4.8.1.2, the fixed chair system, intermediate boarding and alighting stations can be provided at any point on the system. The removable chair system, however, only allows for a station at either end of the system.

4.8.1.7 Maintenance

The drive and return wheels, as well as the wheels of the intermediate supports are designed with replaceable polyurethane inserts to minimise wear on the rope. Consequently, these require regular monitoring and replacement in order to ensure that they do not wear to a point where the rope could be damaged. The rope itself requires replacement on an annual basis, although, with regular inspection, this could be extended.

Horizontal alignment of the intermediate supports is critical as even minor misalignments could lead to derailments of the rope. Any misalignments caused by ground movement must be attended to immediately. For this reason, the supports are designed to be adjustable to allow for quick realignments. Drive and tensioning units require maintenance on a regular basis. Loss of tension in the system could allow the sag between the supports to cause chairs to bottom out on the footwall.

4.8.1.8 Installation considerations

Chairlift systems can be designed to accommodate any mine layout. Horizontal curves of up to 120° and 4 m diameter can be accommodated by the fixed chair system, and gradients of up to 45° are possible. High-speed removable chair systems can operate on inclines of up to 18°. Curves are accommodated by means of special curve stations. However, the curves may require a larger radius than 4 m as the chairs are taken off the rope and guided around the curve on tubes.

4.8.1.9 Compatibility with other systems

Chairlifts are designed for the transport of personnel, but any material that can be carried in a handbag (elephant bag) is permissible. In emergencies, it is possible to attach a stretcher unit to the rope. The chairlift can be designed to have a narrow operational width of 900 mm, which allows for the installation of a parallel rail or conveyor system for the transport of rock.

<u>Alternative system</u>

A variation on the chairlift system is the "Men Assisted Walking System". This type of system is commonly installed in steep or long inclines where labour is assisted to negotiate the incline. This system consists of a motor at one end driving an endless wire rope through a pulley, or set of pulleys. Each person using the system is issued with a handgrip, which is placed over the rope in such a way, that it "grips" the rope, enabling the person to be pulled up the incline. For safety considerations, low speeds must be adopted to prevent persons from being pulled off their feet and injured, particularly in a steep incline where the consequences of a person falling could be fatal. Safety trip wires are therefore included in any system.

4.8.2 Evaluation of chairlift system

4.8.2.1 Size

Figure 4-55 depicts the recommended clearances for the chairlift. A minimum haulage size of 2410 mm in height by 2,100 mm in width is required for the installation of a chairlift system. While the chairs themselves hang only 1580 mm below the rope, other factors such as maximum allowable sag of 200 mm, ground clearance of 500 mm and approximately 300 mm for the rope supports need to be considered. The width of 2100 mm allows for the minimum gauge of 900 mm, plus 600 mm clearance either side of the ropes. At boarding and alighting points, some additional width will be required, but the height requirement can remain the same. The drive and return units require a minimum height of 3500 mm and width of 5300 mm to allow for the installation of the framework, drive motors and walkway.

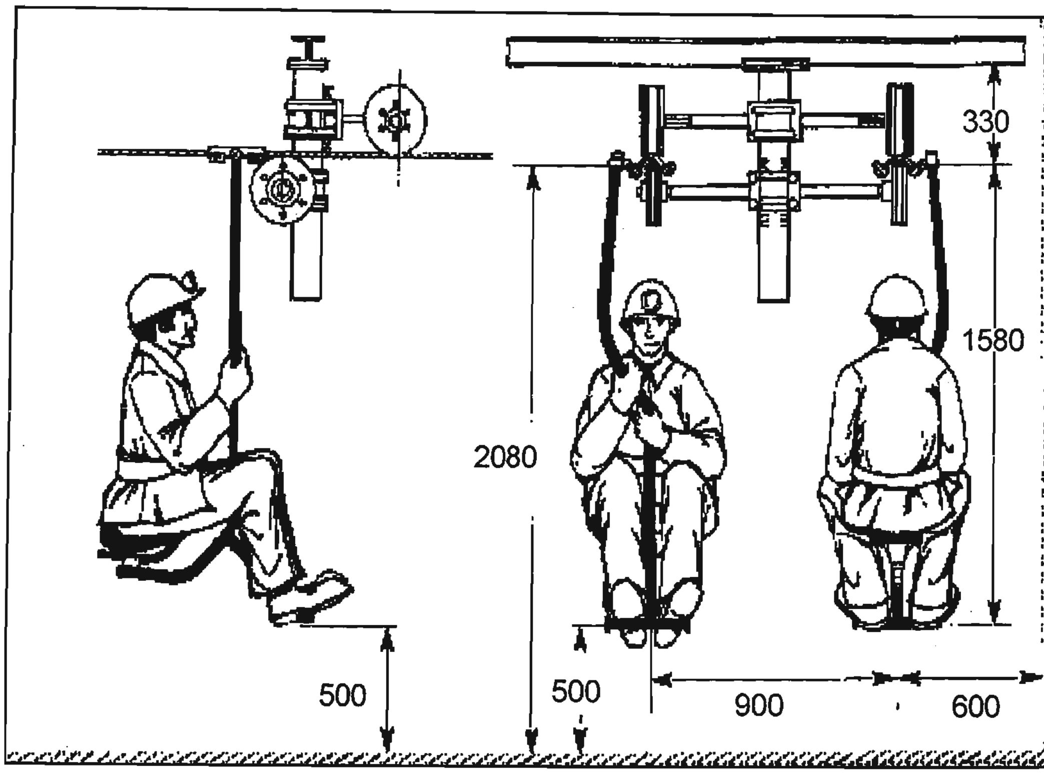


Figure 4-55: Chairlift clearances (Nehrling, 2002)

4.8.2.2 Flexibility

Chairlifts are useful for the conveyance of workers over long distances at speeds of up to 5 m/s and on gradients of up to 45°. Negotiation of horizontal and vertical curves can be accommodated, eliminating the need for straight tunnels. The chairlift is still limited in that it is a simple out and back system. Branches from main haulages require additional or alternate transport systems. A further limitation on the chairlift is that only personnel and hand tools can be conveyed. Rock and materials cannot be conveyed on a chairlift, and so it must be seen as a personnel transport medium only.

4.8.2.3 Environmental compatibility

Electric and hydraulic drives used for chairlifts do not produce significant amounts of heat or noise pollution, and there are no exhaust emissions.

4.8.2.4 Control and automation

As the chairlift system operates in isolation from other transportation mediums, there is no need for a central controller as each chairlift can be started or stopped by an operator without affecting any other systems. Emergency stop cords are provided on the full length of the system, allowing persons riding the chairlift to stop the system if necessary. Some form of communication needs to be provided, but this can be in the form of telephones at suitable intervals to allow persons to communicate with the operator should the system be stopped for any reason.

Sensors can be placed in the wheels of the rope supports to detect possible damage to the ropes, or if the rope should derail, and these can be linked to an emergency stop system and control board.

4.8.2.5 Cost implication

Installation costs are in the region of R1100 per metre for the rope, chairs and intermediate supports, and an additional R240 000 for the power pack, return and tension units. Rope replacement costs are in the region of R55 per metre for a 22 mm diameter rope.

4.9 Trackless Transportation

4.9.1 Audit of trackless system

4.9.1.1 Overview

Load haul dump trucks (LHD) and small articulated dump trucks (ADT) have been used in underground mining for several years. Drill rigs, utility vehicles and light delivery trucks are also in use for the purposes of mining. Recently, light-duty commercial vehicles (LDV) have been commissioned for use in some mines for use as utility and light delivery vehicles (Figure 4-56). The personnel carrier's dimensions are 5,5 m (l) x 1,8 m (w) x 2,1 m (h).

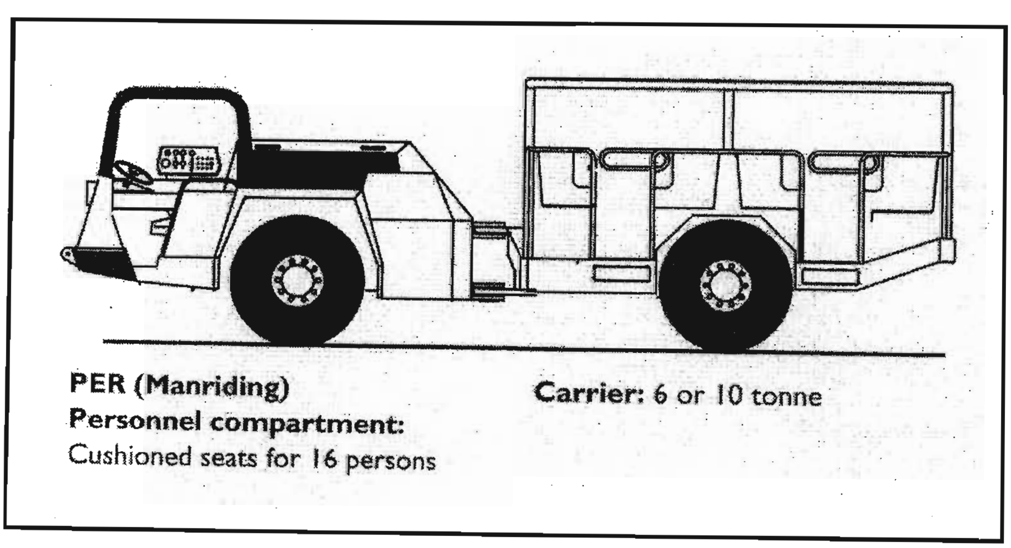


Figure 4-56: Personnel carrier

Mines that are currently utilising LDVs for underground transport include Koornfontein, New Denmark and Tavistock collieries. AngloPlats and Impala Platinum Mines also make use of LDVs in their inclines, but this is for the use of officials and for the transport of small emergency materials, and not for the transport of personnel or materials on a regular basis.

4.9.1.2 Capacity

Theoretically, any size diesel or electrically powered vehicle can be used for underground transportation, as long as the haulages are of sufficient size to accommodate them. In practice, however, only those mines with direct surface access to the levels via declines make use of the larger vehicles. In the ultra deep mine application, the ability to transport any trackless vehicle down the shaft imposes size limitations on the vehicles, and so larger vehicles are not practical.

Smaller vehicles with a maximum length of 3 m that can fit inside the cages could be used, and larger vehicles slung under the cages. The most likely type of vehicle is a specially adapted bakkie with either load carrying or personnel carrying abilities. Bakkies are limited to a maximum of 12 passengers per vehicle. Thus to transport an entire shift would require a fleet of vehicles running several trips each to achieve this feat. The bakkies are able to attain speeds of up to 60 km/hr in theory, but in practice, this is likely to be 30 km/hr or lower.

Figure 4-57: Kiruna Electric Truck (Dreyer, 2001)

4.9.1.3 Limits on length

There are no practical limits on the operational length, as diesel powered vehicles can operate for as long as their fuel capacities allow. Battery powered vehicles are also able to operate over long distances with the only limiting factor being the recharge times for the batteries. Electric vehicles can be used in conjunction with overhead trolley wires, which would allow unlimited range within trolley line equipped haulages. Hybrid battery and electric vehicles give a limited operational range beyond the end of the trolley lines, allowing some flexibility in terms of being able to service cross cuts and development ends.

4.9.1.4 Environmental compatibility

Diesel powered trackless vehicles have the same limitations as those attributed to diesel locomotives (Section 3.2.1.3). Electric and battery powered vehicles do not produce exhaust gases and their heat output is significantly lower than that of the diesel engine. Stray currents and arcing are however one of the features of an overhead trolley line, which need to be considered similar to those discussed in section 3.2.1.2.

4.9.1.5 Powering systems

Diesel engines are the most common type of power units utilised for trackless vehicles. These types of powering units are required to be equipped with flameproofing measures, noise and emission retardation systems.

Electric power units make use of an overhead power supply in the form of overhead trolley lines. Electric power units can also be equipped with nickel cadmium batteries, which are continually charged while the vehicles are connected to the overhead trolley lines. This enables use of the vehicles for limited periods in areas where no power supply is available.

4.9.1.6 Boarding and alighting

For personnel riding purposes, no special boarding and alighting arrangements are required as personnel riding vehicles are able to stop at any point in the haulages to pick up or let off persons.

4.9.1.7 Maintenance

A systematic maintenance procedure is required in order that the vehicles are not allowed to deteriorate to a point where they may become unsafe to operate, or unreliable. Such a maintenance philosophy includes the systematic monitoring of the engine oil for contaminants in the case of diesel vehicles and the scheduling of regular maintenance for the vehicles. These maintenance intervals should be based on hours of use, or be of a set period. Maintenance intervals based on hours in use is the preferred method, as vehicles that have a higher usage are maintained more frequently, and unnecessary maintenance of light duty vehicles is avoided.

Tyres are the highest cost item after labour in the running of a trackless fleet. The abrasive nature of the footwall necessitate that tyres be monitored and replaced on a regular basis. The rocks associated with hard rock mining have sharp edges; tyres, therefore need to be regularly checked for cuts in the tread and sidewalls, and replaced when necessary. Care must be taken in the selection of tyres, as inappropriate tyres will need replacement more frequently than ones suited for their intended purpose.

4.9.1.8 Installation considerations

Trackless mining vehicles for the transport and handling of rock typically require haulages with a minimum size of 3 m by 3 m cross section. Materials handling vehicles and personnel carrying cars on the other hand can be of a smaller size than this, and can be made to require very little headroom. Tunnel sizes can therefore be smaller than those required for LHDs. A minimum size of 2,5 m by 2,5 m cross section is sufficient for the operation of such vehicles, although parallel haulages may be required for traffic moving in opposing directions. Alternatively, passing bays may be provided to allow vehicles to move in both directions in a single haulage.

Footwall and sidewall heave should have little impact on the operation of these vehicles, as they are able to handle undulating footwalls and can negotiate obstacles. Smaller vehicles, such as light utility vehicles, can be particularly effective in negotiating obstacles due to their small turning circles.

Where diesel powered vehicles are utilised, ventilation in the haulages must be such that exhaust gases are quickly removed from the environment. Roadways need to be constructed and maintained to prevent unnecessary wear or damage to the tyres. Therefore, good roadway surfaces are key to the successful operation of trackless vehicles.

4.9.1.9 Compatibility with other systems

Trackless vehicles are able to run in conjunction with other systems. Where track construction is such that it permits roadways to be constructed on the rail tracks, vehicles can operate over the railway tracks. However, where trackless vehicles operate in conjunction with conveyor systems, bridges are required for the conveyors system to operate over roadways and in some instances separate haulages are constructed for trackless vehicles and conveyors. Where this is not possible, roadways can be constructed adjacent to the conveyor systems, but this requires haulages large enough to accommodate both systems.

4.9.2 Evaluation of trackless system

4.9.2.1 Size

Vehicles of virtually any size can be manufactured for use in an underground environment, but these are generally in the order of 1,8 m in width, and up to 2 m in height, dependent on their duty. The ground clearance of the vehicle must be 250 mm to 300 mm. Minimum haulage sizes required will be in the order of 3 m high by 3 m wide, allowing for one-way vehicle travel. Two-way travel will require minimum tunnel widths of 4 m to 4,5 m or passing bays at regular intervals.

4.9.2.2 Flexibility

Trackless vehicles can travel virtually anywhere where tunnel sizes permit. Horizontal and vertical changes in alignment can be negotiated, as can branches from the main haulage. Vehicles are available that can convey personnel, materials or rock, and, with the use of interchangeable modules, it is possible to use the same vehicle for any number of purposes. The trackless vehicle can be said to be the most flexible of all transportation systems.

4.9.2.3 Environmental compatibility

Diesel or petrol driven vehicles are required to be specially adapted for underground use, however, similar to diesel locomotives and monorails, environmental constraints make such vehicles unsuitable in ultra deep mines. Battery operated vehicles are available, but current battery technology will require that large rechargeable batteries be used. These batteries are both heavy and large, thus severely limiting the load carrying capacity of the vehicles. Electric vehicles are relatively inflexible as they operate normally where the overhead trolley line has been installed, although using batteries offers some flexibility. However, there is still a reluctance to utilise electric trucks.

4.9.2.4 Control and automation

Remote control of trackless vehicles is possible, however to date no automation of trackless vehicles has been achieved. A driver will be required for the operation of these vehicles. Communication is still possible as radios can be fitted to the vehicles, allowing communication via a leaky feeder type system. Transponders can also be fitted to these vehicles so that their whereabouts can be monitored.

4.9.2.5 Cost implications

No specific infrastructure is required for the use of trackless vehicles except for the minimum tunnel dimensions to allow these to operate without hindrance. Costs relating to these systems are therefore based solely on the capital costs of providing the vehicles themselves.

On surface, the cost of such machinery ranges from less than R105 000 for light duty vehicles (LDV), to several hundred thousand rands for LDVs. Should the smaller bakkie type vehicles be adopted for underground usage, the necessary flameproofing and exhaust scrubbing equipment will add to the basic vehicle cost. A basic non-flameproof vehicle costs in the region of R150 000, but the addition of special load boxes such as for personnel transport adds additional cost of up to R35 000 to the basic cost. The cost of exhaust gas scrubbers and other flame proofing equipment is in the region of R15 000 to R22 000.

Currently two local manufacturers supply LDVs to the mining industry. These are Toyota, which have adapted their Hilux range to operate in an underground environment, and Samcor, who have adapted the Ford Courier. Both these vehicles are similar to their surface counterparts in appearance, but with derated motors to allow for lower engine operating speeds and temperatures.

4.10 Summary and conclusions

The findings of the literature search and audit and evaluation for horizontal transportation are summarised in Table 4-12. Certain factors of individual transport systems are shown as well as the uses for which the systems are suited.

Table 4-12: Horizontal transportation options

Transport Type	Speed (km/hr)	Personnel Capacity	Material capacity	Rock capacity	Gradient (degrees)	Capital Costs	Operating Costs
High-speed locomotives	30 – 40	High	High	High	2.5 (max)	High	High
Low – Medium speed locomotives	5 – 12	Medium	Medium	Medium	2.5 (max)	Low	Low
Monorail	3 – 14	Low	Medium	Low	12 -36	High	Medium
Trackless	3 – 20	Low	Medium	Medium	11	High	High
Conveyor	9	High	N/A	High	17 (max)	High	Low
Chairlifts	4 - 9	High	N/A	N/A	18 (max)	Med – High	Low
#Walking	4	Low	N/A	N/A	N/A	NA	NA

#(walking included for comparision purposes).

Conclusions from the literature study are as follows:

- The installation, maintenance and management of the underground rail systems in South African gold mines need to be improved. Communication and tagging systems are useful tools to identify corrective actions required and simulation packages should be used to assess changes to the system or the evaluation of new systems.
- Remote/automated systems are being used more frequently in new high capacity rock transportation systems. The technology used in these systems will lead to significant changes to transportation systems.
- Electrically powered trolley locomotives are most suitable to high-speed track installations. For such systems, track installations/maintenance, rolling stock conditions and general safety standards are extremely important.
- Low to medium speed locomotives are inexpensive and have low track maintenance requirements. For these units, the use of electrical powered is preferred to that of diesel.

- Monorail systems are ideal for concentrated material transportation, however they are not suited for high capacity personnel transportation or rock removal. Monorail systems are versatile, negotiating steep gradients and fit into small areas, but are slow moving and expensive to operate.
- Chairlift systems are expensive to install, are limited to personnel, and are only suitable along main transportation routes. They have low operating and maintenance costs.
- Trackless vehicles are unsuited as a means of transportation in ultra deep level gold mines, as they are generally poorly utilised, require large excavations, and are expensive to purchase and maintain.
- Work conducted prior to 1999 focused primarily on the rock handling aspects of the in-stope process with the aim to improve face advance and productivity. Recent research initiatives have expanded the in-stope process to include personnel and material transportation.
- The movement of personnel, material and rock needs to be integrated so that efficient in-stope transportation can be efficiently conducted.

5. In-stope transportation

Currently, there is no significant mechanised transport system capable of moving workers and material easily and efficiently from the stope cross cut to the stope face. Workers proceed via designated travelling ways, which then allow access to the stope through dip gullies, or from travelling ways to the face via the strike gully. Material is commonly delivered into the stope by means of a monowinch. Rock is removed with winches and scrapers, a method that has been used for the past half-century.

In the stoping environment, all modes of transport tend to use the same area. For example, workers and material tend to share the same stope entrance, with workers travelling alongside the monorope. In the strike gully and occasionally in the dip gully, workers, material and rock use the same area, which means that only one system is able to move or operate at any one time. Although this has been the position for many years, these usages need to be altered in order to provide for safe, productive and cost-efficient mining operations.

5.1 Personnel

5.1.1 Audit of in-stope personnel transportation

The movement of workers in the stope environment is one of the most basic and underdeveloped potential means of transportation in ultra deep level gold mines. No mechanised means of transport is currently used in narrow reef stopes.

The workforce congregates at the waiting place and, once given the go-ahead by the supervisor, enters the workplace, usually via a short travelling way (T/w) which is accessed from the cross cut (Figure 5-1 and Figure 5-2). Access into the stope is either by climbing up from the bottom cross cut (Figure 5-3) or by climbing down from the top cross cut. Typically, fewer than 100 workers will be required to access a stope during a drilling shift. Since the workers are split between the top and bottom cross cuts, congestion is unlikely to be a problem. Entrances into the stope are via travelling ways inclined at 34° and typical lengths range from 5 m to 22 m.

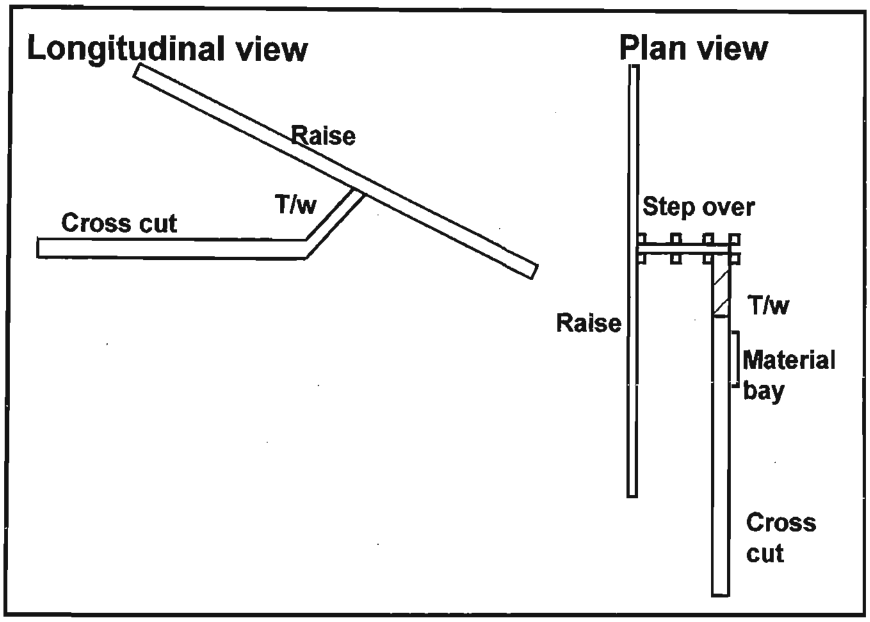


Figure 5-1: Stope entrance via the inclined travelling way

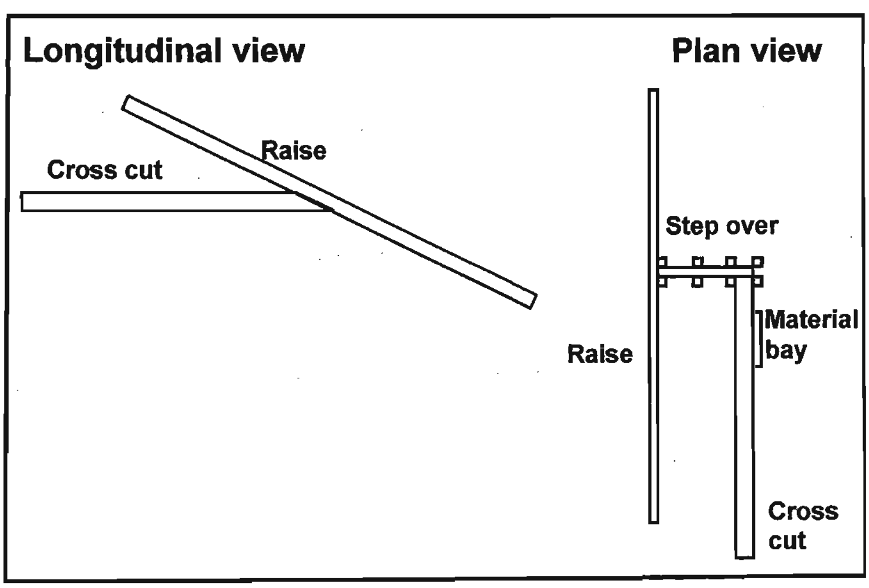


Figure 5-2: Stope entrance via the reef intersection

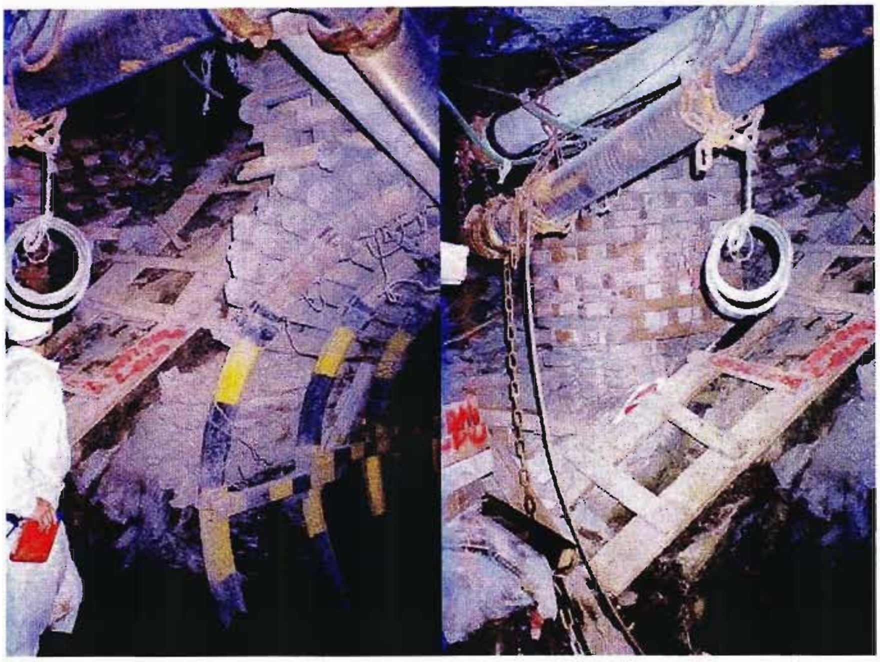


Figure 5-3: Stope entrance

The layout of a travelling way for deep level gold mining is depicted in Figure 5-4, which indicates the travelling routes in a typical longwall (LSP) layout.

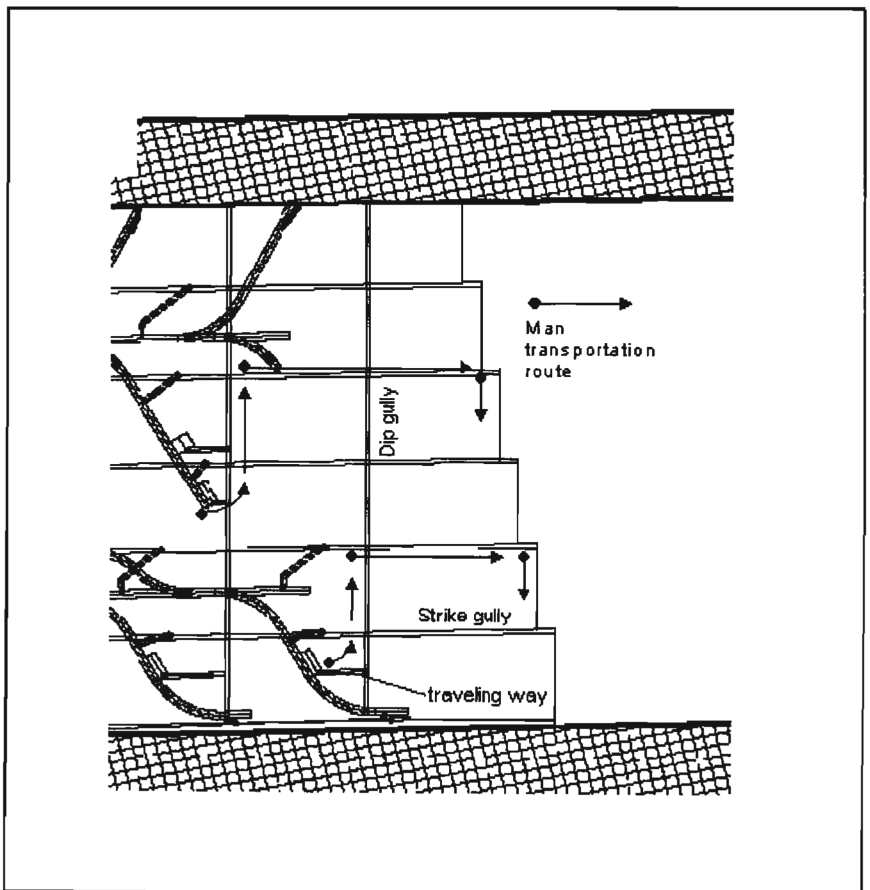


Figure 5-4: Travelling routes in a typical longwall layout

Vertical travelling in the stope is done through dip travelling ways, which are generally the original raise, or through travelling ways established alongside the raise during the ledging of the raise. Typical dimensions of travelling ways for deep level stopes are given in Table 5-1.

Table 5-1: Travelling way dimensions

Description	Units	LSP	SGM	SDD	CSDP
Stope entrance travelling way dimensions (h x w)	m	2,8 x 2,0	2,4 x 2,4	2,4 x 2,4	2,0 x 2,5
Stope entrance travelling way length	m	Varies	15	15	22,5
Stope entrance travelling way inclination	°	34	34	34	34
Dip travelling way dimensions (h x w)	m	2,0 x 1,8	3,0 x 1,8	2,4 x 2,4	2,0 x 2,5
Dip travelling way lengths	m	60	120	240	85
Strike travelling way dimensions (h x w)	m	1,6 x 2,0	1,8 x 2,8	N/A	1,8 x 2,5
Strike travelling way lengths	m	100	80	37,5	70

Note: LSP = longwall; SGM = sequential grid mining; SDD = sequential down dip; CSDP = closely spaced dip pillars.

Movement in the stope can be strenuous, as travelling ways are often obstructed with equipment, material and ore (Figure 5-5). The travelling way may also experience ground movements or deviations of half a metre to several metres due to faults, rolls or changes of dip, which must be negotiated by the worker. Although ladders, handrails or climbing chains may be required, in practice they are seldom provided.

Another general problem with in-stope travelling is the fact that the travelling way commonly shares the same space with the monowinch way. This restricts the travelling space and adds the potential hazard of workers being injured by the rope/pulley or by the material that is being transported. Although some layouts make provision for dedicated travelling ways, often they are not implemented or, when they are installed, they are poorly maintained (Figure 5-6 and Figure 5-7).

Figure 5-5: Travelling way congested by scoop and scraper ropes

Figure 5-6: Dip travelling way – planned but not implemented

Figure 5-7: A travelling way around a stope ore pass
(Note the unsafe openings around the grizzly)

Stope crews use the dip travelling ways to gain access to their work panels, the distances covered being typically as short as 60 m and as long as 240 m. Access to the stope face is usually through the strike gully, which requires the workers to share the gully (Figure 5-8) with the scraper and occasionally the monowinch. Typical strike lengths are from 15 m to 100 m. In many cases, workers travel from panel to panel via the face (Figure 5-9 and Figure 5-10) and the size of the access way is often limited as mining widths are restricted to the reef channel height or to a minimal stoping height of approximately 0,8 m.

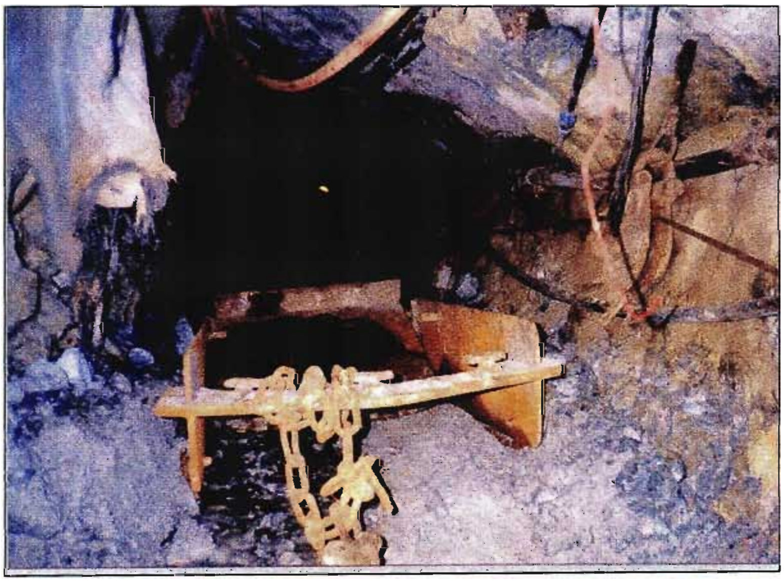


Figure 5-8: Strike gully travelling way

Figure 5-8: Strike gully travelling way

Figure 5-9: Face travelling way – Example 1

Figure 5-10: Face travelling way – Example 2

5.1.2 Evaluation of in-stope personnel transportation

Travelling by foot in the reef horizon should not be regarded as inappropriate. It must be emphasised that in any transport system, whether it be for a city commuter or a stope worker, at some stage of the journey it will become necessary for the person to walk. However, in the stoping environment there is a need to reduce the amount of physical effort and improve the conditions of the journey. Current in-stope travelling has many shortcomings and requires significant changes if safety and productivity are to improve in ultra deep level mines. By designing integrated transport systems and by implementing planned travelling ways, many of the shortcomings currently experienced can be overcome.

5.1.2.1 Size

Movement in the stope horizon takes place via travelling ways, which often serve several purposes. For example, dip and strike gullies, which are main rock arteries, also serve as travelling ways. For this reason, the travelling ways are generally smaller than what is really required. Although a minimum height of 1,8 m is necessary to allow workers to walk upright, the available height is often less than this as travelling ways are not developed to full height, and gullies are frequently blocked by equipment or choked with broken rock.

5.1.2.2 Flexibility

In-stope travelling ways provide access to the working panels via dip and strike travelling ways, which can be entered from either the top or the bottom of the stope. Travelling ways are usually based on the mining layout, but they also can be developed as required and thus are flexible with regard to positioning. However, travelling ways are influenced by geological characteristics, such as faulting, reef rolls, closure, etc., and therefore should be properly planned.

5.1.2.3 Environmental compatibility

Providing good in-stope travelling ways while maintaining airflow control can be problematic. In order to travel on dip, gaps must be left open for people to travel in, however, these same gaps also allow ventilation to be redirected into the back areas and away from the working faces. Thus, ventilation brattices are required to minimise the loss of ventilation in the back areas (Figure 5-11).

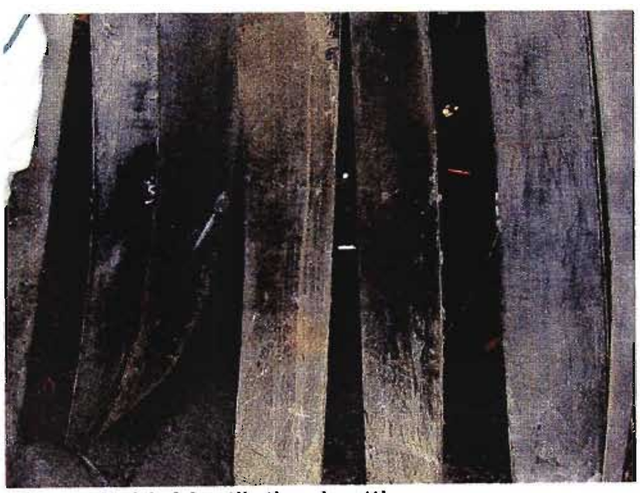


Figure 5-11: Ventilation brattice

5.1.2.4 Costs

Since the majority of travelling ways are developed for other uses, i.e. as strike and dip gullies, their cost is minimal. The cost of a travelling way from the cross cut to the stope (raise) is approximately R4 000 per metre, whereas the cost of an in-stope travelling way (i.e. a dip or strike gully), where the footwall is usually "ripped" or popped out, is in the region of only R350 per cubic metre blasted.

5.1.2.5 Conclusion

Good in-stope travelling is possible in ultra deep level mining layouts. The major problem is the inability to plan and implement these travelling ways properly. This is largely a management issue, which can only be addressed once production personnel understand the importance of getting workers to the face with a minimal amount of physical effort. There should be no need for workers to crawl, climb or manoeuvre around equipment to get to their working places.

5.2 Material handling

In-stope material handling is an important element in mining. Handling involves the transport of all in-stope materials and equipment from the cross cut to (as close as practicable) the stope face, in a safe and efficient manner. Material should be rehandled as little as possible. The handling process must also be adaptable to a variety of mining cycles and layouts, shaft systems and horizontal infrastructure layouts.

In some instances, such as where the mining method involves long backlengths, it would be advantageous if the material handling system could handle all items of equipment used in the reef horizon, including winches, scoops, drill rigs and other heavy items of equipment.

Wilson *et al.* (2000) established that in-stope material handling systems are generally deficient: "The level of technology for in-stope handling is low and there is a clear need to improve in-stope handling systems and methods." They further stated, "bulky and heavy materials need to be moved in hot and confined areas. Further, low levels of productivity and the high safety risk were often the result of poor working conditions." These sentiments are confirmed by AngloGold's 21st

century mining criteria of improved safety, reduced manual exertion and greater productivity.

Timber continues to be the support type most widely used in South Africa and constitutes the bulk of material to be transported, despite the introduction of mechanical steel props, elongates and backfill. Both the handling and transport of multiple timber units are time-consuming and wasteful, with numerous rehandling points along the supply route. For this reason, alternative transport systems are constantly being sought. In addition, heavier material (>35 kg) is required to be transported by chain blocks or other methods, which severely delays delivery.

With the exception of purpose-designed trackless mining operations in South Africa, material is brought into deep mines on trackbound systems and transferred from the cross cut to the reef horizon by using a monowinch system. The following sections examine and evaluate monowinch and monorail technologies, and discuss the practicalities of implementing these two in-stope material-handling systems.

5.2.1 Monowinch systems

5.2.1.1 Audit of monowinch systems

As described by Rupprecht *et al.* (2002), until the 1970s, when monowinches were introduced, material handling was extremely labour-intensive. It was often accomplished either by forming human chains, and/or by dragging material along in either a scraper scoop or a purpose-designed steel basket attached to the scraper rope (these practices still take place). In areas of steep gradients, wetted timber slides were constructed on dip. Transport within the reef horizon was assisted by sliding the timber down to the appropriate strike gully. At that time, the materials transported into stopes were reasonably standardised and of a size and mass commensurate with the manual handling capabilities of the crews.

As mines became deeper, however, the introduction of alternative support elements, such as hydraulic props, composite packs and backfill bags, made manual transportation more onerous. During the late 1970s and early 1980s, the mining industry conducted much research aimed at increasing efficiencies, and monowinches were introduced as an alternative to manual transportation. By the mid 1980s, manual transportation had given way to monowinch systems.

Current stoping operations are served almost exclusively by monowinch installations. Operating in cross cuts as close as possible to the timber supply, monoropes (wire ropes) are reeved through a series of pulley blocks to follow the route chosen for the material to travel. The units are fitted with steel wire ropes with diameters ranging from 10 mm to 16 mm. The power of the drives varies from 3 kW to 22 kW or, in the case of modified scraper winches, up to 37 kW. The purpose of the modified winches has been to service very long backlengths and to serve as main material transporters in some decline operations.

The monowinch is a mature technology that is well known and understood by the mining fraternity. The monowinch system is labour-intensive and, although it is widely used, it carries an inherent risk for workers to injure hands. Experience has shown that the monowinch is capable of supporting face advances in the order of 15 m to 20 m per month over lengths of 240 m. The capacity of a monowinch is about 35 kg, with the actual physical capacity being determined by the size of drive, rope diameter, length of rope, gradient, distance between supports and number of direction changes. By increasing the size of the winch and rope, it has been possible to serve virtually any mining layout currently used.

The performance of the system is, however, very dependent on a number of factors:

- Loading efficiency.
- Routing.
- Interaction with other activities and services, such as rock transportation and the movement of personnel.
- Storage at loading and unloading points.
- Unloading efficiency.

Capacity and limits on length

Determining the limiting length and load of a monowinch system depends on the following:

- The power rating of the monowinch.
- The mass of the materials to be transported.
- The spacing between successive items.
- The dip (magnitude and direction).

- The diameter of the rope used.
- The length of the rope.
- The tension on the rope.
- The number of direction changes (i.e. the number of corner pulleys).

As these factors are mine-specific and some features, e.g. the mass of the materials and the spacing between successive items, vary, a generic value of the load-carrying capacity is difficult to determine. Similarly, the limit on the length of monorope also depends on these factors. In most cases, the limits of length are based on personal experience.

Some general limits that are applied in practice are:

- It appears that 35 kg is the maximum mass of the material that would be transported using a monowinch system.
- One monowinch system should service not more than two to three panels.

The lengths of the monoropes (from the monowinch to the return pulley) that are generally applied in practice are:

- 3 kW - less than 60 m.
- 5,5 kW - between 60 m and 150 m.
- 7,5 kW - less than 175 m.
- 11 kW - less than 200 m.
- 15 kW - less than 300 m.
- 22/37 kW - greater than 300 m.

Loading and unloading

In general, monowinch systems do not require much operating space and are ideal for high-stress environments, as they should not be adversely affected by stope closure or undulating hanging-wall conditions. The largest operating area required is at the loading zone. This area must contain a monowinch, which could require dimensions of up to 2 m (l) × 0,92 m (h) × 0,72 m (w); a loading platform (if necessary, with dimensions of approximately 2 m (l) × 1,2 m (h) × 0,5 m (w)); a material bay, and sufficient space for operators to work clear of any rolling stock. A typical material bay should be approximately 30 m (l) × 3 m (h) × 2 m (w) with the monowinch operating out of the material bay.

Loading materials

Two methods are used for loading materials. The first method uses a loading magazine onto which the item is tied using twine (Figure 5-12). The twine is then slid off the pipe and over the rope. When this method is used, the monowinch motor is not stopped for loading. The second method involves starting and stopping the motor at intervals. In this case, the items are tied directly onto the monorope. In both cases, the distance between the items attached to the rope should be 3 m to 5 m. This distance is dependent on the load: the heavier the load, the wider the required spacing.

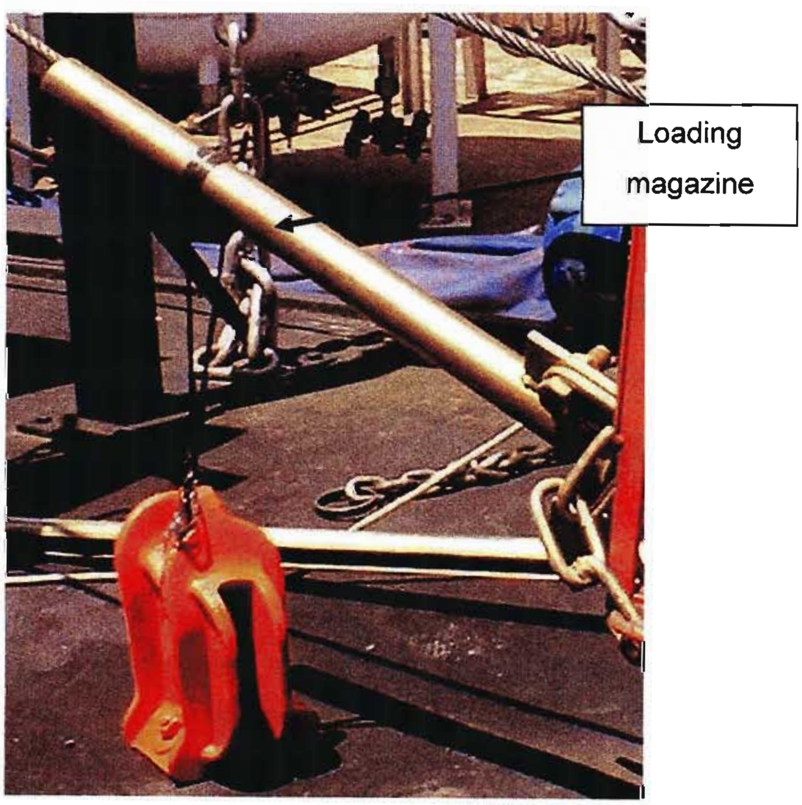


Figure 5-12: Loading magazine (Clarke, 2002)

Unloading materials

Two methods of unloading materials from the monorope are applied in deep level mining. The most common method is manually cutting the twine supporting the material from the monorope along any point of the rope (Figure 5-13). This method is usually applied where a monowinch serves several panels. Where large quantities of material are being delivered to one location automatic cutters are preferred and the twine can be cut by a stationary piece of cutting wire.

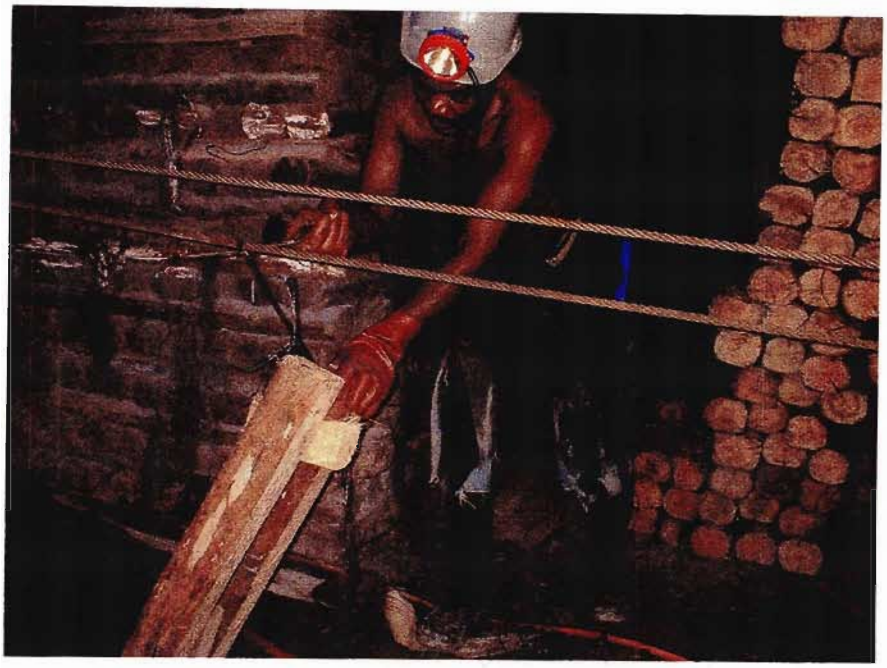


Figure 5-13: Unloading using a manual cutter

Installation and maintenance

In practice, the monowinch is positioned in the cross cut, either in a material bay or along the sidewall of the haulage. From this position, the monorope is fed into the stope along travelling or material ways, where it feeds either the stope face(s) or a position in the centre raise. Monowinch systems are extremely flexible, capable of handling sharp corners and can move in different directions, i.e. up the raise, and through strike gullies to the stope faces.

Monowinch systems require very little maintenance. Daily inspections by the operator are required and include checking the system for broken or damaged components. Component life depends on the types of load and on the quality of the installation and maintenance. Maintenance requirements can be minimised by adhering to installation standards and by using the correct components, for example, by selecting the correct corner pulleys, properly tensioning the monorope, and spacing the pulleys at appropriate distances. Pulleys are designed to deflect a specific range of angles with some pulleys being restricted to a deflection angle of less than 45°. If the monorope is over-tensioned, the contact surface between the pulley and the rope is increased, again increasing the wear on these components. Similarly, if the spacing between successive pulleys is too large, rope tension increases, thereby reducing the life of the rope.

From cross cut to stope face

Some mines operating longwalls and grid layouts route the monoropes from the cross cut up the travelling way and along the strike gully running immediately behind the first line of packs above the gully (Figure 5-14). Thereafter, the rope is routed behind the temporary support parallel to the face and is zigzagged through the panels. Experience has shown that such an arrangement is capable of meeting the support requirements of face advances in the region of 6 m to 12 m per month. This method allows material to be delivered close to the face, typically within 4 m to 12 m, eliminating the need to rehandle material at the raise/strike gully intersection. However, this system requires good control as failure to advance the rope and pulleys forward on a regular basis will cause the monorope to fall behind the face, increasing the amount of manhandling needed to get the material to the face.

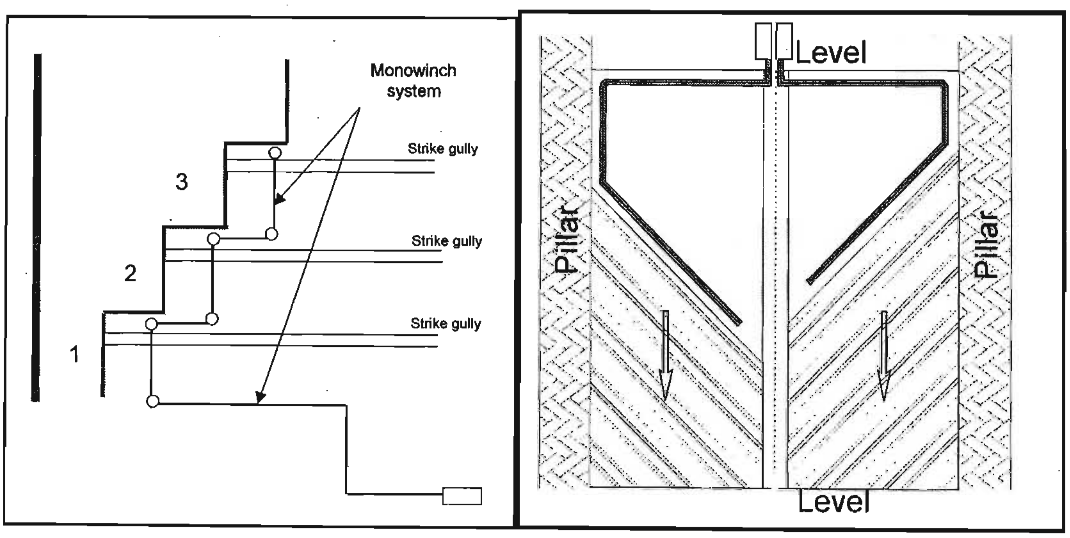
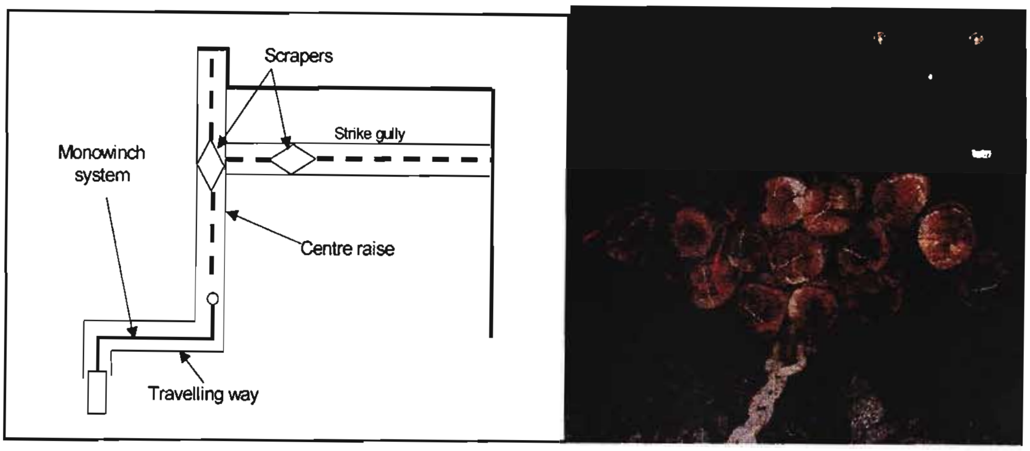


Figure 5-14: Monowinch systems - Cross cut to stope face

From cross cut to raise

An alternative to zigzagging the monorope behind the panels is to operate the monorope in the centre raise (Figure 5-15(a)). Material is cut off at the strike gully intersection and transported to the face by the scraper scoop Figure 5-15(b). In other cases, the material is manhandled by human "trains" to the face, whereby individual units are thrown forward a number of times until the units reach the face. In either case, this system is labour-intensive and a considerable amount of rehandling is required. Although this method is able to support face advance rates in the order of 8 m to 15 m per month, the practice of handling material in the strike gully means that the scraper cannot be used to pull rock while the transportation of material takes place.



(a) (b)

Figure 5-15: Monowinch systems -Cross cut to raise

From cross cut to raise and from raise to stope panel

In the third scenario, the monowinch operates from the cross cut and feeds into the centre raise. Material is cut from the primary monorope, rehandled and connected to an in-stope monowinch that feeds directly to the face (Figure 5-16). The monowinch can be extended per blast, thus allowing material to be delivered to the top of the face or along the panel length. Although this system requires an additional monowinch for every working panel, it does reduce the overall material handling requirements. The monowinch should operate either above or below the strike gully, thus improving rock handling times during the day shift, i.e. the scraper winch is free to operate during the day shift as the material is routed away from rock moving operations.

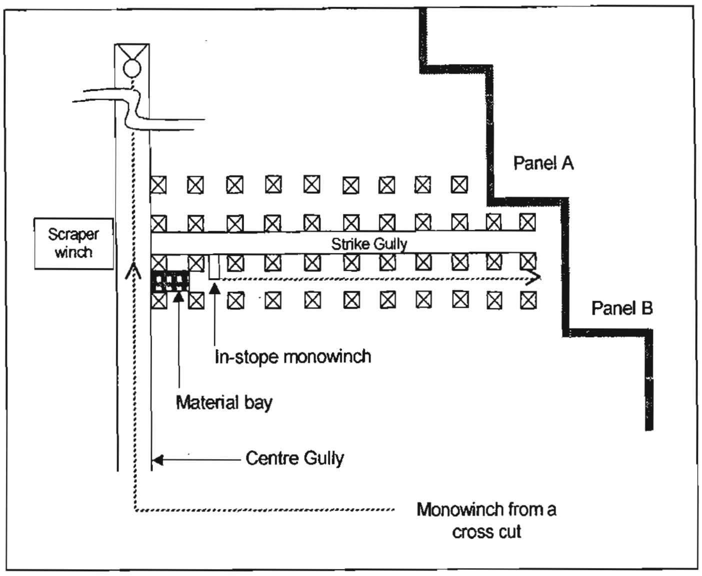


Figure 5-16: In-stope monowinch system

Component costs

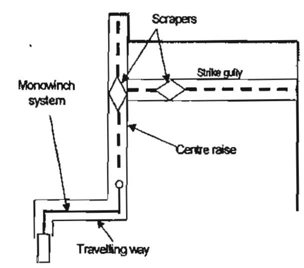
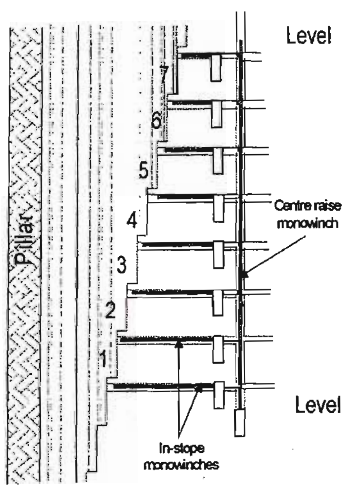
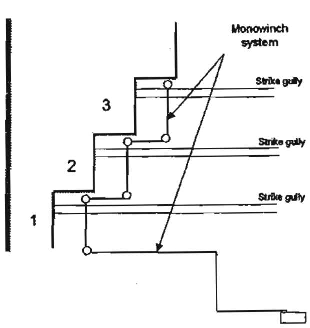
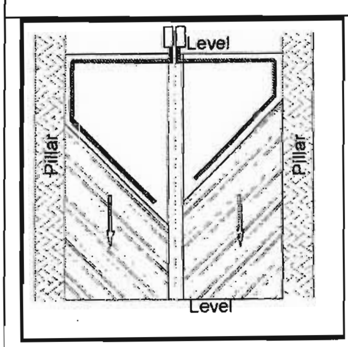
The component costs for the monowinch system are shown in Table 5-2.

Table 5-2: Monowinch component cost breakdown

Component	Monowinch	Cost (2002)
Motor	3 kW	R19 000
	4 kW	R11 000
	5,5 kW	R14 000
	7,5 kW	R18 000 to R20 000
	11 kW	R19 000 to R27 000
	15 kW	R28 000
	22 kW	R48 000
Rope	13 mm	R12/m
	16 mm	R18/m
Pulleys	Corner	R236 to R455
	Inline	R133 to R345
	Brow	R254
	Return	R155 to R288
Tensioner		R216

Table 5-3 presents cost estimates for various monowinch systems that are assumed to have certain parameter values. These costs exclude installation, excavation and labour costs.

Table 5-3: Cost estimates for monowinch systems

<table>
<tr><th>Scenario</th><th colspan="2">Assumptions</th><th>Approximate cost</th></tr>
<tr><td rowspan="7">
</td><td>Length of system</td><td>200 m</td><td rowspan="7">R50 000
Servicing along raise</td></tr>
<tr><td>Number of corner pulleys</td><td>8</td></tr>
<tr><td>Spacing between pulleys</td><td>5 m</td></tr>
<tr><td>Number of in-line pulleys</td><td>32</td></tr>
<tr><td>Return pulley</td><td>1</td></tr>
<tr><td>Tensioners</td><td>2</td></tr>
<tr><td>11 kW or 15 kW winch</td><td>1</td></tr>
<tr><td rowspan="7">
</td><td>Rope length (raise)
Rope length on strike</td><td>200 m
10-80 m</td><td rowspan="7">R250 000
Servicing seven panels</td></tr>
<tr><td>Number of corner pulleys</td><td>8</td></tr>
<tr><td>In-line pulleys (strike)
In-line pulleys (raise)</td><td>66
32</td></tr>
<tr><td>Spacing between pulleys</td><td>5 m</td></tr>
<tr><td>Return pulleys (strike)
Return pulleys (raise)</td><td>7
1</td></tr>
<tr><td>Tensioners (strike)
Tensioner (raise)</td><td>7
2</td></tr>
<tr><td>7.5 kW winch
11 kW or 15 kW winches</td><td>7
1</td></tr>
<tr><td rowspan="7">
</td><td>Rope length</td><td>250 m</td><td rowspan="7">R55 000
Servicing three panels</td></tr>
<tr><td>Corner pulleys</td><td>8</td></tr>
<tr><td>Spacing between pulleys</td><td>5 m</td></tr>
<tr><td>In-line pulleys</td><td>32</td></tr>
<tr><td>Tensioners</td><td>2</td></tr>
<tr><td>Return pulley</td><td>1</td></tr>
<tr><td>Winch (11/15 kW)</td><td>1</td></tr>
<tr><td rowspan="7">
</td><td>Total length</td><td>200 m</td><td rowspan="7">R100 000
Servicing two panels</td></tr>
<tr><td>Corner pulleys</td><td>3</td></tr>
<tr><td>Spacing between pulleys</td><td>5 m</td></tr>
<tr><td>In-line pulleys</td><td>40</td></tr>
<tr><td>Tensioners</td><td>2</td></tr>
<tr><td>Return pulley</td><td>1</td></tr>
<tr><td>Winch (11/15 kW)</td><td>1</td></tr>
</table>

5.2.2 Monorail systems

5.2.2.1 Monorail audit

Monorails (Figure 5-17) for in-stope applications were introduced much more slowly than monowinches. Developments followed the trend and pace of research into non-explosive mining equipment and the associated need to move heavy articles in a confined space. The impact-ripper project conducted in the 1980s by COMRO (Chamber of Mines Research Organisation of South Africa) led directly to the introduction of a simple I-beam-type monorail and these developments culminated in the completion in 1989 of a detailed feasibility study by COMRO/KBC of a rack and pinion-type monorail. During this period, the use of monorails in the coal and platinum industries was restricted to conventional horizontal applications.

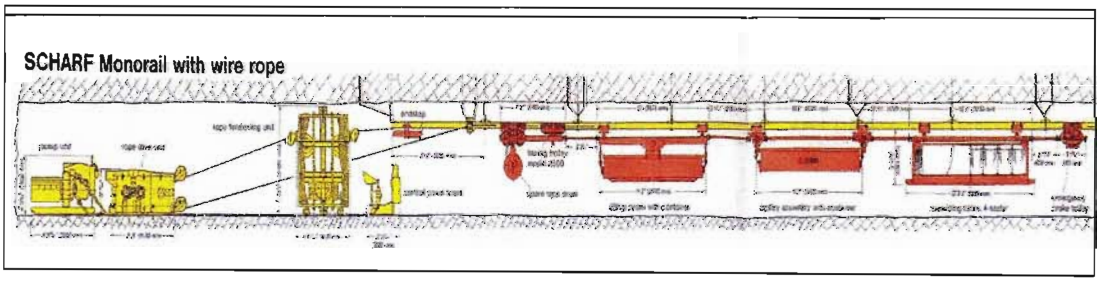


Figure 5-17: Typical monorail train (Nehrling, 2002)

Recent interest by gold mining companies in the use of monorails for the reef horizon is driven by a desire to improve safety and reduce the manual effort required for material handling in stopes, especially those stopes with long backlengths.

Types and installation

Monorail systems have been purpose-designed for material-handling duties. There are three basic types of monorail that could be considered for in-stope operations. These are:

- Large friction drive units: These are designed for horizontal and inclined operations at gradients of up to 20°. Pneumatic, diesel, electric/hydraulic, electrically powered and hydro-powered rack-and-pinion units are available, which can operate up to a maximum dip ranging between 35° and 45°, although 35° is the recommended limit. Current installations are operating at gradients of up to 41°. The capital costs for a large friction drive unit are typically R1 million to R5 million per train. Established suppliers include Walter Becker, DBT/Scharf and Hydro Power Engineering (HPE).

- Rope-driven monorail: The original motivation for developing this unit was to modify the electric or electric/hydraulic monorail to operate in the stope horizon. Due to the cost of such a unit, DBT/Scharf felt it would be more cost-effective to use a rope-driven monorail system. The capital cost for such a unit would be approximately R550 000.

- Manual and powered crawl units: These are small units suitable for horizontal operation only. They can be used as crawl systems in the stope cross cut to improve material handling with monowinch systems. They may also be applied in the strike gully, as a take-over unit for palletised material that has been delivered to the centre raise by the monorail. The capital cost for such a unit would be less than R100 000.

Method of operation

From cross cut to centre of raise

In this scenario, the monorail brings material to distribution points in the centre raise, usually at the strike gully intersection. From the strike gully, the material can be transported in several manners, by hand (human trains), by scraper scoop/basket, by in-stope monowinches or by crawl units operating in the strike gully. Although this system transports the material to the end of the strike gully, the material still needs to be rehandled from the gully to the point of application. Further, while material is being transferred from the gully to the face, scraping operations must be stopped.

DBT rope-driven system

The DBT Rope Drive System is driven off a rope which is driven from a winch located in the cross cut. The master trolley unit is connected to the rope and operates in the raise between the winch and the return unit (Figure 5-18). Potentially, this system could extend the delivery point from the centre of the raise to the end of the strike gully by utilising turntables at the strike gully position. At the centre raise/strike gully intersection, the material can be redirected to the strike gully by means of the turntable. The material is then transported to the face along overhead beams, either manually, by winch or by a crawl unit.

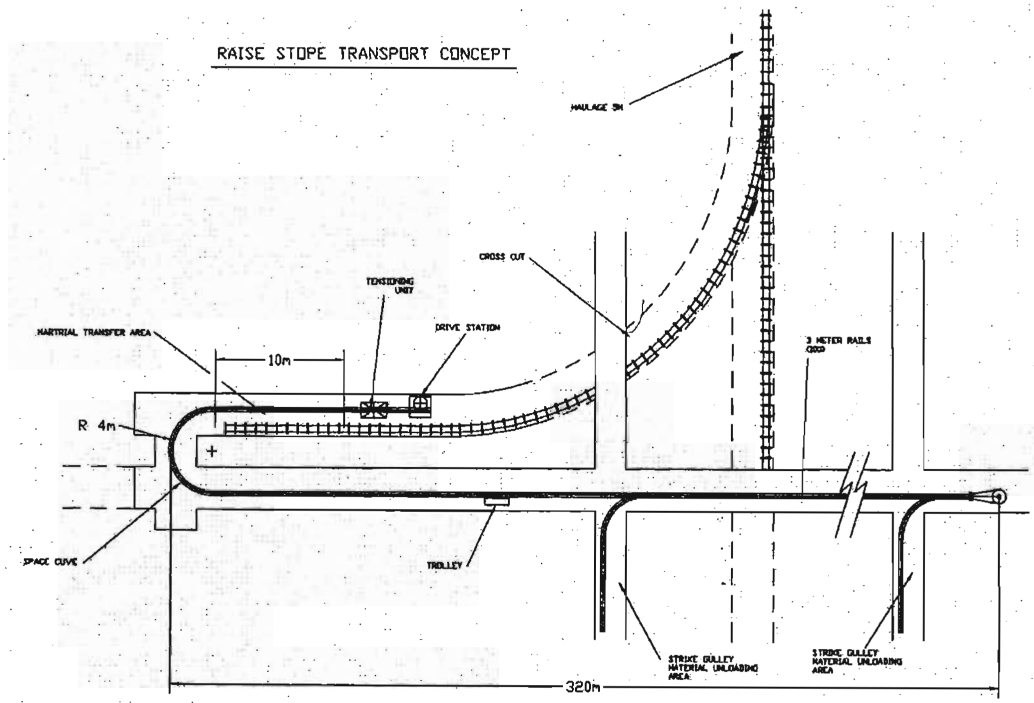


Figure 5-18: Rope-driven monorail system (Nehrling, 2002)

Future implications

Monorail systems from the cross cut to the stope faces may be regarded as the ultimate vision for in-stope material handling. In this scenario, material would be collected at the cross cut, transported up the centre raise and switched into the strike gully, where the monorail would transport the material to the point of installation above and below the strike gully. The material would be packaged in such a way that it could be assembled as a ready-to-install unit, and thus only limited manual labour would be required to build the pack, i.e. to block and prestress the pack. Where other materials, such as elongates and packs are required in the stope, the monorail would transport the items to a face material handling system which would then transfer the material to the point of application.

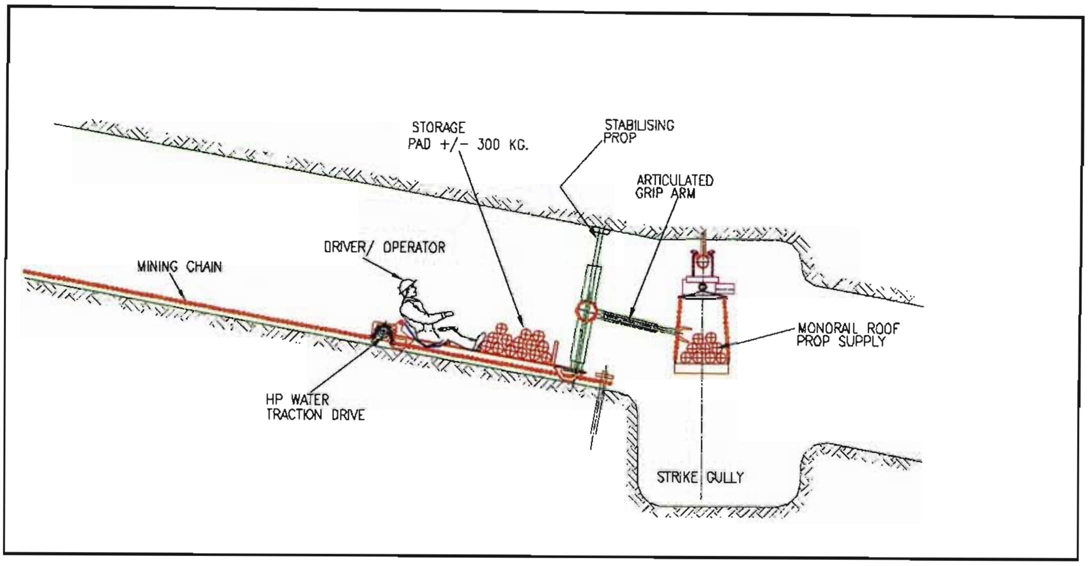


Figure 5-19: In-stope face material handling system (Wilson *et al.*, 2000)

This is a long-term scenario, which it is estimated would take at least a decade to achieve. Based on the experiences of operating monorails in a shallow incline environment (<14°), where two to three years were required to make the monorail system efficient, it is envisaged that similar periods will be needed to resolve each of the remaining technical challenges. For example, it may take two to three years for a monorail to operate efficiently from the cross cut to the centre raise. A further three years may be required to achieve similar results for the monorail operating in the strike gully. The technology for transferring material from the monorail to the desired position above or below the gully would still need to be developed, as would the face handling system for moving the material on the face.

If this vision cannot be realised, the role of the monorail as a cost-effective in-stope material transport system is questionable. The matter of utilising the monorail to move only heavy equipment is debatable, as the time and cost involved in such an installation make this mode of transportation uneconomical. In fact, it may be worthwhile in the first place to query how often these items need to be transported in a well-operated stope. Thus, the question must be asked, "Are monorails necessary or economically feasible in the stoping horizon?" If the monorail does not take material from the cross cut to the point of usage, then effectively the monorail is only replacing the monowinch without reducing the extent of handling requirements. The benefits of receiving palletised material in the raise are questionable, as the material would still require further handling to reach its final position.

This is not to say that monorail systems do not have a role to play in the stoping environment. Indeed, it may still be feasible to use the monorail to develop/raise and equip stopes, thus reducing the time required to establish the stope. In this case, the monorail is used primarily as a development tool, with material handling being a secondary function. In other cases, the monorail may be used to support layouts with long backlengths.

5.2.3 Evaluation of monowinch and monorail

There are currently only two viable means of transporting support material mechanically into stopes. The first is the well-established monowinch system and the second is the rack and pinion type monorail system.

The monowinch system is a relatively cheap system (R50 000 to R250 000) that is well known and applied in the mining industry. For the current status quo of grid mining applications, monowinches are adequate. However, the use of in-stope monowinches is recommended as it reduces the amount of manual labour needed, does not require the use of the strike gully, and is capable of feeding material to the stope panel. As far as the utilisation of monowinches is concerned, practice has shown that the quality of the installation appears to be the dominant indicator for the success of the system.

Nevertheless, it is proposed that current monowinch systems can be more efficient by:

- Making use of material bays.
- Providing separate material ways or barricading travelling ways for dual use.
- Using loading magazines or trailing start/stop monowinch switches.
- Installing monoropes properly and checking their condition regularly.
- Choosing the appropriate pulleys for the appropriate applications.
- Improving discipline of the system.

Each monowinch installation is effectively able to serve up to three stoping panels, although panels that achieve high face advances are generally served by a dedicated monowinch. A disadvantage of the monowinch is that it is labour-intensive and material often requires rehandling in order to reach the face. Furthermore, the system transports material in small units, and hence is unsuitable for handling heavy items (>35 kg). It is therefore unlikely to support any future introduction of mechanised or continuous mining.

In the short to medium term, monowinches can support stoping operations. However, appropriate mine planning must be done to ensure that the support and infrastructure requirements do not exceed the capabilities of the monowinch. For instance, highly concentrated stoping, coupled with high face advances over long backlengths, is not conducive to the use of the monowinch system.

The monorail should be viewed as the system of the future since it is able to handle material in bulk loads up to 3 000 kg. A monorail can handle all in-stope material, such as 6 m and 3 m pipes, winches, scoops, drill rigs, drill steel, vent columns, explosives, etc. In addition, monorails have the potential to transport the workforce in the stope.

The main problem with monorails is the suspension of the rail from the hangingwall, and this problem will be exacerbated in ultra deep level mines or where rock stress is a problem. Means of solving this problem include the installation of bridging beams between packs on either side of the transport way. The demand that material must be delivered to the face also dictates that the system must be capable of negotiating radii with small curvatures. Capital expenditure and operating costs are obviously of importance and the ability to integrate the monorail with other activities, such as raising operations, is an advantage.

The South African mining industry has used monorail systems at shallow inclinations with some success. However, the monorail is an immature technology for material handling in the reef horizon. Currently, trials are being conducted with monorail systems in the reef horizon at inclinations of 20° and 34°. Although it is too soon to reach any conclusions, it appears that it may be technically feasible to operate a monorail near the reef horizon. There are still many questions pertaining to the practical feasibility of operating a monorail in a stope. For instance:

- Will a monorail be able to handle common geological discontinuities or change direction from the dip raise to the strike gully?
- Once operating in the strike gully, will the monorail be capable of operating efficiently without interfering with the rest of the mining cycle?
- Alternatively, should material be transported by means of crawl units or lightweight rail? Once the material reaches the end of the strike gully, is it possible to transport the material as a unit to the place where it is required?
- Finally, will there be a need for an in-stope device to transport the material further along the face, as depicted in Figure 5-19?

Currently, monorails are the only technology available that may allow the vision of a "hands-free" material handling system to be realised. However, they are limited because they have only recently been introduced to the reef horizon and only operate in the vicinity of the centre raise. To achieve the full potential of the monorail, further research and development is required. Development of the in-stope monorail system should be conducted in incremental steps, some of which are:

- Establish the use of the monorail in the reef horizon.
- Switch the monorail from the dip direction to the strike direction.
- Get the monorail to operate in the strike gully without interfering with the rock handling process.
- Develop a lifting device capable of off-loading material from the monorail directly to the required installation position.
- Develop a face handling system for moving and installing material (packs and elongates) in the stope.

5.3 Rock handling

5.3.1 Introduction

In current mining practice, it has become imperative to secure a substantial increase in both the rate of face advance and productivity, compelling the mine engineer to consider better and more productive ways of mining. Current cleaning operations can support moderate rates of face advance, up to 1 m blast per day, using good management principles. Production rates above this level will most likely require changes to the layout and improved cleaning equipment. To improve on the current rates of advance of 6 m to 10 m per month, mine planners must take cognisance of the performance of the cleaning systems being used, taking into account stope width, panel length and shift duration. Furthermore, stope layouts should be designed for the removal of the ore from the panel face to the stope ore pass.

Over the past couple of decades, little progress has been made in improving stope cleaning systems, although attempts have been made to introduce continuous scrapers as a way of increasing rock handling in gullies. The possibility of automating scrapers to operate during the re-entry period has also been considered. Despite intensive efforts, however, the only significant improvement in face rock handling has been the introduction of high-pressure water jets.

The following diagram depicts (Figure 5-20) the stope layout decision process required to ensure that production targets are met. From the diagram, it can be seen that the stope parameters are interdependent and that all the process criteria need to be met for the system to be successful. The following sections review current rock handling practices and their suitability for ultra deep level mining.

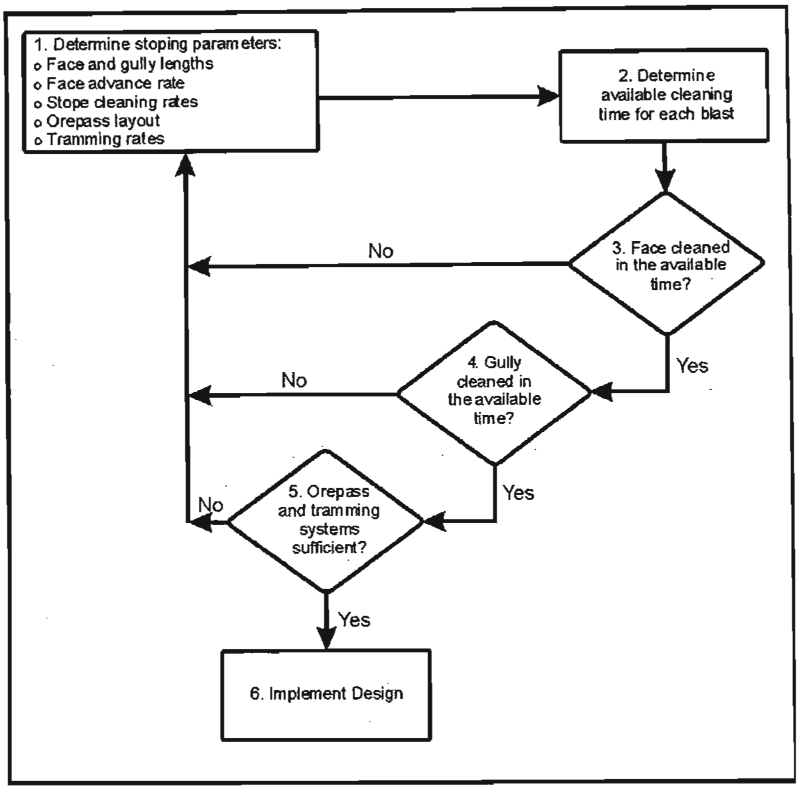


Figure 5-20: Flow diagram of in-stope layout design

5.3.2 Cleaning constraints

The movement of rock in conventional stoping layouts consists of a number of batch processes (face to the strike gully, strike gully to the centre raise, and raise to the ore pass) that have to be synchronised, much like a manufacturing process line. Furthermore, there is a relationship between face advance, face length, stoping width, scraper pull length and the in-stope shift duration in determining the cleaning throughput. The cleaning cycle must be planned to remove most of the ore from the panel during the primary cleaning shift, to ensure a clean face for the next cycle and to minimise the amount of ore locked up as sweepings. The output of such a system, or the face advance rate, is constrained by the batch process that has the smallest throughput. This is the system's constraint (Figure 5-21). The system can be optimised by maximising the flow of ore through the constraint and appropriately scheduling activities that interlink with the constraint. In the above example, the strike gully throughput is the constraint. Thus, production targets must be based on this throughput constraint, and be so designed as to ensure that the strike gully is free of large accumulations of broken rock. This will also benefit the movement of personnel and material.

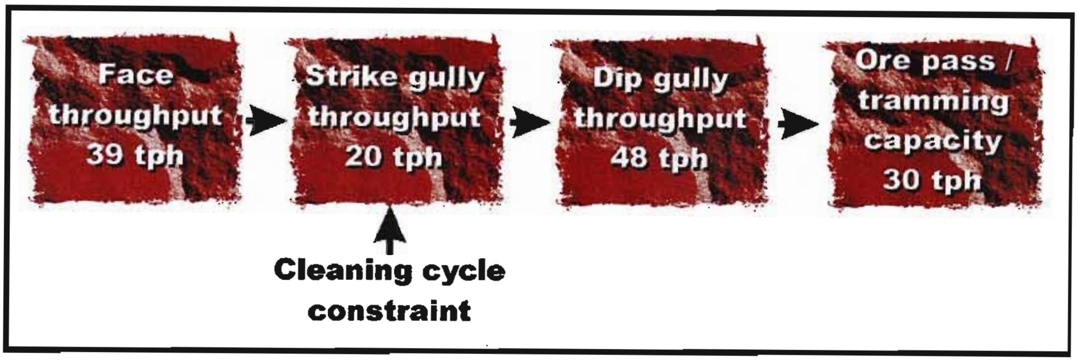


Figure 5-21: Cleaning cycle constraint
(NB. based on a face cleaning rate of 600 t-m/hr)

5.3.3 Cleaning rates

The instantaneous cleaning rates of an in-stope scraper scoop can be calculated using the basic equation that accounts for the rope speed, capacity of the scraper scoop, length of the entire rock moving trip (both empty and full), and turning/digging time. The following is an example of how to determine the tonnage cleaned on a face with a panel length of 30m and a strike gully 90 m in length, utilising a 0,75 ton face scoop and a 2,25 ton gully scoop.

$$\text{Time per trip face} = \frac{\text{Return trip distance}}{\text{Rope speed}} = \frac{30\,\text{m}}{1\,\text{m/s}} = 30 \text{ seconds}$$

Total time per trip = Time per trip + Turning/digging time = 30 s + 5 s = 35 s

Tons per trip face = Scoop/s capacity x Fill factor = 0.75 t x 80% = 0.6 t

$$\text{Tons per hour} = \frac{\text{Tons per trip face}}{\text{Total time per trip}} \times \frac{3600\text{ s}}{1\text{ hr}} = \frac{0{,}6\text{ t}}{35\text{ s}} \times \frac{3600\text{ s}}{1\text{ hr}} = \mathbf{61{,}7\ tph}$$

Equation 5-1: Theoretical face cleaning rate

$$\text{Time per trip gully} = \frac{\text{Return trip distance}}{\text{Rope speed}} = \frac{180\,\text{m}}{1\,\text{m/s}} = 180 \text{ seconds}$$

Total time per trip = Time per trip + Turning/digging time = 180 s + 5 s = 185 s

Tons per trip gully = Scoop/s capacity x Fill factor = 2.25 t x 80% = 1.8 t

$$\text{Tons per hour} = \frac{\text{Tons per trip gully}}{\text{Total time per trip}} \times \frac{3600\text{ s}}{1\text{ hr}} = \frac{1{,}8\text{ t}}{185\text{ s}} \times \frac{3600\text{ s}}{1\text{ hr}} = \mathbf{35{,}0\ tph}$$

Equation 5-2: Theoretical gully cleaning rate

5.3.4 Face cleaning

Face cleaning rates (Figure 5-22) are based on empirical studies done in the late 1980s by COMRO (Pickering, 1987; Morris *et al.*, 1988; Pickering *et al.*, 1988). The work indicates that as panels become longer, the face cleaning rate drops accordingly. In addition, the cleaning rate is dependent on the face conditions and cleaning method, e.g. utilising water-jet-assisted cleaning, and on the application of good mining practices, such as the correct fragmentation, the use of blast barricades, attention to face shapes, etc. Typical cleaning rates of 200 tons-metres per hour (t-m/h) can be expected for faces cleaned without water jets and rates of 400 t-m/h for water-jet-assisted cleaning, while rates as high as 600 t-m/h are possible for faces utilising water jet cleaning when combined with good mining practices.

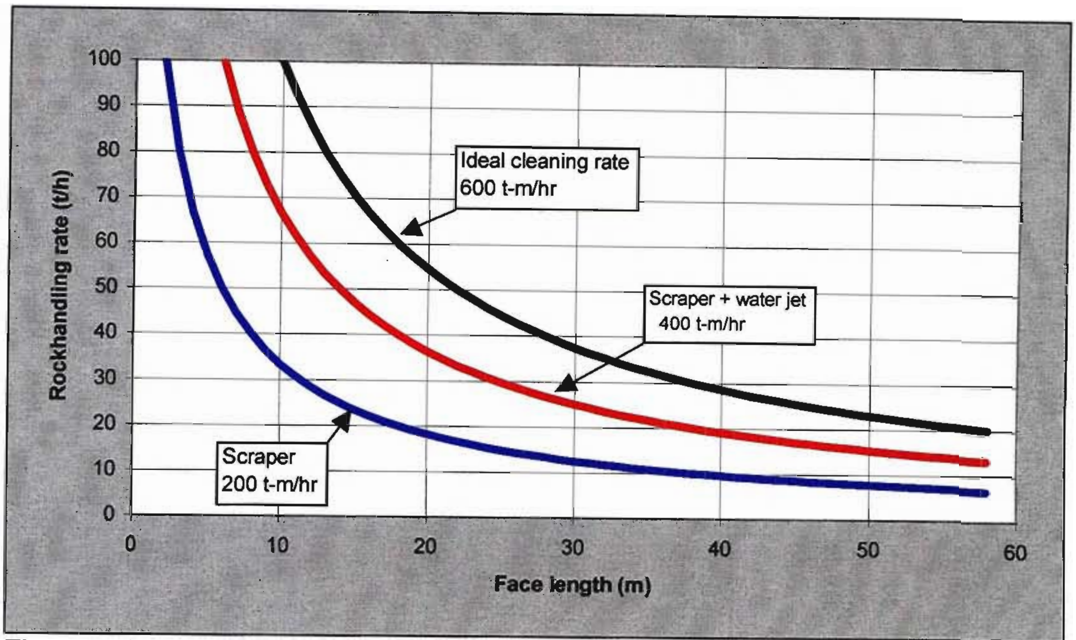


Figure 5-22: Various face cleaning rates (Morris *et al.*, 1988)

Table 5-4 illustrates the impact that face length has on the cleaning rates and thus on the total cleaning time, emphasising that the total cleaning time increases in line with the face length.

Table 5-4: Face cleaning parameters

Face length	25 m	40 m
Face cleaning pull length	28,1 m	43,1 m
Average face cleaning length	14 m	21,5 m
Face tonnage	103 tons per blast	165 tons per blast
Cleaning rate	42,8 tons per hour	28 tons per hour
Cleaning time	144 minutes	353 minutes

5.3.4.1 Face conditions

Face shape

Face shape plays a vital role in the cleaning performance of the face scraper. This is indicated by the potential cleaning rate of 600 t-m/h, which is based on several factors, of which face shape is, perhaps, the most important. Longer panels become more prone to poor face shapes due to out-of-sequence firing, misfires and geological disturbances. Bends in the face cause the scraper scoop to hang up, thus requiring rerigging or water jets to move the broken rock to the scraper.

Rigging

The poor positioning of equipment, such as scraper ropes and scraper scoops, can make cleaning difficult when it is left in a position where it can be buried or damaged by blasted rock. Efficient cleaning is further complicated when rig holes are not drilled or are blasted. It must be noted that rigging is seldom done correctly, which contributes to the poor cleaning rates commonly achieved.

Preparation plays a critical role in the cleaning process. For example, when the equipment is left in the vicinity of the face, it is easy for the stoping crew to commence cleaning operations very quickly. However, when the face has not been prepared, the available cleaning period is shortened as the cleaning crew is required to spend more of the shift on setting up. Also, sharing equipment between faces is a practice that should be discouraged as this results in unnecessary manhandling and transportation of equipment.

Scraper ropes should preferably be rigged into the face and secured to the top of the permanent support prior to the blast, thus obviating the need for the cleaning crew to "snake" down the blasted panel to rig the scraper rope. To further improve efficiencies, all equipment, such as snatch blocks, pins, hammers, hoses and temporary support, should be placed in convenient positions near the stope face.

Placement of support

Incorrectly positioned support (Figure 5-23) can influence cleaning rates by obstructing the scraper path, causing delays and thereby shortening the time available to clean. The positioning of the final row of supports should take account of the face advance and the width of the scoop. Generally, support should be placed a minimum distance of 1,2 m from the face (before the blast) to allow for the free movement of the scoop, which is normally 1,0 m to 1,5 m in width.

Figure 5-23: Face scoop blocked by furthermost prop

Blast barricades

Blast barricades are an effective method of confining the ore on the face, thus reducing the number of re-riggings and the area to be cleaned by the water jet. Local conditions influence the type of barricade selected, as well as its position relative to the face. There is a wide range of preferences for the type and position of blasting barricades. What is important is that a barricade should be used to confine the broken rock and thereby increase the rate of face cleaning.

5.3.4.2 Water jetting

Water jetting plays an important role in face cleaning as it allows a higher cleaning rate by moving the rock into the path of the scraper, thus reducing the number of rerigs involved in the cleaning process. There are several advantages to using water jets for stope cleaning, as listed below:

- A clean face, as no rock is left on the face, removing the need for sweeping, and resulting in better gold recovery and an improved mine call factor (MCF).
- Higher face cleaning rates, resulting in more blasts and an increase in the face advance rate.
- Reduced water consumption when managed properly.

When used correctly, water jets result in lower water consumption compared with low-pressure water systems. The water flow from a 25-mm-diameter hose is approximately 6 litres per second compared with 2 litres per second for a high-pressure, low-volume water jet and 3,4 litres per second for a low-pressure, high-volume water jet. In addition, most water jetting systems have automatic shut-off valves that close off the water when not in use.

5.3.5 Strike gully cleaning

Strike gully cleaning is the key process in the cleaning cycle. The ability to remove the rock from the discharge point of the face to the ore pass or dip gully determines the face panel length and the face advance. For this reason mine planning must concentrate on removing the broken rock from the gully. In general, strike gullies dip at between 5° and 20° and operate between lengths of 15 m and 120 m. Strike gully winches typically have a rating of 37,5 kW, utilising 19-mm-diameter ropes pulling ± 1-ton scoops. The degree of filling is dependent on the dip. A 60% to 80% fill factor can be applied to scraper scoops operating at dips between 0° and 10°, while a fill factor of 80% to 120% can be used for a scraper cleaning in gullies with a dip between 10° and 20°. As a rule, a scraper requires an unbroken smooth footwall, otherwise a broken/rough footwall may damage the ropes or, in extreme cases, the winch. However, this is not a problem in most South African gold mines.

Based on empirical studies conducted by COMRO in the late 1980s and confirmed through recent studies (Rupprecht, 2002), strike gully cleaning rates of 900 tons-metres per hour (t-m/h) are achievable when utilising a single 2-ton scraper or a combination of a pilot scoop and a 1,5-ton scoop (Figure 5-24). The above cleaning rate takes into account factors such as scraper utilisation, scoop fill factor, rigging and rerigging, repairing ropes and breaking rocks at the tip.

With current equipment, it is possible to clean a nominal 112-ton face blast in 11 hours over a 90 m strike gully. It may therefore be concluded that it will be difficult to reliably achieve a 15-m/month (30 m panel) face advance with gullies approaching 90 m in length.

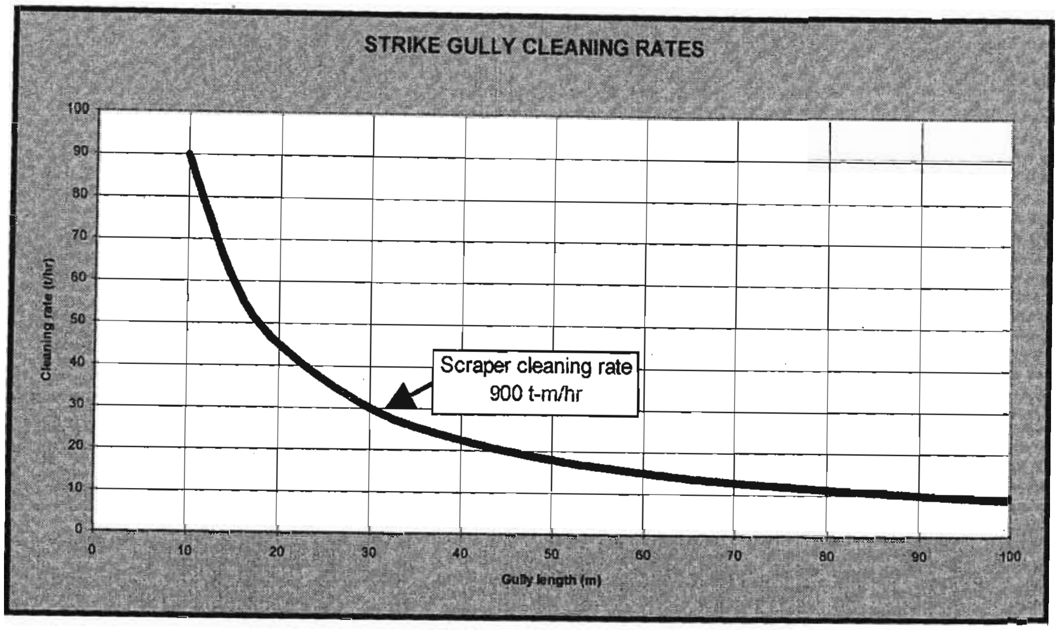


Figure 5-24: Strike gully cleaning rate (Morris *et al.*, 1988)

Currently, emphasis is placed on utilising the strike gully to facilitate high face advances as the gully is used to store (1,8 t/m) broken rock rather than transferring the rock into the ore pass. However, this is a temporary solution and works only until the gully is choked with broken rock (Figure 5-25). Hence, emphasis should be placed on improving the strike gully cleaning rate, with activities being planned so that the gully scraper achieves maximum utilisation.

The strike gully serves as the main artery for not only rock, but also personnel and material transportation. Strike gully cleaning is often delayed by the transportation of material, either because pack units are being transported in the gully or, while the monowinch is operation, as material on the monorope fouls the scraper rope. Therefore monorope installations should be designed to minimise interference with strike gully cleaning. On certain reefs, it may be possible to develop monoways in the hangingwall to allow the monowinch to operate concurrently with the scraper.

Figure 5-25: Strike gully full of ore

5.3.6 Dip gully cleaning

For scraping on an incline, the scraper cleaning system remains unchanged. The power, however, increases for scraping up dip and decreases for scraping down dip. Table 5-5 gives the approximate variation in rope pull. As a base case, a 30% improvement or reduction in the cleaning rate is used for dip scraping, i.e. a 30% decrease in the cleaning rate for up dip scraping and a 30% increase for down dip scraping (Figure 5-26). The decrease in productivity is caused by the scraper having to move against gravity and negotiate uneven footwall. In contrast, down dip scraping is aided by gravity.

Table 5-5: Scraping on grade (after Riemann, 1986)

Angle from horizontal	Scraping up dip	Scraping down dip
	Increase in rope pull	***Decrease in rope pull***
10°	15 % rope pull	20% rope pull
20°	30% rope pull	30% rope pull
30°	35% rope pull	65% rope pull
45°	40% rope pull	Material slides by itself

5.3.6.1 Down dip scraping

Down dip scraping is commonly used in underground layouts for panels above the longest ore pass, especially in layouts with long backlengths (>250 m). The broken rock is delivered by the strike gully winch into the dip gully and then transported to the tip normally by 37,5 kW or 55 kW winches. A down dip cleaning rate of 1 200 t-m/hr is applied for normal dip cleaning, utilising a single 2-ton scraper scoop (Figure 5-26). This cleaning rate can increase up to 1 800 t-m/hr based on the number of scoops, the fill factor, dip of reef, and the size and condition of the scraper equipment.

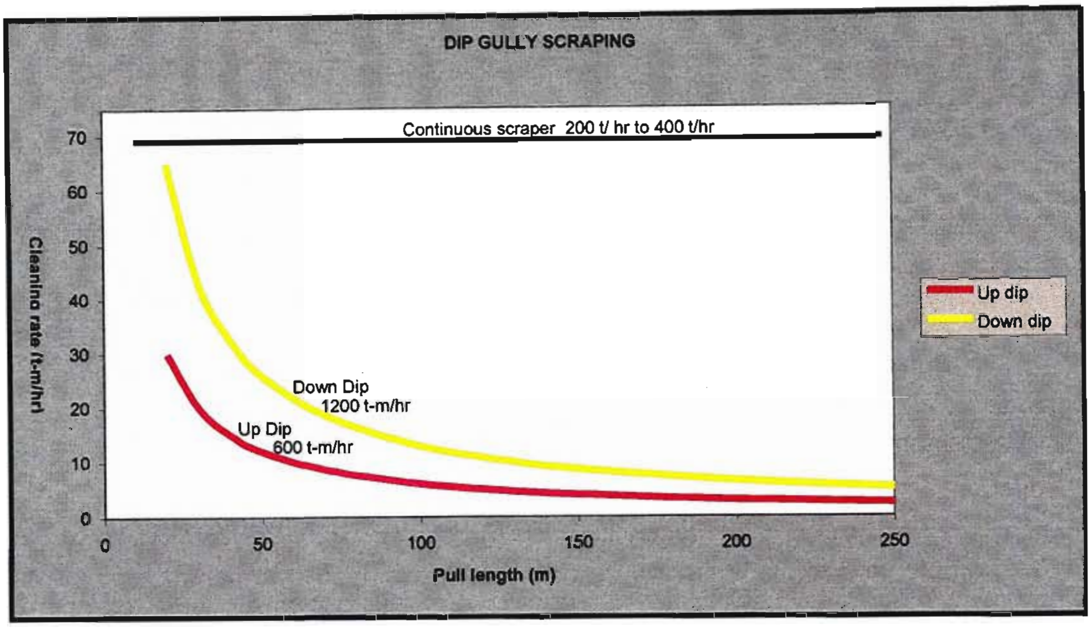


Figure 5-26: Dip gully cleaning rates

In general, it is not the dip gully that is the throughput constraint as the scraper can operate during both the day and night shifts. The scraper is further aided by the dip of the reef and often by the addition of another scoop. In order to ensure that dip gully cleaning does not became a system constraint it is important that mine planners limit the number of panels feeding into the dip gully. In special circumstance dip gully scraping can utilise both up dip and down dip scraping, usually two thirds of the scraping distance applying down dip scraping and one third of the scraping distance applying down dip scraping.

5.3.6.2 Up dip scraping

Up dip scraping is used in special circumstances, for example winzing operations or where the ore body is extracted from below the last access level. Conventional up dip scraping utilising 37,5 kW winches achieves a cleaning rate of 600 t-m/hr. This rate can be improved by utilising larger winches or more scraper scoops.

5.3.6.3 Continuous scraper

The continuous scraper is a slow-moving scraping machine travelling at 0,25 m/s. It operates with quarter-ton buckets attached to a single drive chain; the spacing of the scoops determines the cleaning capacity, which is of the order of 250 t/h (Figure 5-26). As the continuous scraper occupies the entire dip gully, personnel and material have to be diverted away from the centre gully. The continuous scraper can be loaded either at a single or at multiple feed points with the buckets, spaced typically at 1 m, picking up and transporting the rock to the tip (Figure 5-27 and Figure 5-28).

With its ability to convey high tonnages over long distances and up steep dips, the continuous scraper offers the possibility of introducing improved mining layouts. This scraper can operate in both the up dip and down dip direction, although the up dip direction is preferred as it offers the added benefit of separating water from the rock.

To date the continuous scraper has been utilised with limited success. One of the major problems encountered is its inability to handle geological discontinuities, which prevents the continuous scraper from being used over longer backlengths.

Figure 5-27: Continuous scraper in operation looking down dip

Figure 5-28: Continuous scraper head drive unit looking down dip

5.3.7 Sweeping and vamping

5.3.7.1 Sweepings

Sweeping of the panel is the final process in the cleaning cycle. This process removes the last accumulation of broken rock by using waterjets (Figure 5-29) or a combination of water, brooms and brushes to remove the fines from the panel. Thus, it is important for most of the ore to be removed by the initial cleaning operation. In most operations, sweeping falls outside the primary cleaning cycle and is done as a secondary function, with sweepings being performed independently of the drill-and-blast cycle. Typically, sweepings are monitored on a monthly basis and production personnel tend to perform sweepings close to the actual survey measuring date.

Figure 5-29: Sweepings in back area

In stopes utilising backfill, however, the sweeping function becomes integrated with the drill-and-blast cycle, forming part of the support cycle. Because support, in the form of backfill, is installed continuously, it is important that all ore is removed and a final washing conducted before the support (backfill) is installed. If cleaning is incorrectly done, then ore will be left behind and become a source of gold loss. Control of backfill placement therefore needs to be managed appropriately.

5.3.7.2 Vamping

Vamping operations are similar to sweeping, only the activity refers to gullies, ore passes and cross cuts. Vamping is the final stage of a panel's life, when the remaining ore is removed, followed by the reclamation of all useful material, i.e. pipes, tools, winches, scoops, etc. The vamping of strike and dip gullies is conducted in a similar manner to sweepings, employing the same type of equipment. Vamping of the ore pass consists of emptying the box, washing fines into a hopper and reclamation of the ore chute. Cross cut vamping is conducted by removing the top 50 cm of track ballast, thereby collecting any fines lost through tramming operations. Cross cut vamping can be done either by hand or by vacuum machines.

6. Guidelines

6.1 Introduction

The guidelines proposed here provide best practices for the design of ultra deep level underground transport systems. The suggested transport design methodology could contribute significantly to reducing transport-related accidents in South African gold and platinum mines, and also help to raise productivity by improving the durations of face shifts, reducing fatigue, supplying material in a just-in-time approach, and increasing efficiencies by removing ore efficiently.

With respect to the development of transport systems, historically there has been a considerable amount of work and innovation over the years on horizontal transportation, but very little on in-stope transportation. In addition, much of the work has concentrated on rock and material transportation, with little emphasis placed on the movement of the workforce. Furthermore, no real effort has been made with regard to integrating the different areas of transportation.

The guidelines below represent a compilation of best practices from current mining operations and recommendations for horizontal and in-stope transportation in future ultra deep level mines. The envisaged requirements do not necessitate the introduction of new technology but rather the optimisation of current systems by implementing available technologies in combination with proper planning, management and the integration of the systems. It is intended that these guidelines be used by mine designers and planners during the design process of ultra deep level transport systems. In many cases, the guidelines presented are equally applicable to current operations.

6.2 Significant general findings

The research work carried out in the area of transportation has resulted in the following general design findings, which are considered critical to the establishment of a successful system.

Trackbound systems are preferred

Trackbound transport is the preferred system for ultra deep level mines served by vertical shafts and extensive horizontal infrastructure.

Importance of integration

There will be constant interaction between personnel, material and rock in all areas of the transport system and that this must be taken into account when such systems are designed.

Importance of scheduling, communications and control

Many of the problems encountered with current transport systems are caused by inadequate or inappropriate controls. The importance of integrated horizontal transport schedules cannot be over-emphasised as this remains the easiest and cheapest means of addressing existing system difficulties. Effective communication plays a key role in securing and maintaining compliance with planned schedules. The incorporation of technologies such as tracking, automation and communication, along with carefully designed and well-managed transport systems, must be further promoted. Implementing an appropriate transport system, combined with the expectation that the 21st century worker will be better educated and more motivated, will lead to high levels of efficiency, productivity and safety.

Importance of the design of nodes and end-points

Stations and cross cuts constitute nodes in the transport system and the working faces are the end-points. Both nodes and end-points have been identified as requiring considerable improvement with regard to both design and organisation.

6.3 Horizontal transportation

Horizontal transportation takes place from the station area to the end of the stope cross cut. Traditionally, it has been inefficient (± 28%) in comparison with vertical transportation (± 95%) and accounts for a high proportion (± 28%) of mine accidents. This area can be improved significantly through better track work and management systems.

6.3.1 General horizontal transportation issues

6.3.1.1 Management

No transport system, however modern or well designed, will operate efficiently and effectively unless it is operated within a well-organised and managed structure. The following management principles should be considered:

- Scheduling and controlling are tasks that have been identified as being crucial for successful transport systems. Scheduling is required to co-ordinate all the operations within the transport system, thus ensuring that the system operates properly and on time. Transport systems should be prioritised to ensure completely reliable delivery, whether of personnel, material or rock.
- Haulages should be operated in accordance with strict schedules, akin to those used for the transport of personnel, material and rock through vertical shafts. A process management system, combined with prioritised scheduling, is recommended. Scheduling should take account of off-shift workers, including night and afternoon shift workers and special shift workers, as well as of the transportation of large items of material such as oversized equipment, pipes and tracks.
- Emphasis should be placed on establishing and maintaining a continuous transportation process, thereby reducing the need to increase haulage speeds above present norms (16 km/hr).
- Drivers of trains should be multi-skilled (trained) in the maintenance of locomotives, rolling stock and haulages. This would give them greater "ownership" and responsibility.
- An integrated approach that considers all transportation requirements must be adopted in mine design. Design decisions must not be sacrificed at a later stage due to time or financial constraints, and simulations should be used to assist in optimising mine planning.

6.3.1.2 Haulage system

In ultra deep mining layouts, haulage systems must be able to cope with high rock stress and heat loads, and should support higher rates of face advance. The current trackbound systems will be able to cope successfully if significant improvements are made. The following guidelines are proposed:

- Mining engineers should adopt an integrated approach that considers all transportation requirements, including haulage classification and speed requirements. It is again stressed that design decisions must not be sacrificed at a later stage due to time or financial constraints.
- Tracks should be installed to a sufficient standard to enable them to handle the required duty of the personnel, material and rock transport system.
- Maintenance, whether performed by mine personnel or by contractors, must be done on a continuous basis. A maintenance schedule must be drawn up for haulages, which includes daily, weekly and monthly examinations. The personnel required for haulage maintenance should be given realistic productivity targets. One worker per 2 km of track is appropriate for haulages that are shotcreted and where track work is initially installed to a high-quality standard. Some form of quantity and quality assessment must be done to monitor track installation and maintenance.
- Designated parking areas must be provided on the levels for locomotives and carriages. Proper procedures must be instituted to ensure the orderly management of rolling stock, particularly during shift change-over. In the haulage itself, passing loops at regular intervals, able to accommodate at least one train length, should be installed . Signalling devices should be incorporated into the loops to control traffic.
- Better lighting improves haulage conditions as problem areas can easily be identified. All junctions, ventilation doors, personnel loading areas and places where workers congregate should be illuminated and in non-illuminated haulages, the sidewalls should be whitewashed to improve visibility.
- Ventilation doors should be automated or telecontrolled so that the trains do not have to stop to open and close them manually. Stop blocks should be removed from main haulages, thus precluding the driver or guard from having to leave the locomotive to remove or replace the stop blocks.
- Automated switches could be one-way to impose control over the transport system.
- Communication systems are required between the train driver and a central control room. The control room personnel could then make

contingency plans if there is an emergency or a breakdown, and they could also give specific instructions to the driver relating to daily tasks.

- Leaky feeder systems should be considered for all new projects. However, existing operations should base the decision on the economics of installing such a system and on the benefits that would be derived from such a system.
- Signalling devices should be used to control traffic and improve the flow of traffic.
- Fans or other significant sources of noise should not be installed at points where signalling or communications are routinely done.
- It is advantageous to have designated haulages that separate rock from personnel and material transportation. Further work is required on the possible use of return airways for rock transport.

6.3.2 Horizontal transportation of personnel

Locomotive trains are the predominant transportation medium in most deep level mines, handling ± 192 workers per hour at speeds of up to 16 km/hr. A minimum tunnel size of 2,0 m x 2,0 m is required to operate this system at a maximum gradient of 1°.

For effective personnel transportation, the following possibilities should be considered:

- Control pedestrian access in the haulage. The haulage should be free of personnel at all times, except for maintenance personnel and supervisors. Pedestrians should be required to request permission from the controller before entering a haulage.
- To determine the number of trains or trips required on a level, the capacity of the train, coupled with the cage delivery rate and the distance to be travelled to the working places, must be considered.
- Ergonomically designed personnel carriages should be used. Ideally, the personnel train's capacity should match the cage capacity.
- The train must always be controlled from the front. Therefore some means of reducing the time required for shunting the locomotive must be found, either by installing a passing loop, or utilising two locomotives on either end of the train, or utilising remote-control driving from the guard's van.

- New technologies such as transporting personnel in rock hoppers and having carriages air-conditioned merit further investigation.
- Personnel loading bays, near the working places, should be provided on a regular basis, every 600 m to 800 m. These loading bays should consist of a concreted platform or a section of double track for the length of the personnel train. The personnel carriages should be parked in these bays when not in use so that they do not interfere with other transportation activities.
- The environmental conditions at the bays must be conducive to allowing workers to wait for the train in comfort. The area should be well illuminated, with the train schedule clearly posted. The parking area for the train should be well demarcated, as should the places where personnel are to enter the carriages.
- For distances of less than 1 km, personnel should walk to the working places.

6.3.3 Horizontal material transportation

Horizontal material transportation begins at the shaft and ends at the cross cuts or other working places such as development ends or service excavations. To ensure maximum utilisation of the transport car and minimum congestion in the haulages and cross cuts, material cars should be off-loaded quickly, with the cars being sent back to surface as soon as possible.

The design of any material transport system should start with the requirements in the stope panel and work back to surface. Materials should be packaged on surface in a form suitable for handling. Support elements and other materials such as cement bags should be packaged to fit on the cars and in accordance with the requirements of the stoping panel. Items not suitable for direct packaging can be loaded onto pallets or containers that are also sized for the cars. Long items, including pipes, rails and switches, should be transported in specially designed cars or bogies.

Horizontal transport should be organised in such a way that both full and empty cars are handled effectively during one shift. This "material shift" should not coincide with major rock tramming activities. It will be possible to achieve this if sufficient storage space and appropriate unloading equipment are available at the end-points (i.e. the cross cuts).

6.3.3.1 Shaft station

Bank and shaft station designs should be closely matched and standardised. This greatly simplifies operation and enhances safety. The following measures should be considered for the effective and efficient handling of material cars at stations:

- <u>Provide sufficient storage space</u>
  Double track should be provided immediately adjacent to the shaft. The minimum length required is the sum of the average length of a material car plus the maximum number of cars that are expected to be handled during a single shift. For example, if the cars are 3 m long, and 60 cars are handled during a shift, then 180 m of double track is necessary.

- <u>Provide mechanical means of moving cars</u>
  Some means of moving cars forward to the shaft must be provided. Capstan winches, purpose-designed pushers and modified locomotives may be used for this purpose.

- <u>Design effective layouts</u>
  Storage space must be provided in such a manner that it does not affect personnel or rock transportation operations. Material cars should not be stored in the tip cross cut, as this will hamper rock tramming activities.

- <u>Consider shaft safety stopping mechanisms</u>
  The choice, design, configuration and operation of shaft safety stopping mechanisms must take material transportation needs into account. This is especially the case with long material.

6.3.3.2 Cross cuts

The design of cross cuts is critical to the operation of the material transport system. Most cross cuts are required to handle rock loading, material and explosives, as well as access by personnel to the stoping horizon. In addition, mining services such as power, water, compressed air and backfill converge at this point. The following guidelines for good material handling practice are relevant:

- <u>Provide sufficient storage space</u>
  The importance of providing a properly designed material storage bay in the cross cut cannot be over-emphasised. Good practice dictates that material storage capacity for at least two days should be provided in cross cuts. This should not hamper other operations such as rock loading. The storage space can be provided on a separate spur line or in a dedicated storage area. The latter minimises material car cycle time, but increases the frequency of rehandling.

- <u>Minimise unloading time</u>
  Cross cuts should be kept open for rock loading and other purposes (such as ventilation) at all times. Material cars should spend as little time as possible in the cross cut. This can be achieved by providing suitable equipment for off-loading in the cross cuts. Lifting equipment, including crawl beams and chain blocks, can be used at these points. Alternatively, flat cars fitted with frictionless beds could be used, with monoropes operating immediately above them. The load would be pushed horizontally onto a stationary frictionless bed to be used as the storage area. Also, the use of real-time information systems should be considered for tracking and controlling the movement of full and empty cars.

- <u>Position monoropes correctly</u>
  Where monoropes or some other overhead material transport system are used, these should be positioned so as to minimise the amount of subsequent rehandling required in the storage area.

6.3.4 Horizontal rock transportation

Trackbound systems (locomotives and hoppers) are generally suitable for ultra deep level mining. The prevailing poor track conditions are the single biggest cause of the current inefficiencies in underground trackbound transport. Currently, a 28% level of utilisation is normal, with individual trains transporting no more than 70 tons to 100 tons per day, which equates to less than three productive trips per shift. The necessity for increased maintenance of rolling stock and locomotives, the high frequency of derailments, slow operating speeds, increased operating costs, congestion and low utilisation of the infrastructure are all symptoms of current poor track conditions. With ultra deep level mining it will be essential to increase productivity. The number of trains in use should be decreased by increasing the track standards – this will make it possible to use larger units capable of running more efficiently.

The following guidelines are recommended:

- Loading and discharge of rock is a problem area in all mine transport systems. Therefore, the number of loading and discharge points should be kept to a minimum.
- Remote-controlled chutes should be used to improve the overall feeding of ore into hoppers, as well as to reduce the risk of workers being exposed to mud rushes.
- Reducing the amount of water used in the stoping environment decreases the risk of mud rushes or spillage of ore. For this reason, water usage should be limited to a maximum of 2 tons of water to 1 ton of rock.
- For high-production areas, tramming between the points of loading and discharge should be undertaken independently of the loading operation. The tramming system should be based on a "locomotive gathering system" whereby large locomotives gather hoppers from a designated area near the reef horizon and smaller locomotives are used to load the hoppers at the loading points. In this way, one locomotive will operate in conjunction with two or more spans of hoppers, so that tramming can take place simultaneously with loading and discharge.
- Communication systems should be used to increase tramming capacities, as voice communication has been found to reduce congestion and improve the overall efficiency of tramming operations. The use of interactive tags and tag readers also increases management capabilities, as information is supplied on a real-time basis.
- The use of traffic control and communication systems is recommended on single-track haulages planned for tonnages greater than 35 000 t/m. For tonnages in excess of this figure, passing loops or double haulages should be considered.

6.4 In-stope transportation

In-stope transportation is concerned with operations between the stope cross cut and the working face. Historically, in-stope transportation has employed basic technologies, with many activities still being performed manually. Congestion remains a problem in the proximity of the stope entrance due to the continuous movement of personnel, material and rock. This area requires substantial improvement in order to achieve higher production levels.

6.4.1 In-stope personnel transportation

Currently, there is no transport system that will significantly improve the movement of workers in the stoping environment. However, the adoption of mechanised mining methods could necessitate the introduction of systems such as the monorail, which could then be used efficiently for personnel transportation. It must be emphasised, however, that at some stage during the transportation process it will become more appropriate for personnel to walk. Thus, travelling by foot in the reef horizon should not be regarded as being inappropriate.

6.4.1.1 Stope entrance

The following design parameters should be considered to reduce the congestion currently experienced:

- Dedicated entrances should be provided for the stope crews and in a position that prevents rocks from the gully entering the travelling way.
- A clear view of any machinery moving along the travelling way should be provided.
- Inclined travelling ways should be positioned away from box fronts to ensure that rock loading operations do not interfere with the stope entrance.

6.4.1.2 In-stope improvements

The problem of congestion experienced in the cross cut becomes even more acute in the stope horizon. Key problems are the general lack of working height and the dynamic nature of the working environment, such as the advancing face and stope closure. The batch nature of the mining process further aggravates this situation. The following practices should be considered to address these problems:

- Illumination along the face should be considered as there is a direct correlation between safety and productivity, and levels of lighting.
- Where personnel share the travelling way with material transportation, the travelling way should be physically barricaded and have a minimum height of 1,8 m and a width of 0,8 m. In-stope travelling ways should take account of the stope closure and ride that are expected over the life of the travelling way. As a guide, a travelling way should initially be excavated to a height of 2,2 m and a width of 2,4 m, which would provide for 0,4 m of closure and 0,4 m of ride.
- Travelling platforms or by-passes are required for ore passes, and bridge constructions are recommended for safe and continuous travel over strike gullies.
- The panel lengths must match the cleaning rates attainable in the strike gully. This will help to prevent the strike gully from becoming congested with rock, thus restricting the movement of personnel.

6.4.2 In-stope material transportation

6.4.2.1 Monowinches

Material handling systems using monowinches generally provide adequate means of supplying materials and monowinch systems are the only ones currently suitable for ultra deep level mining. However, the monowinch requires rehandling of material and is unsuitable for items heavier than 35 kg. For the handling of material of less than 35 kg, monowinch systems should be used, taking the following into consideration:

- The facility must be so designed as to ensure minimal interference with rock handling operations.
- The importance of providing a properly designed material bay in the cross cut cannot be over-emphasised and the design should take into account storage, the unloading of cars and the movement of materials into the stope. For manual off-loading systems, good practice dictates that material storage capacity for at least two days should be provided in each cross cut. In cases where mechanised handling equipment, such as overhead crawls or monorails, is available, storage capacity can be

reduced to one day. Storage space can be provided on a separate spur line or in a dedicated storage area. Material cars should spend as little time as possible in the cross cut and material from the train should be off-loaded and stored within 30 minutes. Where monoropes or other overhead material transport systems are used, these should be positioned to minimise rehandling of material from the storage area.

- Monowinches from the cross cut to the centre raise should be used in conjunction with smaller 7,5 kW monowinches, operating below the strike gully, to feed the advancing stope face (Figure 6-1). This will allow material transportation to continue while the strike gully is being cleaned. In addition, it should be ensured that the distance between the end of the monorope and the face is no greater than 5 m to 7 m.

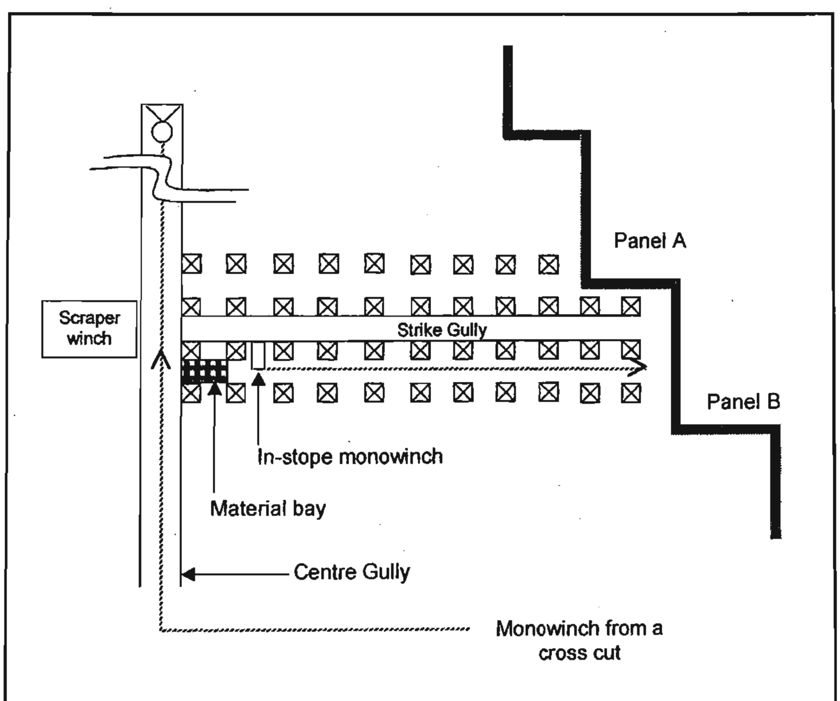


Figure 6-1: Recommended monowinch system for grid and breast mining

- No more than two winches should operate from a cross cut and in no case should a monowinch serve more than four panels. Three panels are considered the ideal maximum.
- 11 kW monowinches supplied with 13 mm ropes should be sufficient for material-handling distances of up to 200 m. For longer distances, 15 kW monowinches with 16 mm ropes should be used. The use of 22 kW monowinches should be restricted to special circumstances as the components of the monowinch are not designed for such large winches.

- Dedicated material ways should be provided to separate the access of workers and material into the stope.
- In-stope material ways sharing the same area with personnel should be installed, together with barricades between the travelling way and monowinch system (Figure 6-2).

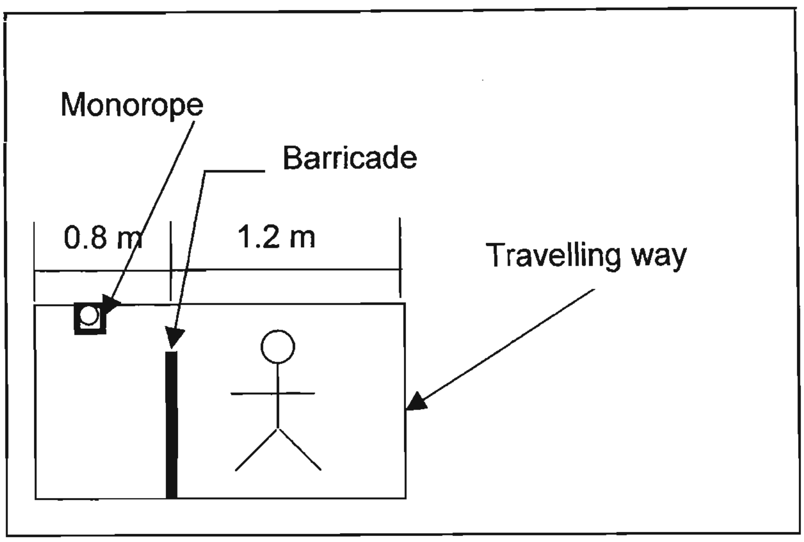


Figure 6-2: Combined material and travelling way

6.4.2.2 Monorail

If the current production rates of between 6 m to 10 m are required, then monowinch installations would, albeit with a revised layout, be sufficient. However, if greatly improved production targets are required, then it is recommended that further development of monorails should be focused on.

The introduction of monorails for in-stope applications has been much slower than that of monowinches. Recent interest by the gold mining industry in monorails for in-stope use is driven by desire to improve safety and reduce the manual effort required for material handling in stopes, as well as by the need to offset the effect of long backlengths. The following guidelines are recommended for the use of monorails in the stope environment:

- Current and future trials of monorails should concentrate on achieving the vision of a “hands-free” material handling system, which will allow material to be placed at the point of use by mechanical means.

- High expenditure on a monorail system may be justifiable if the system is required to accelerate raise development initially and then support material handling.
- The operating parameters of the monorail should be considered when blasting and supporting of excavations are being planned. For example, the operating height requirements of a monorail are in the order of 3,5 m when operating over material cars, while in-stope height requirements are approximately 2,5 m with operating widths of approximately 2 m.
- Further development is required with regard to the transfer of palletised material to the monorail, as currently there is no easy method of doing this.
- Further research is required to develop a lifting device that will allow a pack to be transferred from the monorail and installed on either side of the strike gully.
- A face material handling unit needs to be developed in order to fulfil the vision of a "hands-free" material handling system.

6.4.3 In-stope rock transportation

In current mining practices, it has become imperative to secure a substantial increase in both the rate of face advance and productivity, compelling the mining engineer to consider more productive ways of mining. Here, a way of improving the in-stope cleaning process is identified and proposed.

Current cleaning operations can support moderate rates of face advance, up to 1 m per day. Production rates above this will most likely require changes to the layout and improved cleaning equipment. To improve on the existing rates of advance of 6 m to 10 m per month, mine planners must take cognisance of the cleaning performance of the systems being used, taking into account stope width, panel length and shift duration. Furthermore, stope layouts should be designed for the complete removal of the ore from the panel face to the stope ore pass.

Over the past couple of decades, little progress has been made with stope cleaning systems. Some attempts have been made to improve gully rock handling by introducing continuous and automated scrapers. Despite intensive efforts, the only improvement in face rock handling has been the introduction of high-pressure water jets.

6.4.3.1 Cleaning constraints

The movement of rock in conventional stoping layouts consists of a number of batch processes (face to the strike gully, strike gully to the centre raise, and raise to the ore pass) that have to be synchronised, much like a manufacturing process line. The production output from the stope is dependent on the batch process that produces the smallest throughput. This is the stope's or system's constraint (Figure 6-3), which can be optimised by maximising the flow of ore through the constraint and appropriately scheduling activities that interlink with the constraint. In this case, the strike gully throughput is the constraint and the cleaning rate is required to match or exceed the tonnage generated by the blasted panel, based on the cleaning rate and the duration of the cleaning shift(s). This will ensure that the strike gully remains free of large accumulations of broken rock and is thus available for the movement of personnel and material

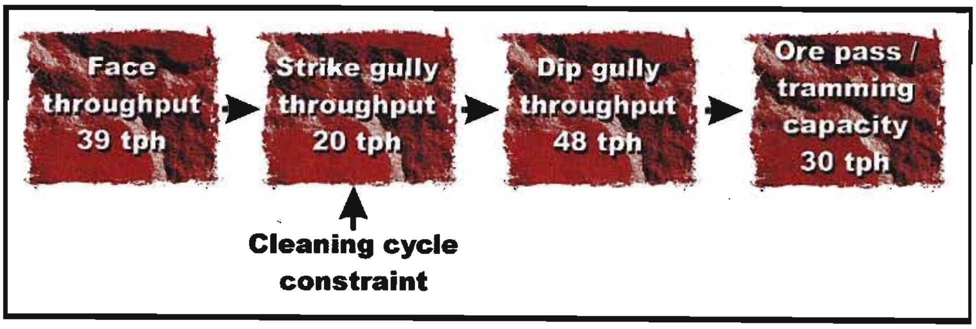


Figure 6-3: Cleaning cycle constraint
(NB. based on a face cleaning rate of 600 t-m/hr)

As a guideline, the mining engineer, production supervisor or mine planner should consider the following when planning a stope layout:

- Shift duration and travelling times.
- Activities of the rock cleaning shift (Figure 6-4).
- Appropriate cleaning rates for face, strike gully and dip gully.
- Panel lengths, which should be based on the smallest cleaning throughput.
- Cleaning cycle throughput, which must be based on the removal of all the ore from the face to the ore pass.

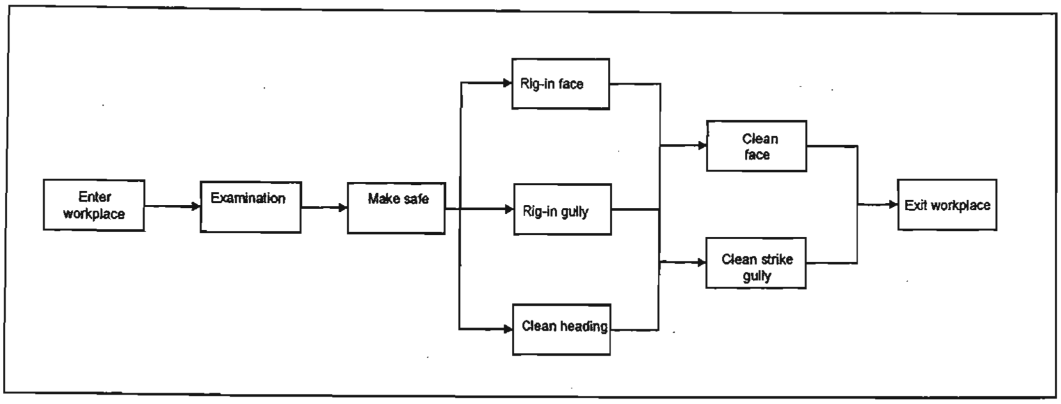


Figure 6-4: Typical activities for a rock moving shift
(Rupprecht and Williams, 2001(b))

The following cleaning rates are recommended (Figure 6-5):

- A face cleaning rate of 600 t-m/hr should be applied when good blasting practices and water jets are being used, the cleaning equipment is in good condition and there is a dip of 23° or greater.
- A cleaning rate of 400 t-m/hr should be applied for face cleaning when water jets are being used but one of the following factors is also present: poor face shape, fragmentation, severe geological disturbance, cleaning equipment in poor condition, or a dip of less than 23°. This cleaning rate can also be applied for panels not using water jets, but where the operating conditions are good, i.e. good fill factor, straight face, smooth footwall, etc.
- A cleaning rate of 200 t-m/hr should be applied for face cleaning not using water jets and where operating conditions are poor, i.e. poor face shape, fragmentation, severe geological disturbance, poor cleaning equipment or a dip of less than 23°.

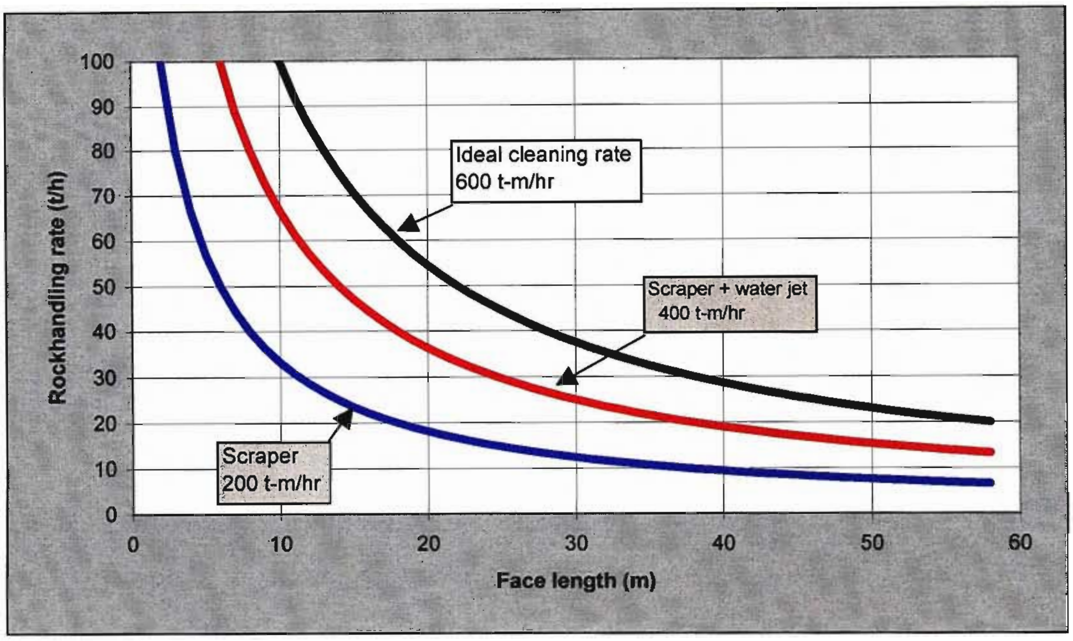


Figure 6-5: Various face cleaning rates (Morris *et al.*, 1988)

- Blasting techniques that improve hangingwall and footwall conditions, and that regulate stope face shapes must be employed.
- The above cleaning rates should be used as a guide for planning purposes, and should be modified in accordance with actual performances.
- The primary cleaning cycle should remove the bulk ore to minimise the extent of sweeping that needs to be conducted.
- The face should be straight and orientated in an underhanded position so that the face scoop can effectively bar (protect) it during cleaning operations.
- Scraper ropes should be rigged into the face and secured to the top of the permanent support prior to the blast.
- All necessary equipment, such as snatch blocks, pins, hammers, hoses and temporary support, should be placed in convenient positions near the stope face.
- Blast barricades should be used to confine the ore, thus increasing the face cleaning rate.
- Low-volume, high-pressure, non-dumping water jets are preferable for face cleaning, sweeping and vamping.
- Open water hoses should be used only for dust-allaying purposes.

6.4.3.2 Determining the strike gully cleaning rate

- A strike gully cleaning rate of 900 t-m/hr should be applied for a cleaning operation utilising a single 2–ton scoop, or a combination of pilot scoop and scraper (Figure 6-6).
- The length of the pull influences the cleaning rate of the strike gully scraper and thus the ultimate face length. Therefore, panel lengths should be designed in accordance with the capacity to remove the ore from the strike gully rather than from the face.
- The strike gully should not be utilised for ore storage, but rather be kept as empty as possible to facilitate the transportation of personnel and material.

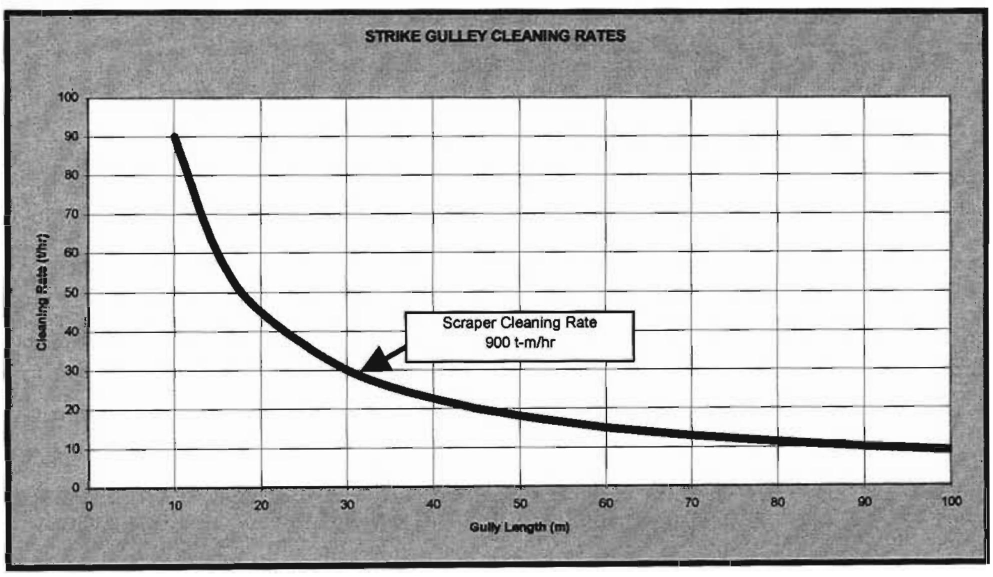


Figure 6-6: Strike gully cleaning rate (Morris *et al.*, 1988)

6.4.3.3 Determining the dip gully cleaning rate

- A down dip cleaning rate of 1 200 t-m/hr to 1 800 t-m/hr is achievable depending on the number of scrapers, fill factor, dip of reef, and size and condition of the scraper equipment (1 200 t-m/hr used for a 2-ton scraper, see Figure 6-7).
- An up dip cleaning rate of 600 t-m/hr to 1 200 t-m/hr is achievable depending on parameters similar to those listed for down dip cleaning (600 t-m/hr used for a single 2-ton scraper, see Figure 6-7).
- The continuous scraper produces cleaning rates in the order of 250 t-m/hr over a backlength of 320 m.

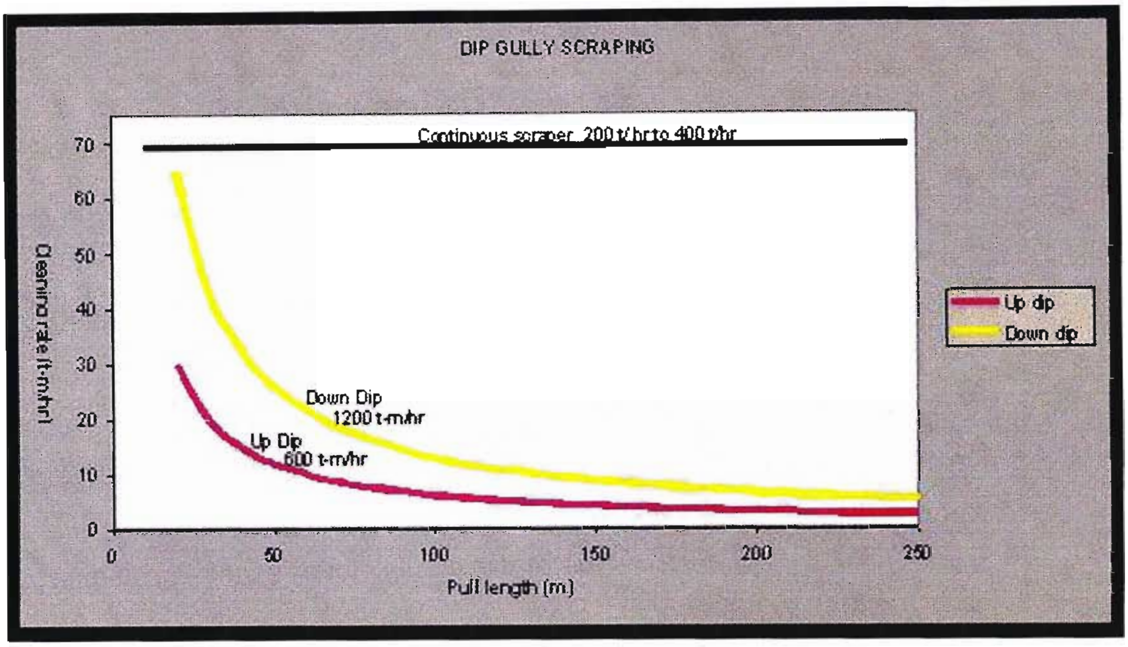


Figure 6-7: Dip gully cleaning rates

6.4.3.4 Sweeping and vamping

- The methods currently applied to perform sweepings and vampings are sufficient to achieve the desired results. Where possible, mechanised means of cleaning should be used to minimise the physical effort required.
- The cleaning cycle should aim at the complete removal of the broken ore to minimise the extent of sweeping needed.
- Sweepings and vampings should be conducted with non-dumping water jets.
- Vacuum systems should also be used to improve sweeping and vamping operations.
- Gullies should be blasted to the design widths to minimise the amount of ore locked up during stoping operations.
- Vamping of an ore pass should be limited to ensuring that the ore pass is empty at the end of the stope's life.
- Before the cross cut is vamped, a sampling programme should be conducted.

Traditionally, mine transport has been viewed as a series of independent activities. However, as already stated, this view is not conducive to an efficient transport system. An attitude of striving to move personnel, material and rock safely, quickly and efficiently must be inculcated and this requires a mindset change.

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