RAPID COAL PRODUCTION SYSTEM
FOR
LOW SEAM APPLICATIONS

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des akademischen Grades
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von
Dipl.-Ing. Manfred FUCHS
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Diese Dissertation entstand während meiner Tätigkeit als Business Development Manager in der Firma Voest Alpine Materials Handling GmbH & CoKG, innerhalb des

Sandvik Mining and Construction

Sektors der Firma Sandvik AB.

Für die Betreuung während dieser Arbeit und für wertvolle Ratschläge bin ich dem Vorstand des Lehrstuhles für Fördertechnik und Konstruktionslehre der Montanuniversität Leoben, meinem verehrten Lehrer, Herrn

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Ich erkläre an Eides statt, dass ich diese vorliegende Dissertation

Rapid Coal Production System for Low Seam Applications

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Rapid Coal Production System
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Contents

1. Introduction

Due to the increasing demand in thermal and coking coal in highly industrialized countries as well as the surfacing of numerous new markets and consumers in so called emerging markets, the coal mining industry is starting to face the fact of having difficulties to supply the quantity of coal needed.
Coal mining, material handling systems and also transport systems built 10 to 15 years ago are rapidly moving in direction their capacity limits and need to be up-graded or replaced to be able to handle the increasing demands.

Hard coal has been steadily increasing its share in the world energy mix over the last couple years. The majority of the growth goes onto the account of the expanding Asian markets and here in the consumption as well as in production of hard coal.
Europe will show the opposite side in a decline in the production (cutbacks in the uneconomic domestic production) and also in a reduction in the consumption of this primary energy source.
North, Central and South America are growth markets in both consumption and production terms. Especially the United States will increase their hard coal consumption significantly to offset the decreasing availability of domestic oil and gas reserves.
In large parts of Africa, for example the Republic of South Africa, the energy consumption will increase exponentially over the next couple years due to the set of a basic living standard for each resident and also for all immigrants to the Republic of South Africa. Most of the electric power generated in this country is produced through the use of hard coal. Also petrol is mainly produced from hard coal which also effects the coal consumption dramatically.
Basically the same tendency can be observed in the CIS.

Key forecasts predict an ongoing growth in coal production and world trade. In the steam coal sector, coal will increase its importance for the use in power plants continuously, whereas the use of coal in the heat market will continue to decline. Coking coal consumption will grow together in the same pace as the pig iron production, therefore the trade of coking coal will move forward due to the increasing demand in steel.

To be able to compare all different primary energy sources directly the so called ‘coal equivalent’ in short ‘ce’ was introduced.

COAL EQUIVALENT (ce)	
1kg gasoline	1.59kg coal equivalent
1kg fuel oil	1.52kg ce
1kg natural gas	1.35kg ce
1kg anthracite	1.14kg ce
1kg hard coal	**1.00kg ce**
1kg hard coal coke	0.97kg ce
1kg lignite briquette	0.72kg ce
1kg firewood	0.57kg ce
1kg fire peat	0.56kg ce
1kg crude lignite	0.34kg ce
1kg kWh	**0.123kg ce**

Table 1: **Coal Equivalent**;

The Coal Equivalent is a reference unit for the evaluation and comparison of the energy contents of various energy carriers. 1 kg coal equivalent corresponds to a value specified as 7,000 kilocalories (7,000kcal ~ 29.3MJ ~ 8.141kWh) which is approximately the calorific value of hard coal (depending on the type).

World Energy Mix:
In 2004 the world energy consumption was laying at around 15 billion tce (tons of coal equivalent), (also see table 2).
Hard coal consumption worldwide grew by approximately 750 million tce (+26%) from 2001 and 3.65 billion tons (2.9 billion tce) to records showing a consumption of already around 4.6 billion tons (3.65 billion tce) in 2004. Future hard coal consumption by the end of 2010 is forecasted to reach around 5.9 billion tons (4.68 billion tce).

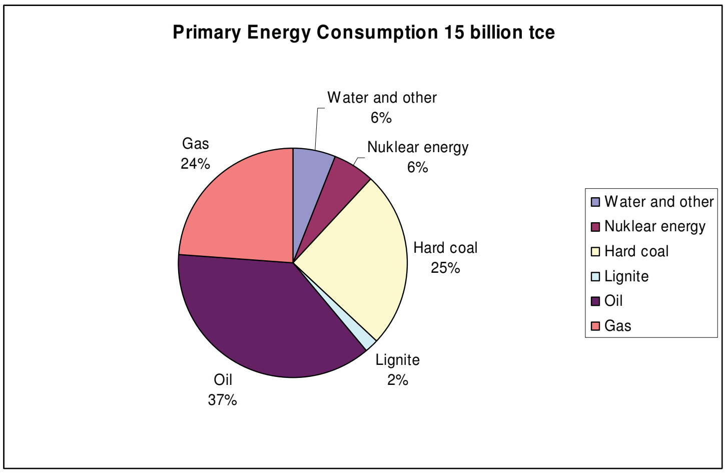


Table 2: **Primary Energy Consumption**; Source: BP Statistical Review of World Energy, June 2005; RWE World Energy Report 2005

At the present hard coal is accounting for around 25% of the global primary energy consumption.

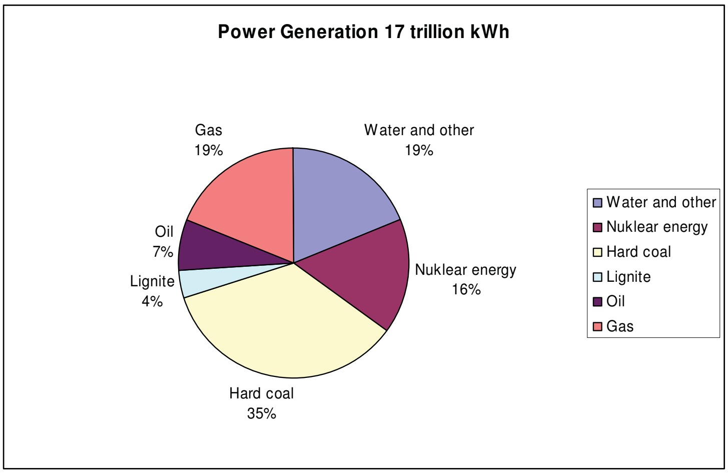


Table 3: **World Power Generation**; Source: BP Statistical Review of World Energy, June 2005; RWE World Energy Report 2005

About 70% of the total global coal production goes into power generation which covers about 35% of the total electricity requirement (see table 3).

The remaining 30% of the total global hard coal production are distributed in around equal shares to the steel industry as coking coal and the heat market which covers customers outside the electricity and steel sector in the likes of cement industry, paper mills, etc. as well as for domestic heating.

World Hard Coal Consumption

by sector

	1980		2004	
	bn t	%	bn t	%
Power Plants	1,00	36,00	3,35	73,00
Steel Industry	0,60	21,00	0,55	12,00
Heat Market	1,20	43,00	0,70	15,00
TOTAL	2,80		4,60	

Table 4: **World Hard Coal Consumption**; Source: Association of Coal Importers; RWE Power, World Market for Hard Coal, 2005 Edition

Even though this domestic fuel sector decreased from around 43% in 1980 to now around 15% of the world hard coal production, it is still a significant market in Eastern Europe, Turkey, China, North Korea, CIS, South Africa and such countries. In current times of high oil and gas prices this decline could actually been slowed down significantly (table 4).

Developments in World Energy Consumption by Energy Source (in bn tce)

	1980	1985	1990	1995	2000	2004	2010
Mineral Oil	4,35	4,05	4,48	4,71	5,13	5,46	5,70
Natural Gas	1,96	2,15	2,52	2,81	3,18	3,51	3,70
Nuclear Energy	0,24	0,50	0,74	0,76	0,85	0,90	0,91
Hydro Power	0,54	0,67	0,73	0,82	0,89	0,92	0,95
Hard Coal	2,50	2,85	2,12	2,90	2,79	3,70	**4,70**
Lignite	0,42	0,42	0,38	0,34	0,33	0,33	0,28
TOTAL	**10,01**	**10,64**	**10,97**	**12,34**	**13,17**	**14,82**	**16,24**
Share of Hard Coal [%]	29,20	30,70	27,40	26,30	23,70	27,20	**29,00**

Table 5: **Developments in World Energy Consumption**; Source: BP Statistical Review of World Energy; RWE Power, World Market for Hard Coal, 2005 Edition, own calculations

The hard coal market is predicted to grow with a rate of 1.3-1.5% per annum up to the year 2025 (see also table 5).

Reserves / Output / Consumption of Hard Coal

	Reserves 2005		Output 2004		Reach	Consumption 2004	
REGION	bn t	%	mill t	%	years	mill t	%
Europe	14	1,8	184	4,0	76	400	8,7
CIS	192	24,5	360	7,8	533	290	6,3
Africa	50	6,4	252	5,5	198	170	3,7
North America	253	32,2	962	20,9	262	1000	21,7
South America	16	2,0	67	1,5	238	50	1,1
PR China	96	12,2	1.956	42,5	48	1700	37,0
other Asia	97	12,4	549	11,9	177	910	19,7
Australia NZL	67	8,5	270	5,9	248	80	1,7
TOTAL	**785**		**4.600**		**170**	**4600**	
Total in bn tce	642		3.650		180	3650	

Table 6: **Reserves, Output, Consumption of Hard Coal;** Source: Federal Institute of Geosciences and Natural Resources, Hanover 2005; RWE Power, World Market for Hard Coal, 2005 Edition

On the production/supply side North America and China are currently the leaders accounting for more than 60% of world's hard coal production in 2004. Comparing the production figures with the actual consumption figures show again China and North America are using almost 60% of the world's hard coal production (table 6).

When developing all currently known hard coal recourses and presuming that the consumption stays at the value of 2004, the world wide hard coal resources would supply us with this primary energy source for the next 180 years (table 7).

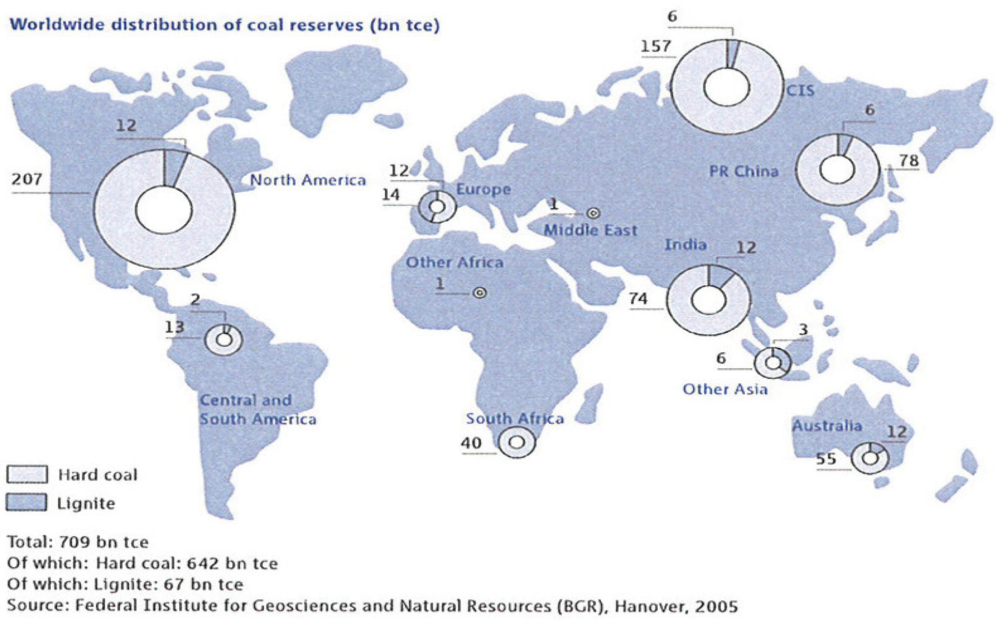


Table 7: **World Distribution of Coal Reserves;** Source: Federal Institute of Geosciences and Natural Resources, Hanover 2005; RWE Power, World Market for Hard Coal, 2005 Edition

The world wide trade of hard coal is nicely balanced over many markets which show a significant advantage over other primary energy sources in the likes of mineral oil and natural gas (table 8).

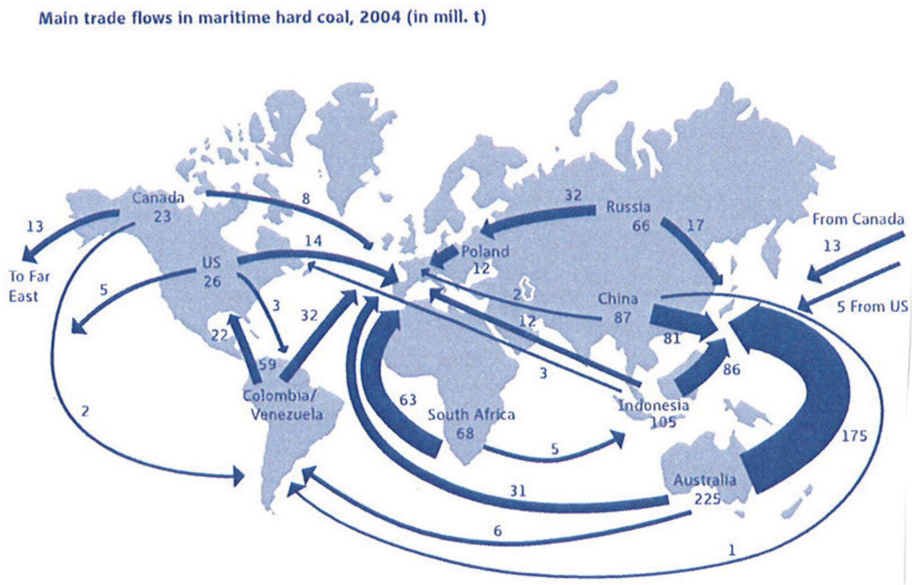


Table 8: **Main Trade Flows in Maritime Hard Coal;** Source: Associations of Coal Importers; RWE Power, World Market for Hard Coal, 2005 Edition

With the Kyoto Protocol signed and in force (other protocols of this kind will definitely follow) it remains unknown how the task to reduce the greenhouse gases, CO_2 in particular, will influence the coal industry around the world. New developments in direction of 'clean coal' with the aim to increase the efficiency of power plants as well as the capture and storage of CO_2 in underground cavities will be one way of meeting the requirements of the agreement.

Further expansion in the global hard coal trade will require mining companies to invest in replacement machinery and also in additional mining and transport capacities.
The international mining potential of hard coal is widely dispersed around the world (not like the oil and gas deposits) and is still in the position to meet the world's future energy and raw material requirements.

Coal markets are amongst the most robust markets nowadays. By 2030 it is projected that the global energy consumption has increase by about 70% as the emerging economies have developed their coal resources and have increased their pro-capita energy use to same level as seen in the leading industrial countries.

Such huge growth before us means an industry and also a workforce well prepared and trained for the future and the changing tasks and duties coming.

It is now a shared task between the mining companies to develop the new capacities and the machine and system suppliers to provide the mining and material handling equipment to produce the hard coal in an economical, efficient and reliable way.

The following mining system was thought about, developed and engineered by the author to set one step in the direction of those increased mining and transporting capacity requirements in the future.
The invented system is taking up the task of reaching a highly increased efficiency as well as a highly increased economical level in underground coal production.

2. Initiating Problem

Underground hard coal deposits like all other natural mineral deposits are not following any kind of regulation or order during the period they are getting formed.

Without going into too much detail in the actual formation of coal, only the essential basics and details important for the solution finding process of the problem are being discussed here in short.

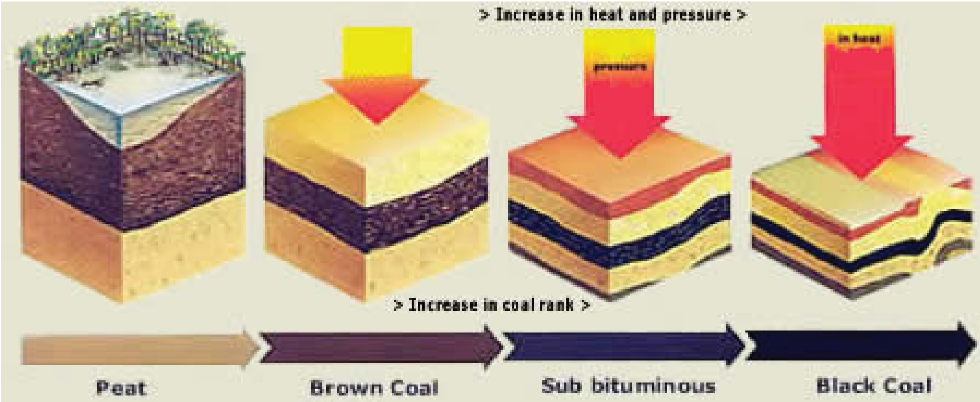
Coal formations are always starting with an accumulation of organic matter (trees or other plants) in low oxygen settings in the likes of a peat bog. The organic matter compacts and starts forming a bed of peat. The peat bed gets buried by sediments and under the influence of heat and pressure begins a chemical changing process, a metamorphosis.

Picture 1, Picture 2: **Early stages of the Coal Metamorphosis;** Source: Coal Formations; www.athro.com/geo/trp/gub/coal.html

Picture 1 and picture 2 above show early stages of such a metamorphosis.

Peat exposed to heat and pressure from burial beneath other sediments becomes compressed and also chemically changed into low grade coal such as lignite and under further heat and pressure is converted to higher grades of coal.

The pressure from the weight of the overlaying sediments that bury a peat bed will compact the coal. Peat transforms to low grade lignite when they are compressed to about 20% of their original thickness (see picture 3 and picture 4).



Picture 3: **The Coal metamorphosis**


Picture 4: **Stages in the Coal Metamorphosis**

Picture 3, Picture 4: Source: Australian Coal Association, www.australiancoal.com.au/origins.htm

Lignite typically transforms to bituminous coal as it is compressed even further and heated to between 100 and 200deg Celsius. This drives much of the water and other volatiles from the coal. Longer exposure to such surrounding conditions will further drive the volatiles from the coal and drive the chemical reaction to produce anthracite. This process takes more than ten million years.
Anthracite coals are very hard coals and are typically compressed to about 5 to 10% of the original thickness of the peat bed and contain usually less than 10% water and volatiles.

What is influencing the 3 dimensional expansion of the resulting coal seam?

- The original peat bed is usually varying in thickness and composition, also the pockets where the organic matter accumulates is never of the same symmetrical shape and this explains the variations of the resulting coal seam thicknesses we experience everywhere.
- The covering sediments compressing the organic matter with their weight are also varying in thickness and composition which again influences the compression ratio and again the resulting seam thicknesses.

These two above mentioned influences are therefore responsible for the variation of the coal thickness over the whole length of the seam. In an extreme way such a coal seam can have vertical dimensions varying from more than 10m down to only a couple of centimeters. The actual thickness of the covering layer is also determining the selected mining method. In case the top layer is shallow enough the coal deposit can be accessed from the top by removing the covering sediments. In case the top layer gets to thick an underground mining method has to be used.

- Geological activities and movements in the likes of the building of mountainous formations, the dynamics of the geological plates, faults, cracks as well as earth quakes are affecting the coal seams in the vertical development as well as all other layer the coal seam is sandwiched in.
- The horizontal dimensions are obviously depending on the initial size of the peat bed with a starting and finishing area wherein the coal seam is shallowing out to zero.

Vertical and horizontal dimensions of the resulting coal seam, the undulation of the coal deposit as well as the quality, the subsequent market price of the coal and the location where the coal mine would be established is subject to intense investigations to determine whether a coal deposit is economical to mine or not.

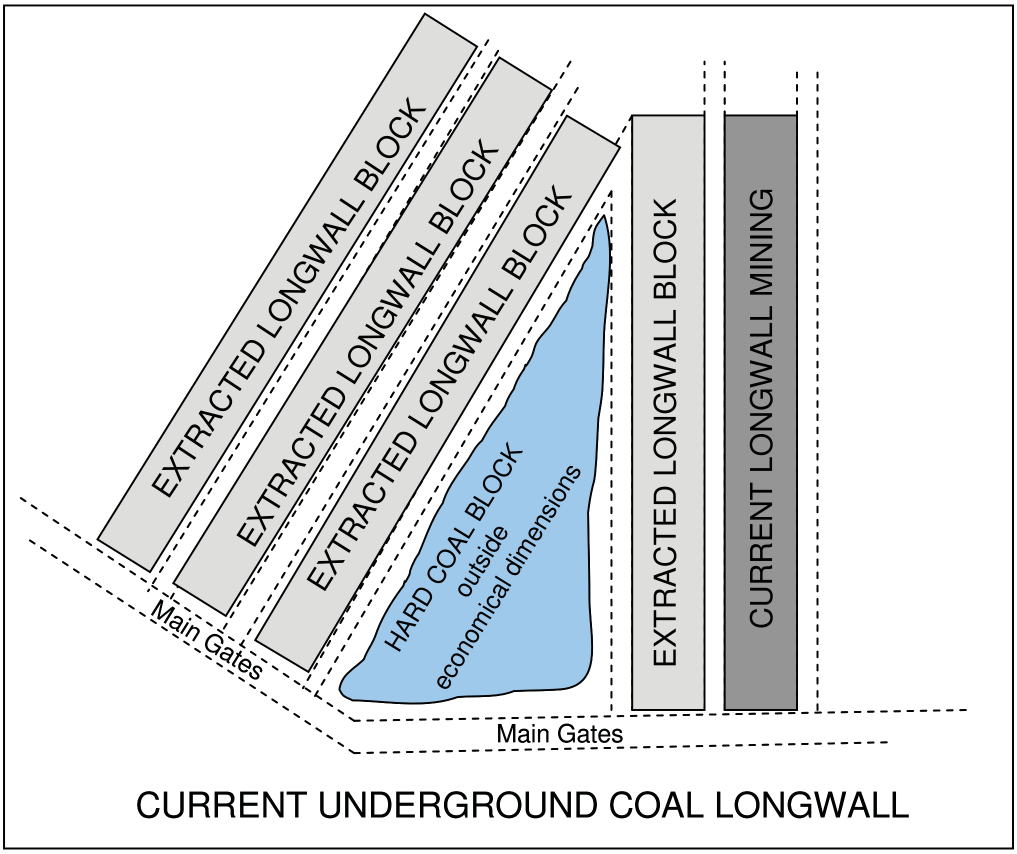
Assuming all the technical and financial investigation had been done in a proper and correct way and a coal mine has been established and running in a financially healthy way, there will however be a line which has to be drawn to where the mined coal seam will not be economical to mine anymore.
This could be because of too extreme undulation in the coal seam, due to the coal quality fading away to lower grade coal types, due to increasing ash contents or due to not manageable gas or water ingress - the list is very long.
However, there is also the possibility of the coal quality is extremely good, the undulation of the seam is within the limits, all other aspects influencing a commercially viable coal mining process are also fine only the seam thickness is shallowing out beyond the initially commercially set limits.

The following picture shows such an underground coal seam where there was an area discovered right in the middle of the deposit where the coal is shallowing out to a thickness of around 1200mm. The actual surrounding coal deposit has got a thickness between 3500 and 2500mm and the horizontal dimensions as well as the thickness of the covering layers have justified the use of an underground longwall operation.

This longwall operation has been successfully in operation for many years.

The blue colored section in picture 4 indicates the shallow hard coal seam area which was left behind and the normal longwall operation was moved over to the area where the seam thickness continued within the economical dimensions.

The main reason for this left over coal deposit, as indicated before, is to be found in the commercial sector. Technically the mining industry provides solution for such low coal seams but for this specific operation the cost factor was moving such an operation outside economical limits.



Picture 4 **Schematic of an Underground Coal Mining Operation**

In other words we can say that under normal mining conditions this blue colored shallow hard coal section would remain untouched!

The development of an economic mining method and the necessary machines to extract this low seam hard coal deposit was the main target and the aim of the forthcoming investigation, research and development done by the author. The method as well as the machines to mine the coal seam will have to be flexible enough to be installed and removed in a quick way and to have the possibility to be utilized in as many as possible different underground conditions to justify the investment and to make such small coal seams commercially viable.

3. Basic system lay-out requirements

3.1. Current coal mining methods

To be able to determine the best and most effective way of extracting the left over hard coal deposit, an investigation and analysis of current underground coal mining methods was performed.
As previously discussed the determining factor, whether the coal extraction will be done by removing the overburden material to access the deposit from the top or by entering the coal seam via underground tunnels, is the actual thickness of the coal covering sediments. This decision is mainly driven by the cost factors.

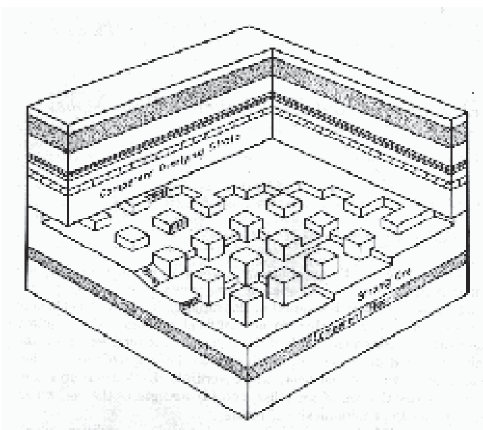
Based on the depth of the mine and the minimum thickness of the cover layer over 300m only the underground mining methods will be addressed in the following investigations.

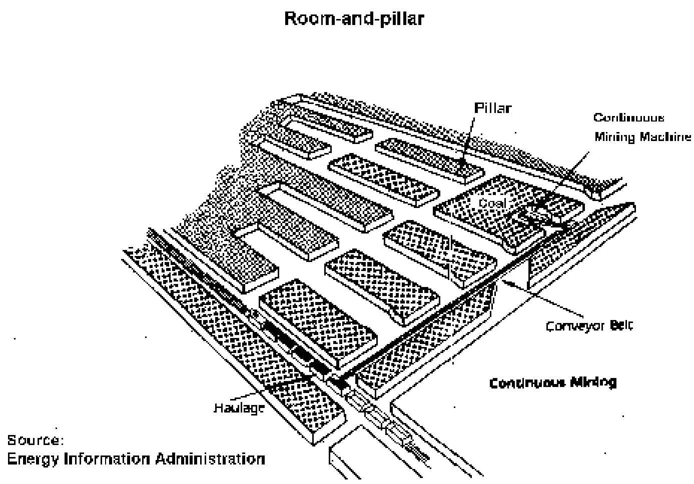
The two main underground coal extraction methods are:

- Room and Pillar Mining
- Longwall Mining

<u>Room and Pillar Mining:</u>

About 60 to 70% of all underground mined coal is extracted with the Room and Pillar Mining method (see picture 5 and picture 6).





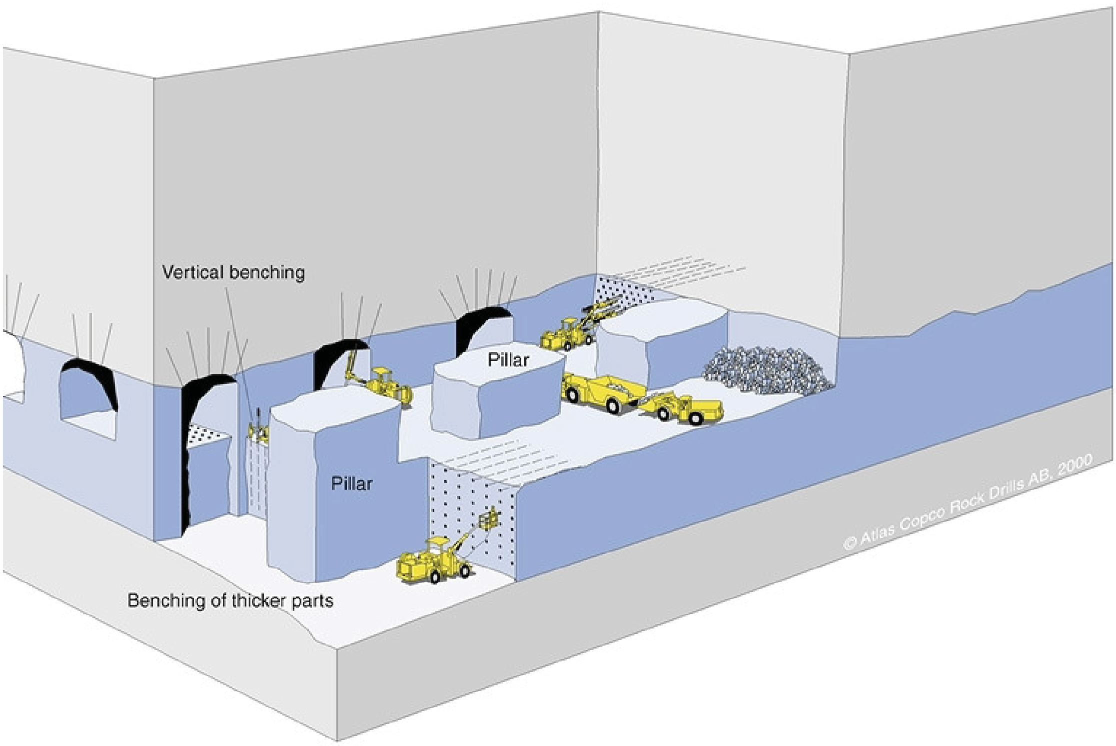
Picture 5 **Room and Pillar Mining** Picture 6 **Room and Pillar Mining**

Source: Energy Information Administration, Underground Mining, room and pillar mining, www.umwa.org/mining/ugmine.shtml

Rooms are getting cut into the coal bed leaving a series of pillars or columns of coal which have the task of supporting the mine roof and also to channel and guide the air flow through the mine. Generally, the rooms are between 6 and 10m wide whereas the pillars are normally in the range of 25 to 35m wide.
As the mining advances the mining action leaves a grid-like pattern formed by the rooms and pillars.
Additional roof support and also in some occasions rib support is required to prevent the cavities to fall in. This additional support is done with roof and rib anchors set right after the mining process.
After the mining has reached the end of the coal seam the next following mining sequence could be the retreat mining. This mining method is not always followed as the occupational health risk as well as the risk to damage the mining equipment is high. When retreat mining is chosen, the workers start cutting out the pillars in a controlled manner until the roof falls in. After the retreat mining is finished the mining area will be abandoned.
The normal room and pillar mining will reach an extraction rate of about 65% which can be increased significantly with the retreat mining method.

The room and pillar mining method is performed in two types, the **conventional room and pillar mining** and the **continuous room and pillar mining**.

The conventional mining method is the oldest form wherein the coal seam is drilled, blasted and then loaded into cars for transport (see picture 7).



Picture 7: **Conventional Room and Pillar Mining**; Source: www-1.atlascopco.com/Websites/RDE/website.nsf

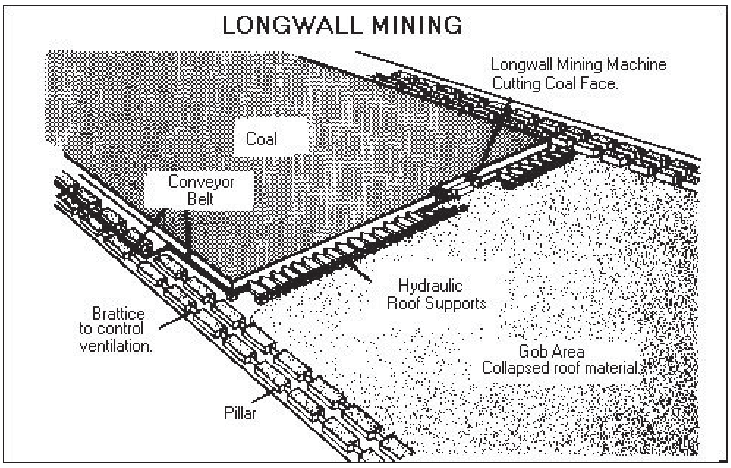
When the continuous mining method is used the coal seam is being mechanically cut with a drum type mining machine. This mining machine (see picture 8) is also picking up the material and is loading the transport cars behind for further transport. The actual coal cutting action is only interrupted by the necessary strata control measurements in the likes of roof and rib bolting (setting of roof and rib anchors) to prevent the mine roof and walls from falling in.

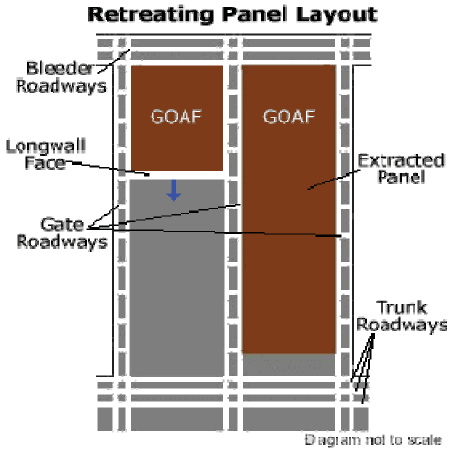
Picture 8: **Continuous Mining Machine ABM14**

Room and Pillar mining has got the advantage of low capital cost and is very flexible in regards to relocations or changing conditions of underground seams. This becomes very important when facing extremely undulating coal seams or when the seams vary in thickness over very short distances or the coal seam is disturbed by faults and selecting mining is necessary to avoid rocks. The actual production figures are much lower than for the following longwall mining method and rank around the 1,0Mio ton figure per year (depending on the seam height).

Longwall Mining:

Longwall mining accounts for around 30% for all underground coal production worldwide.





Picture 9: **Longwall Mining** Picture 10: **Longwall Mining Schematic**

Picture 9: Source: Underground Mining, longwall mining, www.umwa.org/mining/lwmine.shtml
Picture 10: Source History and Methods of Longwall Mining, www.uow.edu.au/eng/longwall/history.html

In longwall mining, large blocks of coal are defined during the development stage of the mine and are then extracted in a single continuous operation. Generally each block of coal is about 200 to 250m wide and up to 2000m long depending on the mine lease dimensions or the actual coal seam itself (see picture 9 and picture 10).
Those coal blocks, also called panels, are created by driving a set of headings from the main roadways in the mine to a certain distance into the panel. The roadways, as said before, generally 200 to 300 apart, are getting joined at the end to form the starting face of the longwall operation.
The longwall face is supported by hydraulic roof supports which provide a safe working area for the workers and the machine (see picture 11).

Picture 11: **Longwall Shearer / Hydraulic Supports;** Source History and Methods of Longwall Mining, www.uow.edu.au/eng/longwall/history.html

The machine which is moving forwards and backwards along the coal face taking out a slice of coal every time it passes is either a ‘shearer’ or a ‘plough’. The coal is falling onto a chain conveyor and transported out of the panel. The coal cutting machine and the hydraulic roof supports are advancing automatically behind the cut coal face. The mine roof (goaf) behind the hydraulic roof supports will eventually collapse.

Longwall equipment is extremely high in capital cost and also very inflexible when installed and running. A system like this cannot be used in coal seams with rapidly and often changing thicknesses and neither in undulating coal seams. The consequence would be the cutting of rock as well coal and the therefore higher washing costs.
Longwalls, however, will get the production figures up to 8,0Mio tons per year when used in the right conditions (depending on seam height).

As long as the coal block is within certain dimensions it is economical to mine a hard coal deposit with the longwall mining method. The dimensions become commercially critical when falling short of approximately 100m in width, 500m of length. This also depends on the quality of the hard coal and the market price when sold.
The height is also critical but not as such. Here it is more important to find the right longwall equipment for the average height of the coal seam altogether. This is due to the fact that the cutting height of a longwall system is almost fixed and a change would result in extremely high investment costs.

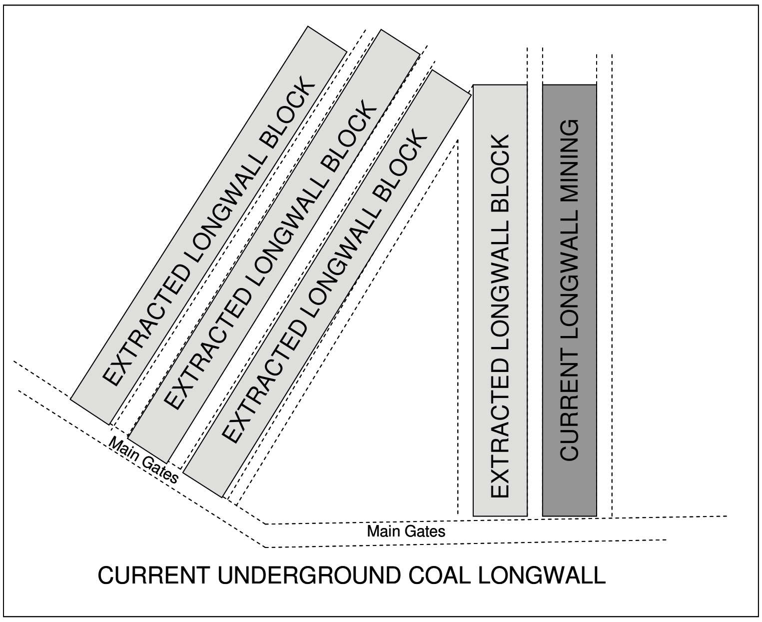
Intensive commercial investigations are necessary to determine whether a longwall mining operation justifies the effort of installing and running it. Whilst the length and the width of the block (extracted coal volume) are only balanced to the actual costs of installing, running, taking out of the longwall system, the actual height faces the physical limits of such a machine.

Due to the fact that a longwall shearer including the chain conveyor and the hydraulic roof supports is designed for a certain extraction height (approx. 0,6 up to 6,0m) a coal seam lower as it was designed for means, that the longwall system is going to cut not only coal but also the covering sediments. This will reflect in most likely higher running costs (wear and tear) as well as higher cost in the subsequent coal washing where the stones have to be separated from the coal again.
In case the cutting height is too low, precious hard coal will be left in the seam and the extraction rate will be lower than actually possible.

Extraction rates with proper designed longwalls can be above 90%.

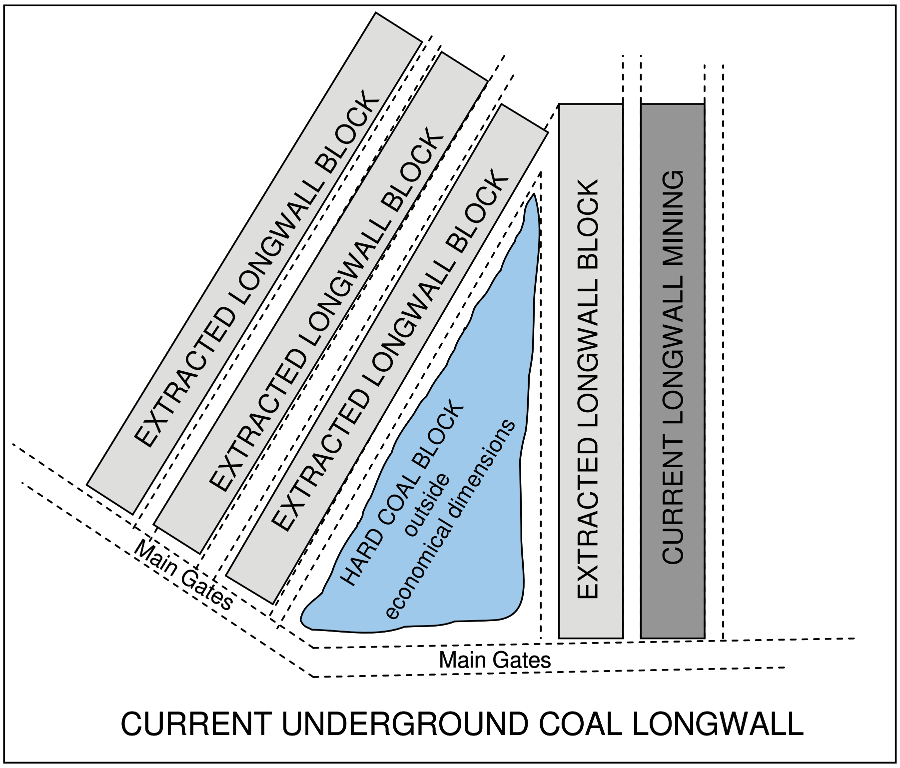
A longwall mine as described above is serving the purpose to demonstrate the normal longwall operation as well as the limits to such a mining method.
The whole hard coal bed is very steady around the 3,5m mark and therefore the longwall system installed is tailor made for the conditions.
The length (around 2500m) and width (around 250m) of the individual longwall blocks was possible to be accommodated in the lease in a commercially variable way.

The arrangement of the longwall blocks is sketched up below in picture 12.



Picture 12: **Schematic of the Underground Longwall Mine with Longwall Mining Blocks**

The sketch does not show the whole underground mine. It only shows the area around a coal block which had to be left over due to a significant, locally confined shallowing out of the coal seam. The left over hard coal block in the shape of a triangle (picture 13) is impossible to be cut with the longwall system available.



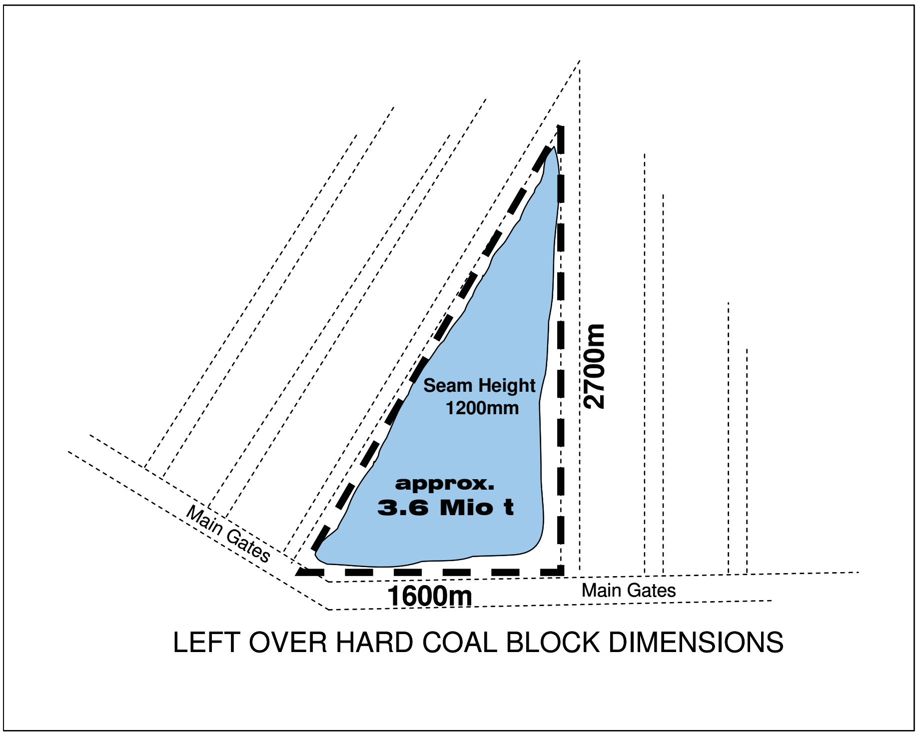
Picture 13: **Left over Hard Coal Block**

The size of the left over block (roughly around 4,2 Mio t of hard coal) also does not justify an investment of a new longwall nor changes to the existing one.

3.2. Mining system requirements

The left over coal seam is covering a hard coal volume of above 3,6 Mio tons of 'clear' coal (picture 14). 'Clear' coal in these regards means coal which does not have to be washed or does not have to be run through a beneficiation plant at all.
The coal when mined within the below shown dimensions is only supposed to be cut, sized and can be sold straight away without any refinement processes.

Avoiding the beneficiation costs is besides other issues critical for this low height deposit and this could be the determining factor between mining the deposit and leaving it behind untouched.



Picture 14 **Dimensions of the Left over Hard Coal Block**

Following rough mining system requirements can be established as a guide line for the following research.

To provide overall engineering and design criteria, a successful mining system must have:

- Operating height below 1200mm

- High efficiency
- High flexibility (height wise and for relocation purposes)
- Certified to underground mine regulations
- Low investment costs
- Low running costs
- Minimum amount of operators necessary

3.3. State of current technology

To be able to select a suitable mining system it was necessary to look into the market offerings and determine what machines and systems are available which actually work to our benefit.

What mining machines are available in underground coal mining suitable for our case?

Longwall Shearer and Longwall Plough

As described before the longwall mining equipment is one of the most effective and efficient coal extraction systems available.
The roof is held up with hydraulic roof supports protecting the actual coal shearer or coal plough from falling in rock.
The longwall extraction system combines the roof supports (picture 15), the chain conveyor and the shearer (picture 17) or plough (picture 16). The whole system advances automatically, the roof is collapsing behind the roof supports.

Picture 15: **Longwall Roof Supports;** Source: www.dbt.de/media/bildmaterial/produkte-serviceleistungen/strebbau/hobelsystem/bild4agr

The cut out coal falls onto the cross chain conveyor which again dumps the coal onto a belt conveyor for the transport out of the panel.

Picture 16: **Longwall Plough with Chain Conveyor** Picture 17: **Longwall Shearer with Chain Conveyor**

Picture 16: Source: www.dbt.de/media/bildmaterial/produkte-serviceleistungen/strebbau/schildausbau/schildausbau.html
Picture 17: Source: www.dbt.de/media/bildmaterial/produkte-serviceleistungen/strebbau/walzenlader/walzenlader.html

Advantages:

- Very high production figures (up to 8 Mio tons per annum)
- Almost continuous coal extraction
- Very efficient and reliable
- Only needs the temporary roof supports to protect working area
- Relatively simple design
- Runs fully automated
- Low running costs

Disadvantages

- Very high capital cost (up to 15 times the costs of a Continuous Bolter Miner)
- When installed very inflexible
- Coal block has to be developed before the longwall gets installed
- Only effective in close to even seams
- Cannot be used in undulating seams
- Only cuts the height it is designed for
- High effort to install or relocate

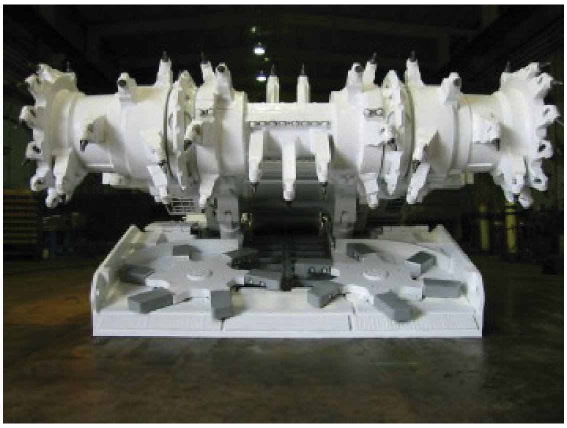
Continuous Miner

Continuous Miners (picture 18, 19, 20) are mainly used in the room and pillar mining method when talking about coal production. The miner cuts continuously coal out of the face.
To be able to clear the cut material out from the cutting area, the continuous miner is equipped with a loading table below the coal cutting drum where the coal is picked up by loading arms and thrown onto a chain conveyor.

This chain conveyor runs through the machine to the back of the machine and dumps the coal either onto the floor or onto a hauling machine (truck, shuttle car, etc).
The miners are powered via electrical power cables supplied from underground load centers.
The machines are running on tracks and are therefore very flexible during relocation or advancing processes.

Due to the fact that the miner is cutting out the coal block and leaves an unsecured cavity behind, the distance the Continuous Miner is allowed to advance into the coal block is restricted according to the underground strata control rules of the mine.
In a normal case we are talking about distances between 10 to 20m. The machine has to be pulled out after this distance and permanent ground support has to be installed before entering the mining face again. This is done by special bolting rigs which are running independently to the miner.
The Miner is relocating to another mining face at that time and is cutting/mining the coal there for a distance of 10 to 20m before returning to the previous mining face to proceed there again.
This mining method is only effective when operating at least two cutting faces at the same time.

This mining action is intermittent and subject to many relocation movements and therefore time without mining coal.



Picture 18: **Cutter Drum / Loading Table;** Source: www.dbt.de

Picture 19: **Drum Miner;** Source: www.dbt.de

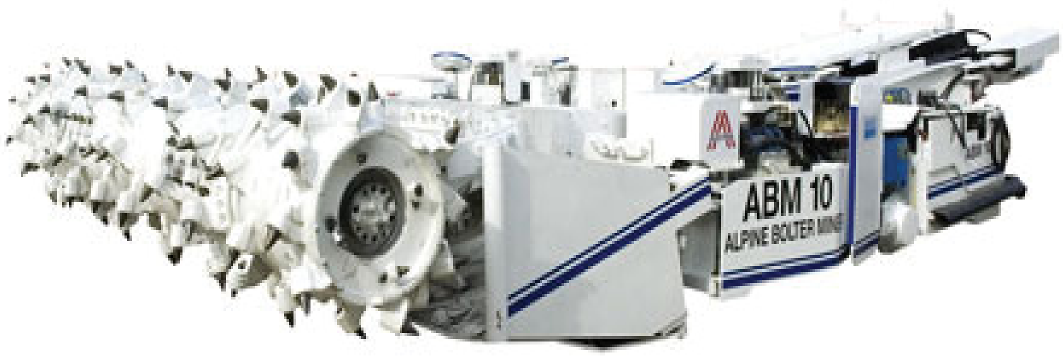
Picture 20: **Continuous Drum Miner;** Source: www.joy.com

The mentioned advantages and a solution for the disadvantages of the Continuous Miner are getting combined in the Continuous Bolter Miner.

Continuous Bolter Miner

The Continuous Bolter Miners (picture 21) have one main advantage over the standard Continuous Miner. They are able to cut the coal and install the ground control measurements at the same time.
The Continuous Bolter Miner does not have to be pulled out of areas due to the need of installing ground support and therefore gains valuable time for cutting. The cutting/mining operation could, assuming there are no ventilation restrictions, advance continuously in the real sense for a more or less unlimited distance.

Continuous Bolter Miners are being used for the development of the coal blocks for the longwall or in the above mentioned board and pillars mining method.



Picture 21: **Continuous Bolter Miner ABM10**

Advantages:

- Very flexible
- Easy and quick relocation process
- Onboard roof and rib drill rigs for strata control (Continuous Bolter Miner)
- Can be used for selective mining when following the coal seam
- Much lower capital costs compared to longwall

Disadvantages

- Much lower production rates than a longwall (up to 1Mio tons per annum)

- Needs strata control measurements to operate
- Operators under unsecured roof (Continuous Miner)
- Difficult to automate
- Need a very flexible material clearing system behind the machine
- Needs at least 3 people to operate (1 off driver, 2 off drill rig operators)
- Needs additional people to operate the material handling equipment
- Many relocation processes and therefore roadway damage (Continuous Miner)

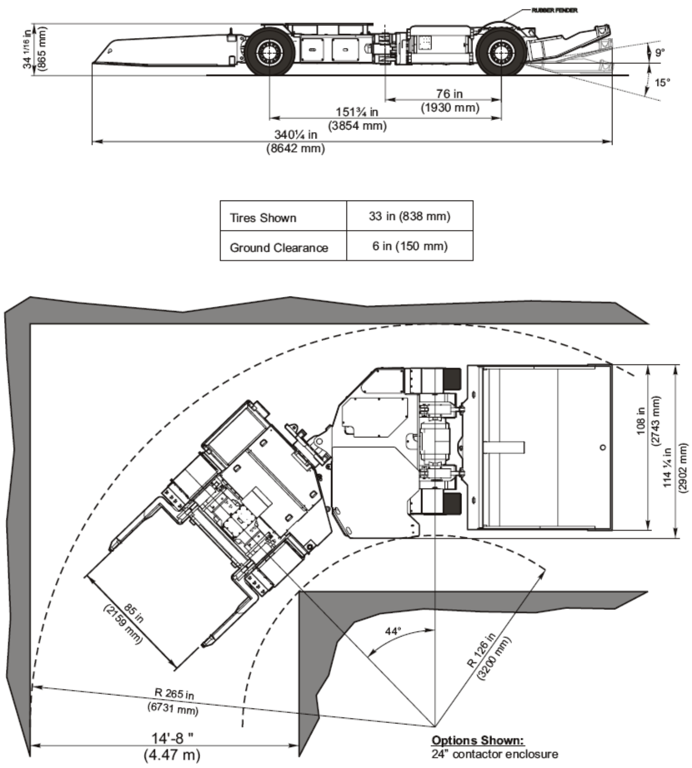
LHD (Load-Haul-Dump) Rubber Tired Vehicles

In case the coal is dumped onto the floor the behind the cutting machine running hauling machine has to pick up the coal, transport (haul) the coal to an area where it is getting handled and transported out of the panel.
Those machines are called LHDs for ‘Loading’, ‘Hauling’ and ‘Dumping’ the material cut by the mining machine (pictures 22, 23).

Advantages:
- Low capital cost
- Usually more than one as spare unit in case of outages
- Very flexible in where they are getting operated
- Diesel or Battery powered therefore independent to power supplies
- Can serve more than one dumping area
- Independent from hauling distance
- Coal lump sizes do not effect operation
- Simple design

Disadvantages:
- Intermittent material transport
- Long hauling distances cause long waiting times
- Limited capacity (up to 8 tons in one load)
- Needs one operator
- Require high road maintenance
- Difficult to automate

Picture 22: **LHD**



Picture 23: **LHD Dimensional Schematics**

Shuttle Cars

Shuttle cars (picture 24) are similar to LHD with but without the ability to pick up the coal by itself. This means that the mining machine has to load the shuttle car. This also means that the coal cutting operation has to stop when there is no shuttle car behind the miner which increases in certain situations the down time of the mining operation due to waiting.

Picture 24: **Shuttle Car**

Advantages:

- Low capital costs
- Spare units normally available
- Higher capacity than LHDs (up to 16tons in one load)
- Simple design
- Flexible in regards to road turns

Disadvantages:

- Powered via Power Cable
- Limited length of power cable
- Limited capacity
- Needs one operator
- Difficult to automate
- High roadway maintenance necessary

<u>Conveyor Systems</u>

Conveyor systems (picture 25, 26) are the conventional material handling system within production panels. They are normally found in any underground coal mining application where longer distances and high capacities have to be managed.

Picture 25: **Underground Belt Conveyor**

Picture 26: **Underground Drift Belt Conveyor**

Advantages:

- Very high production figures (over 4000 tons per hour possible)
- High efficient
- Low capital cost
- Low running cost
- Simple design
- Very reliable

- Low dust, noise and spillage
- Safe operation
- Fully automated
- Insensitive to roadway condition

Disadvantages:

- Have to be loaded by other means (LHD, Shuttle Car, mining machine, etc.)
- Sensitive to over size lumps
- Inflexible when installed
- Fixed length or limited in extension length (drive power)
- Difficult to relocate

To analyse the research on currently available mining machines for underground coal mining we have to summarize:

To extract the coal out of our hard coal area it is not only necessary to look at the technical aspects. The actual selection of a suitable mining system is mainly depending on the commercial outcome of the mining process.

None of the currently available underground mining systems on the market will serve to mine the hard coal deposit in our application in an economical way.

Economically means that there would be a mining systems available on the market *but* the capital investment to purchase such a machine, in this specific case a longwall shearer or plough, are not in any relationship to the rather small hard coal deposit in which it has to be used.

Economically means that the running costs of the mining system have to be as low as possible. Economical also means the use of as little as possible supporting mining material and low relocation / set-up cost. All unnecessary mining equipment, personnel and supporting machines shall be avoided. Also the mining system needs to have the flexibility to be used elsewhere.

Economically means further more the flexibility of a mining system to be able to follow the coal layer in a way that the cut material is only consisting of coal and the sediments are left behind. Impurities / ash content in form of sediment rock have to be avoided by all means to avoid the washing of the coal before selling it.

3.4. Cost / performance aspects

As mentioned before, the cost factor is the determining factor in our application. Only if the owner of the mine is making profit with the sold coal the mining process is justified. The Total-Cost-of-Ownership will be determining what system can be used.

The next step was to have a good look at the current hard coal price achievable on the market which will be the starting point for the investigations.

The hard coal deposit we are having on hand for our investigations is located in the Pittsburgh (Pennsylvania, USA) underground seam.
This hard coal type is located in the category ‘thermal and coking coal’, is used in power plants to generate electrical power and also certain percentages of the coal deposit are used to make coke.
In the table 9 the price development of this type of hard coal on the spot market is highlighted.

The type of hard coal is specified as Northern Appalachia (NAP) and is, in respect to the BTU index, one of the highest quality hard coals available.

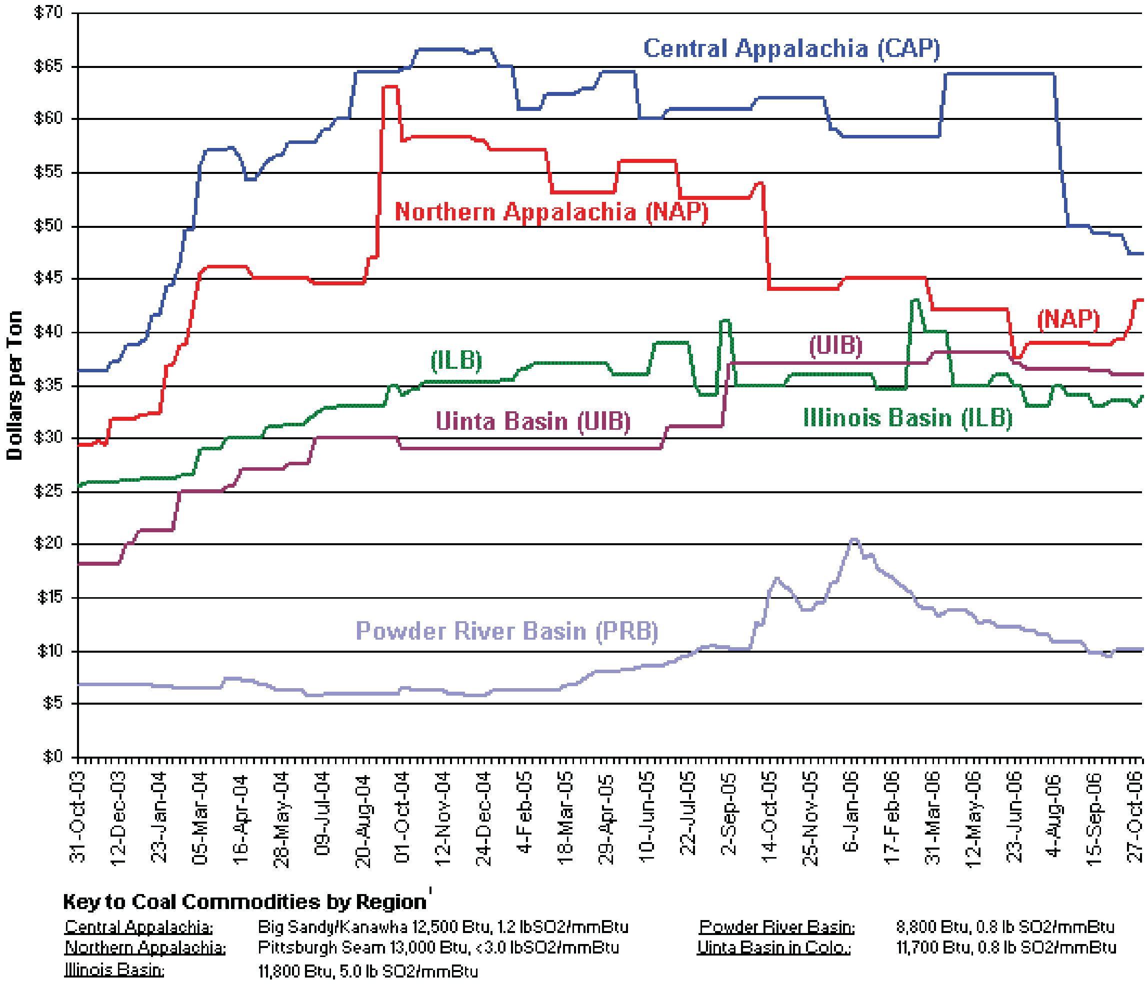


Table 9: **Coal Commodity Prices;** Source: Energy Information Administration, www.eia.doe.gov/cneaf/coal/page/coalnews/coalmar.html

Definition: A **British Thermal Unit (BTU)** is the amount of heat energy needed to raise the temperature of one pound of water by one degree F. This is the standard measurement used to state the amount of energy that a fuel has as well as the amount of output of any heat generating device. All combustible materials have a BTU rating. For instance, propane has about 15,000 BTUs per pound. Charcoal has about 9,000 BTUs per pound and wood (dry) has about 7,000 BTUs per pound (defined by the Energy Information Administration).

The import price into Europe of this type of hard coal has risen sharply since the fourth quarter of 2003. The price rise is less steep in the EMU countries due to the strong Euro. The price rise is due in part to the sharp increase in the use of coal in Asia, in particular in China. The price has now stabilized.

Coal import prices per ton (table 10)

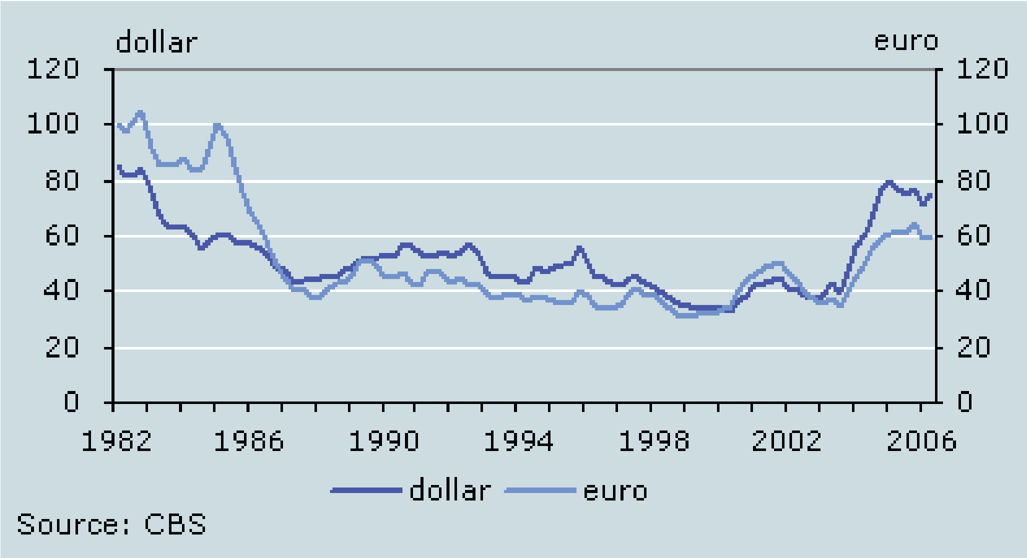


Table 10: **Coal Import Prices into Europe;** Source: CBS-Coal price stable and high, www.cbs.nl/en-GB/menu/themas/bedrijfsleven/energy-water/puplicaties/artikelen/2006-2050-wm.htm

On average, in the first half of 2006 the hard coal price per metric ton (including freight and insurance) on the Rotterdam spot market, this means imported into Europe, reached approximately USD 61,00.

The sea freight rates (table 11) are still very comparable to a multi year average and amount to USD 13 – 15,00 per metric ton for standard overseas transport.
Following there is an indication of the price development of freight charges per ton of coal.

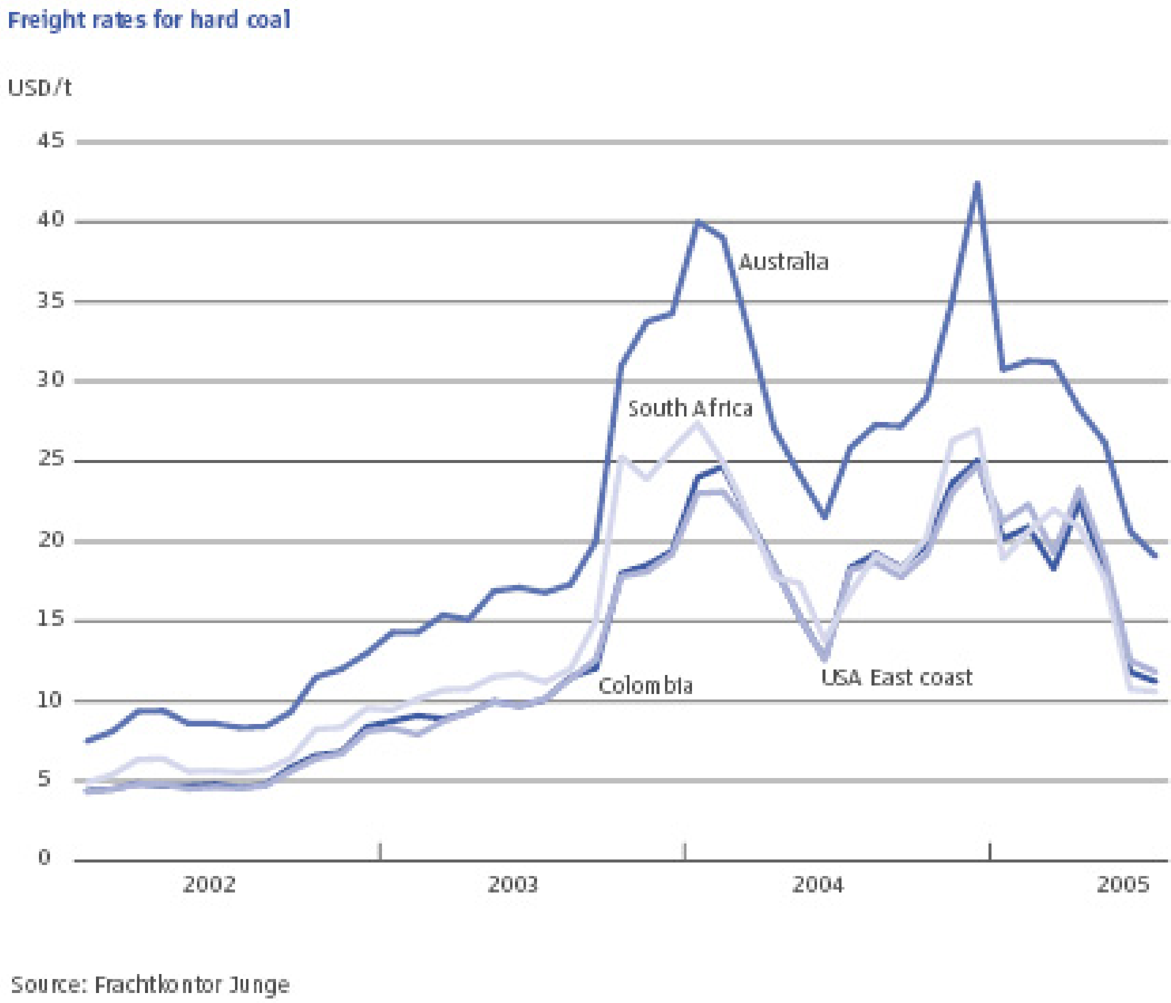


Table 11: **Freight Rates for Hard Coal;** Source: Frachtkontor Junge; World Market for Hard Coal, 2005 Edition, RWE Power

The BAFA (Bundesamt für Wirtschaft und Ausfuhrkontrolle) prices which are relevant for long-term purchasing agreements in the electric generation sector for German hard coal are around the Euro 63,00 per metric ton of 'hard coal equivalent' figure in the first half of 2006.
A similar figure is expected for the second half of 2006 (see table 12).

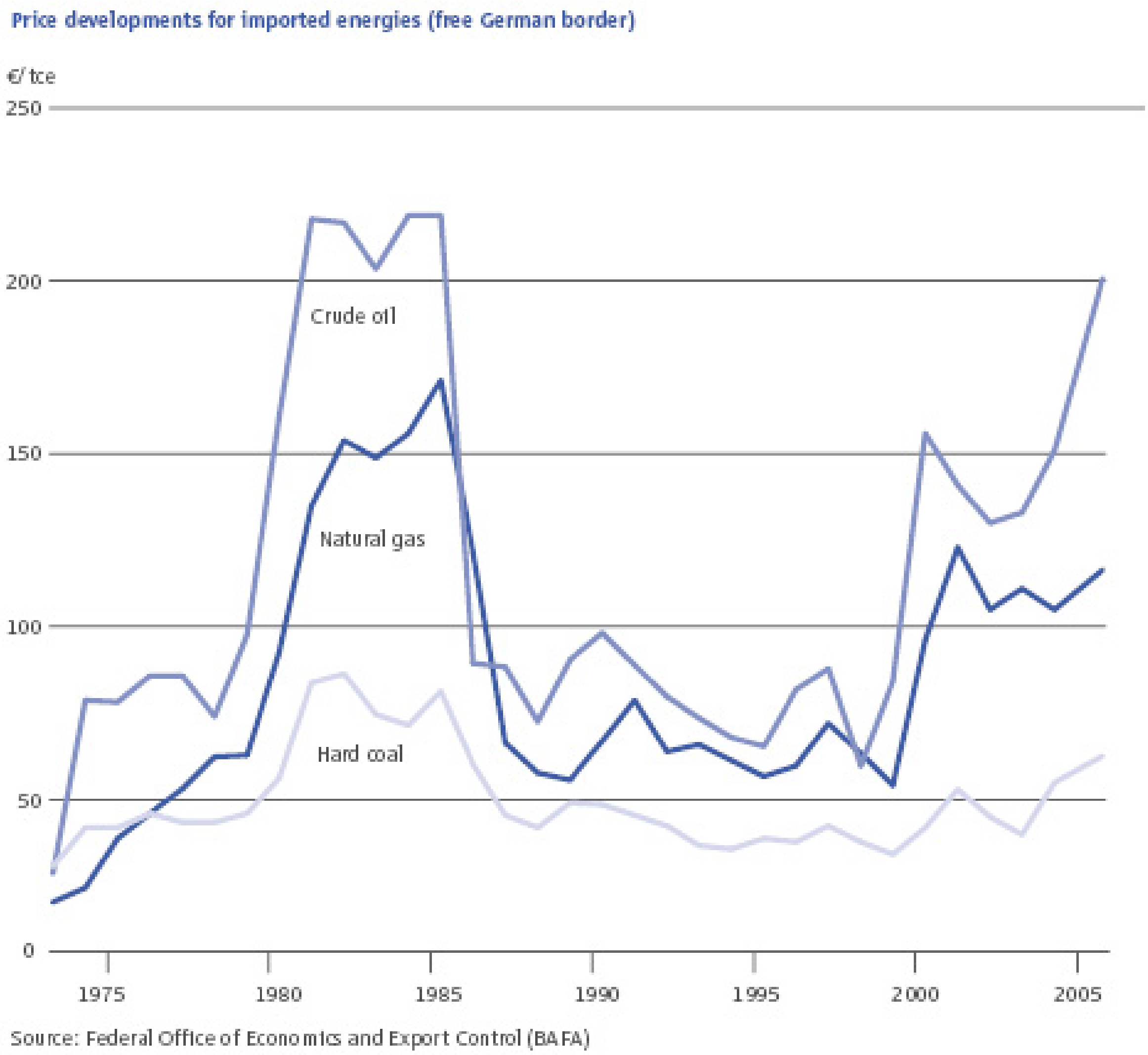


Table 12: **Price Developments for Imported Energies;** Source: Federal Office of Economics and Export Control (BAFA); World Market for Hard Coal, 2005 Edition, RWE Power

The shown figures in table 13 are separating the actual cost prices for the consumer or user of the coal and have nothing to do with the actual real cost price of the individual component.

Based on the market price of thermal coal on the spot market we can now calculate the actual rough market value of the hard coal deposit which was left over between the longwall areas due to the ‘shallowing’ out of the coal seam.

The achievable spot market price of the Northern Appalachian hard coal is around the USD 43,00 mark. The hard coal deposit we base the investigation on has got a clean volume of approximately 2,57Mio m^3 @ 1,4 kg/ m^3 or in weight 3,6 Mio tons.

Representative costs in the coal chain (2004/5), cif ARA

Export country	Region Extraction method	Costs free mine USD/t	Transport domestic USD/t	Port handling USD/t	Sea freight Aug 05 USD/t	Total costs free ARA port USD/t
1. Steam coals						
Australia	Queensland Opencast	11-32	6-14	2-3	14	33-63
	New South Wales Underground	19-31	3-10	2-3	17	41-61
	New South Wales Opencast	17-29	3-10	2-3	17	39-59
South Africa	Opencast	16-24	6-10	1.5-2	9	32.5-45
Colombia	Opencast	22-24	2-3	3-5	9	36-41
Russia	Opencast	15-16	10-20	2-3 (6-8)*	8	35-47 (39-52)*
China	Underground	29-36	6-9	2-3	12	49-60
Indonesia	Opencast	14-26	2-7	2-4.5	12	30-49.5
Venezuela	Opencast partly plus transit/post-treatment	16-20	5-6	3-5	9	33-40
2. Coking coals						
Australia	Queensland Underground	22-33	8-10	2-3	14	46-60
	Queensland Opencast	20-28	6-9	2-3	14	42-54
	New South Wales Underground	20-40	4-6	2-3	17	43-66
	New South Wales Opencast	22-27	5-7	2-3	17	46-54
Canada	British Col. Opencast	26-36	20-22	3-5	17	66-80
US/Central-Appalachians	Underground	30-48	15-26	2-3	10	57-87

* incl. transit, post-treatment
Source: International Energy Agency, Coal Information, own calculations

Table 13: **Representative Costs in the Coal Supply Chain;** Source: International Energy Agency; World Market for Hard Coal, 2005 Edition, RWE Power

We have to remember the normal height of the coal bed in our investigation. The current normal mining height wherein the longwall equipment is used is around the 3,5m mark. The operating longwall equipment was specifically built for these conditions and is impossible to be changed to a cutting height below 1,5m seam height.
The left over hard coal area has got seam height of around 1,3 to 1,4m whereof only 1,2m are possible to mine out. This means in case we would like to utilize a longwall extraction system in this operation, a new or may be second hand machine would have to be procured.

A longwall system for such an application will range in a capital cost area of above USD 25,0 Mio.

We have 3,6 Mio tons of hard coal with a possible extraction rate of approximately 80% (due to the shape of the left over hard coal block) available is therefore reduced to a realistically mine-able hard coal volume of around 2,5 Mio tons.

The fact that we have basically only 2,5 Mio tons of coal available for longwall extraction means, that after the extraction of the coal deposit, there is no further use for the low height longwall system. The system will have to be written down over the extraction process.
Simplistically, for every ton of mined hard coal USD 1,0 would have to be taken into account on equipment depreciation only. This amounts already 2,3% of the sell price and most likely almost 5% of the cost price to mine the hard coal deposit.

Even though a longwall system would have this hard coal deposit most likely extracted within 10 to 12 months and the coal would be available for selling much earlier than with another mining method the investment cost of a longwall system based on the extraction volume is much too high to achieve a competitive market price for the hard coal.

Therefore another mining method had to be investigated and the view went in direction board and pillar mining.
Continuous Mining will be the preferred way of operation. Using a continuous mining method the cutting rate can be as high as 400,000 tons per year considering the dimensional restrictions of our hard coal deposit.
Also the capital cost of a continuous mining system, mining machine and material handling system, will most likely not be in the range of 20 to 25% of a longwall system.

With a possible extraction rate of 55 to 58% of the 3,6 Mio tons, we have a volume of roughly 2,0 Mio tons sellable hard coal to mine using the board and pillar method.

This means that our
2,0 Mio ton left-over-hard-coal-deposit is worth approximately USD 86,0 Mio
on the current spot market.

Assuming we will be able to mine/produce 400,000 tons of hard coal per year, the operation will run over a period of 5 years and will generate a revenue figure of around USD 17,0 Mio per year.

The above investigated achievable spot market price for hard coal will now form the basis of all further researches and developments.

It has to be mentioned that the spot market price is a selling price of the coal which means this price contains the *actual costs to produce the coal* including the *sales and administration costs* as well as the *profit* the mine owners want to make to form a healthy business.

This section contains references to the following:
- World Market for Hard Coal, 2005 Edition, RWE Power
- Energy Information Administration

3.5. Suggested mining method

In the chapter 'cost and performance aspects' parameters are shown which demonstrate that the use of a longwall system is commercially not viable in the hard coal deposit we had to analyze.

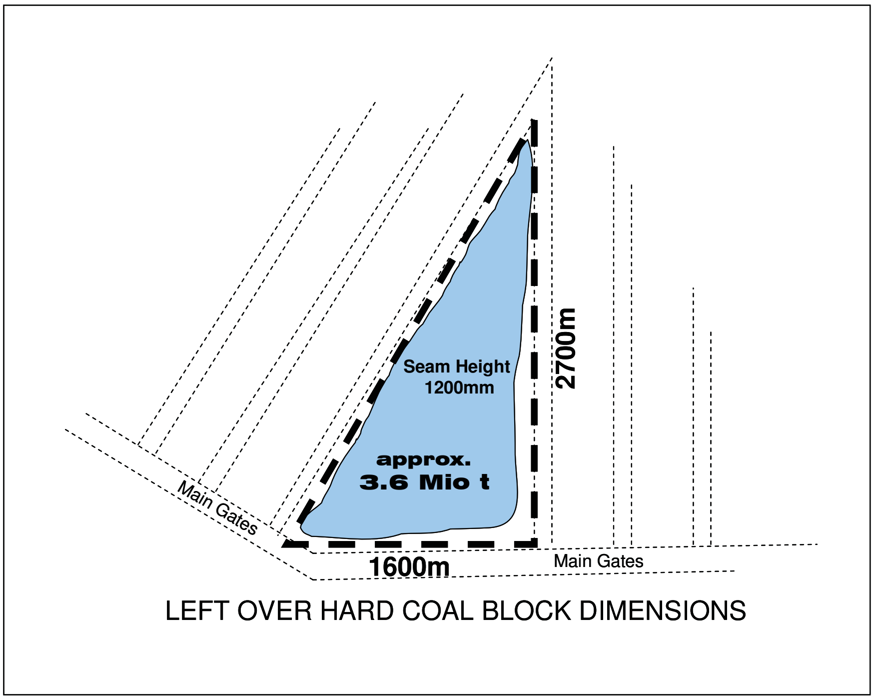
The use of the **Room and Pillar mining method** will present many advantages over the longwall:

- Much lower capital investment
- High flexibility of the operating equipment
- Unaffected by geological changes
- Mining machine can do development of /access to coal deposit and production work
- Easy to operate and to relocate
- Mining system can be used elsewhere (complete or in parts) when finished

Due to the fact that we are using the standard already existing underground mine materials handling system (conveyors, bunkers, crushers, etc.) to transport the cut coal out of the mine means:

- No changes to underground material handling system necessary

The left over hard coal deposit has got the described shape and dimensions (picture 27):

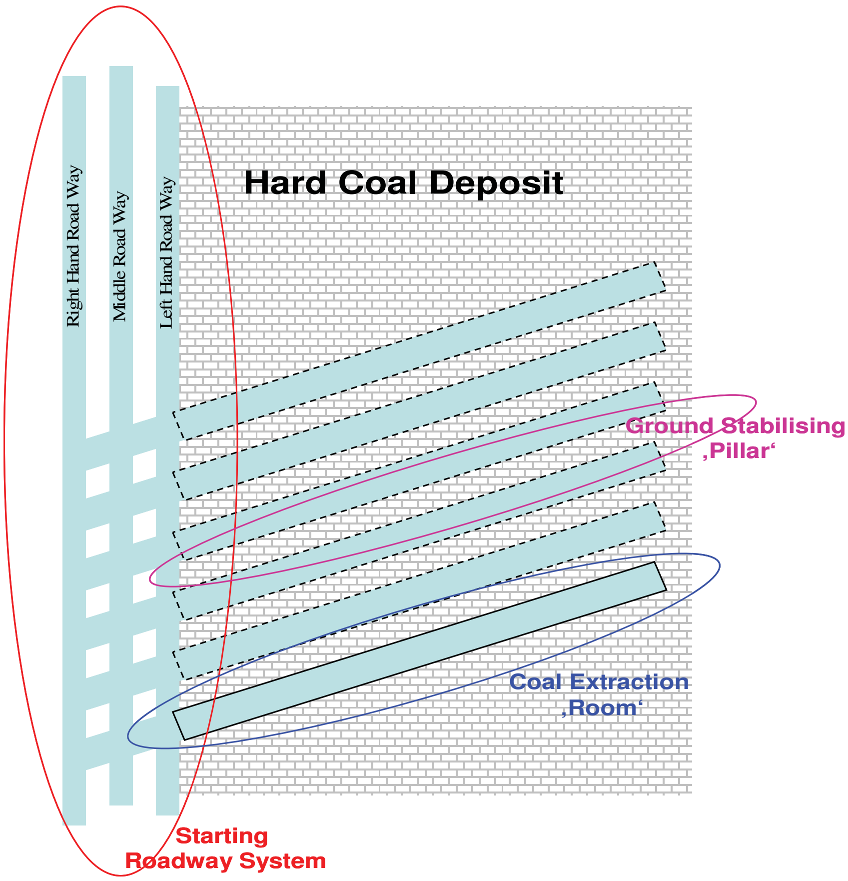


Picture 27: **Dimensions of the Left Over Hard Coal Deposit**

A room and pillar system will have to consist of the following:

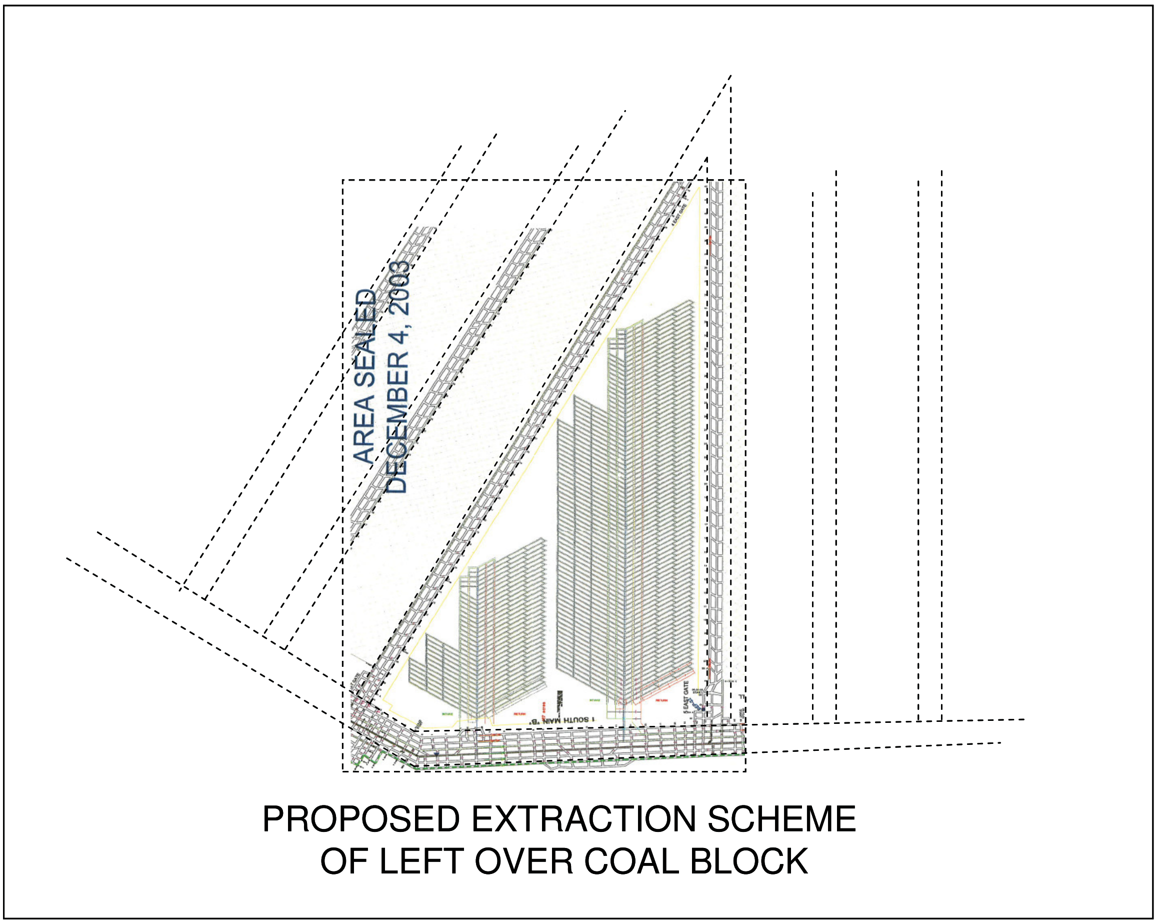
- Roadway system from where the mining will start from. This underground roadway system can require the one, two or three parallel tunnels which have the purpose of, for example, accommodating a material handling system (belt conveyor system, shuttle car traffic), underground ventilation (intake or fresh air as well as return or exhaust air), serving the general underground transport traffic (railway, people transport, material transport other than the mined minerals, etc.) or storage (transformers, material in the likes of ground support material, equipment in the likes of pumps, drills and other tools).
- Rooms from where the mineral deposit will be extracted from (actual mining face)
- Pillars to guarantee the integrity of the mine itself
- May be additional ventilation tunnels connecting the rooms

The following schematic in picture 28 shows the principle lay out of a room and pillar mining method where the actual basic ideas to extract our left over hard coal deposit are already incorporated in. Three starting roadway will be used due to the local underground regulations the mine is subject to in the United States of America.



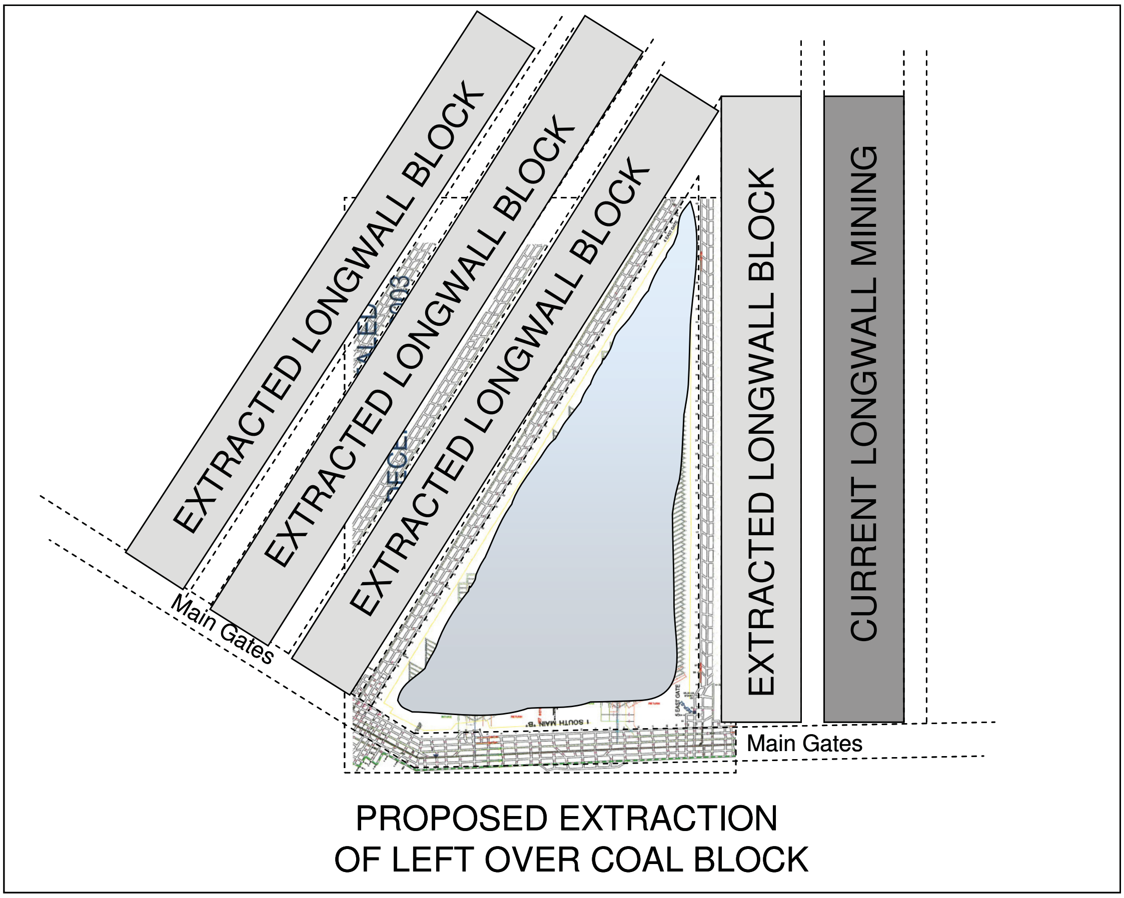
Picture 28: **Schematic of a proposed Room and Pillar Mining Method**

Inserting the sketched up left over hard coal deposit into the existing running mine plan (picture 29) and superimposing a room and pillar mining plan shows the following picture of an suggested underground mine operation.



Picture 29: **Proposed Room and Pillar Mining Method superimposed into the real Longwall Mining Operation**

After overlaying all the discussed plans and layouts the following basic mining plan is established and visualized in picture 30.



Picture 30: **Proposed Mine Plan**

A detailed dimensional layout of the proposed room and pillar mining plan will be worked out at a later stage when all local underground mining regulations regarding strata control, ventilation, turning radii, roadway length and width, etc. had been considered and complied with.

3.6. Underground coal mine regulations

The governing institutions of all underground coal mining have their origin in the Occupational Health and Safety (OH&S) Acts of the individual countries.
In the United Sates of America the federal enforcement agency for the health and safety of mine workers is the Mine Safety and Health Administration (MSHA) in Australia it is the Australian Institute of Mining and Metallurgy (AusIMM) and in Germany the Safety of machines used in underground mining application is governed by the Deutsche Industrie Norm (DIN).

The very strict acts, rules and laws for underground mining in those countries are leading the way for all other countries as well at the present moment.

Even though the actual regulations are different in each country, the main leads are the same for all underground mines worldwide. The individual rules are somewhat built onto each other and are therefore related to each other in regards to the wordings. It has also to be mentioned that the OH&S code in each country is subject to continuous additions, omissions, corrections, and rewordings.
They are living documents and it is the duty of each Original Equipment Manufacturer (OEM) to comply with the codes specific for the machines they supply.

All machines entering an underground coal mine have to be designed and built in conformity to the local underground mine regulations and their codes.
The machines have to be checked out and approved by a government authority before the machines are allowed to be operated in the underground mine. The machines and also any other equipment have to hold this approval with a specific approval number as well as a plate fixed to the machine showing the necessary data of this approval. In case the equipment is built in exactly the same manner, the approved machine can be built in any quantity without further checks. They only have to have the plate with the approval details engraved permanently fixed to the machine.

As said before, the approval of the machine is aiming at the safe operation of the machine and all its functions. All OH&S regulations and acts are mandatory to comply with, with the only aim to create a safe working environment for the workers the mine employs.

The risk which has to be reduced or eliminated is two fold

- OH&S risk
- Financial risk

The actual machines and equipment as well as the mineral deposit and the access to it are very important to protect too but treated secondary to OH&S.

Standard steps towards getting an approval from the underground mining authorities granted in a more or less reasonable time frame are to check out the following:

- Are all components built into the machine approved by the local underground authority?
- Are the local design rules for the specific equipment followed?
- Has a design review been done?
- Is the operation of the equipment safe?
- Has an operational risk assessment been done?
- Have all identified OH&S risks been eliminated?
- Have all other operational risk issues been addressed?
- Is the machine able to operate according to all other local underground regulations
- Can the machine fulfill the local underground mining requirements?
- Have all relevant people been involved in the risk assessments and reviews?

The approval plate has to show all for the machine important details (for example picture 31):

Punch Mining System PMS-500	
Manufacturer:	Sandvik Mining and Contruction
Serial No.:	#1
Date of Manufacturing:	November 2006
Approving Authority:	MSHA – Pennsylvania (USA)
Approval No.:	…………………..
Approval Year:	2006

Picture 31: **Approval Plate for an Underground Mining Machine**

In case of any technical changes to the equipment or any other related condition, the approval has to be amended accordingly.

Also the design of the underground mine itself with the dimensions of the roadways (width and height), the turning radii from one road into the other, the ventilation of the mine to last but not least the underground strata control measurements have to comply with the governments underground mine regulation acts.
For all mining actions the coal mine has to obtain an approved ground (roof, rib, face) control plan before starting any rock cutting.
In parallel the local mine regulation put in place by the mine managers shall complement the overall OH&S system of the underground mine.

One of the main issues in an underground mine and specifically in the underground coal mine in which the above mentioned left over hard coal block was under investigation, whether we are allowed to extract the coal or not, are the strata control or in other words ground control measurements.
The strata control measurements are measurements to keep the integrity of the mine for safety reasons after the coal bed or coal layer was cut out. The strata control measurements prevent the sediments or rock in which the coal deposit was bedded in to collapse. Those also called ground supports partly regain the strength of the missing rock even though they will never be able to substitute the full capacity. This limited substitution of rock strength is the reason for the determination of restricted roadway and dimensions of the mining chamber/room.

In the beginning years of mining technology the use of wooden poles was common to prevent the roof from falling in.
In modern mining those poles are still in use but only in certain applications which do not form a part of this investigation and will not be further discussed.
The ground control measurements which will form part of this work are done with steel, fiber or wire robe anchors (mainly with threaded ends) which will be inserted in a pre-drilled hole, subsequently chemically (resin) and mechanically (expansion rings) retained in the hole and pre-tensioned against the rock face (torqued up nut against roof washer) to compensate for the lost (cut out) structure.

Following is an excerpt of the MSHA (Code of Federal Regulations, Title 30, Part 75) regulations in the USA concerning 'roof bolting'.

---

30 CFR § 75.204 (Code of Federal Regulation)

Roof bolting.

(a) For roof bolts and accessories addressed in ASTM F432-95, ``Standard Specification for Roof and Rock Bolts and Accessories," the mine operator shall--

(1) Obtain a manufacturer's certification that the material was manufactured and tested in accordance with the specifications of ASTM F432-95; and

(2) Make this certification available to an authorized representative of the Secretary and to the representative of miners.

(b) Roof bolts and accessories not addressed in ASTM F432-95 may be used, provided that the use of such materials is approved by the District Manager based on

(1) Demonstrations which show that the materials have successfully supported the roof in an area of a coal mine with similar strata, opening dimensions and roof stresses; or

(2) Tests which show the materials to be effective for supporting the roof in an area of the affected mine which has similar strata, opening dimensions and roof stresses as the area where the roof bolts are to be used. During the test process, access to the test area shall be limited to persons necessary to conduct the test.

(c)(1) A bearing plate shall be firmly installed with each roof bolt.

(2) Bearing plates used directly against the mine roof shall be at least 6 inches square or the equivalent, except that where the mine roof is firm and not susceptible to sloughing, bearing plates 5 inches square or the equivalent may be used.

(3) Bearing plates used with wood or metal materials shall be at least 4 inches square or the equivalent.

(4) Wooden materials that are used between a bearing plate and the mine roof in areas which will exist for three years or more shall be treated to minimize deterioration.

(d) When washers are used with roof bolts, the washers shall conform to the shape of the roof bolt head and bearing plate.

(e)(1) The diameter of finishing bits shall be within a tolerance of plus or minus 0.030 inch of the manufacturer's recommended hole diameter for the anchor used.

(2) When separate finishing bits are used, they shall be distinguishable from other bits.

(f) *Tensioned roof bolts.* (1) Roof bolts that provide support by creating a beam of laminated strata shall be at least 30 inches long. Roof bolts that provide support by suspending the roof from overlying stronger strata shall be long enough to anchor at least 12 inches into the stronger strata.

(2) Test holes, spaced at intervals specified in the roof control plan, shall be drilled to a depth of at least 12 inches above the anchorage horizon of mechanically anchored tensioned bolts being used. When a test hole indicates that bolts would not anchor in competent strata, corrective action shall be taken.

(3) The installed torque or tension ranges for roof bolts as specified in the roof control plan shall maintain the integrity of the support system and shall not exceed the yield point of the roof bolt nor anchorage capacity of the strata.

(4) In each roof bolting cycle, the actual torque or tension of the first tensioned roof bolt installed with each drill head shall be measured immediately after it is installed. Thereafter, for each drill head used, at least one roof bolt out of every four installed shall be measured for actual torque or tension. If the torque or tension of any of the roof bolts measured is not within the range specified in the roof control plan, corrective action shall be taken.

(5) In working places from which coal is produced during any portion of a 24-hour period, the actual torque or tension on at least one out of every ten previously installed mechanically anchored tensioned roof bolts shall be measured from the out by corner of the last open crosscut to the face in each advancing section. Corrective action shall be taken if the majority of the bolts measured--

(f)(5)(i) Do not maintain at least 70 percent of the minimum torque or tension specified in the roof control plan, 50 percent if the roof bolt plates bear against wood; or

(f)(5)(ii) Have exceeded the maximum specified torque or tension by 50 percent.

(6) The mine operator or a person designated by the operator shall certify by signature and date that measurements required by paragraph (f)(5) of this section have been made. This certification shall be maintained for at least one year and shall be made available to an authorized representative of the Secretary and representatives of the miners.

(7) Tensioned roof bolts installed in the roof support pattern shall not be used to anchor trailing cables or used for any other purpose that could affect the tension of the bolt. Hanging trailing cables, line brattice, telephone lines, or other similar devices which do not place sudden loads on the bolts are permitted.

(8) Angle compensating devices shall be used to compensate for the angle when tensioned roof bolts are installed at angles greater than 5 degrees from the perpendicular to the bearing plate.

(g) *Non-tensioned grouted roof bolts.* The first non-tensioned grouted roof bolt installed during each roof bolting cycle shall be tested during or immediately after the first row of bolts has been installed. If the bolt tested does not withstand at least 150 foot-pounds of torque without rotating in the hole, corrective action shall be taken.

[53 FR 2375, Jan. 27, 1988, as amended at 55 FR 4595, Feb. 8, 1990, Apr. 22, 1998]

Following underground mining regulations have to be kept in sight specifically when developing a mining system which is supposed to extract the left over hard deposit we have got on hand to investigate.

- Machine engineering and design for underground coal applications
- Strata control requirements
- Ventilation requirements

Due to the fact that the underground coal mine this left over hard coal deposit is located in the USA, the underground coal mining regulations of the United States of America (MSHA) in specific the ones from the areas around Pennsylvania, West Virginia and Ohio have to be considered and complied with. This comprises the complete system layout which is the aim of this research and development program.

4. System evaluation

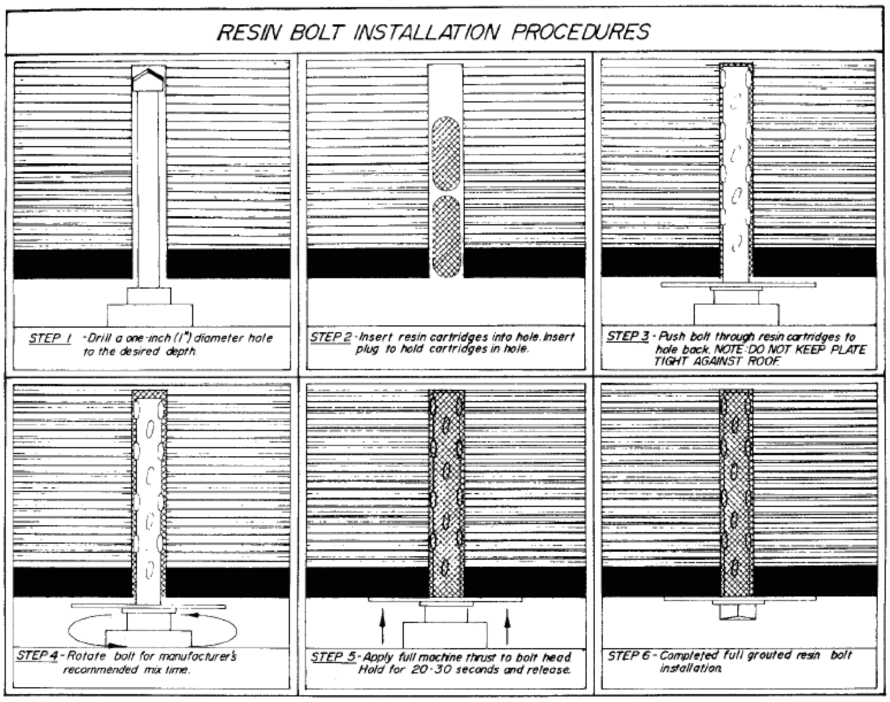
4.1. Specific local underground regulations

Besides all main underground coal mining regulations (Code of Federal Regulations, Title 30, Part 75) which have to be complied with in this coal mine in the United States of America, only the three very relevant rules which are effecting our discussed mining system will be high lighted by the author.

<u>Strata control (ground support)</u>

As discussed before the underground mine regulations request the proper installation of ground supporting measurements to prevent the collapsing of roof and ribs in the mined out areas.
Such ground supporting measurements are mainly roof bolts and rib bolts. For deep mines there are also steel arches as well as ground bots used for strata control.

Due to the reason that the coal mine which is subject to our investigations is only around 300m below surface level the use of roof anchors (installed into to roof) and rib anchors (installed into the side walls of the cut out sections) are sufficient.



Picture 32: **Resin Bolt Installation Procedures;** Source: Roof Bolting in underground mining, stat-of-the-art-review, Syd.S.Peng, D.H.Y.Tang, International Journal of Mining Engineering, 1984, 2, 1-42; Mc.Cormick et al. 1974)

Roof and rib anchors are available as mechanical and chemical types (picture 32).
Whilst the mechanical anchors restrain themselves via, for example, an expansion shell, the mainly used type of anchor is the chemical type. A chemical mix between a resin component and a hardener or curing component glues the anchor into the drilled hole.

Both anchor types will be tensioned via a nut and a steel plate against the roof. The tensioning of the steel or fiber glass bar is compensating for the lost tension due to the cut out coal or rock.

The actual roof and rib bolting pattern will be determined by the geologists responsible for the mine and checked and approved by the mines department (MSHA, table 14).

§ 75.220 Roof control plan.

(a)(1) Each mine operator shall develop and follow a roof control plan, approved by the District Manager, that is suitable to the prevailing geological conditions, and the mining system to be used at the mine. Additional measures shall be taken to protect persons if unusual hazards are encountered.

(2) The proposed roof control plan and any revisions to the plan shall be submitted, in writing, to the District Manager. When revisions to a roof control plan are proposed, only the revised pages need to be submitted unless otherwise specified by the District Manager.

(b)(1) The mine operator will be notified in writing of the approval or denial of approval of a proposed roof control plan or proposed revision.

(2) When approval of a proposed plan or revision is denied, the deficiencies of the plan or revision and recommended changes will be specified and the mine operator will be afforded an opportunity to discuss the deficiencies and changes with the District Manager.

(3) Before new support materials, devices or systems other than roof bolts and accessories, are used as the only means of roof support, the District Manager may require that their effectiveness be demonstrated by experimental installations.

(c) No proposed roof control plan or revision to a roof control plan shall be implemented before it is approved.

(d) Before implementing an approved revision to a roof control plan, all persons who are affected by the revision shall be instructed in its provisions.

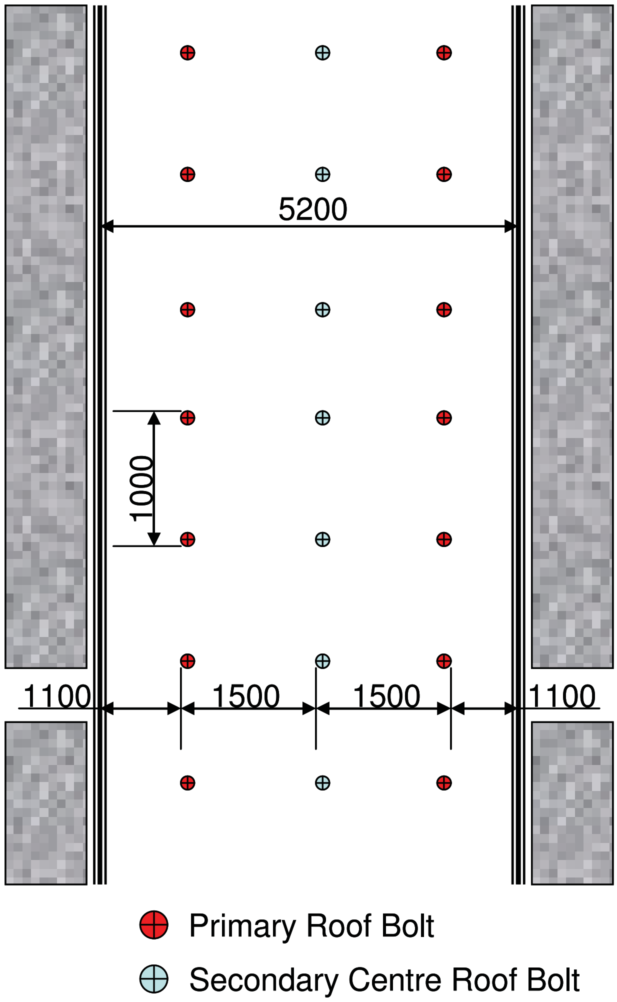
(e) The approved roof control plan and any revisions shall be available to the miners and representative of miners at the mine.

(f) Existing roof control plans that conflict with this subpart C shall be revised to meet the requirements of this subpart C by September 28, 1988. This paragraph (f) shall expire March 28, 1989.

[53 FR 2375, Jan. 27, 1988; 53 FR 11395, Apr. 6, 1988 as amended at 60 FR 33723, June 29, 1995]

Table 14: **MSHA Roof Control Plan;** Source: MSHA (Code of Federal Regulations) 75.221 of Title 30 – Mineral Resources

Following the very simple roof bolting pattern with three roof bolts across the roadway spaced in a longitudinal distance of 1,0m is shown in picture 33.



Picture 33: **Suggested Roof Bolting Pattern**

The bolting can be done in sequences and is directly dependent on distances from the mining face and time elapsed since the cutting process.
Roof bolts are spaced by fixed maximum dimensions as well as the time frame in which a roof and rib anchor has to be installed is directed by a maximum fixed duration which it is allowed to take. The actual values of the distances and the time frame are only dependant on the geological conditions in the underground mine, the rock mass, the faults, etc. and the covering layers above.

For this investigation following underground strata control rules have been filtered out as being the most relevant and effecting ones for the selected mining process in our left over hard coal seam.
The following rules are not quoted in the same words as in the actual underground coal mine regulations which MSHA has put in place:

- The maximum road way width is 5200mm. No road way is allowed to be wider as this dimension due to the danger of not being able to control the covering layers with conventional anchors.
- The roof and rib anchors have to be installed within a maximum longitudinal distance of 1000mm.
- Centre roof bolts have to be installed along the centre line of the cut out road way in the above set maximum distance to each other.
- Across the width of the cut out cavity the roof and rib bolts have to be installed within +/- 25mm tolerance around the pre determined bolting pattern.
- The maximum distance coal is allowed to be mined out beneath unsupported roof (no strata control measurements installed) is 15m. After advancing the cutting out process for 15m into the coal bed the mining action has to be stopped and ground control measurements have to be installed before commencing any further.
- Continuous mining or cutting out work is allowed when installing temporary roof support anchors whilst mining. Temporary roof bolting means a minimum amount of roof anchors have to be installed during the excavation work which re-establishes a certain roof support function. Those temporary roof bolts are also called primary roof bolts
- Within a time frame of 72 hours the pre-determined ground support measurements have to be completed. The completed roof and rib bolt pattern is also called permanent roof and rib support.
- In case the roof and rib anchors are not installed within this 72 hour time window, the mined out section must not to be entered by any means anymore. If these sections are supposed to be accessed because of whatever reasons in the future, the permanent ground support has to be installed first.
- In excavations higher than 1300mm rib anchors have to be installed 500m above ground level spaced vertically by 1000m and in a maximum distance of 500mm from the roof level down.
- The roof and rib bolts have to be of a grouted in (chemical) type.
- The minimum length of the roof anchors is 1300mm and for the rib bolts 1100mm

Continuous checking of the underground roof and rib conditions and in specific the behavior of the rock together with the installed strata control measurements over the time is one of the most important routines in an underground mine.
Permanent measurements of the movements of the mined out section are giving the necessary information of the long time behavior of the system.
In case the rock is starting to cave in, additional ground support has to be installed as well as the strata control rules for the underground mine adjusted to manage the changed conditions.

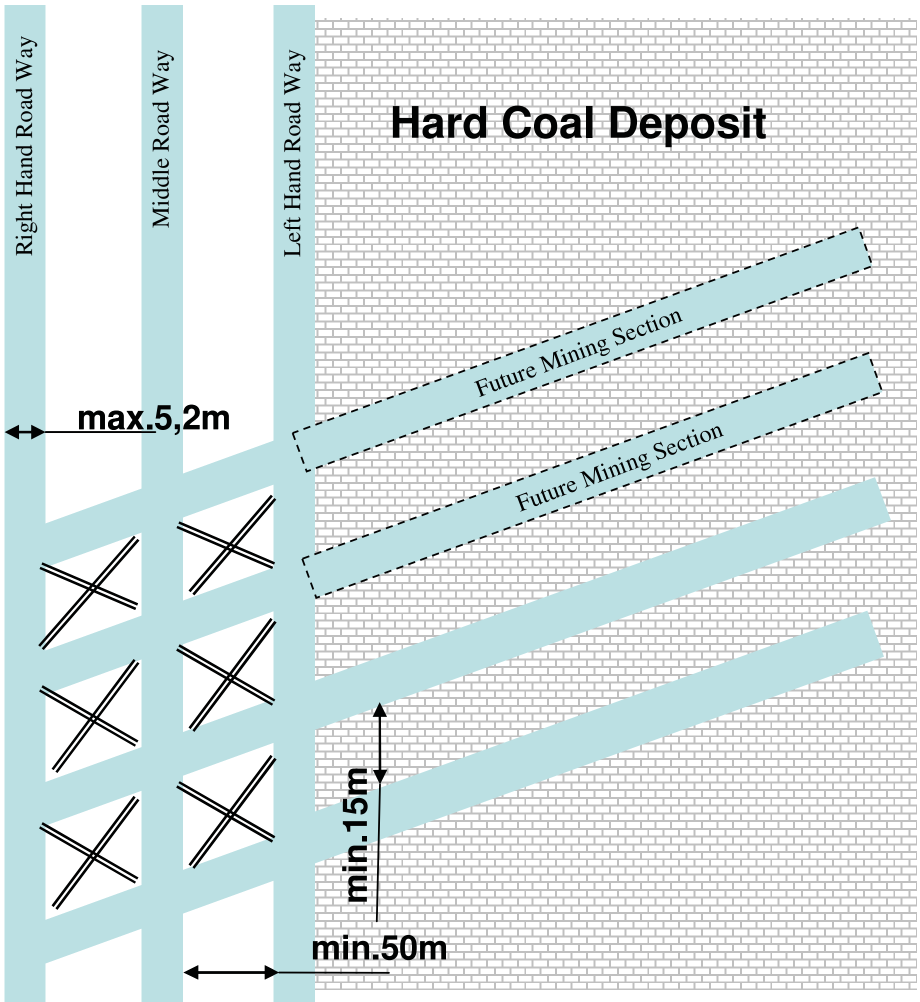
Preventive investigations of the in front laying geological conditions are also an important part of mine geologist. This is done via exploration drilling from the surface down into the mineral deposit as well as drilling into the mining face to get the right picture.
The strata control measurements will be verified and confirmed or adjusted according to the gained information. The geological conditions are a permanent varying system and it is of utmost importance to keep the underground mine body subject to continuous tests and checks.

Pillar Size:

In the room and pillar mining operation the pillars are playing an integer role as a crucial part in the underground support system.
Pillars are taking most of the load applied from the above sitting rock layers in an underground coal mine. This is why it is very important to keep the predetermined pillar sizes well above the minimum dimensions set by the geologists.

It has also to be accounted for that the loss of coal in the side walls, the ribs, due to the ground pressure and the subsequent effect of coal breaking out over the time as well as over-cut due to the natural fracturing of the coal when mined out is reducing the size of the pillars. These effects are considered in a safety factor applied to the pillar size calculation.

In the following picture 34, the minimum dimensions of the pillar sizes for our underground research area are illustrated.



Picture 34: **Pillar Dimensions**

The final pillar size will be subject to specific underground stability calculations and will be optimized by the mine operator and their geologists. The final dimension will have to be determined when starting to lay out the mining system is its dimensions.

Ventilation:

Mine gases such as CH_4 Methane and CO Carbon Monoxide are the most dangerous gases in underground coal mines and have been responsible for many mine fatalities. Their effect on the environment is similar and the result of having the gases present in a critical concentration is a high risk to occupational heath and safety. (includes references from www.cdc.gov)

Methane:

Methane is a colorless, odorless but highly flammable gas. It is lighter than air and will be found near the mine roof.
When diluted with air to a concentration between 5 and 15% the mixture becomes explosive.
The methane gas occurs naturally in all coal mines. The gas is trapped in the pores within the coal bed and is a by-product of the coal metamorphosis. When the coal is getting broken out, the methane gas is being released into the underground environment. The amount of methane gas released by the coal is depending on the geological age, the type of coal, the depth of the coal deposit as well as other means by which the gas is kept within the coal, for example water.
The flammable mixture of methane and air can be ignited by electric arcs and sparks, open flames or by friction heat between coal cutting tools and the sediment rocks laying above the coal seam.

Due to the fact that Methane gas is extremely dangerous permanent checks for this gas have to be performed whilst people work in the underground environment.

Gas samples are taken on a regular basis in all mining sections of the underground mine. This can be done either with hand-held or stationary devices. Only certified and authorized mine employees are allowed to perform such test which has to be done before and during workers are working underground.

MSHA and the Federal safety standards (Code of Federal Regulations, Title 30, Part 75) mandate that (quote of this code),

'*when 1.0 percent or more methane is present in a working place or an intake air course […] electrically powered equipment in the affected area shall be de-energized, and other mechanized equipment shall be shut off.*'

Carbon Monoxide:

Carbon Monoxide is a toxic gas that is produced from the incomplete combustion or explosion of substances containing carbon such as coal, natural gas or gasoline. Large quantities of CO are generated during mine fires or explosions.

CO is colorless, tasteless, odorless and slightly lighter than air. It is flammable and explosive in mixtures with air in concentrations between 12.5 and 74%. It is toxic because it blocks the ability of the hemoglobin in the blood to carry oxygen from the lungs to the muscles and other tissue in the human body.

The recommended exposure limit (REL) for CO is 35 ppm, measured as a time-weighted average (TWA) for up to a 10-hour workday during a 40-hour work week. The ceiling concentration (not to be exceeded during any part of the workday) is 200 ppm. CO in concentrations of 500 ppm or 0.05% can be fatal in 3 hours. Higher concentrations can lead to coma and death in minutes. Carbon monoxide is known as a "silent killer".

Carbon Monoxide can be detected by hand-held sensors. Stationary sensors may also be installed at strategic points in mine airways.

Underground coal mine ventilation:

The main purpose of mine ventilation is to dilute, render harmless, and carry away dangerous accumulations of explosive and toxic gases and dust from the working environment and especially the mining section (this is where most of the gases and dust are created) in underground mines.

Federal safety standards (Code of Federal Regulations, Title 30, Part 75) for ventilating underground coal mines mandate that (quote of this code),

"*the air in areas where persons work or travel […] shall contain at least 19.5 percent oxygen and not more than 0.5 percent carbon dioxide, and the volume and velocity of the air current in these areas shall be sufficient to dilute, render harmless, and carry away flammable, explosive, noxious, and harmful gases, dusts, smoke, and fumes.*"

Hazardous concentrations of methane underground can be controlled by dilution (ventilation), capture before entering the section air stream (e.g., methane drainage), or isolation (seals, air locks and stops).

Explosions can be prevented or mitigated by eliminating ignition sources (no open fire or flames, specially designed compartments to accommodate electric gear, etc.), by minimizing methane concentrations and coal dust accumulations, and by using passive and active barriers to suppress propagating explosions.

In coal mines, methane explosions can cause subsequent, violent explosions of coal dust. This will after all the harm the explosion causes in the effected area also create huge quantities of CO due to the incomplete combustion and no remaining oxygen in the air. This effects the main air stream and also the by the explosion none effected sections in the underground mine.

To prevent such explosions, covering of the floor, rib and roof surfaces of mine openings with large quantities of inert rock dust such as fine limestone dust is most effective. Rock dusting is mandated by and subject to federal safety standards.

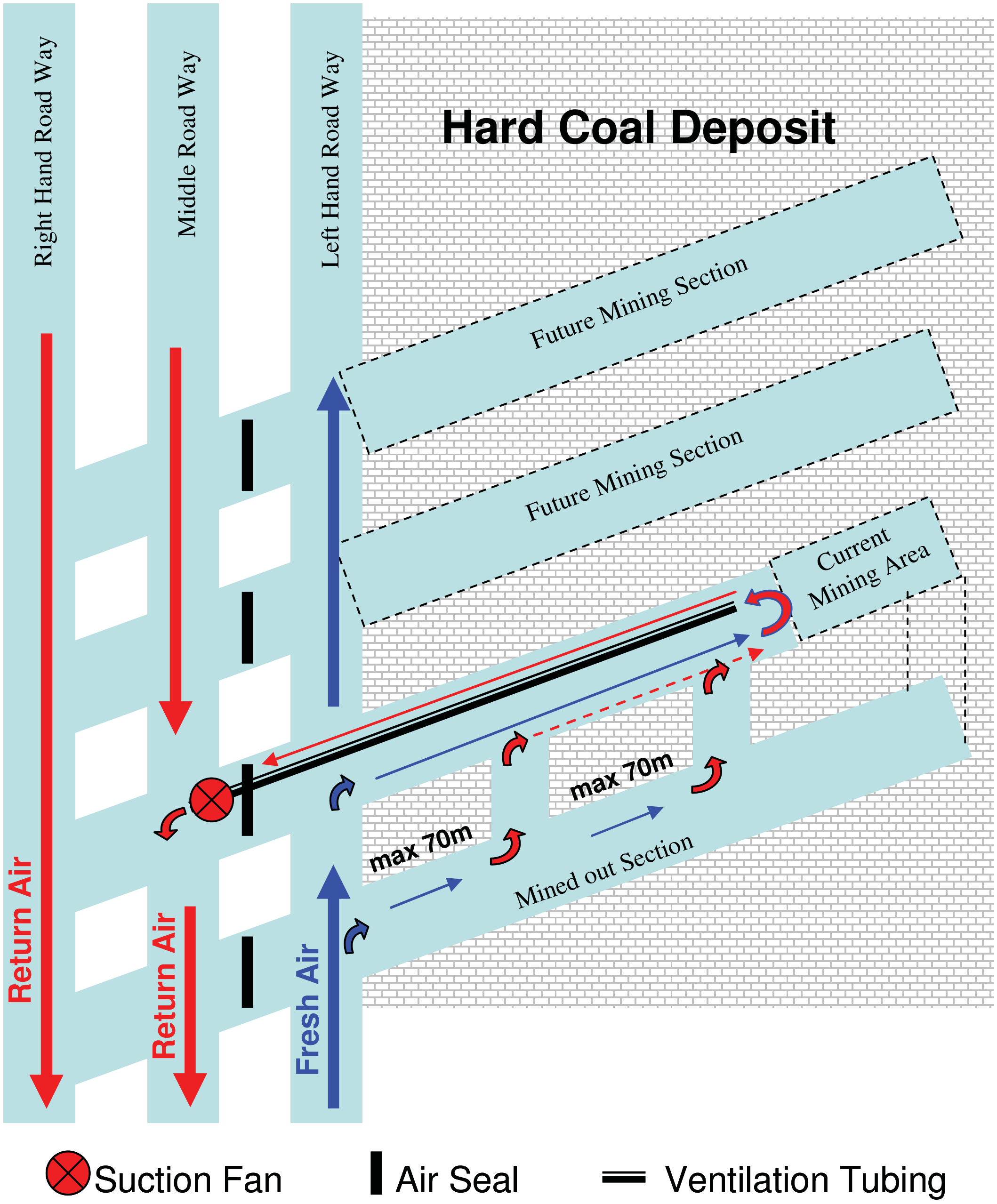
The air stream in the underground mines is forced through by big stationary fans. The fans provide the fresh air into the mines.
The air stream is than guided through the mine in a way that all areas of the underground tunnel system are provided with fresh air and the exhausted air as well as gases and dust are captured and guided out of the mine. Fresh or intake air and return air streams have to be separated and the individual air courses isolated from each other to avoid short circuits.
This guidance of the air stream and to avoiding a premature mixture (short circuits) of the fresh air with the exhausted air, gas and dust and also to reach the furthest corner of the underground roadways are done with fix installed brick walls, seals and air locks.
All those walls, seals and locks are constructed and built to the federal standards.

Also regulators are being used to adjust the airflow quantity into certain sections.

Areas where no airflow can be maintained have to be ventilated separately with mobile fans. Areas like this are for example the actual mining face at the time.
This ventilation of this mining faces can be done in a forced way (fresh air is being pressed into the area) or the return air is being sucked out of the area. Both methods are being used in underground mining, however, in coal mines predominantly the ventilation with a mobile fan sucking the exhausted air out of the mining section is being used.
In this case the mobile fan is sucking the return air, gas and dust from the mining face via ventilation tubes (glass fiber ducts) and is connected at the other end to the main return air stream from the mining sector.

The whole underground tunnel and roadway system has to be ventilated at all times.

To apply the above said to our hard coal deposit leads us to the following suggestion of a ventilation system presented in picture 35 below.



Picture 35: **Suggested Ventilation System**

Room and pillar mining and the accompanying ventilation of the mining area are also done with seals and walls to guide the air stream through the rooms. The mining faces have to be ventilated at all times to extract the deliberated gases as well as the created dust to create a good working/mining area.

The federal safety standards for underground coal mine ventilation also mandate the creation of cross cuts into the previous excavation tunnel after a maximum of 70 meters to guarantee a proper working ventilation system for the previous tunnel.

4.2. Commercial facts and assumptions

4.2.1 Hard coal cost/sell price components

To be able to evaluate the commercial aspects of our left over hard coal deposit we have to analyze the actual cost structure of the coal production over the transport to the final destination the end user in more detail.

Following cost components are relevant for the investigation:

- Coal mining cost
- Coal washing and beneficiation
- Domestic transport supplier end
- Port handling fees supplier end
- Insurances
- Sea freight
- Port handling fees receiver end
- Import duties and fees
- Domestic transport receiver end

The international hard coal trade is only looking at the hard coal price delivered to the border. In the case of Europe this is ARA (Amsterdam – Rotterdam – Antwerp). This average price for hard coal delivered 'CIF (cost, insurance, freight paid) to the European border is around USD 61,00 per ton (RWE Interim Report Jan – Jun 2006, Hard Coal Prices, www3.rwecom.geber.at).

All other prices on the receiver end will not be considered for the hard coal trading price.

Average prices for the supporting functions to get the mined coal from say an underground coal mine in the USA to Europe (ARA) are:

• Domestic transport	>	USD	15,00 per ton
• Port handling fees	>	USD	2,00 per ton
• Sea freight incl. insurance	>	USD	10,00 per ton

This means that whatever the price of the hard coal inclusive profit for the mine owners is and additional USD 27,00 per ton have to be added to get the sell price on the European spot market. In other words the hard coal on the American sport market will be around the USD 34,00 per ton mark on an average basis.

As mentioned at an earlier stage, the very high quality US North Appalachian hard coal from our investigated coal seam is traded on the sport market for up to USD 43,00 per ton. Spot market prices are the maximum prices achievable on the market where the buyer turns up with his trucks at the mine and is getting loaded with the coal amount purchased.

Due to the fact that the real cost price for the ton of hard coal is a very tight kept secret of the mine owner assumptions have to be done by the author.

To get a realistic sell price for the hard coal on hand we have to look at an average price between the spot market and normal long term supply contracts (USD 30,00 to 35,00 per ton). For the purpose of this research the price will be assumed (again due to the very high quality of the hard coal) to be USD 35,00 per ton. This is an achievable price for a long term contract over the 5 years it will take to mine the 2,0 Mio tons of hard coal which is left over in the discussed underground section.

This USD 35,00 per ton is the average value we have to work with when analyzing the profitability of the left over hard coal seam.

4.2.2 Mining cost / profit assumptions

USD 35,00 per ton is the actual sell price on the market considering the current market situation in 2007. It contains the profit for the mine owners and also all the costs to produce the coal.

Rough cost figures for a normal average underground coal mining operation are known and valued around USD 13,00 to 15,00 per ton of hard coal.
This is the cost value to get one ton of coal found, accessed, mined including equipment depreciations, transported, washed and placed onto a stock pile.

Exact cost structures and figures are very difficult to get as this are one of the most secret information around a mining operation due to the obvious fact that their real profit can be seen quick and directly. All mine operators want to avoid this situation.

All other costs like administration and sales costs of the mining house, depreciations of non mining equipment, insurances, other financial costs, etc. have also to be added to these mining costs.

The domestic transport cost of approximately USD 10,00 per ton to deliver the coal to the purchaser have also to be added to the equation as well to get to the real cost price for the profit/loss calculations of the mine operation.

To bring these commercial aspects into perspective with our left over hard coal deposit we also have to consider something else.

USD 30 to 35,00 per ton of hard coal is the assumed long term sell price achievable over the life time of the whole coal mine.

The USD 10,00 per ton for transport are a figure which can hardly be reduced due to the fact that we cannot negotiate much better prices with the transporting companies as for continuously increasing prices for fuel, taxes, etc.

This gets us to the mining operation cost figure of assumable USD 15,00 per ton of hard coal.

The cost components of one ton of coal placed on the mine owned stock pile are as following:

- Exploration cost (drilling, localizing, etc. the coal bed)
- Infrastructure establishment cost
- Mine site establishment cost
- Coal seam access cost
- Mining cost
- Coal handling cost
- Coal beneficiation cost
- Coal stocking cost
- Infrastructure and mine site maintenance cost
- Overhead cost
- Rehabilitation and recreation of the mine site after finishing mine operation

All above mentioned cost components will be calculated into the price of the production of one ton of hard coal sitting on the stock pile.

At the very beginning of the commercial evaluation of the explored coal bed, the amount of extractable and sellable coal has to be determined and all cost components put into relation to this overall amount of tons of hard coal.

The commercial success of the coal mining operation will be calculated based and depending on the achievable sell price of the complete coal volume minus the cost to build the mine, costs to run the mine operation and the costs to extract the coal. Including other financial costs common for this industry this will give the owners the anticipated profitability of the operation over its life time.
During the initial exploration of the hard coal bed, a thinning out area of the coal seam had been revealed. It was planned at the time, that this low seam area will not be mined, therefore the longwall operation stopped before this area.
Roadways to the remaining coal bed will have to be established around the shallow coal bed section, after which the longwall operation will be able to continue.
This low seam area within the overall hard coal bed is the area the investigated and now left over hard coal deposit lays in.

To be able to mine this section and to extract the hard coal in this area it is to find a suitable mining method, organize the necessary equipment, recruit the necessary people and calculate the profitability of this area as an island operation within the existing and running underground coal mine.

The only way to make the mining of the left over hard coal seam lucrative for the mine owner is to stay below the current cost per ton figure of maximum USD 15,00 per ton of hard coal achieved in the overall operation. This will guarantee that the extracted coal can be sold on an adequate basis compared to the longwall operation.
Challenging is the factor that the left over hard coal deposit contains only 2,0 Mio tons of extractable hard coal.
As a direct result of this fact and to be safe in regards to all financial aspects, all equipment (additional and specific to this confined operation) which has to be organized for this limited mining operation has to be covered with that USD 15,00 per ton.

A standard room and pillar mining operation will need the following additional minimum equipment:

- Continuous Mining Machine (CM)
- Material Handling equipment between CM and section conveyor (SC)
- Section conveyor
- Bolting Equipment
- Mobile Fans inclusive air ducting
- Transformers and electrical equipment
- Safety installations

All other costs are the same as for all the other mining operations in the underground coal mine.

Budget cost estimation in very rough terms:

• Continuous Mining Machine (CM)	USD	3,000,000
• MH equipment between CM and SC	USD	2,000,000
• Section conveyor (2000m long)	USD	2,500,000
• Bolting Equipment	USD	700,000
• Mobile Fans inclusive air ducting	USD	200,000
• Transformers and electrical equipment	USD	500,000
• Safety installations	USD	300,000
TOTAL budget cost (additional)	**USD**	**8,700,000**

With the 2,0Mio tons of hard coal available a rough budget investment factor of USD 4,35 per ton of extracted coal has to be considered (not considered here are financing costs of the investment). The targeted mining operation will run for a period of 5 years to extract the 2,0 Mio tons of hard coal. In case the mining equipment can be used again afterwards (what will most likely happen!), this will be an additional benefit to the operation.

To put the cost figures into the right perspectives it has to be mentioned that the USD 15,00 per ton coal sitting on the stock pile is also including the depreciations and overhaul cost of the mining equipment over the life time of the coal mining operation. Therefore a certain amount has to be deducted from the USD 15,00 before adding the just estimated investments of USD 4,35 per ton of hard coal in the left over hard coal deposit.

Standard underground mining equipment will be depreciated over a time span of around 5 to 7 years.
However, in this time frame a standard underground mining operation will produce about 8 to 10 time as much coal as the left over hard coal deposit contains.

An estimation can be done and it can be determined that 1/10 of our investment cost of USD 4,35 per ton (USD 0,43 per ton) most likely less are already included and must be deducted from the maximum allowed mining cost.

This means taking the USD 15,00 and deducting USD 0,43 before adding the new investments of USD 4,35 will lead us to a very rough budget estimation of the mining costs in the investigated left over hard coal deposit of approximately USD 19,00 per ton of hard coal.

The challenge to render our left over hard coal deposit into a profitable business for the mine owner is to reduce the cost side of the normal underground mining operation by USD 4,00 per ton extracted hard coal to achieve at least the same financial result as for the extracted hard coal currently mined.

4.2.3 Cost reduction opportunities

The continuous mining machine as well as the materials handling equipment and the auxiliary equipment are a necessity for the extraction of the coal, the underground coal mining regulations are mandatory.
Cost reduction could be realized by purchasing second hand equipment.

The standard mining operation is based on experience and also measurements to permanently increase the efficiency of the operation to drive the costs to mine the coal down. Most of those savings only offset the start-up losses due to an unknown and new mining operation though.
Realizing this, there are not so many areas money can be saved.

Possible solutions can be found in avoiding unnecessary operations and processes, in the reduction of man power to operate the unit to a minimum and possibilities to operate the mine and still complying with the underground regulations.

Coal washing cost:

The first possible process to avoid is the coal washing or coal beneficiation. In case the mined coal is extracted inside a 1200mm mining height, the hard coal will not be diluted with sediments and does not necessarily have to be processed in a coal washing plant. This will create a <u>reduction of approximately USD 1,30 per ton of coal</u>.

Strata control measurement cost:

Detailed investigations of the underground mining regulations by the author found a possible way to safe money in the area of the strata control measurements, in specific the time frame allowed in between the installation of temporary ground support and the permanent ground support.
72 hours are allowed between the installation of a temporary and the installation of the permanent roof anchors. Incase we mine out a cavity with a height of 1200mm no rib anchors are needed in any case.

Within the time window of 72 hours the mining system has to mine and transport as much coal as possible and leave the mining area behind not to be entered again afterwards. This will avoid the necessity to install the permanent roof anchors and will safe therefore cost.

One meter of mining distance at a cutting height of 1200mm and a cutting width of 5200mm will extract 6,24 m^3 of coal or in other terms approximately 8,74 tons of hard coal.
The cost to install one (1) roof bolt including labor, machine cost and material is estimated at approximately USD 10,50.
Only 60% of the Coal seam can be mined in such a way (the main roadways still have to be secured permanently) so an additional reduction of USD 0,72 per ton of coal is possible.

Labor cost:

The continuous mining machine will need one person to operate and due to the fact that the miner has got the drill rigs on board no additional drill rig crew is needed to operate or relocate the drill/bolting rig.
The machine and the drill/bolting rigs (one each side of the machine) will therefore need three operators to do the work.
The material handling system shall only need one, maximum two operators.

All up the mining system shall not utilize more than 5 workers to operate. No other worker shall be necessary in the sector during the whole mining process.
This shall reduce the labor cost component significantly. The exact factor will be determined in the performance requirements for the mining system.

Chapter 5.5.1 will demonstrate that there is a minimum of 6.5 mine workers needed to operate the system.

It also gives us the fact that those workers will cost us around USD 0,58 per person and ton of hard coal. This will ad up to USD 3,77 per ton of coal.
According to information given by mine operators of different mines the actual labor cost factor per ton of coal averages around USD 5,50 to 6,00 per ton. This is around 36% of the actual overall cost price to produce one ton of coal.
Our savings could therefore be around USD 1,73 per ton of coal.

Combined cost savings:

Coal washing cost savings:	USD	1,30/t
Strata control measurement savings:	USD	0,72/t
Labor cost savings:	USD	1,73/t
Total savings	**USD**	**3,75/t**

This means the production costs of hard coal with the new mining system will be hard at the limits of an acceptable mining system for the mine operator.

Due to only USD 0,25 higher cost per ton of coal compared to the normal mining operation the profitability will be only reduced by 0,72% (USD 35,00 sell, USD 25,00 cost compared to cost of USD 25,25 now) and therefore worth the further development.

4.3. Performance requirements

A continuous mining system has to mine as much as possible out of the hard coal deposit within the allowed time frame of 72 hours. Within the 72 hour time window the mining system also has to retreat out from the temporary secured area and relocate.
This and the contents of this chapter 4.3. is the key element for the success of a new mining system found by the author.

This means the mining system has to avoid downtime due to:

- Intermediate relocation processes
- Waiting periods for material clearance
  - Intermittent shuttle cars
  - Belt conveyor extensions
- defects
- ventilation duct extension time
- re-stocking of roof bolting material

In real life operations, downtime will occur depending on how good the operation was planned.

Additional time has to be accounted for:
- Cutting of the cross tunnels from one roadway to another in a distance of maximum 80m along the length of the mining tunnel for ventilation purposed
- Maintenance to the equipment
  - Change out of cutter picks
  - Oil changes
  - Electrical checks
  - Mechanical checks
  - Maintenance on material handling system
- Repairs
- Stone dusting

- General inspections

The decision had to be made, that due to all the above mentioned reasons the allowed mining/cutting time window will be set at 16 hours with a relocation time of 8 hours – all up 24 hour for one single mining sequence.

The remaining 48 hours will be used for the ventilation cross cuts, maintenance, stone dusting, etc. which should leave enough time in case of unpredictable repairs or other happenings which cannot be predicted.

16 hours are two complete 8 hour work shifts and one 8 hour work shift will be used for the relocation of the machine.

The target was set at a mining rate of <u>200m</u> of hard coal a day through a tunnel of 1200mm high and 5200mm wide. This will result in an extraction volume of 1,248m^3 or approximately 1,747 tons of hard coal per day.
The operation will run 24 hours a day 240 – 260 days a year (Saturdays / Sundays and public holidays excluded) and will produce minimum 400,000 tons of sellable hard coal in average.

The continuous mining machine with the two roof bolting rigs on board will have to be able to mine and temporary secure the area with a speed of minimum 12,5m per operating hour (100m per 8 hour) shift or in other terms 1,0m/4,8min.

The material handling system has to be able to transfer the mined hard coal to the section conveyor without any time delay. A shuttle car transport would result in waiting times and is therefore not suitable. A continuous haulage system has to be developed to receive the cut material on a permanent basis and transports it continuously back to the section belt conveyor and further out of the section to the main underground conveyors.
A mining speed of 12,5m/operating hour will result in a continuous material flow of 78m^3/hour which is very low. Peak rates will be up to 300m^3/hour (approximately 420t/h) which the material handling system will be developed for.
A minimum amount of time is allowed to be used for the extension of the material handling system when following the continuous bolter miner. The time to advance and to extend shall not stop the mining machine from cutting coal at any times.

The mining machine and the material handling system shall be track mounted to be able to relocate self powered with a maximum amount of workers available during the mining process (5 workers).

The whole relocation process from finishing the 200m extraction tunnel back to the initial starting position over to the new starting area for a new extraction tunnel must not take longer than one 8 hour working shift.

All other work mandated by the Code of Federal Regulations, Title 30, Part 75 can and shall be done after the mining system has finalized the retreating process out of the extraction tunnel.
As mentioned before all work has to be completed after a maximum of 72 hours and the extraction tunnel must not be entered at any times after that.

5. Rapid Mining System

5.1. Technical system parameters

The mining machine, the ground support equipment and the material handling equipment have to be considered as one united system, the mining system. Even though the individual components will be addressed individually, the whole package of equipment needed for the production of the available hard coal bed and the speed it needs to produce the coal, the newly developed system will be referred to as **‘Rapid Coal Production System’.**
Even though the individual components of the system are state of the art and generally available on the market, the compiled system of all those individual components, the **‘Rapid Coal Production System’** is a novelty and developed by the author to meet the previously described ambitious targets of a new coal mining system.

Based on the previous investigations it became clear that the actual coal mining process and the ground control installation have to be done simultaneously as there will not be enough time and especially space to do them in sequence.

From the time perspective it is clear that a mining machine with roof drill rigs installed on board can do both operations (cutting and bolting) within the time frame the longest process of the two will take. This brings an instant time advantage.

In regards to the space available in one of the discussed mining tunnels and the fact that the cut out tunnel is only allowed to have a certain length without ground control the mining machine would have to be pulled back to let the bolting equipment pass to install ground control measurements. This means that there has to be enough space available to pull the miner back from the fresh cut mining face and park it whilst the drill rig gets moved to the face to bolt the unsupported ground.

To avoid as much downtime as possible during the time when the mining machine is available for cutting, the material handling system has to clear the mined out material continuously.
The mining machine will start cutting at the starting face of the tunnel and will proceed in a straight line for around 200m. The mined out hard coal has to be transported from the cutting face to the fix installed panel conveyor which then transports the coal via the already available material handling network to the surface.
The transport of the coal from the cutting face to the rear end of the mining machine will have to be done with onboard conveyors on the machine.
The coal will have to be handed over to the material handling system which is continuously following the mining machine through the mining tunnel.
After the 200m long mining tunnel is finished the mining machine including the material handling system have to pull back out of the tunnel and relocate to the next starting face.

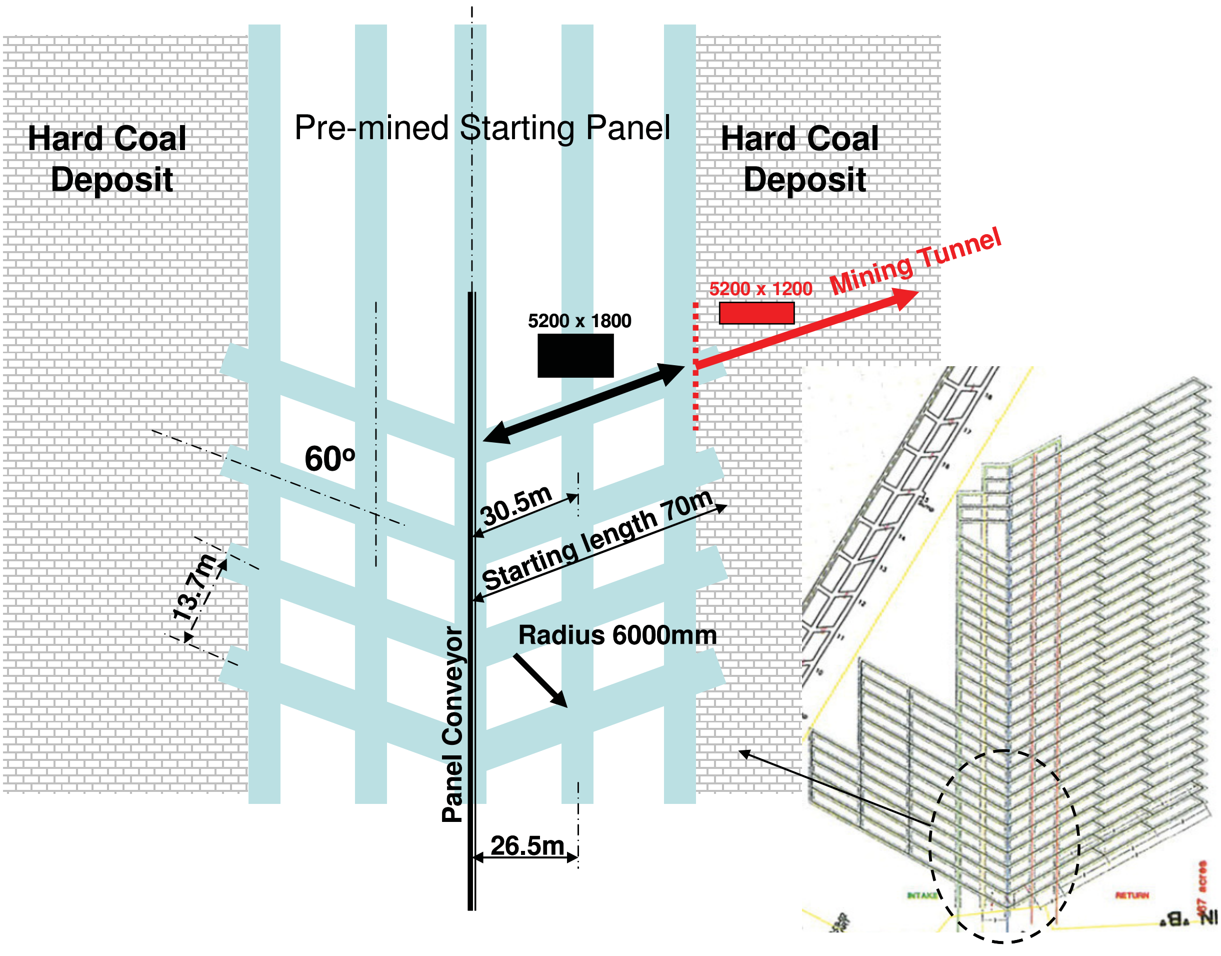
The time frame given for such an operation (start mining > 200m mining > retreating > relocating to new start) is 24 hours or 3 working shifts!

In the following sections the technical configuration of the system will be out lined.

- Dimensional parameters
- Operational parameters
- System components

5.2. Dimensional system parameters

The first step in direction towards a suitable mining system layout, the coal mine operator with their employed geologists had been approached to determine the real and final room and pillar dimensions for the future excavation and mining work (picture 36).



Picture 36: **Final Room and Pillar dimensions**

It was decided that the available hard coal deposit will be mined starting from a pre-mined starting panel which will have permanent ground support installed.
This starting panel will consist of one centre roadway which will accommodate the fix installed panel conveyor to clear the coal out of the panel.

Two parallel roadways along the left over hard coal deposit will be cut for ventilation and transport purposes. The distance between the 5200mm wide and 1800mm high parallel roadways was set at 26.5 meters from centerline to centerline.

Starting from the centre roadway the starting tunnels for the future mining tunnels will be driven in an angle of 60deg, also parallel to each other and in distances of around 13.5 meters to each other. Height and width will be the same as for the centre roadway. The future mining tunnels will be started into the coal block for about 10 meters and already cut with the allowed excavation height of maximum 1200mm.

All corners will be cut by a mining machine with a radius of maximum 6000mm.

The above shown picture of the determined underground starting panel illustrates the combined dimensions given by the mine representatives.

Based on the performed investigation, calculations and research work the mining system has to fulfill the following dimensional parameters.

- Coal mining system has to be able to operate within a cutting height of 1200mm
- Coal mining system has to be able to operate within a width range of 5200mm
- Roof bolting equipment has to be able to install 1200mm long chemical roof bolts
- Coal mining system and roof bolting equipment (in case they are individual machines) has to be able to pass each other within tunnel to secure ground at any time and any location
- Material handling system behind coal cutting system has to be able to operate within 1200mm height
- Mining machine and material handling system has to operate independently
- Material handling system has to be able to reach 0 to 200m (incl. the coal cutting system)
- Mining and materials handling system has to be able to move around 90 deg corners with a maximum of 6000mm radius in the corners
- Mining / cutting side of the overall system has to operate within a roof height of 1200mm and a roadway width of 5200mm
- Discharge end of the overall system has to be able to operate within a roof height of 1800mm and a roadway width of 5200mm
- 500mm dia ventilation tube has to fit beside the mining system
- Mining system has to allow at least 1000mm space each side for accessibility
- A minimum 100mm clearance at the roof (1200mm) in the fully collapsed overall height of the system shall be maintained
- Starting length of the mining system shall be at the maximum 70m

5.3. Operational system parameters

Besides the dimensional parameters of the "Rapid Coal Production System" it is also essential to determine the operational parameters of the system to be able to select the right components for it at the end.

We remember the already worked out requirements and complement them to be able to achieve the necessary production results:

- High safety
- One 200m long excavation tunnel has to be finalized within 16 hours (two working shifts)
- One meter excavation done within 2-3 minutes
- One row (two off) 1200mm long roof bolts installed within 4-5 minutes
- Combined cutting and bolting of one meter mining distance within 4 to 5 minutes
- Minimum amount of operators necessary to run the system
- Continuous material clearance from the cutting face to the panel conveyor
- Maximum 500m^3/operating hour material flow
- Mining system has to advance continuously
- Mining machine and material handling system work independently
- All necessary equipment for the operation on board
- Mining system has to be relocate able within 8 hours (one working shift)
- From starting cutting of one mining tunnel to the start of the next within a time frame of 24 hours (three working shifts)
- Mining system has to retreat and to relocate with onboard equipment
- Maintenance friendly

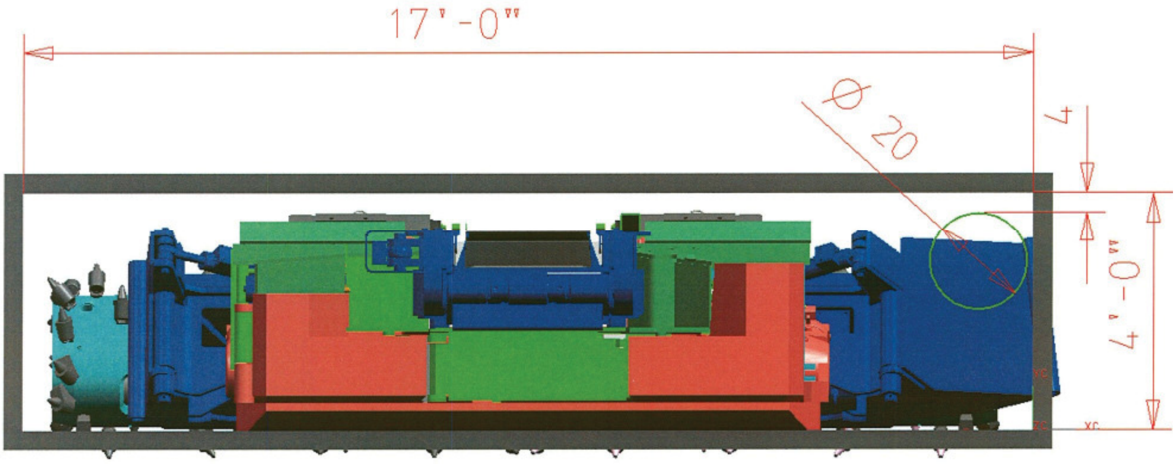
5.4. Rapid Coal Production System component selection

With the above worked out overall system parameters in mind the selection of the components can be started.

The mining machine is the first component in the chain. The next decision will have to be made in regards to the material handling equipment.
During this selection process the system approach must not be forgotten to be able to fulfill all necessary parameters

5.4.1. Mining Machine

To start with a mining tunnel with the minimum available height and width dimension is drawn and a real on the market available machine with only minor adaptation done to it will be drawn into the picture to visualize the dimensional relations inside such a mining tunnel (picture 37).



Picture 37: **Mining machine in mining tunnel 5180mm x 1219mm (W x H), 500mm diameter vent tube**

<u>The market research has shown two available systems</u>

- **Continuous Miner** (picture 38) **plus independent bolting rig** (picture 39)

Picture 38: **DBT Continuous Miner 25M0**, source: www.dbt.de

Picture 39: **JOY RAMTRAC**, source: www.joy.com

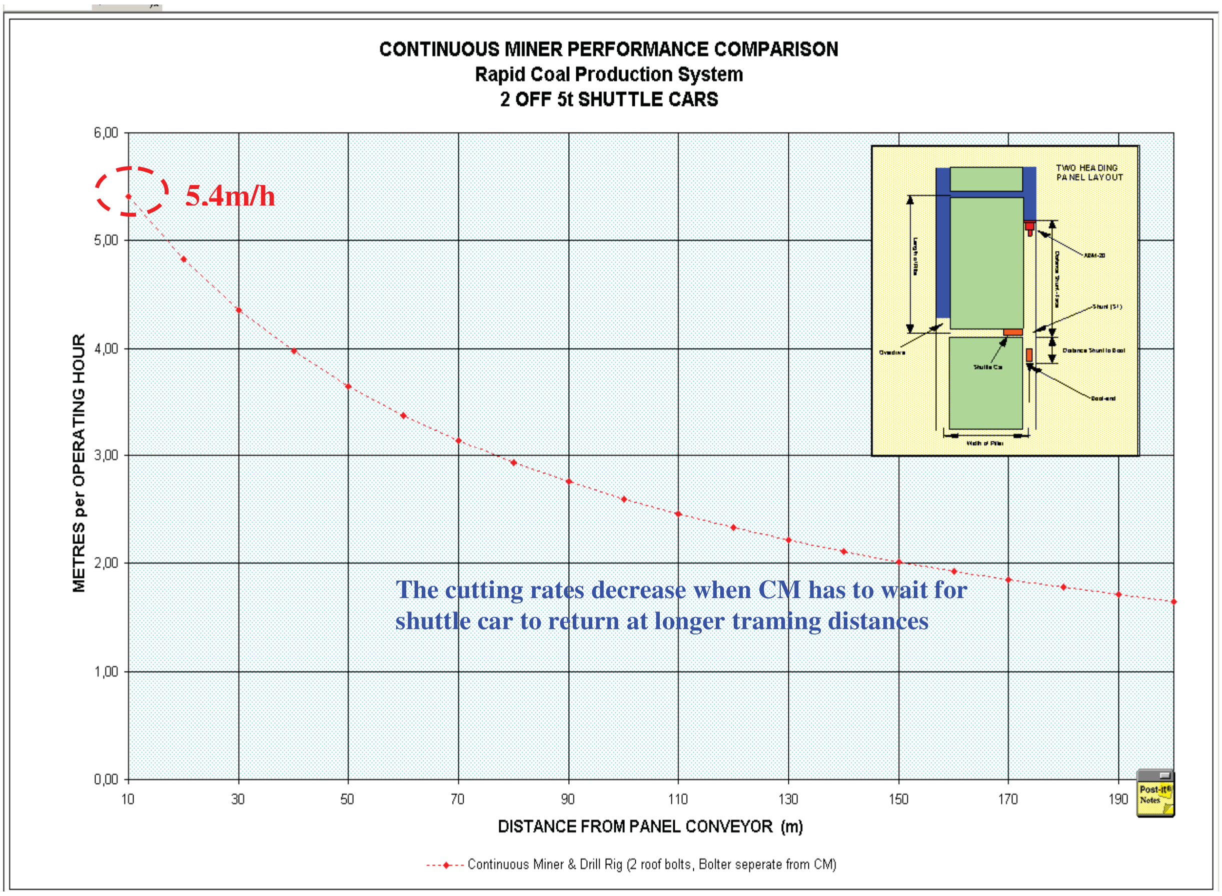


Table 15: **Continuous Miner Performance**

The chart in table 15 clearly shows that the continuous miner with an individual bolting rig will only be good for an advance / cutting rate of about 5,4meters per operating hour.

The above shown graph shows the machine with an intermittent working material clearing system switched behind, so only the highest 'meter per operating hour' shows the capability of the machine and the bolting rig together.

- **Continuous Miner with bolting rigs on board**

The only chance to get the mining sequences as well as the strata control measurements done in the required time frame is the mining machine to be able to cut and roof bolt at the same time.

This is possible with a so called 'Bolter Miner' where the mining machine has got he roof drill rigs mounted on board. This avoids the permanent change out of cutting machine and bolting machine at the mining face to be able to proceed in a long straight line and not exceed the maximum distance the mined out tunnel is left without ground support.

To further reduce the time the mining machine is not able to cut coal (during the roof bolting process where the bolting rigs must not move in any direction) we are looking for a miner which can operate the coal cutting components and the drill rigs at the same time.
This enables the mining system to utilize the available time as effective as possible and comes therefore as close as possible to a continuous mining process.

Such a continuous bolter miner was found in the ABM10 Alpine Bolter Miner manufactured by the mining equipment manufacturer Sandvik AB.

This machine has got the roof bolting equipment mounted independently to the coal cutting and loading components in one and the same machine frame.

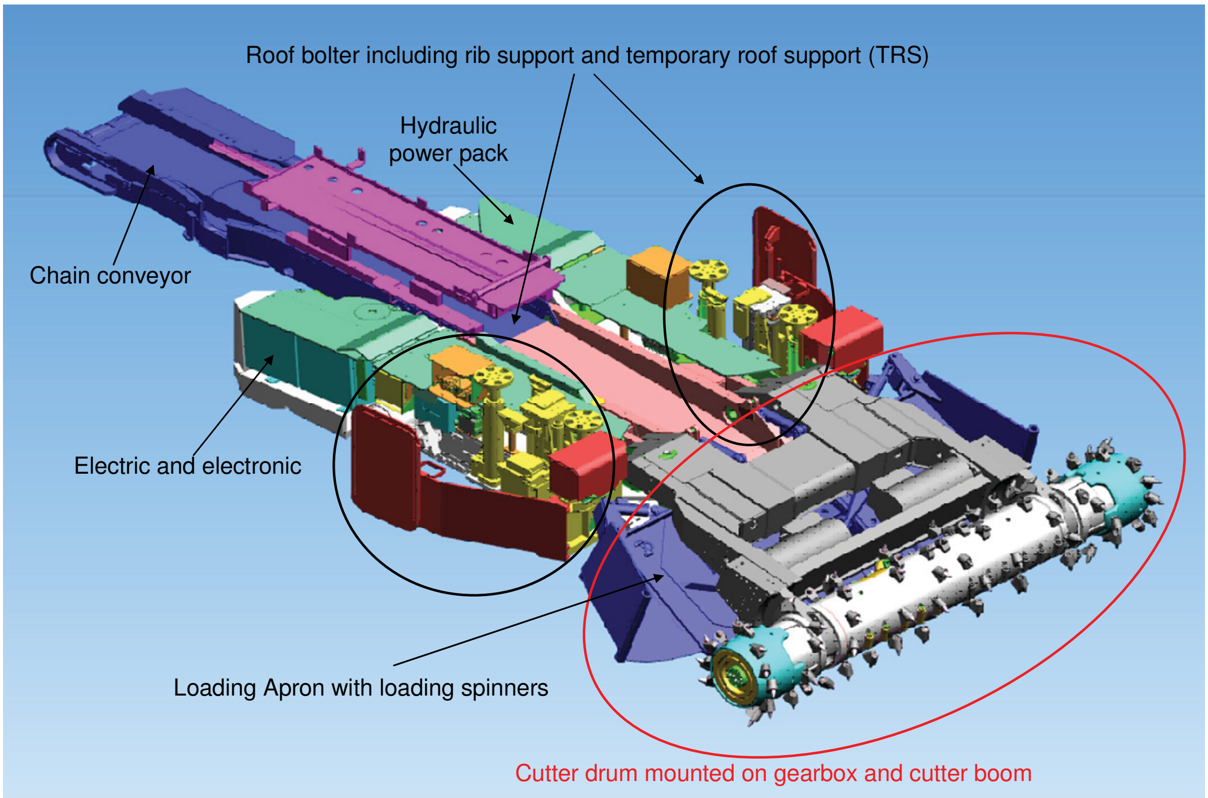
The cutting and loading components, in specific

- Cutter drum including motor and gearbox
- Cutter boom including shearing cylinders
- Loading apron including loading spinners
- Chain conveyor from the apron to the rear of the machine

are mounted on a sliding frame inside the machine frame and are moved forwards and backwards via a summing cylinder to perform the cutting action.

The roof drill rigs, one on the left hand side and one on the right hand side are mounted on the machine frame which is standing still and stabilized during the cutting and bolting action. The onboard electric and hydraulic system is supplying the cutting and loading components as well as the drill rigs at the same time.

Also part of the roof bolting unit are two TRS units (temporary roof supports) which are a safety device holding the roof up during the bolting action and active rib supports. The rib supports are also a safety device protecting the bolting operators and miner driver from material breaking out of the rib during the time they are standing in the vicinity.



Picture 40: **ABM10 Alpine Bolter Miner**

SPECIFICATIONS (table16): **ABM10 Alpine Bolter Miner** (picture 40)

The ABM10 is designed for rapid entry roadway development and coal production in low to mid seams reaching from as low as 1.2m and up to approx 2.3m.

The ABM10 is designed with a sumping head and loading pan mounted on a slide to enable simultaneous cutting and bolting. Bolter stations are located on both sides of the machine directly behind the loading pan for maximum operator comfort and safety. This also enables roof bolts to be installed very close to the face providing a more stable roof.

Two support units brace the miner between the roof and floor, thus stabilizing the machine against reaction forces during the cutting and bolting process.

The emphasis of this new development is the integration of well proven technology into one machine which gives better advance rates under tough mining and restricted space conditions.

Standard features available on the **ALPINE BOLTER MINER ABM10**:

- Simultaneous cutting and bolting operation.
- Integrated high pressure sprays.

- Low rotation speed of the cutter drum and high installed power, in order to reduce dust development to a minimum.
- Electronic controlled sump and shear movement to optimise the cutting sequence as well as the work of the cutter motor.
- Radio remote control system for all functions, except the bolter controls.
- Low ground pressure of crawler tracks.
- No movement of crawler tracks during sump process (thus no damage to floor in soft ground conditions).
- Fully supported machine to give stability for cutting.
- Fully automated greasing system for low service requirements.

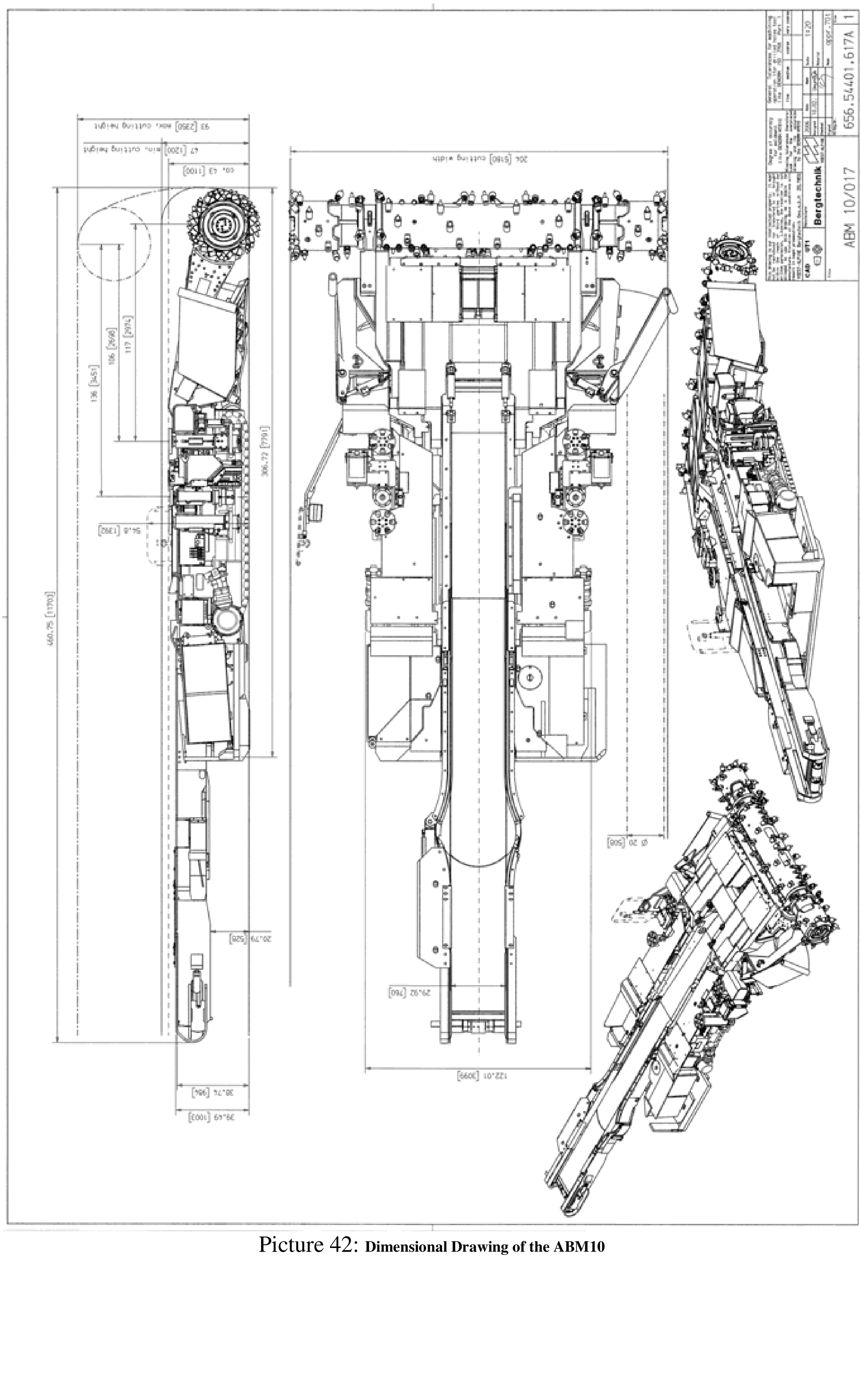
General				
Total length	**[mm]**	**11703**	**[ft]**	**38.4**
Total width for transport	**[mm]**	**3400**	**[ft]**	**11.15**
Total height for transport	**[mm]**	**900 (1000)**	**[ft]**	**2.95 (3.28)**
Total weight, approx.	**[mt]**	**68**	**[tons]**	**75**
Total height in operation over ATRS	**[mm]**	**1100**	**[ft]**	**3.61**
Width of the main frame max.	**[mm]**	**2260**	**[ft]**	**7.41**
Ground clearance	**[mm]**	**175**	**[in]**	**7**
Cutter System – Wide Head Drum Miner				
Working height	**[mm]**	**1200 – 2300**	**[ft]**	**4 – 7.5**
Working width	**[mm]**	**5180**	**[ft]**	**17**
Drum retraction, hydraulically operated, approximate	**[mm]**	**250**	**[in]**	**10**
Drum diameter	**[mm]**	**1000**	**[in]**	**39.4**
Drum speed	**[rpm]**	**41 (2.1 m/sec)(6.8 ft/sec)**		
Sump distance	**[mm]**	**1525**	**[ft]**	**5**
Pick spacing		**Customer specific**		
Installed power	**[kW]**	**2x150**	**[hp]**	**2x201**
Bolting System				
Distance roof support to face (approximate)	**[mm]**	**3000**	**[ft]**	**9.84**
Distance drill rig to face (approximate)	**[mm]**	**3500**	**[ft]**	**11.5**
No. of bolts/cycle			**2**	
Drilling, dry drilling			**Rotating**	

Tightening torque	[Nm]	270 – 380	[lbft]	200-280
Drilling thrust, approximate	[kN]	40	[lbft]	8600
Support ATRS System				
Total force supplied to the roof: variable	[kN]	4x110	[lbft]	4x24728
Loading Device W/2 Conventional Spinners				
Loading capacity	[mt/min]	20	[st/min]	22
Loading width	[m]	5280	[ft]	17.32
Installed power (electric)	[kW]	100	[hp]	134
Retraction each side	[mm]	400	[in]	15.75
Position of loading device above ground level	[mm]	200	[in]	7.87
Position of loading device below ground level	[mm]	100	[in]	3.93
Conveying System				
Conveyor, width	[mm]	760	[in]	30
Conveyor chain speed	[m/sec]	2.2	[f/min]	433
Conveyor capacity	[mt/min]	20	[st/min]	22
Conveyor overhang	[mm]	3912	[ft]	12.83
Discharge height from floor level, (min)	[mm]	500	[ft]	1.64
Discharge height from floor level, (max)	[mm]	1500	[ft]	4.92
Heavy duty chain	[mm]	83	[in]	3.25
Tram System				
Tram, 3 speeds	[m/min]	5 to 20	[f/min]	16 to 65
Ground clearance	[mm]	175	[in]	7
Ground pressure operating	[N/cm²]	25	[psi]	19
Width of track chain	[mm]	530	[in]	21
Dust Suppression System				
Dust collection on board			No	
Water spray system			Yes	
Hydraulic System				
Power	[kW]	100	[hp]	134
Pressure, max.	[bar]	250	[psi]	3626

Cooling System				
Open cooling system with water return		**Yes**		
Electrical System				
Electric potential	**[V]/[Hz]/[A]**		**1000/60/425**	
Total installed power	**[kW]**	**500**	**[hp]**	**670**
Cutter drive	**[kW]**	**2x150**	**[hp]**	**2x201**
Hydraulic drive	**[kW]**	**100**	**[hp]**	**134**
Loader and conveyor drive	**[kW]**	**100**	**[hp]**	**134**
Radio remote control		**Yes**		

Table 16: **Technical Specifications of the ABM10**

Picture 41 (also compare to picture 40): **ABM10 Alpine Bolter Miner**



Picture 42: **Dimensional Drawing of the ABM10**

5.4.2. Material Handling

The material handling equipment has to work independently from the ABM10, has to receive the cut material and transport it to the fixed installed panel conveyor. This can be done by different means

- LHD (Load – Haul – Dump) Car
- Shuttle Car
- Conveyor

The LHD and the shuttle car are transporting machines which are running intermittent whilst the conveyor is able to transport material in a continuous way.

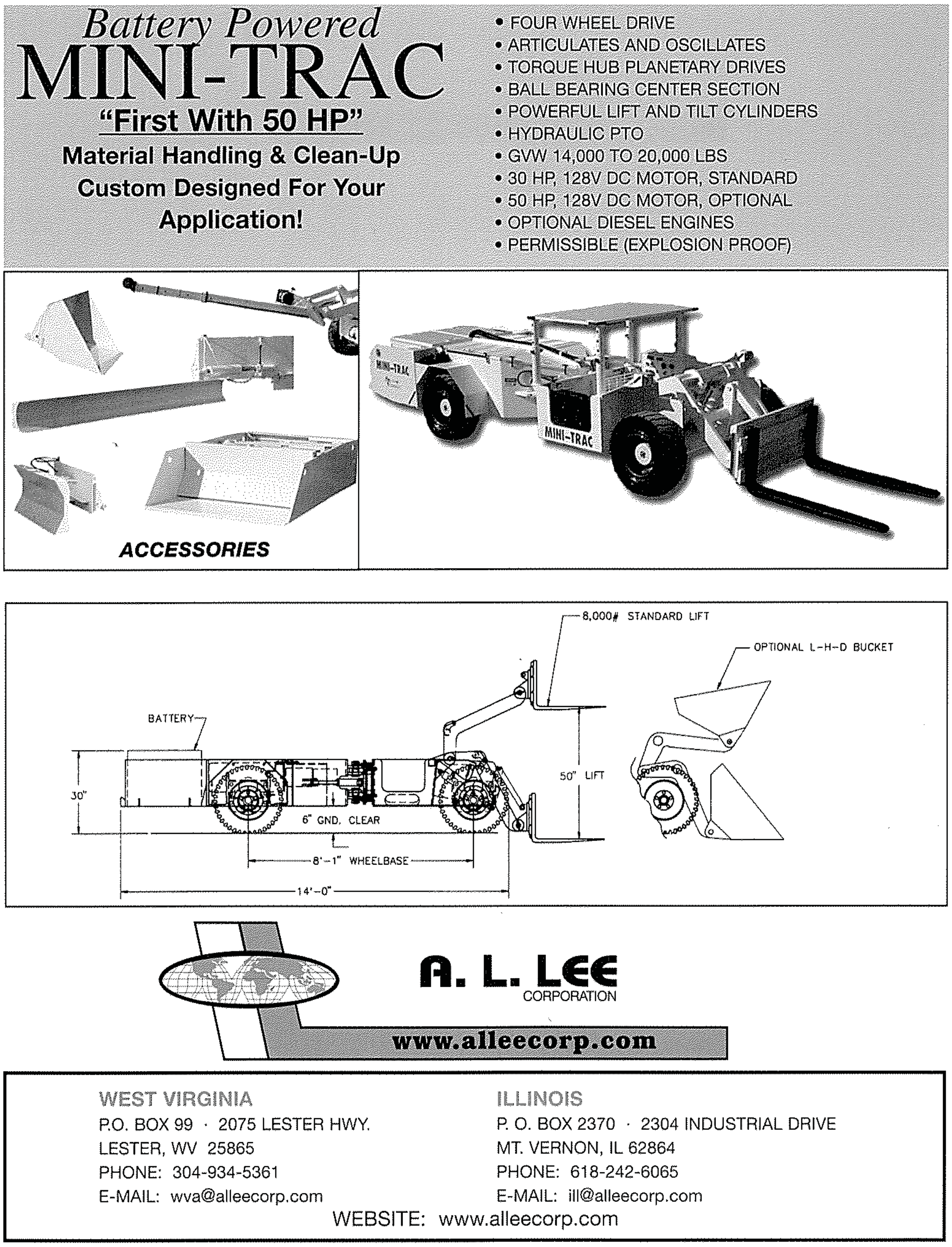
As discussed before some down time due to the intermittent process with LHDs and shuttle cars (compare table 15, cutting performance decrease from 5,4 down to 2,0m/h when transport distance increases from 10 to 150m to the conveyor) can be compensated by using two off LHDs or shuttle cars. One is getting filled by the miner and transports the coal to the panel conveyor. During the transporting time of the first, the second one is getting filled; the downtime is getting shortened but not eliminated.
Keeping the cost factor of the capital investment in mind the first and best option is to use a LHD or a shuttle car for our operation.
The capital costs will be low due to the fact that those machines are serial products of some mining machine suppliers such as SANDVIK, JOY or DBT.

The ABM10 with the average cutting rate of 12.5 m/operating hour will produce an average of $78m^3$ of coal/operating hour which is around 110 tons/hour on an assumed continuous basis.
The real fact is though, that this average production figure also contains the time of roof bolting, relocating and setting up the machine for the new cutting and bolting sequence.
In reality this means for the material handling system a load of around double the material flow to be handled as the 110 tons might be transferred within only 30min.
This means around 220 tons/hour in average on the receiving side of the transporting equipment.
Peak rates, again in reality, will reach around 420t/operating hour though.

To be able to load the car with the ABM10 the car has to be able to stand under the chain conveyor boom of the ABM10 which means that the overall height of the loading shovel of the LHD or the loading area of the shuttle car has to be as low as 500mm.
The remaining machine has to fit into the 1200mm high roadway with enough clearance to the roof to allow for uneven floor conditions during the haulage process.
Also the capacity of the machines has to be large enough to be able to clear the coal from behind the ABM10 in an effective way.

The first option of the use of a **LHD** will be investigated.

A Market research by the author found a machine from the USA based company A.L.Lee Corporation which would have fit the dimensional parameters very well (picture 43).



Picture 43: **MINI-TRAC;** Source: www.alleecorp.com

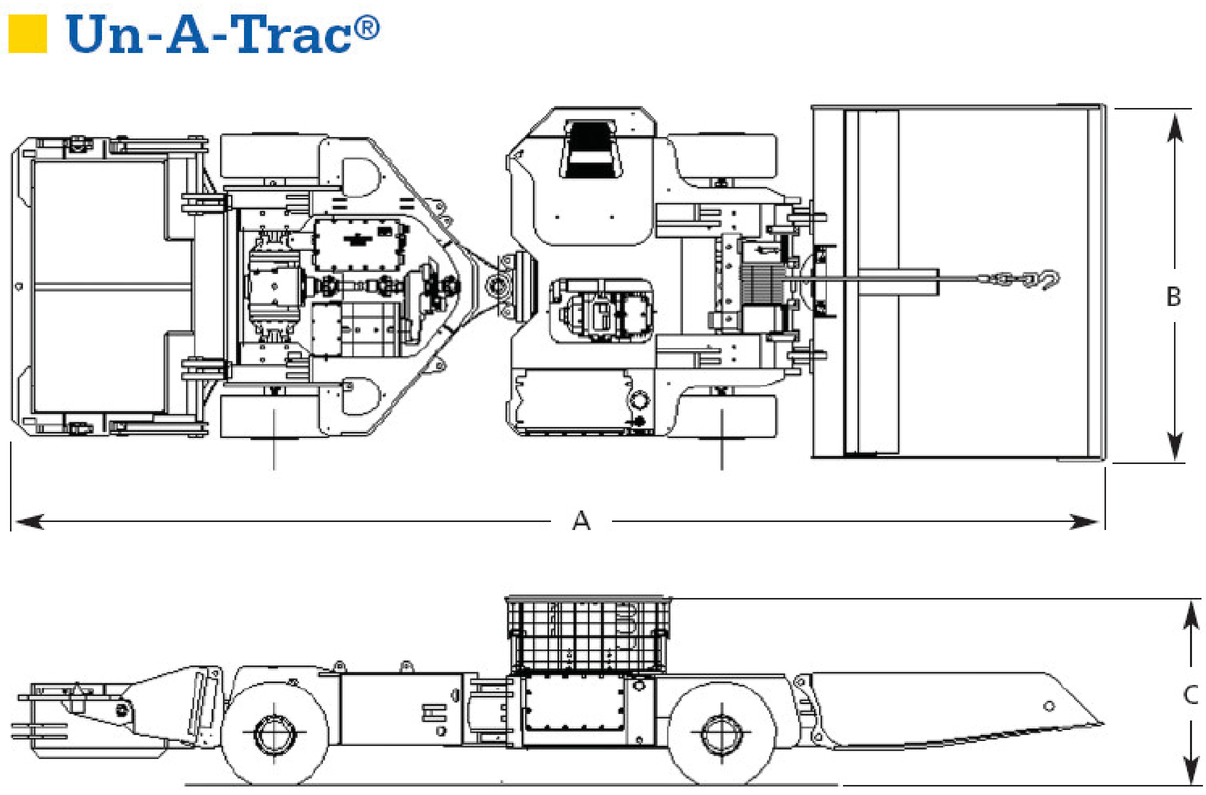
The overall machine height is around 760mm and about 2500mm wide. The machine is articulated via a centre joint which gives it a very good flexibility to adjust to all sorts of drive and loading situations.

The machine scoop will be loaded from the ABM10, the LHD will drive in reverse to the vicinity of the panel conveyor, has to turn around and than discharges onto the panel conveyor.

The scoop in the front of the machine is 500mm high and would have fit nicely underneath the chain conveyor of the ABM10.
The down turning issue with this type of loaders is the capacity of the scoop which will be in the best possible option 1,5m^3 or 2,1tons of hard coal per load.

With a practical average driving speed in the mining tunnel of around 50m/minute, the loader will be able to transport around an average of 25,2tons per operating hour over the 200m long mining tunnel which could be doubled to an theoretical average of 50,4t/hour in case of using two units to reduce the waiting time of the ABM10 for the LHD.

Another option (picture 44) would be the UN-A-TRAC 488 L-DM from DBT (Germany).

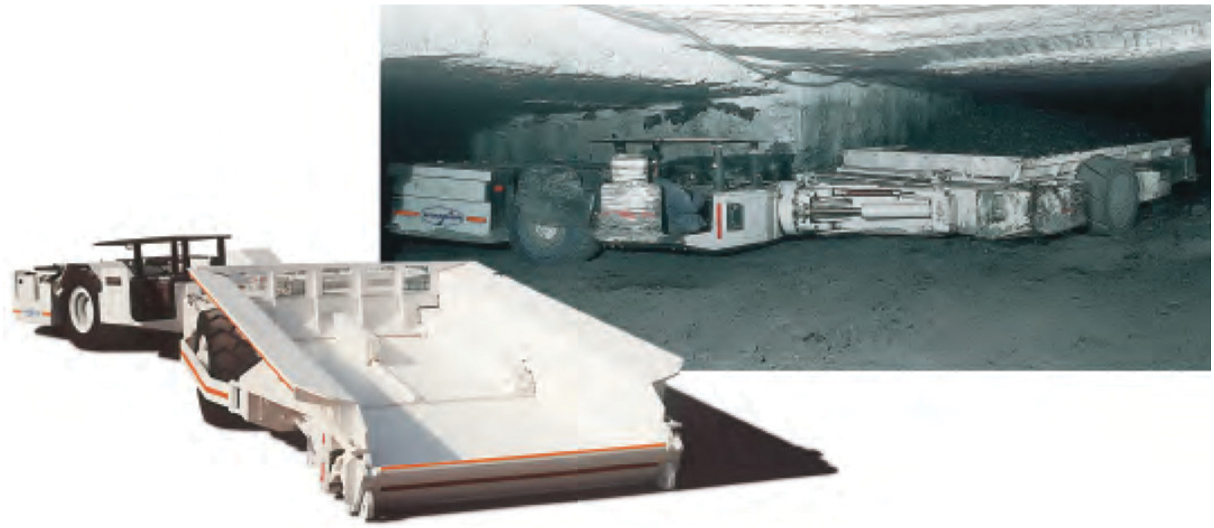


Picture 44: **UN-A-TRAC;** Source: www.dbt.de

With A= 9300mm, B=2900mm and C (adjustment to canopy arrangement)=900mm the resulting pay load nominal around 6,0m^3 would practically be as low as 3.5m^3 or in other terms approximately 5,0tons per load in our conditions.
This would result, when using a two car operation, at around a theoretical average of 110,0 tons per operating hour.

The issue of turning the car around from the loading into the discharging position will be the same for all the LHDs.
The driving or better hauling speed will be same for all cars as the underground road conditions will dictate this maximum speed possible.

A Stamler BH-Battery Hauler (picture 45) would range in about the same performance figures as the DBT machine.



Picture 45: **BH Battery Hauler;** Source: www.joy.com/jmm/products/stamler.html

The performance rate of the LHD in the underground conditions in the left over hard coal deposit will prolong our actual mining time and will therefore shift the time to finish one mining tunnel out of the available time window.
The LHD is therefore not an option.

The second option to investigate is the **Shuttle Car**.

The mining equipment supplier JOY (USA) as well as DBT (Germany) have shuttle cars in their portfolio which would suit the overall dimensions in our underground application in case of minor adjustments done to them.

Shuttle cars depending on the length and width of the car could also load up to 3.5m^3 (approximately 5,0tons) of coal. The advantage of this type of car is that it is slightly more flexible due to the fact that it does not have to turn around when discharging at the panel conveyor side.
With two cars this could bring us into the range of a theoretical average of 120,0tons per operating hour and therefore the same as the LHDs but with a lower transit time (do not have to turn around when discharging).

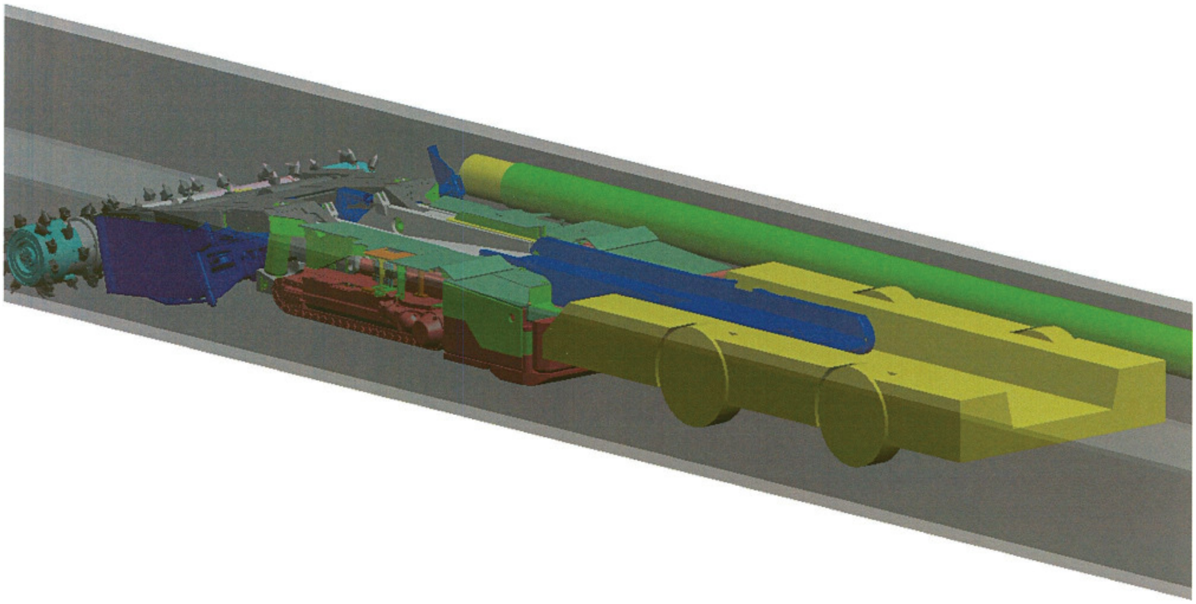
Considering again a two shuttle car operation the down time is still there but slightly lower so the shuttle car seems to be the better solution so far, still not real satisfying therefore further investigations are worthwhile.

Practical investigation in regards to dimensions and material transport performance:

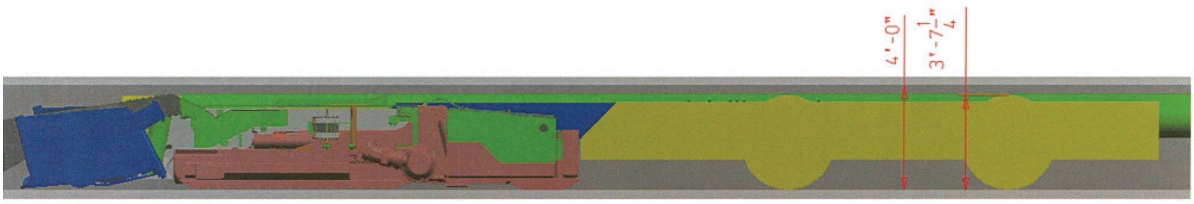
To be able to get practical performance data we have to see the mining machine inclusive the bolting equipment (ABM10) and the material handling system (Shuttle Car) acting together over the distance from 0 to 200m mining tunnel length.

The next step was to create a drawing showing the ABM10 and the shuttle car together in the tightest situation. This is when the shuttle car is getting loaded behind the ABM10.
Also the ventilation tube will be drawn into the picture to see the relations in the right scale.

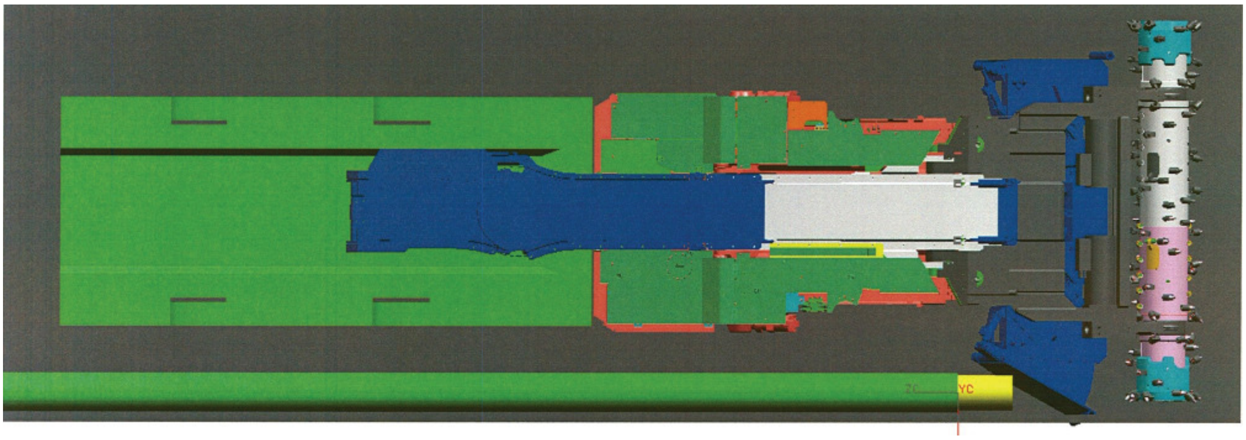
For simplicity purposes the shuttle car will only be drawn in a schematic way but the outside dimensions exactly to scale as shown in picture 46, 47, 48.



Picture 46: **ABM10 with Shuttle, Car 3D view**



Picture 47: **ABM10 with Shuttle Car, side elevation**



Picture 48: **ABM10 with Shuttle Car, top view**

The space filled out by the machines and the ventilation tube and specifically the space left for the operators is very concerning when looking at the operational point of view. No space is left for any decent movements from the operators, not even talking about OH&S.

Before getting into too much trouble changing around dimensional parameters a practical view onto the material handling performances will be done.

Following data will be taken into perspective:

- ABM10 with two roof bolters on board, one each side
- Two chemical roof bolts, 1300mm long, will be installed every meter
- Each drill rig will install one roof bolt per meter
- Roof bolts will be installed in double pass (1300mm roof bolt length versus 1200mm cutting height > drilling with two stages necessary)
- Two shuttle cars with a pay load of 5tons will be used
- Average driving speed of the shuttle car will be around 50m/min
- Loading and discharge time of the shuttle car around 40 seconds
- Overlap of 3m between ABM chain conveyor and HC to give the necessary flexibility

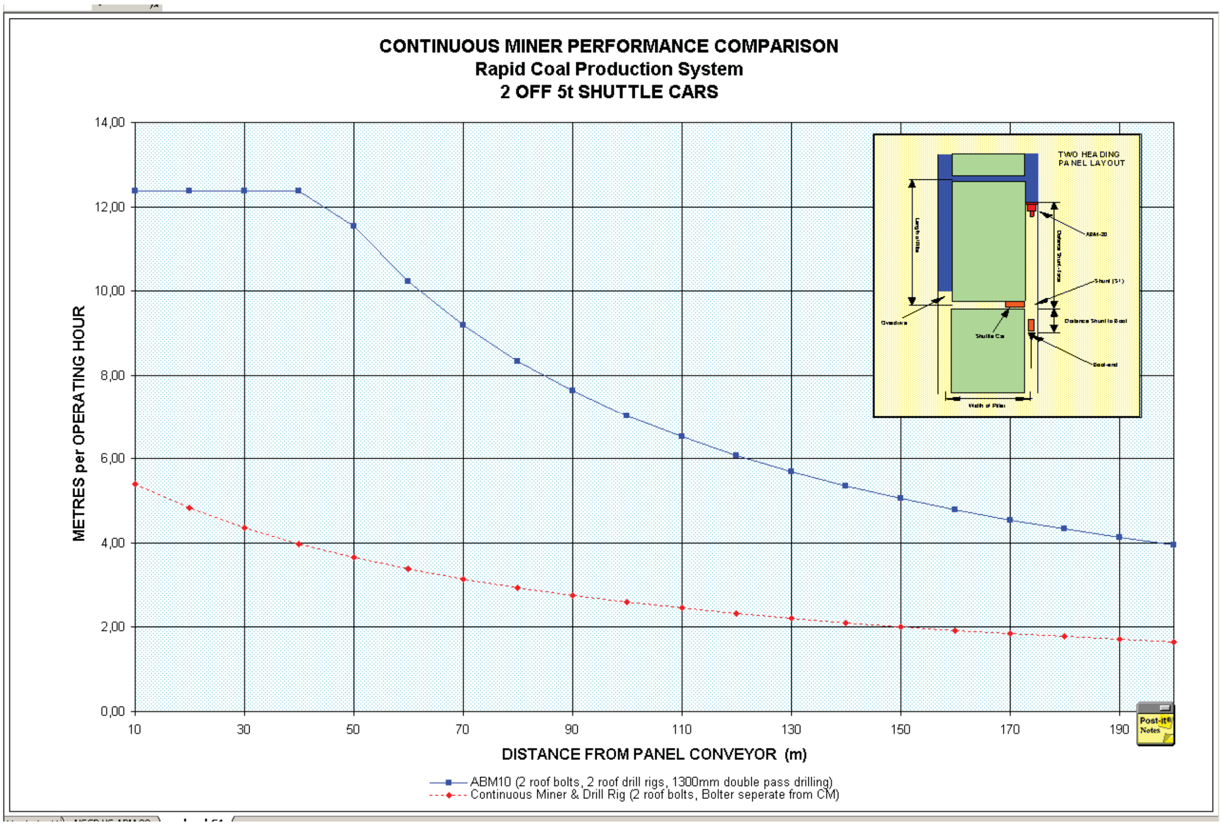


Table 17: **Continuous Miner ABM10 Performance**

The blue graph in table 17 shows the ABM10 and the two 5ton shuttle car working over the mining distance of 0 – 200m.
The ABM10 is, up to the mining distance of approximately 40m, able to cut to its maximum required performance but, due to waiting times for material clearance, will fall down to around 4 meters per operating hour when the machine is at the 190m distance and the shuttle cars have to transfer the material over such a long distance.
A practical average mining performance of around 8meters per operating hour will be the result and this is significantly falling short of the required 12,5 m/hour.

The graph in table 17 also shows the performance figures (red graph, compare to table 15) of a continuous miner together with an individual bolting rig and the combination with the two shuttle cars to be able to compare the ABM10 to it.

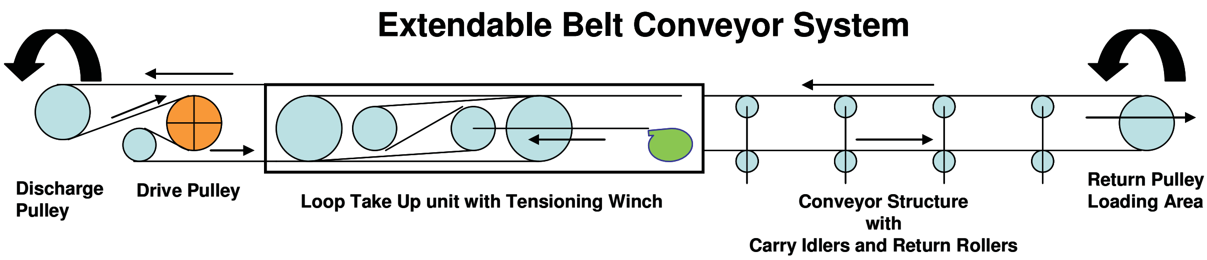
All further investigation around the use of intermittent material handling equipment is therefore not beneficial and the concentration will be aimed at the continuous material handling systems.

Conveyors (continuous material handling systems) have the ability to transport the material from the mining machine to the panel conveyor in a continuous manner; no waiting time is the consequence of that.

Following parameters are essential for the material handling system:

- Continuous advance rate 12.5m/operating hour
- Material handling range 0-190m (ABM10 length subtracted)
- Maximum material flow 420t/hour (300t/hour in case a bunker is used)
- Starting length inclusive ABM10 maximum 70,0m
- Starting tunnel height 1800mm
- Mining tunnel height 1200mm
- Roadway width 5200mm
- Retractable and relocate able within 8 hours
- Corner radii maximum 6000mm

Whilst the material flow of 420tons per hour or in other terms 300m^3/hour is not a problem at all for a conveyor system all other parameters have to be closely looked at.



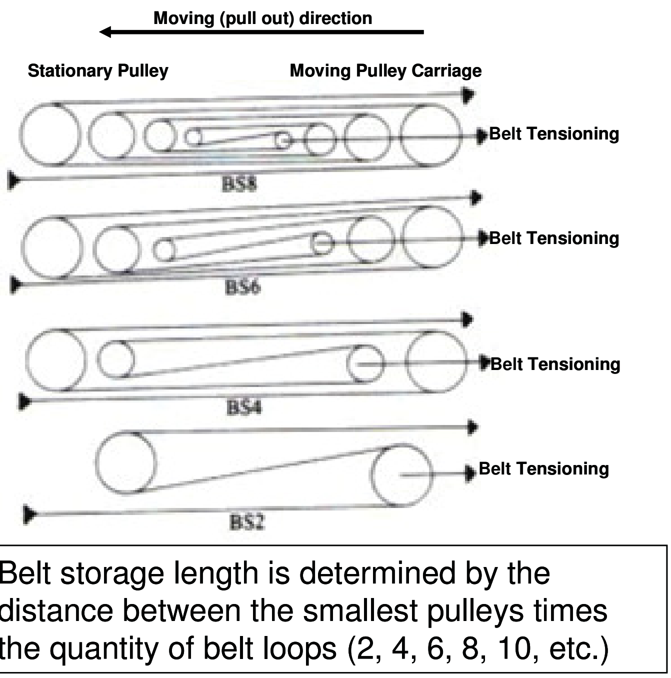
Picture 49: **Extendable Belt Conveyors**

Continuous advance rate:

- Continuous advance rate 12.5m/operating hour
- Material handling range 0-190m (ABM10 length already subtracted)

The only way to have a continuous advance rate or speaking in conveyor terms, a continuous extension rate of 12,5m/hour, is to have a conveyor belt storage in form of a loop at one end of the conveyor and a pulling device at the other end. This pulling device has to pull the conveyor belt out of the loop in a continuous way (see Picture 49).

The loop take up systems in extendable conveyor belt systems are state of the art and are being used in conveyor applications all around the world (picture 50).



Picture 50: **Loop Take Up Systems** Source: www.nepeanconveyors.com/products/belt_storage.html (modified)

The belt for extension distances of up to 300m conveyor length can be stored inside such a loop take up (LTU) unit which only depends on the length of the LTU unit and the quantity of loops employed.
Also the tensioning of the conveyor belt will be done inside this LTU unit.

The return pulley of the conveyor system will be moved forward (in standard cases by underground machines), the belt pulled out of the LTU unit, conveyor structure including the idlers will be added (in normal case this is a manual work) and as long as there is enough belt inside the LTU unit, the conveyor can be extended.

The LTU unit is able to provide us with the necessary flexibility in regards to mining distance/extension range from 0 to 190m. For our application we have to store 380 meters of conveyor belt inside the LTU which contains a problem when considering the 70,0m maximum starting length of the mining system (including the ABM10). This issue will be discussed at a later stage.

The LTU unit will also contain a belt tensioning device. Due to the fact that a gravity tensioning device is out of question for extendable conveyors (permanently changing length of robe which is holding the tensioning weight) a winch system will be taken into consideration.

Picture 51: **Electrical Robe Tensioning Winch;** Source: www.nepeanconveyors.com/products/winches.html

The typical winch (picture 51) consists of a drum where the tensioning wire robe is spooled onto. The drum is attached to a hydraulic or electric motor. Via this motor and a gearbox the tensioning torque is applied to the drum and subsequently a tensioning force is applied onto the conveyor belt by the wire robe which is attached to the moving carriage of the LTU.

As mentioned before the belt conveyor system gets extended when the conveyor belt is pulled out of the LTU because of the return pulley getting moved forward and conveyor structure including carrying idlers and return roller is added in.
This work involves lifting gear in form of forklifts and chain blocks as well as transporting machinery to move the return pulley structure forward and the deliver the conveyor structure to the adding position.
It also involves at least three mine workers for running the machines as well as adding the structure in manually.
Further more this process takes quite some time, in specific around 1/2 hour per meter advance distance assuming all components are at the right spot and fit without problem.

For our application 12,5m/hour advance rate means 4.8 minutes would be available per meter conveyor for the extension!

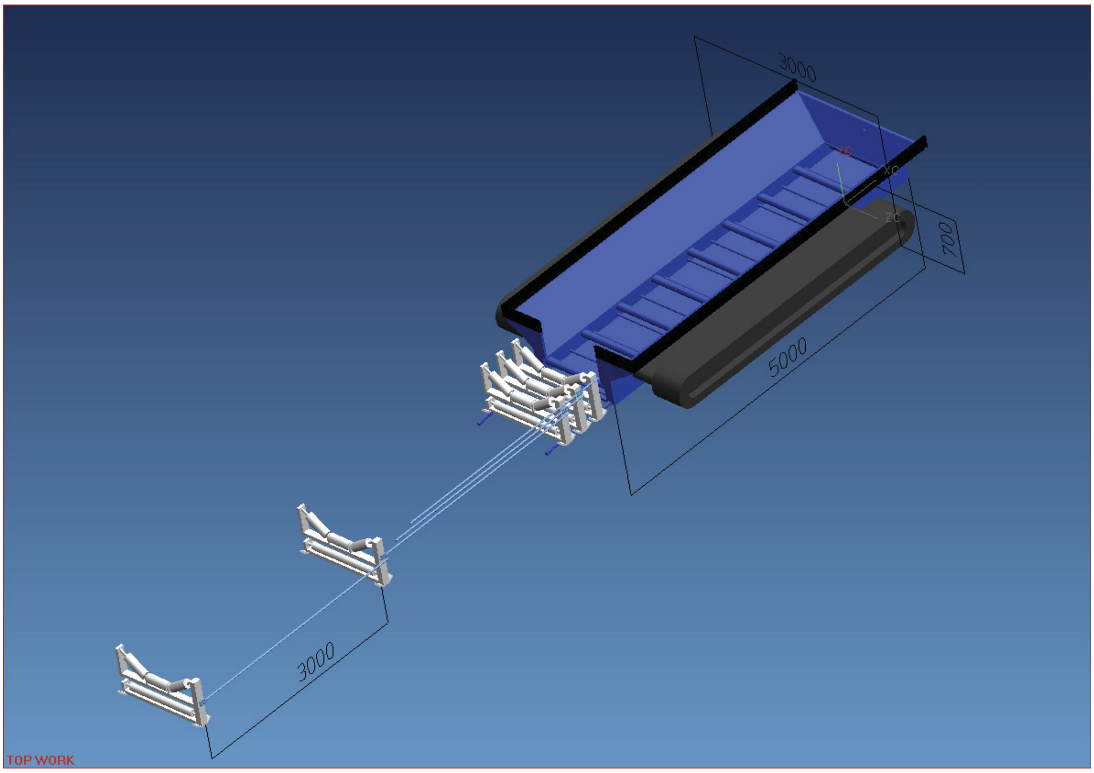
The labor consumption and specifically the time to conservatively advance one meter into the mining tunnel is not acceptable for the application on hand. Another factor is the manual handling of components in a tunnel height of 1200mm and an available width of possibly only 1000mm is almost impossible to do over a longer period of time.

The advancement of the return pulley (loading station) and the adding in of conveyor structure has to become simple, quick and if possible automatic.

The only way thought of by the author was to move the return pulley/loading area automatically is to mount the whole structure accommodating the receiving chute and the return pulley on tracks. The tracks will have to move the structure, pull the belt out of the LTU unit and as well act as the counter weight to the conveyor belt tension applied by the winch on the other end of the conveyor.

Taking the shuttle car dimensions as a first guide line for the dimensions of such a moving device, in the future call Hopper Car, a rough lay out was done.

5,0m long, 700mm high and with 3000mm width the same as the ABM10 the first sketches will be created.



Picture 52: **First Hopper Car lay-out**

Storing the conveyor structure including the idlers inside this Hopper Car (HC) would save the logistics around the transport of such conveyor structure and in case a way could be found to have the stored conveyor structure pulled out of the HC whilst it advances, also the adding in problem of the conveyor structure could be solved (picture 52).

Every 3,0-3,5m one conveyor structure assembly has to be added in. This adds up to around 60 structure assemblies over the full mining distance which has to be stored inside the HC.

Also the maximum material flow of 420t/hour can be reduced to 300t/hour because of the ability of the HC to bunker the absolute peak loads coming from the ABM10.

Dimensional checks of the allowed maximum height of the hopper car show that the space available below the ABM10 chain conveyor boom is only around 700 to 800mm. The hopper car will be loaded by the chain conveyor boom and therefore the hopper car has to fit underneath it.

Overall dimensions of the system:

- Starting length inclusive ABM10 maximum 70,0m
- Starting tunnel height 1800mm
- Mining tunnel height 1200mm
- Roadway width 5200mm

The Hopper Car will always operate within the 1200mm mining tunnels and all dimensions for getting loaded by the ABM10 as well as the advance mechanism, storage areas, conveyor structure, etc. have to allow for those conditions.

The problematic of the dimensional criteria sits in the 70,0m starting length, the 0-190m advance range and the 1800mm starting tunnel height.
The LTU unit has to fit into those dimensions by still giving enough clearance to the roof to convey material over the top.
The Hopper Car will most likely be around 8,0m long, the ABM10 has got a length of 13,0m.
The conveyor belt has to be brought from the around 500-700mm high conveyor structure assemblies in the tunnel up to the top of the LTU unit which will take about 6,0metres (gradient structure) when sticking to the maximum possible 15deg inclination angle of a conveyor.
Also the drive and discharge area of the conveyor with approximately 6,0meters will have to be covered within those dimensions.

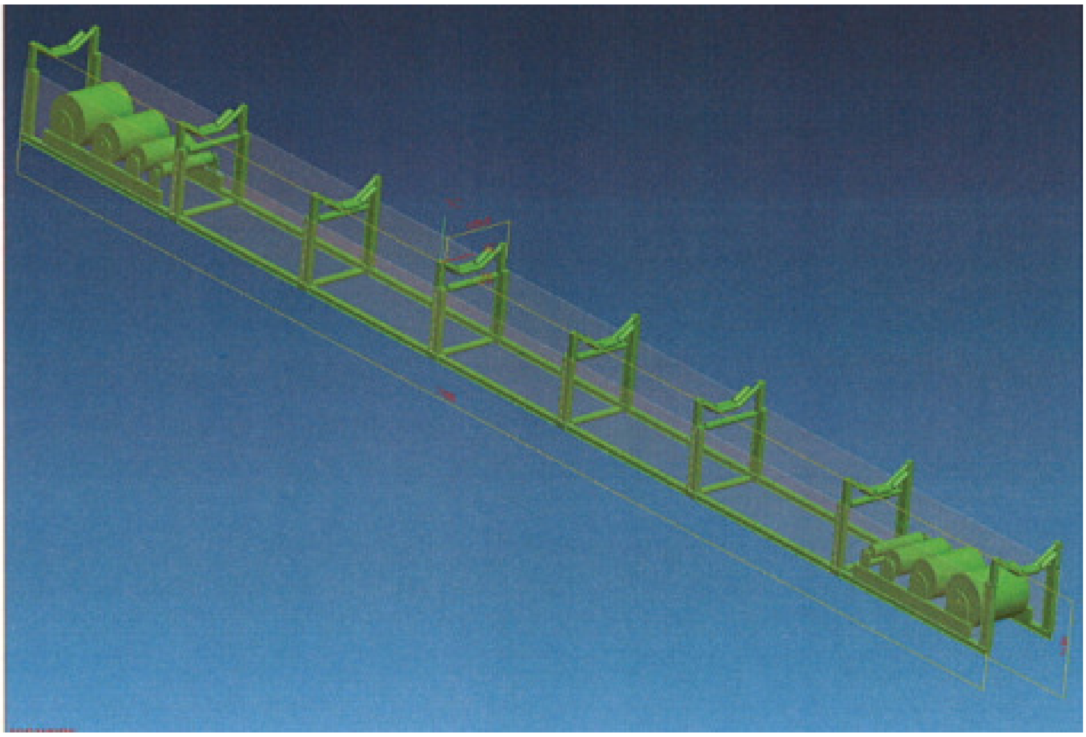
This will leave us approximately 37,0m for the LTU / Drive / Discharge unit including the space we will need to move the components.
The LTU unit should therefore fit within 30,0m and according to previous estimations has to store 368m (380m minus the length inside the Hopper Car which is 2x6,0m) of conveyor belt inside.

Due to the fact that material has to be conveyed over the top of the LTU unit and still have some clearance to the roof the height of the LTU has to stay below 1600mm.

A conveyor belt width of 1200mm will be taken into account for the basic estimation. 1200mm wide belts are normally used for material flows up to 500m^3/hour @ standard conveyor speeds of around 2,0m/sec.

In case we will find space to accommodate around 5 LTU pulleys each side the storable length will get up to around 200+ meters inside the LTU. Still around 150+ short of what we need to reach the 200m mining tunnel length.
Belt will have to be added in somehow during the process and dimensional precautions have to be taken into account for that as well.

A basic layout of the LTU unit can be done with what we know so far and is shown in picture 53.



Picture 53: **Basic LTU lay-out**

<u>Retraction and relocation of the system:</u>

- Retractable and relocate able within 8 hours
- Roadway width 5200mm
- Corner radii maximum 6000mm

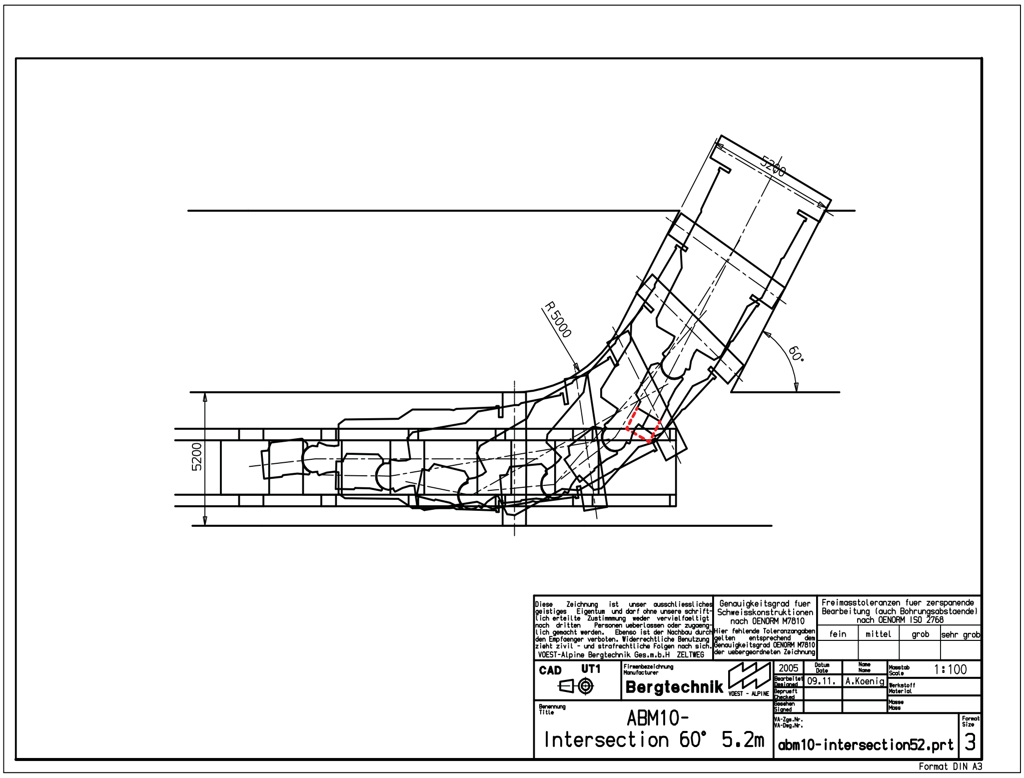
After reaching the end of the mining tunnel the ABM10 and the following Hopper Car have to retracted out of the tunnel and relocate to another starting tunnel within an 8hour working shift.

When reversing the HC out of the tunnel the conveyor structure will have to be collected back up into the storage area of the HC and the conveyor belt surplus developing shall be taken up by the LTU unit via the tensioning winch.

Outside the mining tunnel the ABM10 can move on its own and relocate to the next starting position independently to the rest of the system.

The ABM10 is able to cut and also drive around corners below the 6000mm radius so the relocation of the machine will not be an issue.

A typical corner sequence of the ABM10 is shown in picture 54 below.



Picture 54: **ABM10 minimum cutting/turning radius**

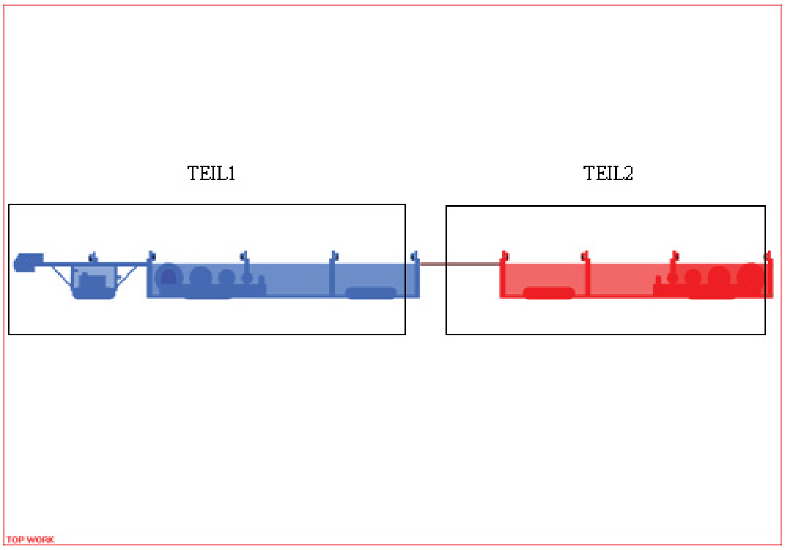
The conveyor system reveals the challenge.

Whilst the Hopper Car will be able to move around corners via the tracks, the around 30+meter long LTU unit including the drive and discharge are physically too long to negotiate around corners present in our underground application.

When looking at the starting tunnel dimensions and the roadway width of 5200mm components with a length of around 13, 0-14,0m would be able to move around corners with the 60deg corner angle and 6000mm corner radii.
One of the major problems is the fact that the conveyor belt will always have to stay inside the system as the removal of the belt would spoil the timing of the relocation process completely.

The LTU unit including the drive/discharge station could be broken up into two 14,0m long pieces still connected in the centre via the conveyor belt which has to stay inside the LTU unit during the relocation process. The belt inside is flexible and might allow the broken up unit to maneuver around the corners without the belt getting damaged. The following pictures will demonstrate the difficulties of such maneuvers as well as the emphasis on a maximum opening angle between the two pieces to avoid damaging the belt.

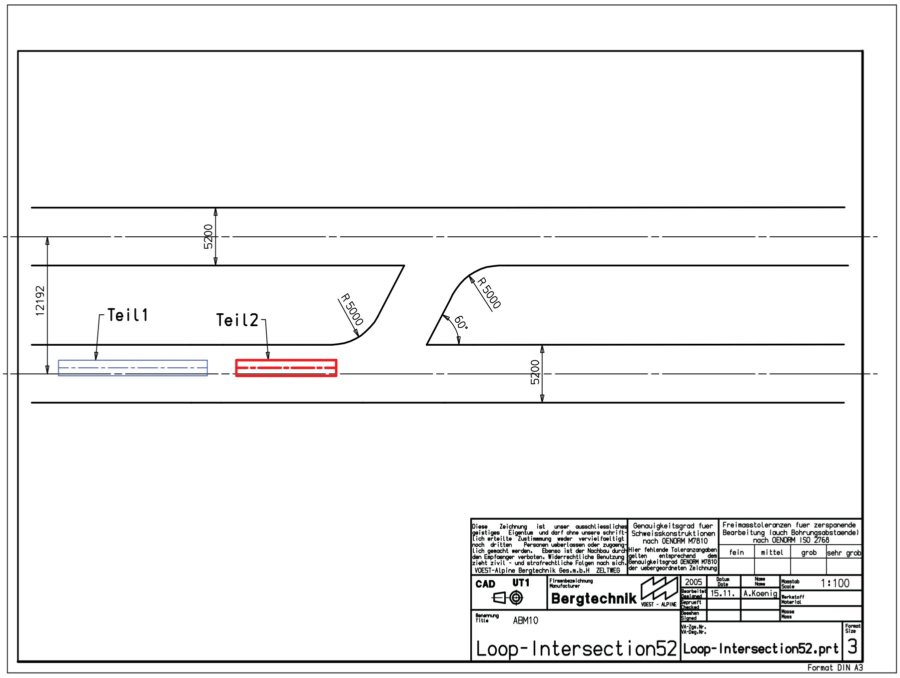
The complete rear area will be divided into to two parts, one consisting of the drive and discharge area and half of the LTU unit, the second one the second half of the LTU unit.



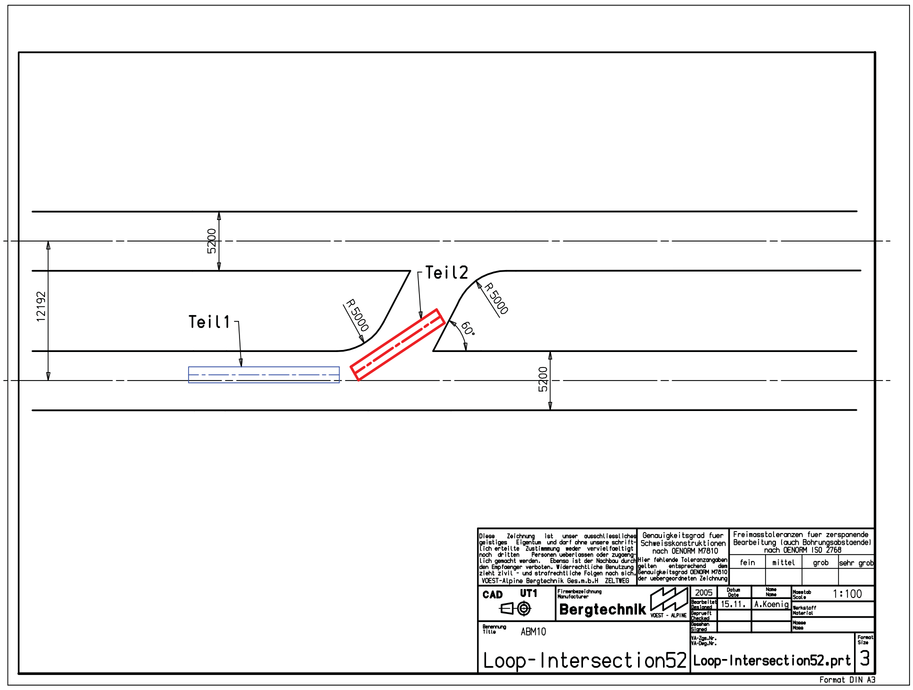
Picture 55: **possible split up of the LTU Unit**

The whole unit, drive and discharge station (Part 1) together with the LTU unit (Part 2) must therefore not be longer than roughly 28,0m in total. This will reduce the LTU unit length and subsequently the stored conveyor belt length inside the LTU unit (see picture 55).

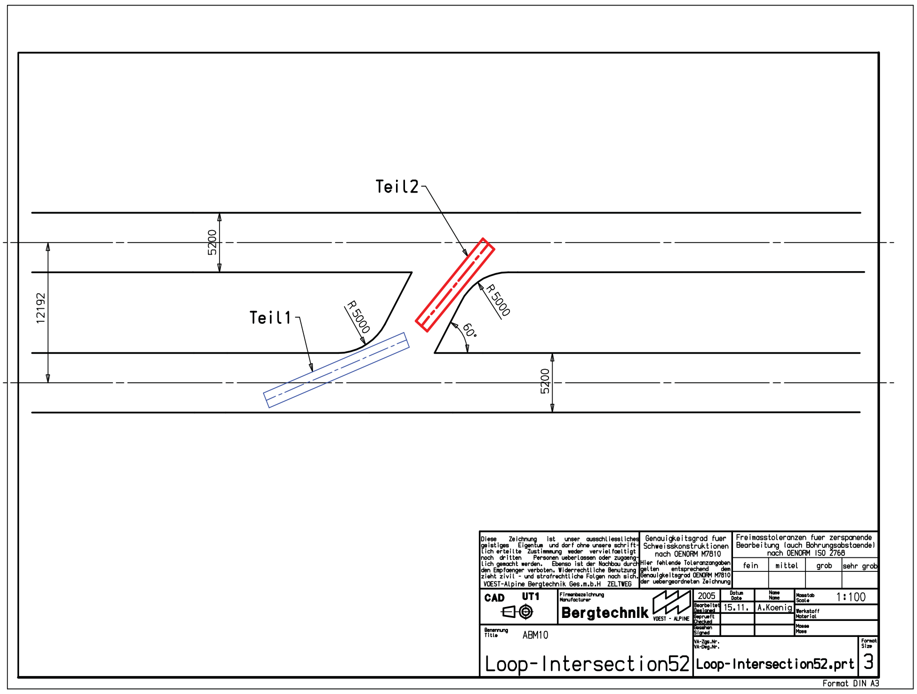
A preliminary relocation sequence was investigated and visualized in the following pictures 56 - 62:



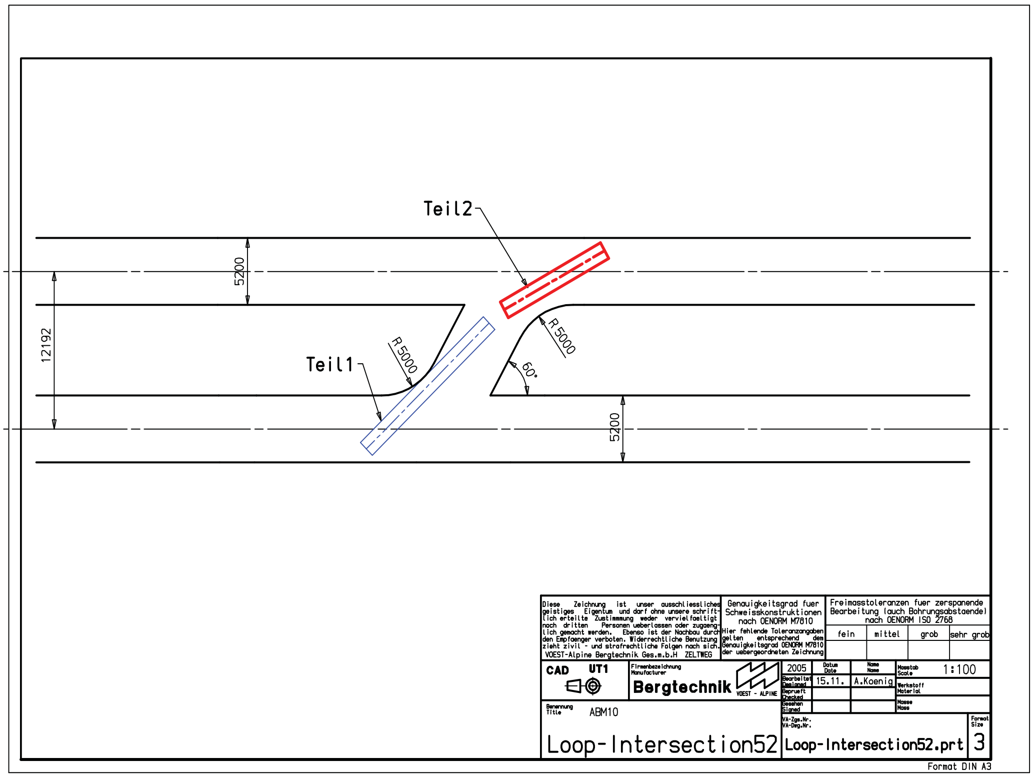
Picture 56: **LTU Unit relocation sequence -1**



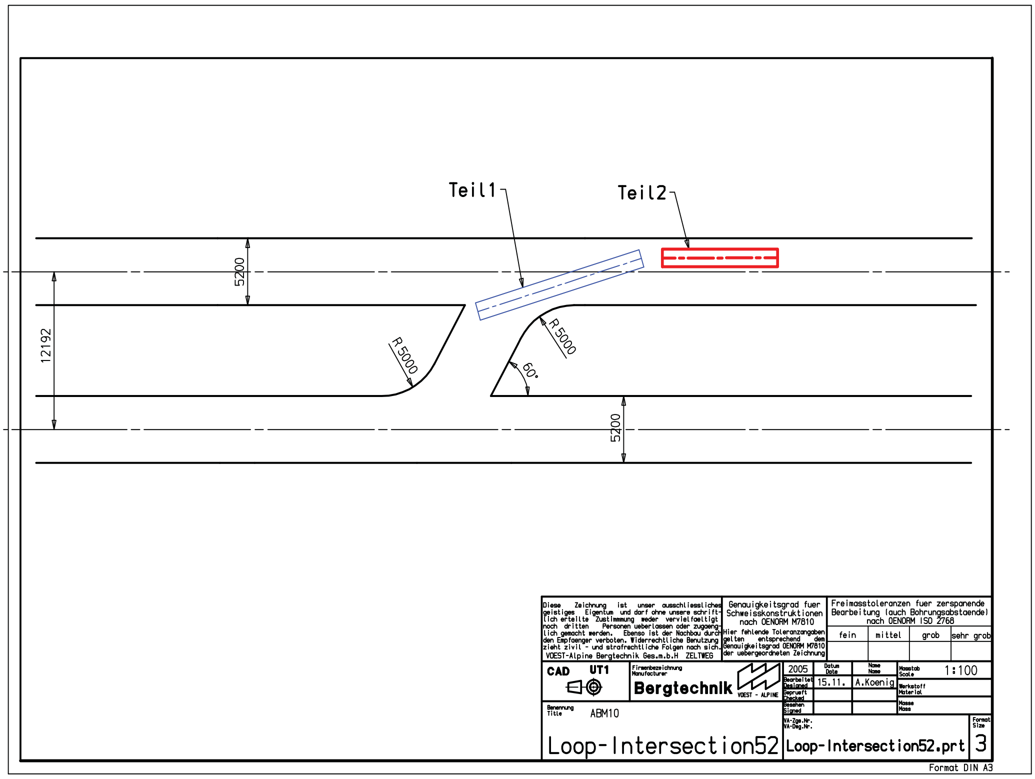
Picture 57: **LTU Unit relocation sequence -2**



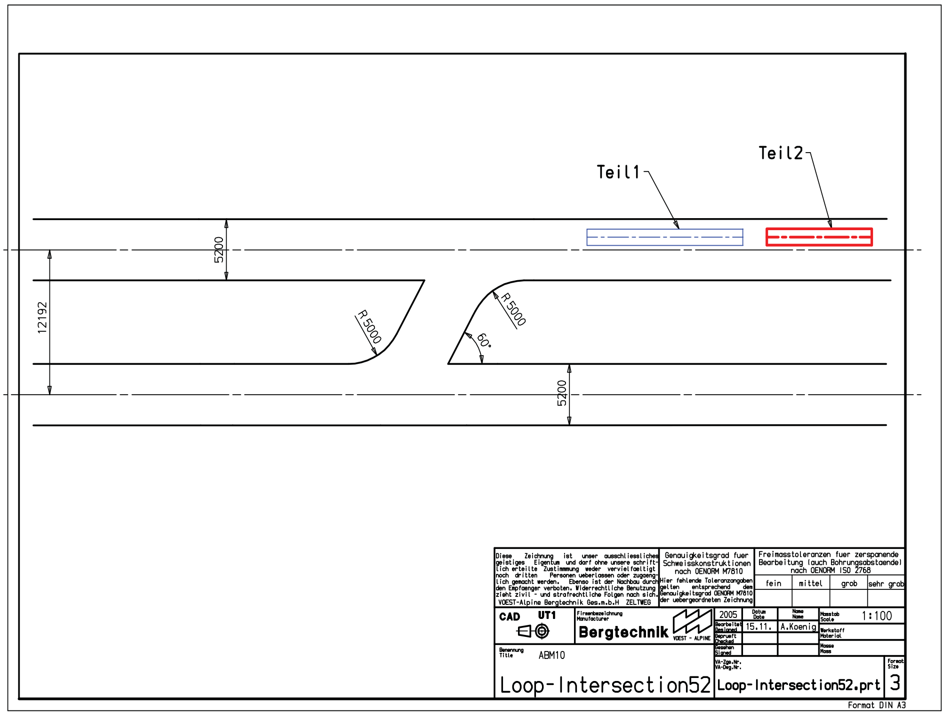
Picture 58: **LTU Unit relocation sequence -3**



Picture 59: **LTU Unit relocation sequence -4**

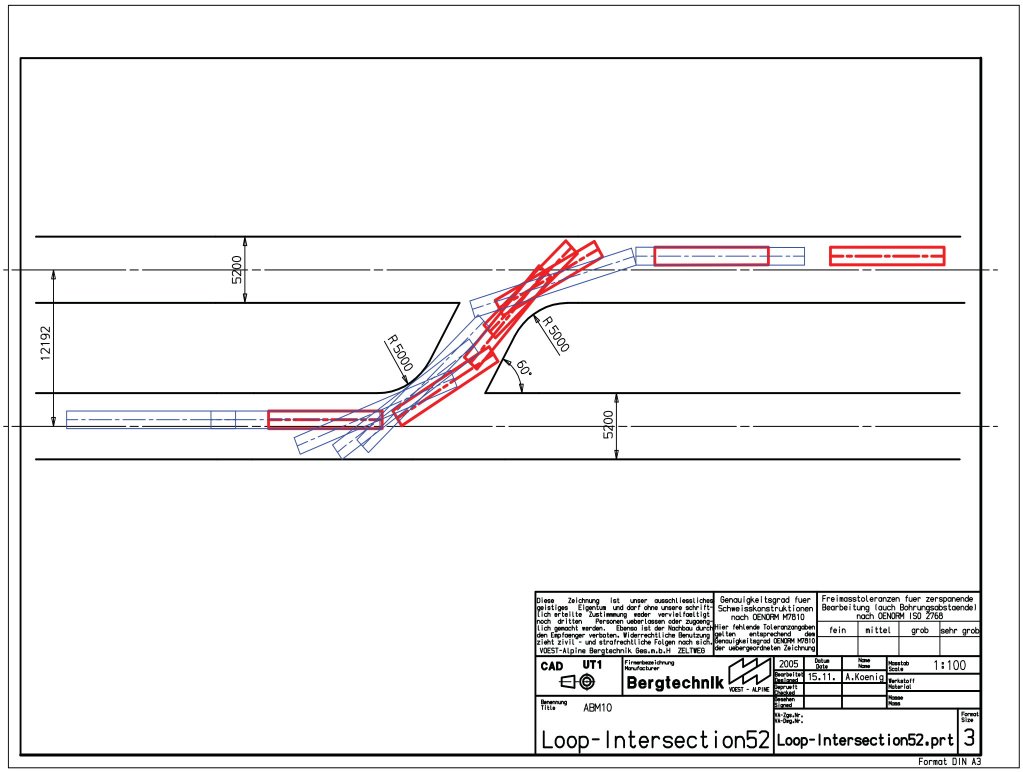


Picture 60: **LTU Unit relocation sequence -5**



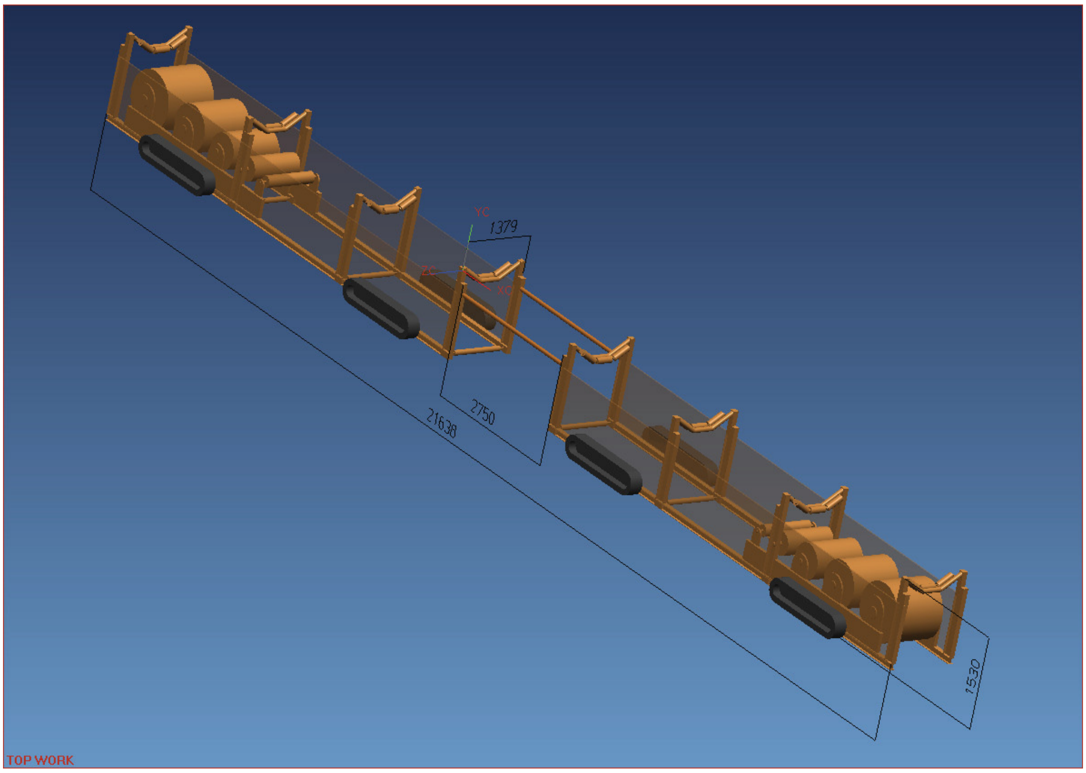
Picture 61: **LTU Unit relocation sequence -6**

The whole sequence superimposed shows the following picture 62:



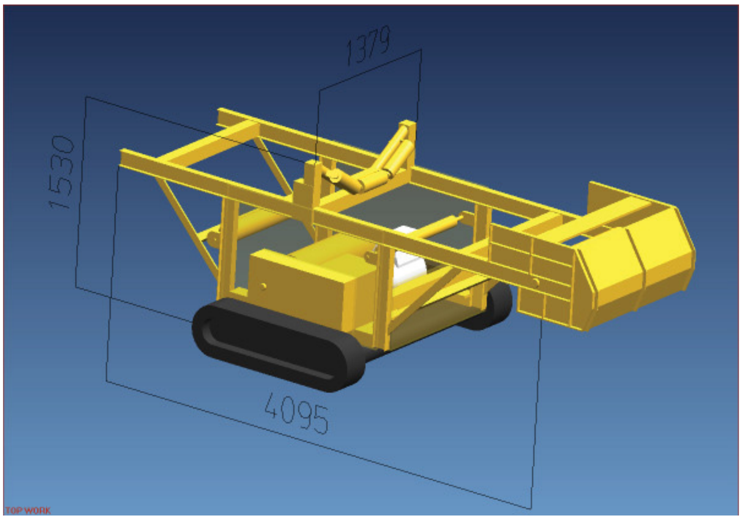
Picture 62: **LTU Unit relocation sequence –in full**

Superimposing the found facts the LTU unit will have to look like similar to the following sketch in picture 63. To be able to relocate all components of the material handling system everything will be track mounted and powered by an on board hydraulic power pack.



Picture 63: **LTU Unit development stage II**

Drive and discharge station are shown in picture 64.

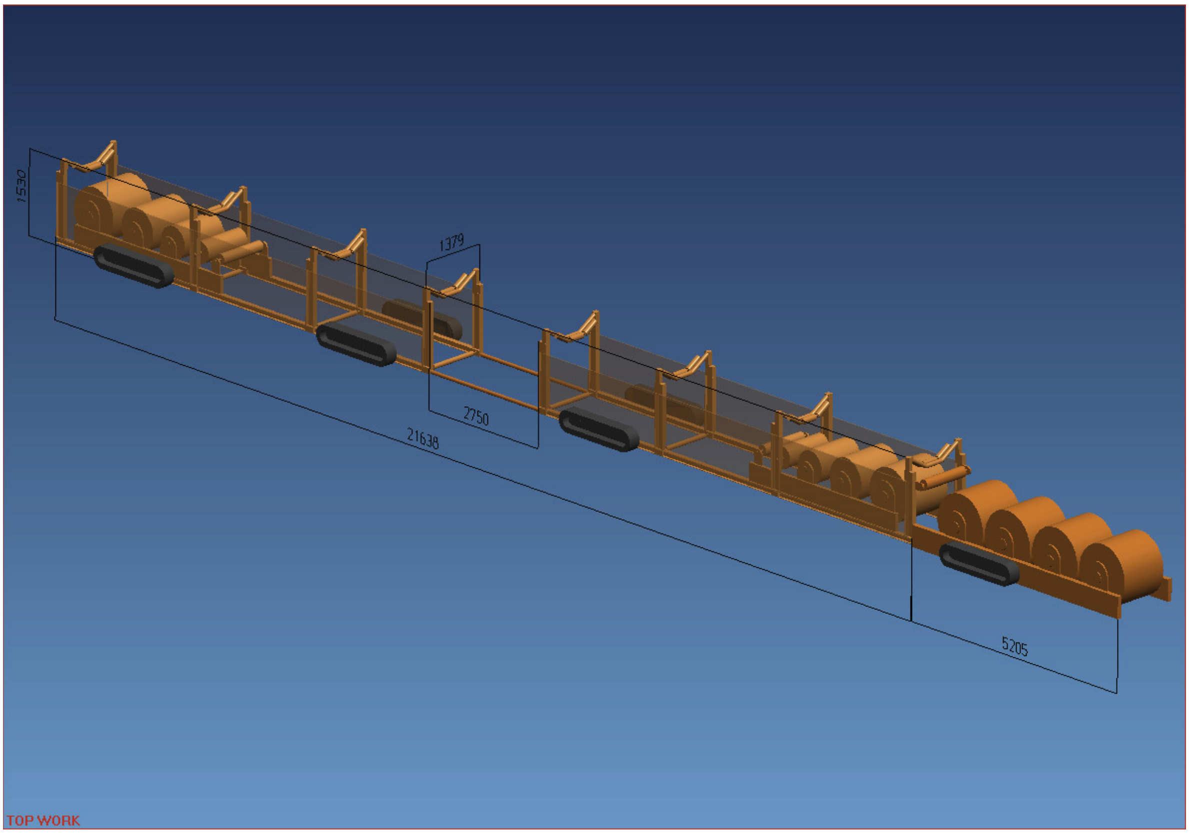


Picture 64: **Drive and Discharge Unit**

Both units together will have a length of around 27,0m. A removable piece in the centre with around 2750mm length will serve as the component to assure the flexibility / separation of the unit when taken out.

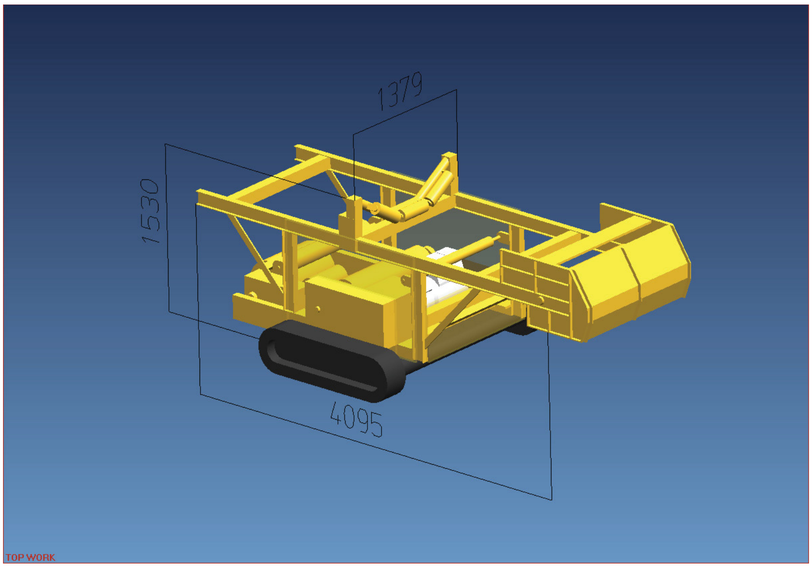
Additional length of the LTU unit has to be considered though for some conveyor belt rolls to be stored inside the LTU unit which would be drawn into the unit when the storage is empty. The handling of the conveyor belt rolls would be avoided with this measurement taken.

The LTU unit (picture 65) might have to be shortened again and may be up to four rolls of belt stored somewhere (the front end might be the best location for the additional belt rolls) close to the LTU unit for quick access when needed.



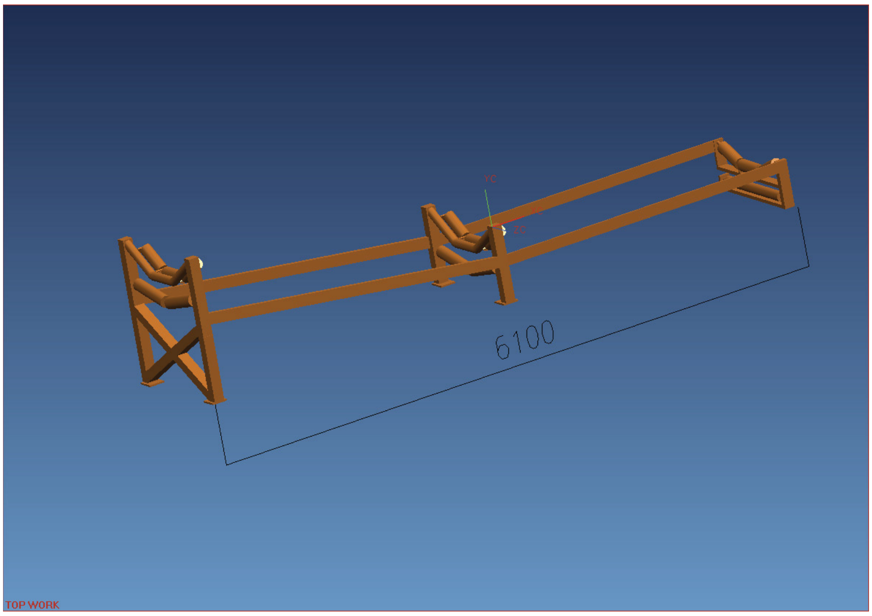
Picture 65: **LTU Unit with Conveyor Belt storage**

The drive station in picture 66 will stay as before with the initially laid out dimensions.



Picture 66: **Drive and Discharge Unit**

Also the gradient section (picture 67) will be laid out so we can combine all necessary components together to be able to make the overall dimensional check. The gradient section will guide the conveyor belt from the mining tunnel height of around 500-700mm conveyor structure height up to the top of the LTU unit staying below the maximum incline angle for conveyor belts to avoid material rolling backwards.



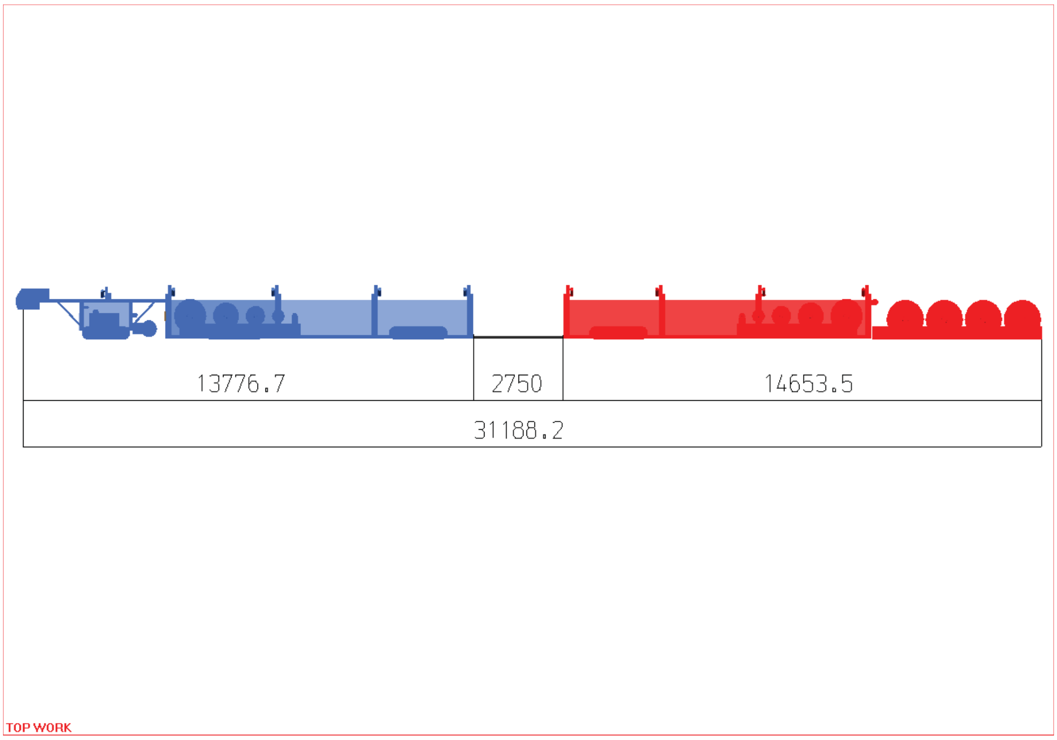
Picture 67: **Gradient Section**

The layout of the initial dimensions was an iteration process for the author covering all parameters at the same time and checking the operational side as well to optimize all dimensional parameters.

Combining all the new gathered information and research results together the two parts of the LTU / Drive / Discharge unit will lead us to the following modified material handling system dimensions (picture 68).

Without the gradient section the rear end of the material handling system will have a length of around 31,2 meters with a 2750mm long rigid but removable frame work in the middle of the LTU.
Taking this rigid frame work out during the relocation sequence will give the essential flexibility in the middle to drive around the 6000mm corners. This will be the same as the conveyor belt sitting in between the Hopper Car and the LTU unit so the principle has got a very good chance to be successful.

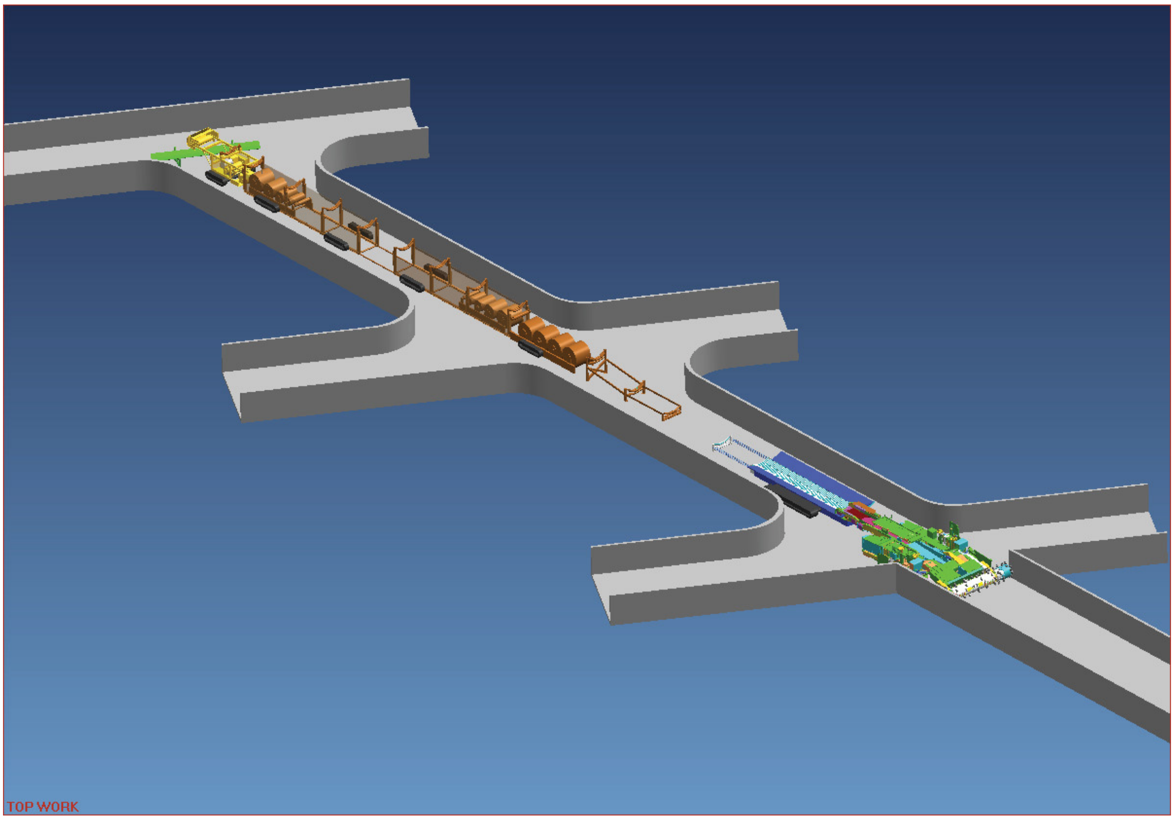
Special attention has to be given to the conveyor belt when going around the corner so that the belt does not get overstressed at the edges and damaged as a consequence.
A stopping system has to be installed most likely to prevent the belt edges from being ripped due to the opening angle between the two carriages getting to wide or due to the belt getting squashed up when moving too close to each other.



Picture 68: **LTU Unit incl. Drive- and Discharge Unit**

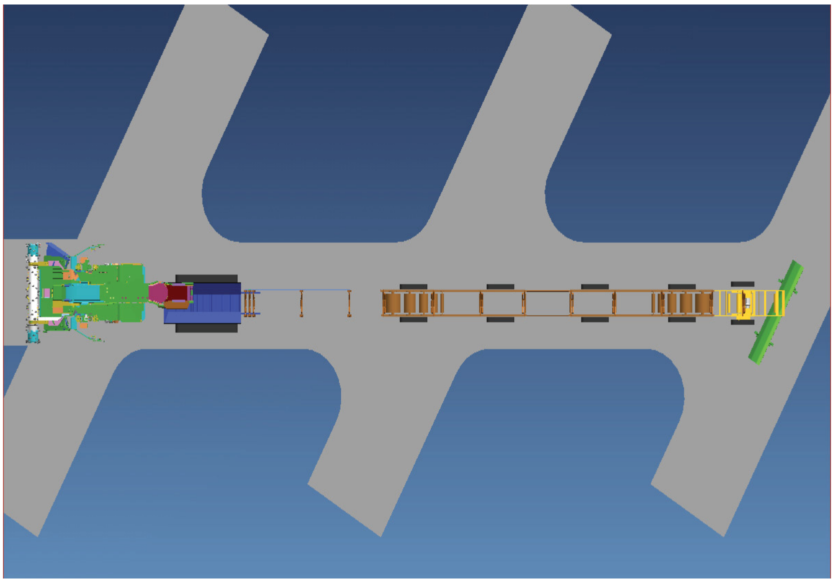
Taking the preliminary set dimensions and layout of the material handling system and combine them with the ABM10 the following 'Rapid Coal Production System' is set up.

To be able to visualize the mining system in its environment in a better way the whole starting tunnel system as well as the mining machine and the material handling system will be modeled into a 3-dimensional picture as shown in picture 69.



Picture 69: **Rapid Coal Production System – First lay-out**

In the overall picture the mining system will now fit into the 70,0m starting length with enough distance left in between the components to be able to move them individually (top view, picture 70).


Picture 70: **Rapid Coal Production System – First lay-out, Top View**

5.5. Operational system parameters

Besides the technical parameters also mining or better operational parameters with the consideration of local underground systems, the handling of the system when interfacing with other machinery and the interaction between workers and machine have to be taken into account before finalizing the lay out of the mining system.

5.5.1. Operators

A minimum amount of operators shall be utilized for the operation of the mining system.
The reason for this is the cost factor. Even though it would be an excellent thing from the technical and operational point of view to have lots of mine workers operating the system, the commercial part of the game dictates us to use as few as possible people only to operate the machines.
Approximately 800 tons of hard coal will be mined out of the mining tunnel (also called 'Punches' from now on) within one 8 hour working shift.

Underground mine workers according to their skill level inclusive all the additional wage components for dangerous work, dust and dirt exposure, noise exposure, etc. can cost the mining company with all wage costs, duties, fees and taxes applied up to approximately and depending on the region USD 10,000 per month without overtime.
An average working month consists of around 21 working days with one 8 hour shift each day.

This would mean that the labor content in the hard coal cost price will as high as around the USD 0,58 per person and ton of hard coal (USD/p, ton).

This quite significantly cost figure is when putting it into perspective to the sell price of hard coal, USD 35,00 (calculated previously in chapter 4.2.1) around 1,65% per person working on the mining system.

Planning the workforce for the new panel in the left over hard coal block will therefore have to be handled very carefully.

How many operators will be necessary to operate the system?

- 1 off ABM10 miner driver (cutting, driving, loading, relocating)
- 2 off ABM10 bolter operator (roof bolt installations LHS and RHS, ventilation duct installation)
- 1 off Hopper Car operator (driver, conveyor structure handling RHS, additional conveyor belt installation into LTU, relocation)
- 1 off Hopper Car assistant (conveyor structure handling LHS, additional conveyor belt installation into LTU, relocation)
- 1 off panel supervisor
- 1 off mechanical / hydraulic fitter (part time, only on request)
- 1 off electrician (part time, only on request)

All up we have to have a minimum of 6,5 mine workers operating the machine and the whole panel.

The cost for labor to operate the new panel will be around the USD 3,77 per ton of hard coal mined out of the left over long wall block which is more than 10% of the actual current hard coal sell price but only 25% (compared to 36% for a normal operation) of the actual cost price of one ton of hard coal.

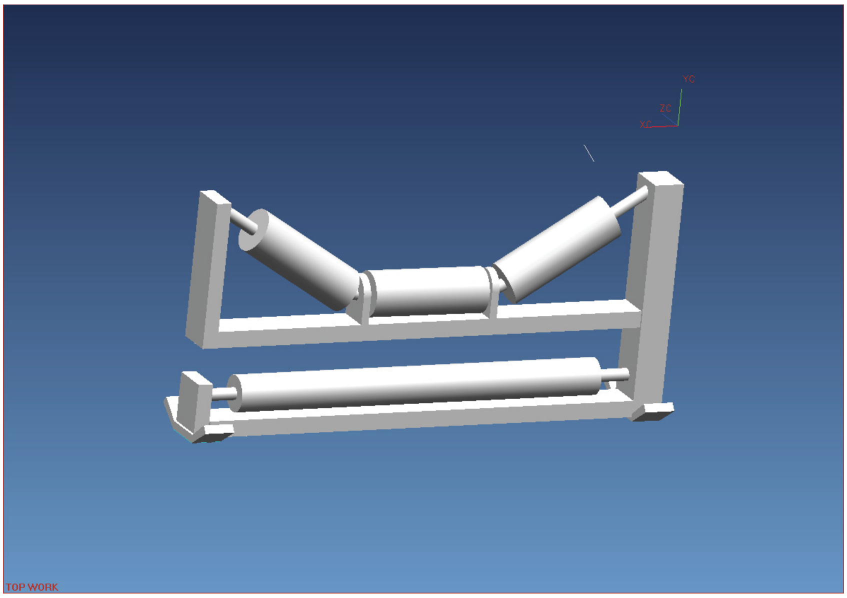
5.5.2. Advance systematic

The Hopper Car is acting as the advancing system for the haulage system.
It will be driven via tracks either electrically or preferably (because of the size) hydraulically.
The hopper car will be powerful enough to pull the conveyor belt against the belt tension force applied by the winch. It will also be heavy enough to counteract the tension force of the winch during the operation.

The Hopper Car will follow the continuous bolter miner and will receive the cut material from it.
An overlap of 2 to 3 cutting cycles of the ABM10 (cutting of 1000mm longitudinal distance plus installation of the roof bolts) between the chian conveyor boom of the ABM10 and the hopper (receiving chute) of the hopper car will be aimed for. This should give the necessary flexibility between the two processes of the ABM10 and the material handling system when advancing at different times.

Even through out the advancing process, the conveyor of the material handling system will be able to transport the cut coal to the panel conveyor.
The necessary additional conveyor belt structures (including the carry and return rollers) for the full length of one Punch or mining tunnel, will be stored inside the Hopper Car and shall be released in distances of approximately 2,6 to 3,0m to each other when the Hopper Car advances.
The distance between the individual conveyor structure assemblies is kept as short as possible due to the extremely low height conditions and to avoid the return belt strand from rubbing along the floor.

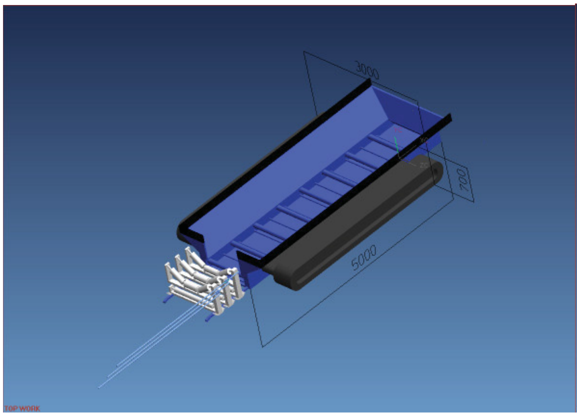
The following picture 71 shows a first idea of the belt conveyor structure assembly for the application. The Hopper Car when receiving the cut material from the miner will have to be below the chain conveyor of the ABM10. This means the available maximum height for the Hopper Car of 700 – 800mm.



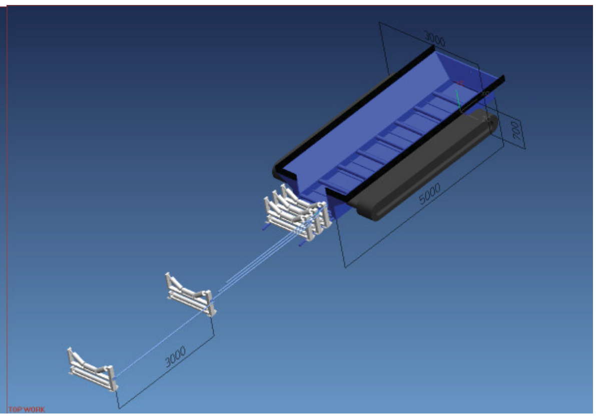
Picture 71: **Conveyor Structure Design**

As a consequence of that and the need of achieving some filling/bunkering height in the hopper car the conveyor structure assemblies must not be higher than maximum 500mm to be able to fit into the hopper car body.
Angled carrying idlers as well as return rollers have to be accommodated by maintaining a certain ground clearance when stored inside the hopper car body.

The installation of stringers between the structure assemblies should also be considered to give the conveyor structure the necessary rigidity and the stringer could assist the structure assemblies getting pulled out of the hopper car body (picture 72, 73).

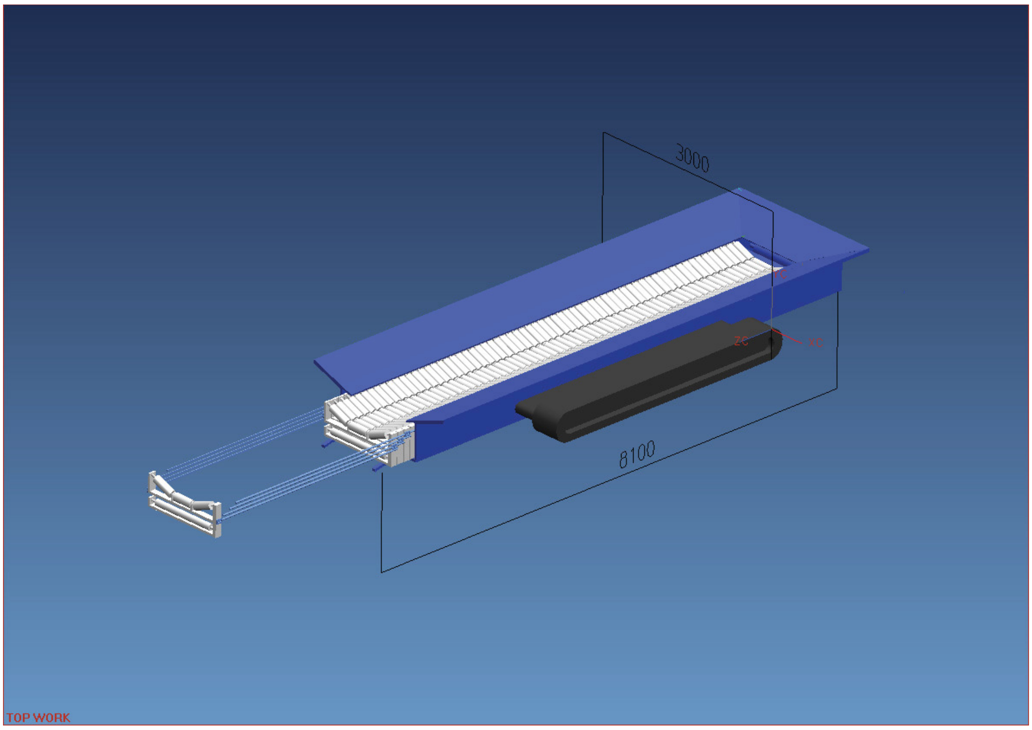


Picture 72: **Advance Process Step I**



Picture 73: **Advance Process Step II**

When estimating the necessary quantity of conveyor structure assemblies which have to be stored inside the hopper car, we end up with most likely 63 units which, when arranged shoulder to shoulder inside the frame will push the hopper car to around 8000mm in length (picture 74).



Picture 74: **Hopper Car with all Conveyor Structure stored inside**

The Hopper Car, when advancing forward, will pull the additional conveyor belt out of a Loop-Take-Up unit, located at the other end of the haulage system.
During the advance movement of the hopper car, the winch will maintain a constant belt tension to be able to run the system even when advancing forward or in reverse. This advancing system is an absolute novelty and one of the main features developed by the author for this system.

5.5.3. Conveyor Belt Extension systematic

The Loop-Take-Up system with the estimated length of around 21,0m can store effectively approximately 160 metres of conveyor belt, which gives an advancing distance of about 80 metres. The additional necessary conveyor belt length to reach the 200 metre mark is stored in form of belt rolls inside of the Loop-Take-Up unit and will be pulled into the LTU when needed.

It will be pulled into the Loop-Take-Up via motors driving the belt rolls, a belt clamping device and the conveyor belt tensioning winch.
Whenever the LTU unit is empty, the stopped conveyor belt (joined together to an endless belt with mechanical links) will be clamped around one of the mechanical joint to keep the joint in a fixed position.
The joint will be opened up and the additional belt length attached to one loose end via this mechanical joint. The winch will than pull the moving carriage back to the starting position pulling the additional conveyor belt into the LTU unit.
The rear loose end of the additional belt will than be joint to the initial belt end and tensioned via the winch. This is another novelty engineered by the author especially for this mining application.

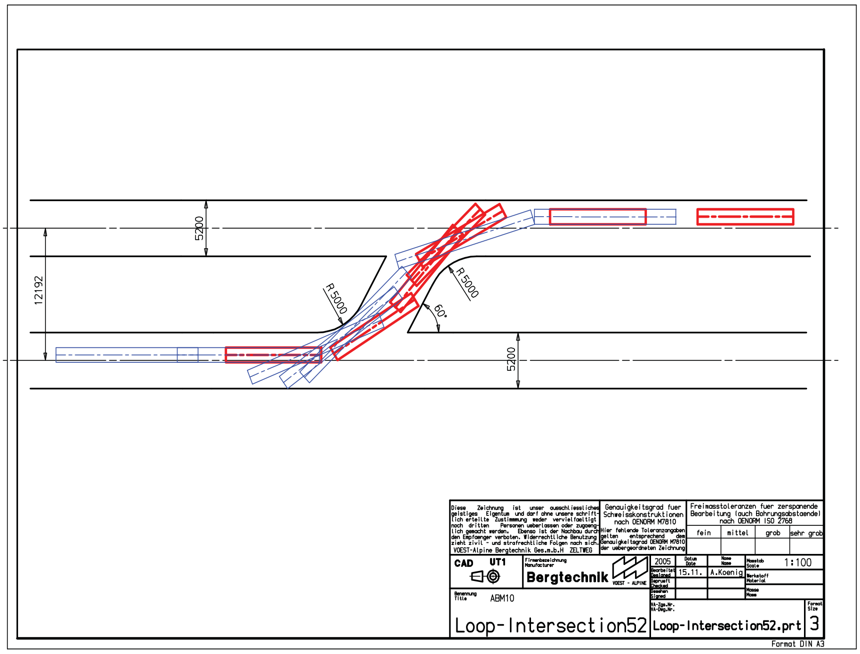
5.5.4. Retreat systematic

After finalising a 200m punch, the mining system has to be retrieved out of the section, relocated and set up for a new 200m cutting/mining sequence.
For this the continuous bolter miner and the hopper car will return to their starting position, collecting all conveyor structure assemblies (back into the storage area of the hopper car) on their way back. The stringer elements will be stored back onto the hopper car.
Also the conveyor belt surplus (from the belt extension process) needs to be reeled back onto the rolls via on-board hydraulic driven reelers. Before doing so, the winch has to release the belt tension and the hydraulic motor winds the conveyor belt back onto the storage spools.
This whole retreat and belt-taking-out process works the same way as advancing forward in mining direction only in reverse and was invented and engineered for this mining application only.

5.5.5. Relocation systematic

Back in their start-up position and therefore out of the latest cut punch, the whole mining system will be split by separating the continuous bolter miner ABM10 away from the hopper car.
The two separate parts will manoeuvre individually up the Right Hand Roadway by taking out the rigid frame work in the middle of the LTU unit to give the flexibility to get around the corners.

The following picture has been shown before (picture 62) in all individual steps and shall serve to complement the description here.

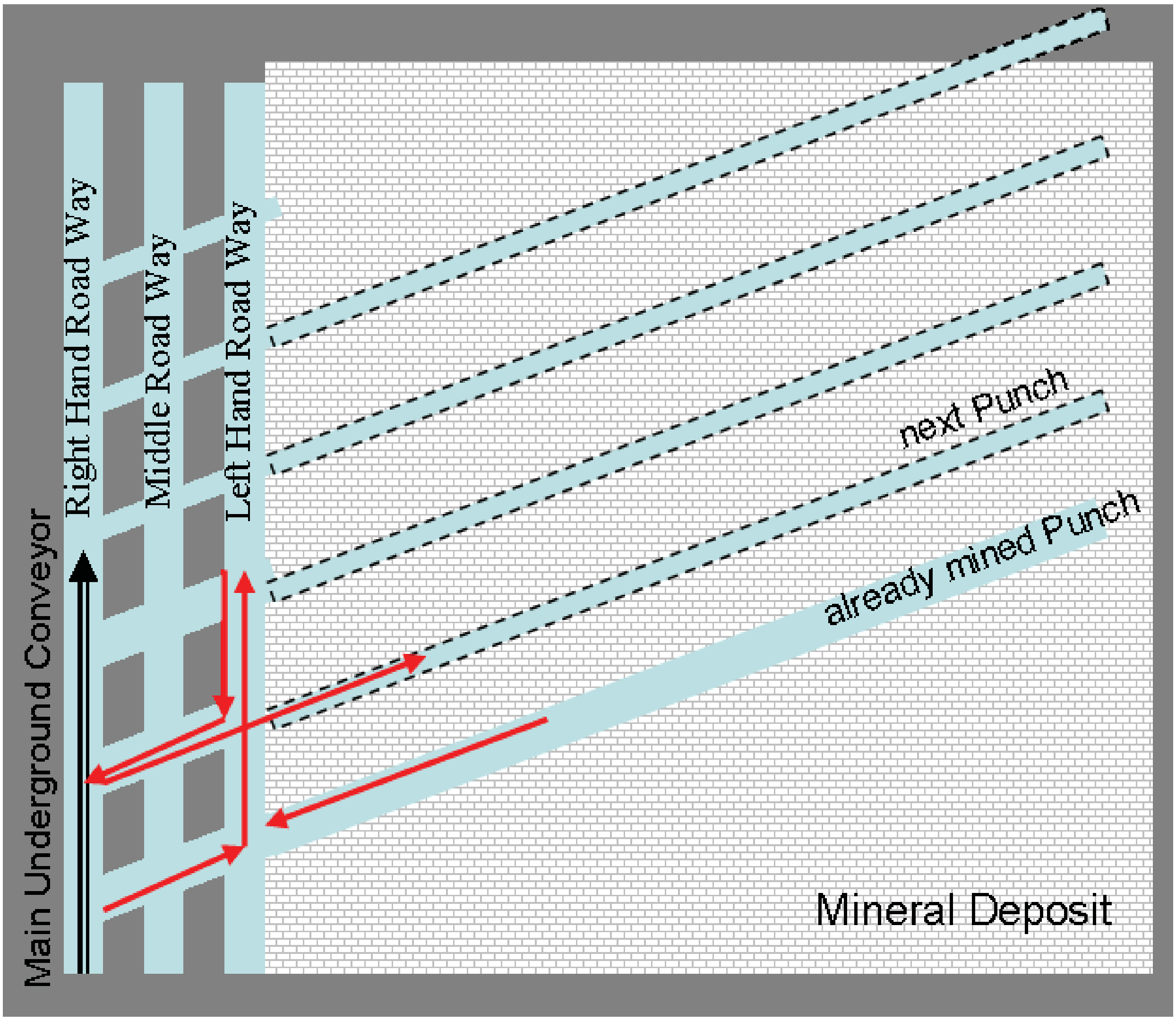


Compare to Picture 62: **LTU Unit relocation process in full**

The system will move upwards the right hand roadway till the Drive/Discharge Station is on equal height of the next “start-up” road-way; will then reverse backwards towards the fix installed panel conveyor till the transfer chute and the panel conveyor line up.

After the way is cleared the ABM10 can move to the new starting position with the hopper car moving underneath the chain conveyor boom.

In this position the continuous bolter miner as well as the Hopper Car will be fully in line with the new “start-up” roadway and the new cutting/mining sequence, punch, can be started in the described way (picture 75 visualizes the whole relocation sequence).



Picture 75: **Reloction Sequence of the Rapid Coal production System**

5.6. Time study

Full concentration had to be given to this process by the author. As mention before, the time it takes to perform the mining, the retreat and the relocation work is utmost important and essential for final success of the operation.

To repeat the main requirements:

- 2 times 8 hours (one working shift) to mine out the 200m excavation tunnel
- 8 hours (one working shift) to retreat and relocate the mining system

Within 24 hours or in other terms 3 working shifts the mining system has to start operating and mining till the machine has finished the mining process and is positioned right in front of its new mining position ready to start cutting.

Following operational steps will be investigated and time estimations calculated

- One hopper car advance step of 3 meters (ABM10 / HC overlap = 3 meters)
- Conveyor belt extension process
- Retreating process
- Relocation process

Table 18 illustrates the timing calculations/estimation done by the author in a simple but effective way in form.

Advance operation

The requirements for the ABM10 are to work and mine the coal with an advance speed of 12.5m/h which in return requires the hopper car to advance one meter within around 4 to 5 minutes.
Due to the fact that the chain conveyor of the ABM10 and the hopper car have got an overlap of 3000mm (the machine and the hopper car can move independently for a certain distance and time; the distance between two sets of conveyor structure inclusive idlers is 3000mm), the requirements for the hopper car are to advance 3000mm within a time span of around 12 minutes.
The investigation shows that the hopper car would be able to advance 3 meters within 3minutes. This allows for the hopper car to install/release one set of conveyor structure inclusive idlers and stringers as well as moving the hopper car to its new position and to set it up in proper running position (level and direction).
This time frame which is necessary to advance the hopper car for the discussed 3 meters is by far quicker than the ABM10 will advance over the 3 meter distance and is there fore suitable for the task.

ADVANCE PROCESS (ca. 10ft)					
Step	Description	time [sec]		time [sec]	total [min]
1	Lift Hopper Car	10		10	
2	Advance Hopper Car 10ft	40		50	
3	Assist Conveyor Structure to glide out		40	(50)	
4	Insert Stringers		60	(110)	
5	Adjust Hopper Car horizontally via Laser	60		110	
6	Lower Hopper Car	10		120	
7	Adjust Hopper Car Vertically via "spirit level"	60		**180**	**3,00**

EXTEND BELT (by 2 rolls >> ca.190ft)					
Step	Description	time [sec]		time [sec]	total [min]
1	Slow down belt to meet joint	20		20	
2	Clamp one end	5		25	
3	Relax belt	10		35	
4	Clamp second end	5		40	
5	Remove joint	180		220	
6	Connect belt to first roll	300		520	
7	Open front clamp	5		525	
8	Pull-in of first belt roll	240		765	
9	Connect frist belt to second belt roll	300		1065	
10	Pull-in of second belt roll	240		1305	
11	Connect second belt to initial belt	300		1605	
12	Open rear clamp	5		1610	
13	Tensioning of belt	10		1620	
14	Start-up of system	15		**1635**	**27,25**

RETURN PROCESS 600ft (ca. 6.5ft/min plus belt storage)					
Step	Description	time [sec]		time [sec]	total [min]
1	Lifting of Hopper Car	10		10	
2	Take out stringer and store	30		40	
3	Return Drive 10ft	120		160	
4	Guide Structure into Hopper Car		120	(160)	
				↓	
5	Process for 600ft			9600	
6	Rewind Belt Storage rolls 1 & 2	1635		11235	
7	Rewind Belt Storage rolls 3 & 4	1635		**12870**	**214,5**

RELOCATION PROCESS					
Step	Description	time [sec]		time [sec]	total [min]
1	Release belt tension	10		10	
2	Collapse Intermediate Structure and secure	500		510	
3	Remove and store Loop-take-up rails	500		1010	
4	Move system forward and up third driveway (4ft/min > 400ft)	6000		7010	
5	Return system back into next start-up section (4ft/min > 400ft	6000		13010	
6	Move system forward to new cutting face (4ft/min > 100ft)	1000		14010	
7	Reinstal Intermediate Structure	500		14510	
8	Reinstall Lookp-take-up Rails	500		15010	
9	Tension belt	10		**15020**	**250,3**
	COMPLETE RELOCATION PROCESS			**27890**	**464,8**
					↓
					7.7 hours

Legend
main duration
secondary duration (in parallel to main duration)
task
total duration of task in seconds
total duration of task in minutes
total duration of complete process (mining and relocation)

Table 18: **Time Study**

Extend belt process

4 additional conveyor belt rolls are needed to reach the end of the mining tunnels. 27,25 minutes times two equals 54,50 minutes time is need to extend the belt. In other words this time is needed to feed the additional requireded belt rolls into the loop take up unit, the LTU unit, to get to the end of the 200m mining tunnel distance. This includes the time to find the mechanical link between two conveyor belt ends, to clamp the belt, release the tension, and relax the belt to be able to take the mechanical link or joint apart to connect and pull the belt rolls in and to rejoin the belt ends for tensioning and start conveying again.
After that the ABM10 needs about 13,5 minutes to mine out 3 meters of tunnel face, the lost time will be limited to around 51,00 minutes. 200 meters times 4.5 minutes means that in about 900min or 15 hours to mine out the full tunnel distance. One full hour will be spare in the worst case so the 51,00 minutes lost time will be fully covered for.
The extension of conveyor belt will be well in between the available time frame and therefore also suitable.

Return process out of the mined tunnel or also called punch

After finishing the mining process after the distance of 200 meters the ABM10 and the hopper car have to return to the starting position. On the way back the hopper car will dictate the returning speed as it has to collect all the set conveyor belt structure back into the storing area of the hopper car. Also the added conveyor belt rolls have to be reeled back onto the belt spools for storage.
The maximum anticipated return speed of the hopper car is around 2 meters per minute which will result in a time of 214,50 minutes to return back to starting position. 214,50 minutes are around 3,6 hours.

Relocation process

The relocation process brings the mining system into the new starting position. This process comprises the separation of the hopper car from the ABM10, the relocation of the ABM10 and the materials handling system to the starting position of the new mining tunnel or punch.
The time estimated for this process is around 250,3 minutes or 4,2 hours.
The return process and the relocation process have to be seen as one process so the times have to be added as this overall process has to happen within one working shift or 8 hours.
250,3 plus 214,5 equal to 464,8 minutes or 7,75 hours which is below the required 8 hour mark and therefore also suitable for the proposed mining process.

All calculated process durations have been calculated by the author in a very reasonable and conservative fashion not to risk the process outcome due to the too tight time estimations.

Some of the work steps will have to be performed in parallel. They are called secondary duration in the time study. Only one (the longer one) of the time durations is taken into account in this situation.

5.7. Active patents

An intensive review of the current state of technology was done by the author and active patents had been researched for not getting in conflict with any of them. It is very important to know all state of the art technologies to be able to get a good picture of the investigations and engineering to be done for the mining system used in the on hand hard coal deposit.

Further more it will be necessary to protect the new design with most likely new patents so it is even more important to know the already available systems. Even though the Rapid Coal Production System will be an assembly of individual state of the art technology available on the market, the actual Rapid Coal Production System as a whole system will be a new invention by the author.

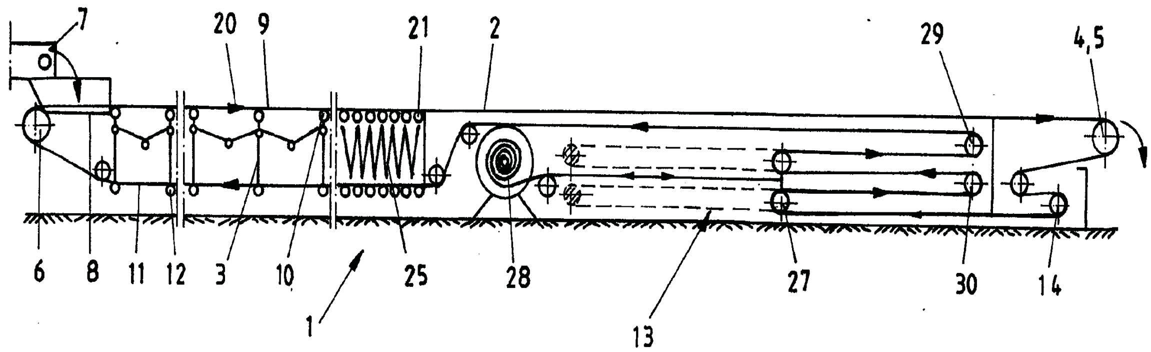
In the following all relevant found patents are summarized by investigating and comparing the new mining concept with already working underground mining systems.

17 relevant active patents had been found which could be in violation with the new design for the mining system for the low seam hard coal deposit.

Telescopic conveyer belt for subsurface mining operations

Abstract of DE 19810988

The telescopic conveyer belt (1) may be extended or shortened by the operation of a scissor lever arrangement which repositions the appropriate rollers. The conveyer assembly holds a belt extension section (13) on an internal carriage (27) in readiness for an extension which may be up to 750m. Additional lengths of belt may be held on a roll.



Reversible extensible belt conveyor

Abstract of US 2,925,901 not available

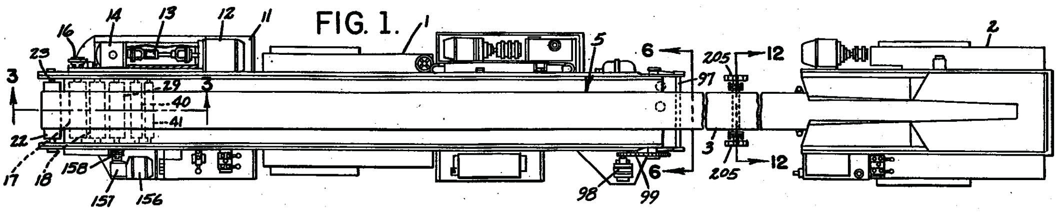


FIG. 1.

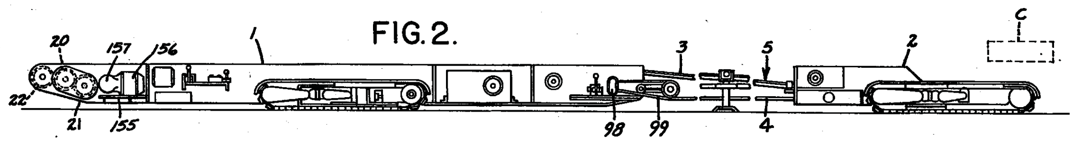


FIG. 2.

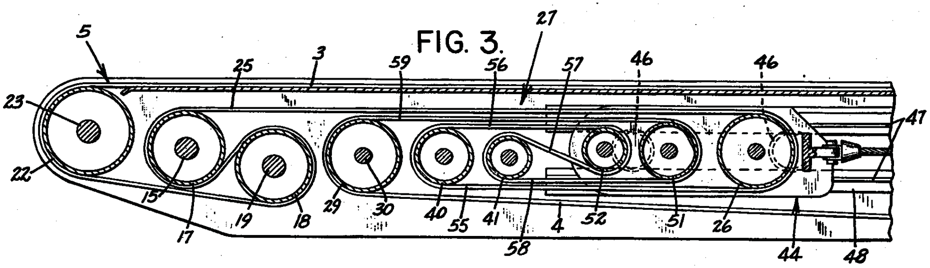
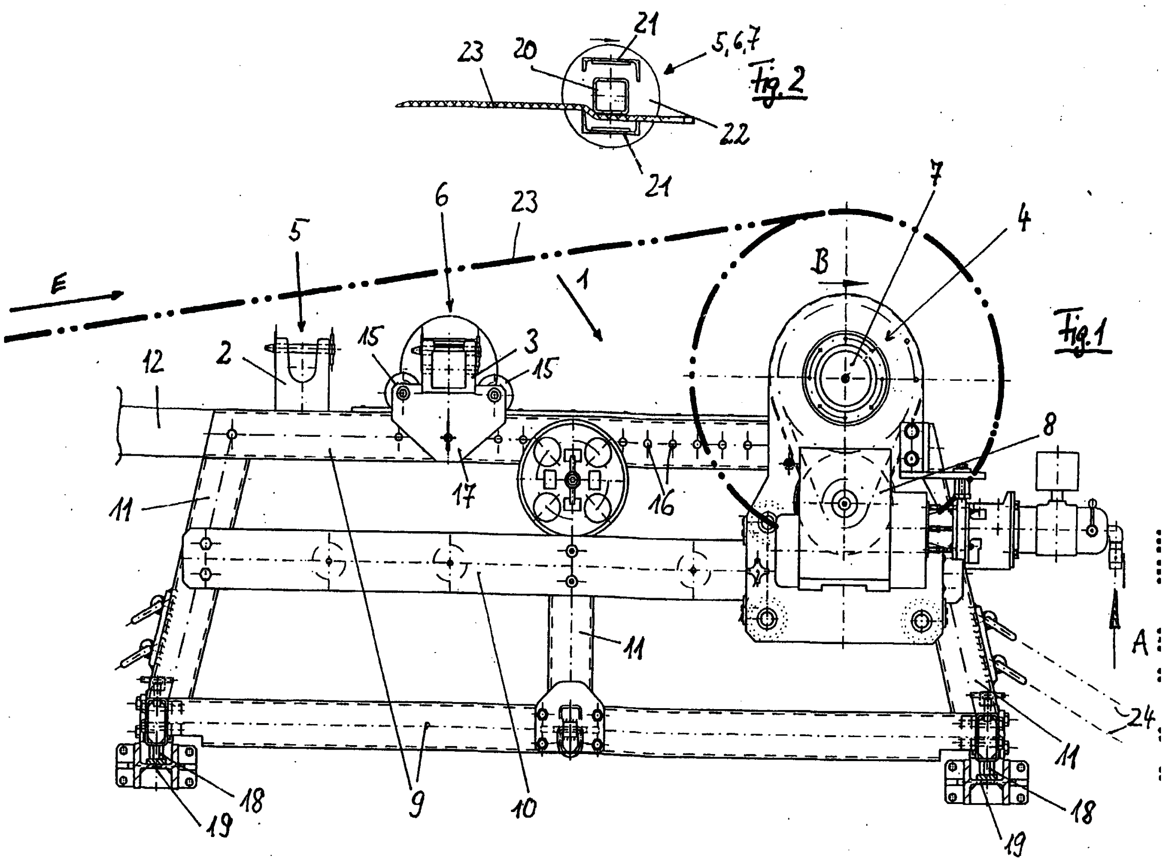


FIG. 3.

Vorrichtung zum Wickeln von Fördergurten (Apparatus for spooling of conveyor belt)

Abstract of DE 29818423 U

Die Einrichtung zum Wickeln von Fördergurten besteht aus einer rahmenartigen Wickelmaschine, in welcher beidseitig an Längsträgern Lagerböcke zur Aufnahme von mindestens 2 Wickelkernen zur Herstellung eines Doppelwickels und einseitig am Rahmen ein verschiebbarer mit den Wickelkernen kuppelbarer Antrieb vorgesehen ist, wobei die Wickelmaschine mit drei im Abstand voneinander angeordneten austauschbaren Wickelkerne aufnehmenden Lagerböcken versehen ist und mindestens der mittlere Lagerbock in horizontaler Ebene verschiebbar ausgebildet ist.



Conveyor belt storage device

Abstract of EP 0196196

The belt to be stored is entrained between a set of rollers (11) on a fixed frame (10) and a set of rollers (13) on a movable frame (12). Separation of the frames increases the amount of belt stored as lengths (31, 32 etc.) extending between pairs of rollers (24-25, 26-27, 28-29 etc.). The sets of rollers (11 and 13) are arranged at an acute angle to the direction of separation to minimize the height of the assembly. In modified assemblies the inclined sets of rollers form a V-shape, with two groups of rollers inclined at acute angles above and below the horizontal, or a Z-formation.

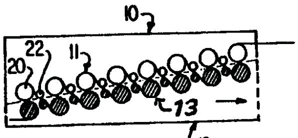


FIG. 1

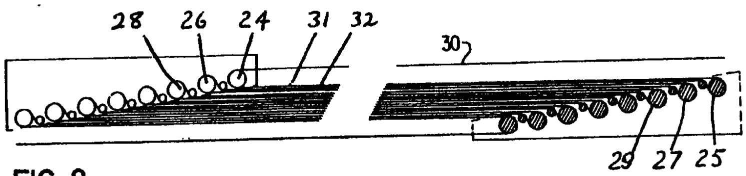


FIG. 2

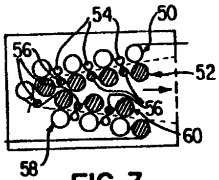


FIG. 3

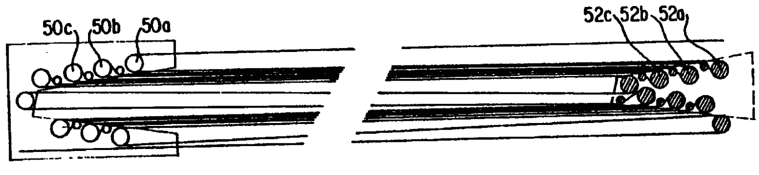
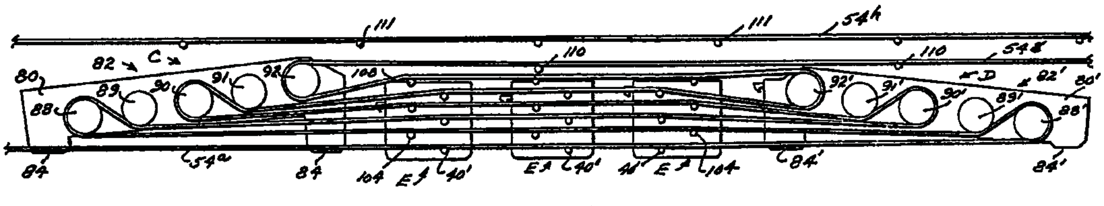


FIG. 4

Belt take-up and storage unit for extensible belt conveyors

Abstract of GB 1086031

A for storing and paying out conveyer belt to permit variations in spacing between terminal points of an endless belt conveyer comprises roller means C, D for supporting a plurality of vertically spaced belt runs, with the uppermost run shorter than any other run and each succeeding run below the uppermost run being longer than the next preceding run there above. Each roller means C, D comprises parallel horizontal idler rollers 88-92 mounted between side frame members 80, slidably guided on rails of a floor frame. Feet on the side frame members which engage the rails may be adjustable to vary the height of the frames above the rails. The belt is supported between the two roller means C, D by rollers 104, 108 mounted between side frame members of separator means E. The latter means are slidably guided on the rails of the floor frame. Other support rollers 40, 110, 111 are mounted independently of the sliding roller and separator means. A hydraulic ram coupled between the roller means C and the floor frame tensions the belt reeved through the unit A. The roller means D is coupled via a cable and pulleys, to a hydraulically operated winch (63), Fig. 4a (not shown). The roller means D is movable relative to the roller means C to wind in or pay out belt from the unit. Latches on the roller means and separator means facilitate spacing of the latter during winding in of the belt. Cups are provided on the floor frame to receive the lower ends of roof jack assemblies.



Conveyor belt/cutting head advance mechanism

Abstract of US 5366068

Disclosed is a conveyor belt/cutting head advancement mechanism (10). Mechanism (10) comprises a conveyor belt support structure (11) connected at one end to a conveyor belt gathering structure (12). The conveyor belt support structure (11) is typically interconnected at its forward end to a self advancing conveyor return end (not shown).

Material removed from a mine face by the cutting implement is delivered to a conveyor belt supported by the support structure (11). Structure (11) comprises a plurality of structural elements (18) pivotally interconnected in a scissor-like configuration. The mid points of elements (18) are pivotally interconnected at a point coinciding with the mounting location of a conveyor belt return roller (16). At the upper pivotally interconnected extremities of elements (18), the conveyor belt advancing rollers are supported. Mounted at a location nearby the lower ends of elements (18) are ground engaging wheels (17). The conveyor belt gathering structure (12) comprises a frame (19) within which is mounted a pair of gathering roller groups (21 and 22). One or both of the gathering roller groups may be mounted upon sub-frames which may longitudinally traverse frame (19). A winch or hydraulic ram (20) may be employed to effect such longitudinal traverse. As roller groups (21) and 22) are caused to move apart, the length of belt (14) gathered thereby is many times greater than the distance through which the roller pairs (21 and 22) have separated.

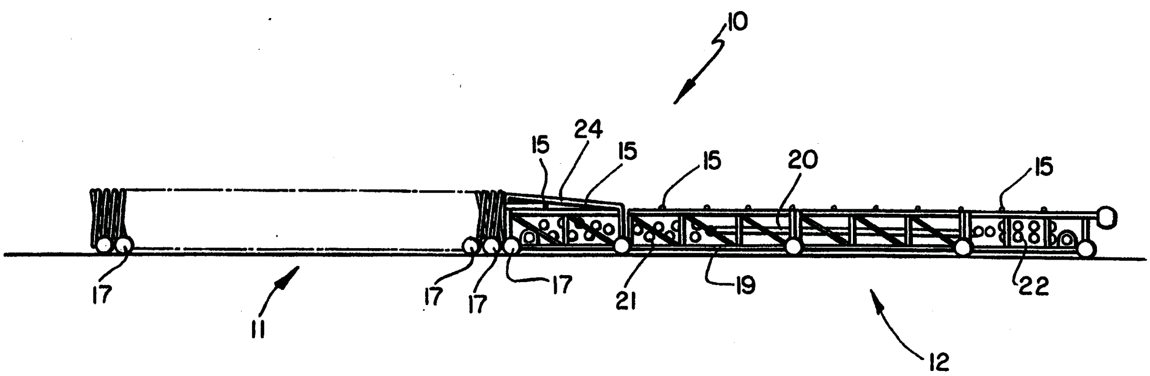
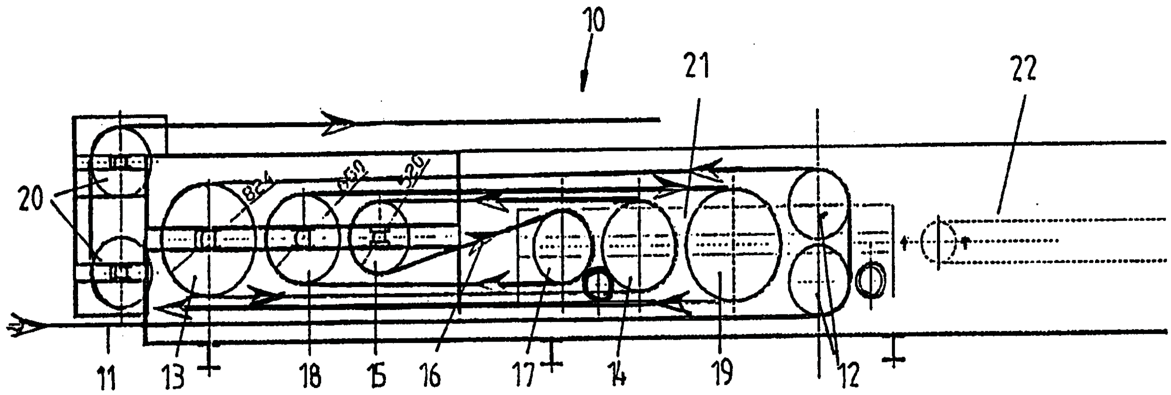


FIG. 1

Continuous conveyor with belt storage loop

Abstract of WO 98/42603

The invention relates to a continuous conveyor with a belt storage loop (10) for compensating the length of the belt. The loop has at least four adjacent tail pulleys via which the belt is fed, and which are in part variable in length, thus making it possible to vary the length of the stored belt. In order to obtain a relatively compact and yet simple structure with high belt storage capacity, the belt (11) is first of all fed inwards in spiral fashion on gradually narrowing tracks, then deviated and finally fed outwards in spiral fashion via gradually widening tracks.



Endless belt conveyors

***Abstract of** GB 1391612*

The tail end 3 of a mining conveyer is slidably supported on longitudinally extending side rails 30 carried by the conveyer supporting structure, the rails 20 being made up of sections so as to enable the length of the conveyer to be increased or decreased The supporting structure comprises longitudinally spaced stools 15 supporting idler rollers (7, Figs 1 and 2) carrying the return run of the conveyer belt, and sockets 19 receiving the guide rails 20 Lengths 24 of angle iron are removably secured to the rails 20 to provide a running surface for the tail end 3 Troughing roller sets (5) are also carried by the rails 20. The stools also carry rails 10 engaged by rollers 13, 14 on ancillary equipment such as a stage loader or a feed bunker.

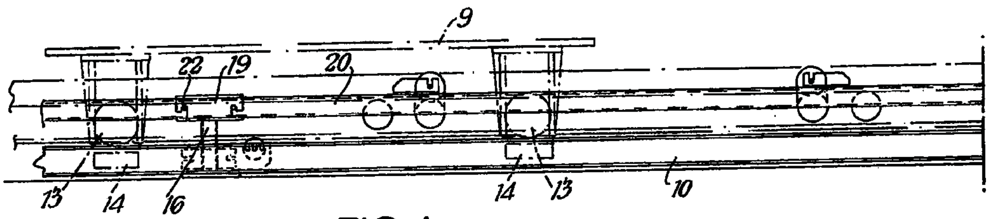
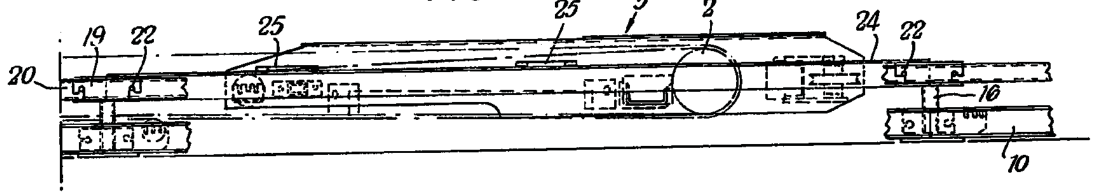


FIG.4

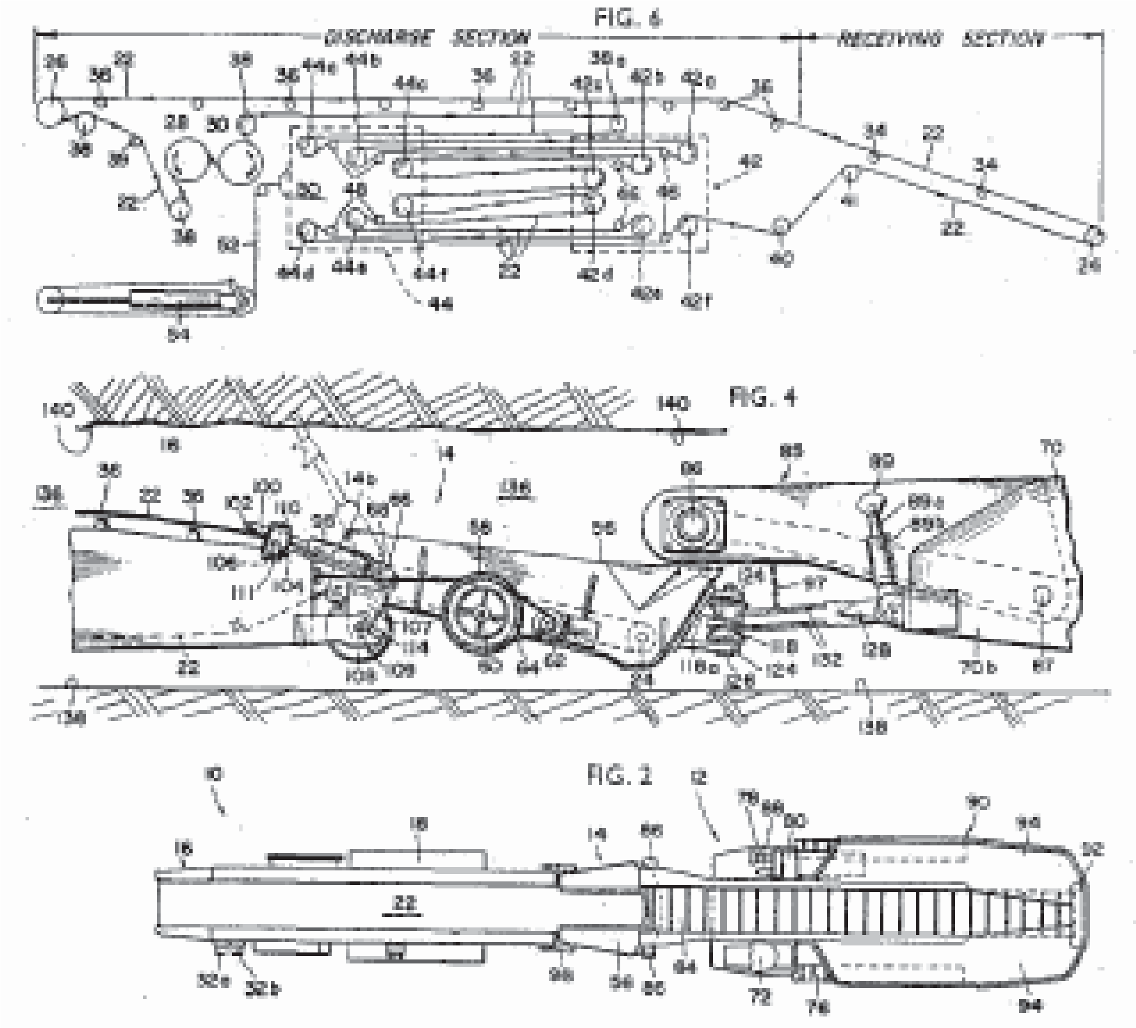


Apparatus for Conveying Mined Material

Abstract of GB1150696

Apparatus for conveying mined material comprises a conveyer unit 10 of the extendible type which is detachably connectible to a self-propelled loading unit 12. A receiving section 14 of the unit 10 is provided with anchor means 100 for engagement with the mine roof 140 and a discharge section 16 includes a belt storage means for automatically adjusting the effective length of a conveying belt 22 during extension and retraction of the unit 10.

Conveyer unit 10.-The belt 22 is driven by rollers 28, 30 and is stored in the discharge section 16 by a stationary roller bank 42 in cooperation with a movable roller bank 44, the latter being carried by a carriage 50. The receiving section 14 is linked to the discharge section 16 by pins 109 which engage in locking slots 114 and is connected to the loading unit 12 by means 96 comprising an hour glass-shaped member 118 and a pair of arms having opposing accurate engaging portions (122), Fig. 5 (not shown). A pair of cables 43 are wound on driven drums 58 and, at one end, are attached to the discharge section 16. The effective lengths of the cables are adjusted during relative movement between the sections 14, 16 and maintain the position of stands (142), Figs. 9, 10 (not shown), which carry idler rollers for guiding the belt 22. A hopper 56 directs material on to the belt 22 and the unit 10 is movable by driven treads (20), Fig. 1 (not shown). Loading unit 12.-An endless chain 84 is driven by hydraulic motors 86 and, at its discharge end, is adjustable in height over the hopper 56 by a pivoted assembly 85, Fig. 4. For temporarily storing mined material when the chain 84 is not driven, an elongated hopper 90 is provided. The unit 12 is movable by treads 76. Operation.-Other than the relative positions shown in Fig. 1, the receiving section 14 may be separated from the discharge section 16, and the loading unit 12 may be uncoupled from the receiving section 14.



Method of providing temporary support for an extended conveyor belt

Abstract of US 5938004

A method of providing a temporary support for a length of belt added to extend an existing fixed conveyor belt having a tail roller. A plurality of separate discrete conveyor belt support stands are provided. The stands are collapsed together as a group and are moveably supported on a moveable support structure. The stands are joined to adjacent stands by a load bearing connector incapable of supporting vertical loads which permits separation of the stands to a predetermined distance. The tail roller is advanced. The moveable support structure is advanced toward the tail roller. At least one support stand is advanced on the support structure toward the tail roller thereby providing support for any additional length of conveyor belt that is added to the system.

Fig.1A.

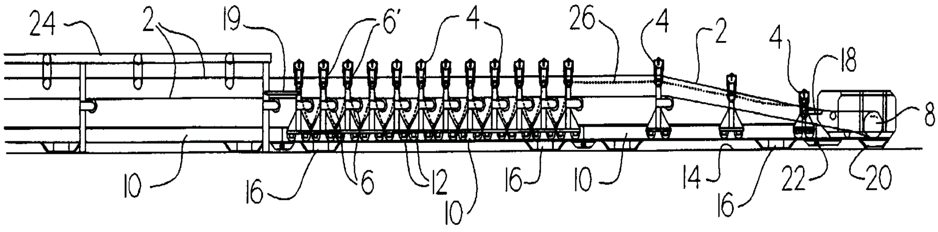
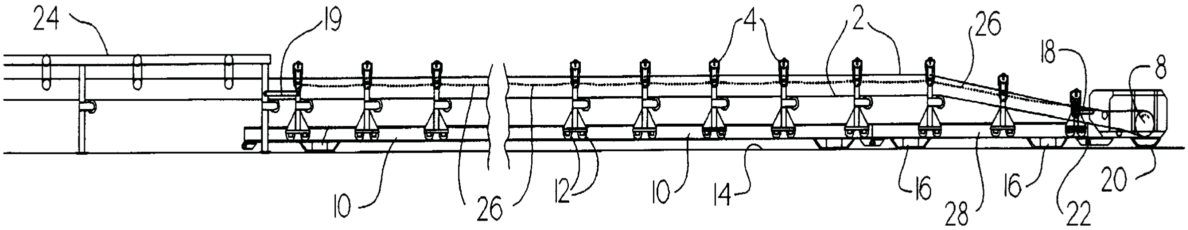


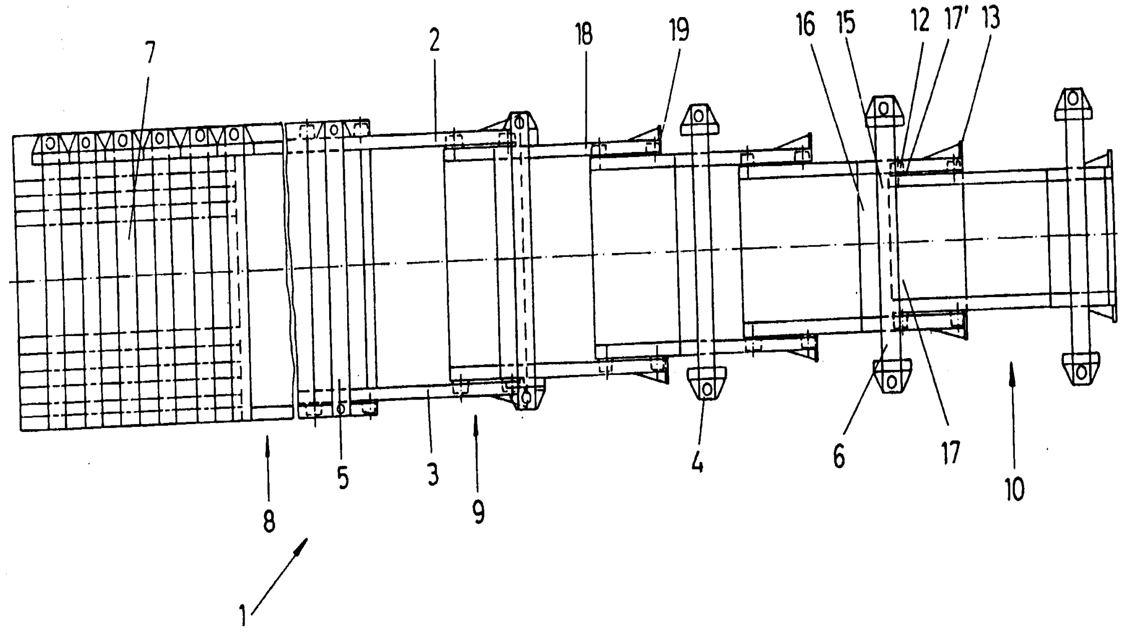
Fig.1B.



Supporting structure for conveyor bands

Abstract of DE 3837986

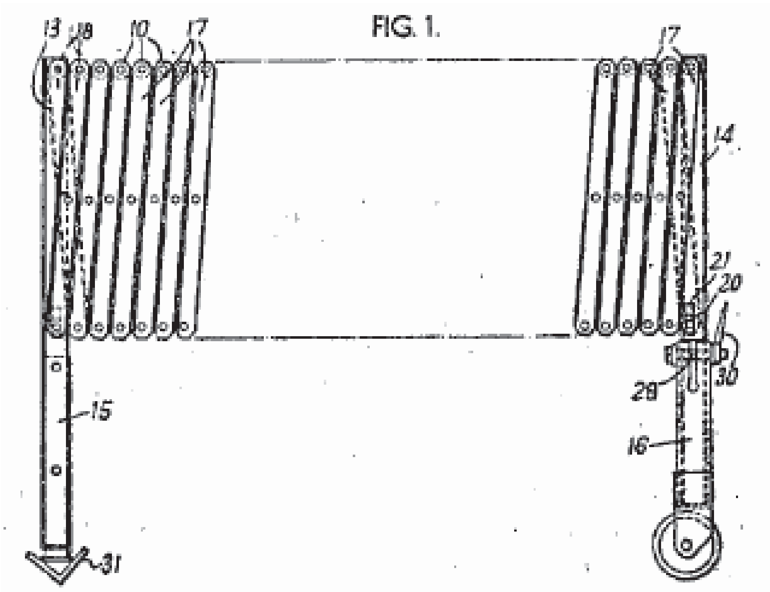
A supporting structure provided for conveyor bands is suitable especially for use with storage-band loops and similar belt-type storage devices, because the supporting spars and therefore the individual portions of the supporting structure are designed to be capable of being pushed telescopically in one another. The individual roller holders for the carrying-run and return-run rollers are designed as roller mounts connected releasably to the supporting spars, so that they can be assigned respectively to a telescope of this type which can be extended individually or in succession in order to lengthen the supporting structure. Conversely, it is possible, in turn, to displace a respective individual telescope or else all the telescopes in succession, in order thereby to shorten the supporting structure correspondingly. The individual telescopes or the supporting spars capable of being pushed in one another are assigned rollers which roll respectively in the larger telescope, so that an advantageous ease of movement of the telescopic supporting structure during extension and retraction is guaranteed.



Extensible roller conveyor

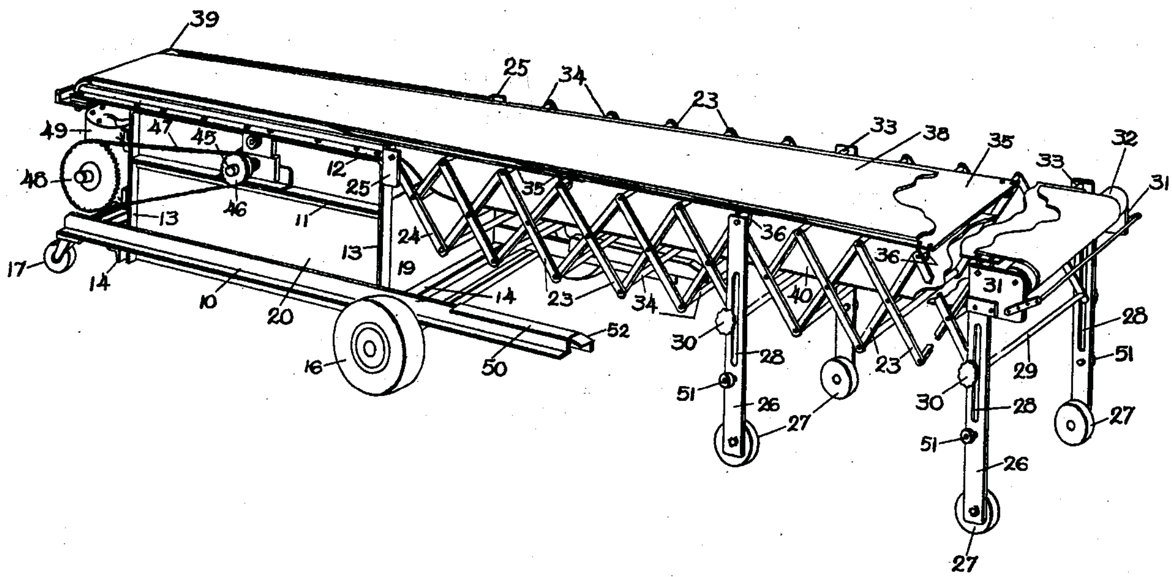
Abstract of GB 1037539

An extensible roller conveyer comprises rollers 10 which are mounted on two pivoted lattice frameworks 11, 12, Fig. 2 (not shown). End supports 13, 14 are vertically adjustable in relation to legs 15, 16 and are slotted to receive slidable blocks 20. These are pivoted to the end lattice members 17, 18 and allow for vertical displacement as the conveyer is expanded and contracted. A modification. Figs. 4-7 (not shown), is arranged to take heavier loads by additionally pivoting the lattice framework members at three intermediate positions. The lattice is prevented from over-extending by check chains 65, Fig. 6. Alternatively a latch 67 may be used, Fig. 7. The conveyer may be in cantilever form, with screws, hydraulic or gear means to control extension of the cantilever.



Extensible belt conveyer

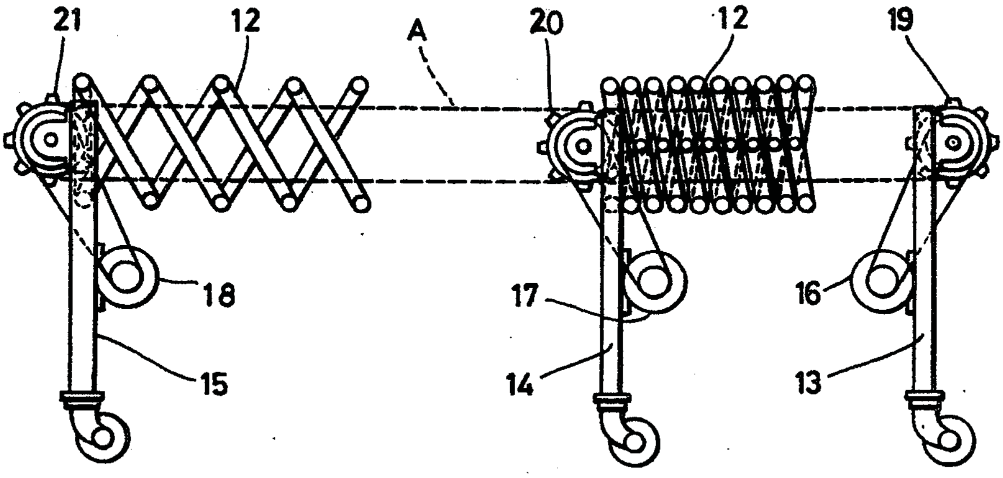
Abstract of US 2576217 *not available*



Flexible conveyor belt

Abstract of US 4260053

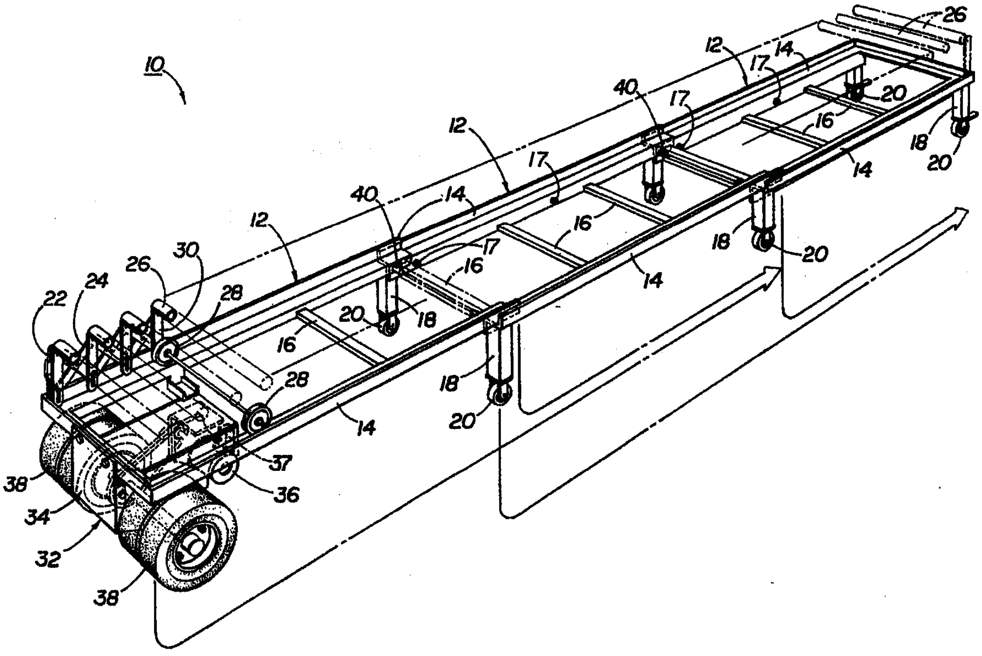
The invention relates to a flexible conveyor belt wherein two pairs of links at one end thereof are respectively and rotatably fitted onto each end of transverse rods disposed with regular spacings from each other. One pair of the said links are rotatably connected with a pair of links fitted onto a preceding transverse rod, while the other pair thereof is rotatably connected with a pair of links fitted onto a succeeding transverse rod. These are characterized in that the centre of a connecting spindle rotatably connecting the links of the said preceding and succeeding transverse rods is adapted to be located below the centre line connecting the centres of the transverse rod receiving holes in the state in which the connected links are rectilineally extended. This enables the flexible conveyor belt to expand, contract and bend systematically with a small curcature radius and smooth and noiseless movement.



Telescoping rigid frame and scissor conveyor

Abstract of US 5490592

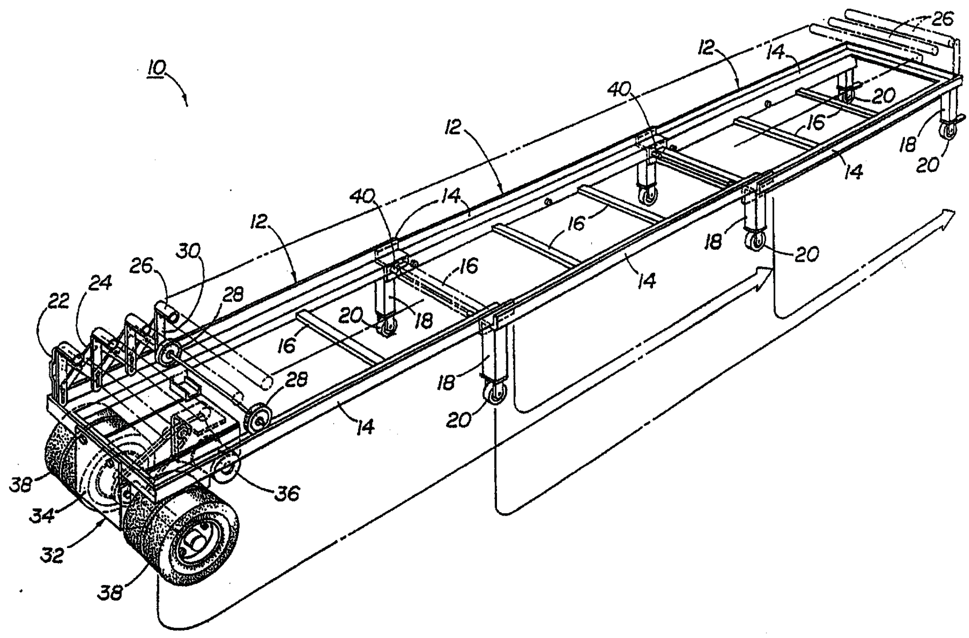
Telescoping rigid frame/scissor conveyors. Such conveyors are formed of a plurality of telescoping frame segments which may be positioned at any number of desired nested/unnested configurations. The frame segments bear a lazy tong or scissors conveyor structure which assures uniform roller spacing, and rollers positioned at one height along the conveyor rather than at different heights for each frame segment. Counterintuitively, the increased structural complexity is more than outweighed by the benefits resulting from the uniformly positioned rollers of a uniform height, which are thus adapted to convey articles in a reliable and jam-free fashion unlike previous expandable rigid frame conveyors. Conveyors according to the present invention may contain automatic locks to prevent overextension, powered rollers and motive devices for automatic and reliable extension and retraction.



Telescoping rigid frame and scissor conveyor with suspension

Abstract of US 6003658

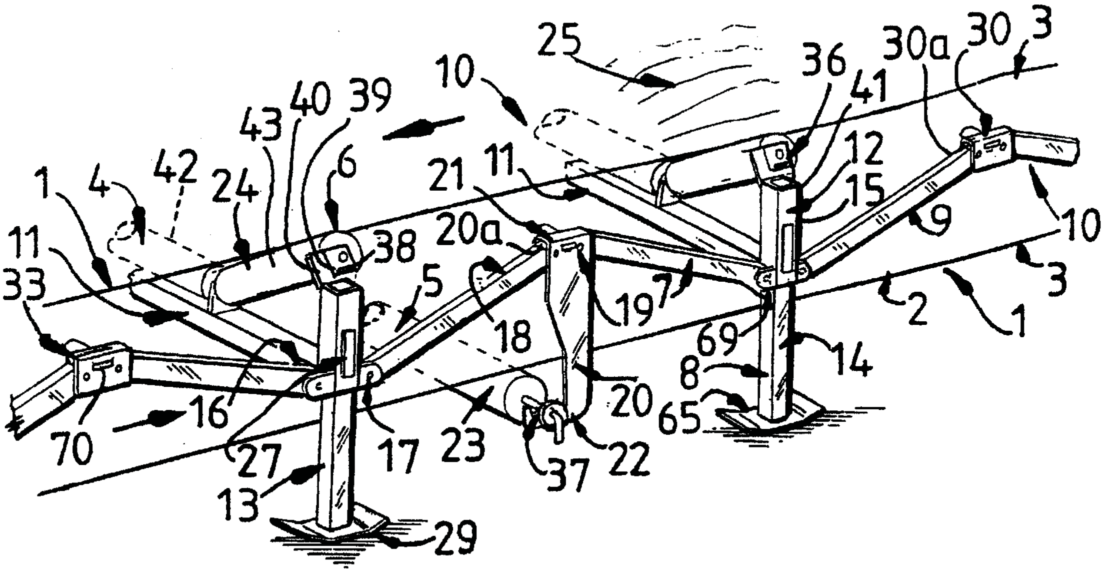
Telescoping rigid frame/scissor conveyors with suspension structures to accommodate less than totally flat surfaces, and for easy and ergonomic repositioning. Such conveyors are formed of a plurality of telescoping frame segments which may be positioned at any number of desired nested/unnested configurations. The frame segments bear a lazy tong or scissors conveyor structure which assures uniform roller spacing, and rollers positioned at one height along the conveyor rather than at different heights for each frame segment. The conveyors are adapted to bear greater weight than conventional lazy tongs conveyors, but contain suspension systems in order to conform to irregularities in factory and warehouse floors and slabs and for easy movement. The suspension systems may include motorized drive units, swivel mounted roller units, jacking roller units and more conventional rigid legs that contain springs or other suspension members to distribute the loads of and borne by the conveyors evenly and efficiently, and to allow the conveyors to positioned, expanded and retracted easily and efficiently.



Conveyor belt support

***Abstract of** US 5850903*

The present invention relates to extensible endless conveyor belt support structures and systems. A support structure has vertical support members (8) connected by V-shaped arrangements of pivotally connected (17, 19) link members (7, 9). By mounting at least the upper belt run rollers (4) on the vertical supports (8) the load can be more effectively supported whilst providing a relatively high ratio of extended to contracted length. By using telescopically extendable (14, 15) vertical supports (8), any increase in overall height of the support structure upon contraction thereof is minimized or avoided. By also having detachable lower belt run rollers (23) the support structure can be inserted into an endless conveyor belt installation without dismantling or even stopping running thereof. The invention also provides systems including setting devices for holding the support structures in their contracted form and deploying and collecting them into and from their extended conditions. Preferably the setting device includes a retractor means for drawing in the support structure during collection thereof.


25
36
30a
30
3
10
41
43
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39
12
1
4
42
24
6
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20a
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18
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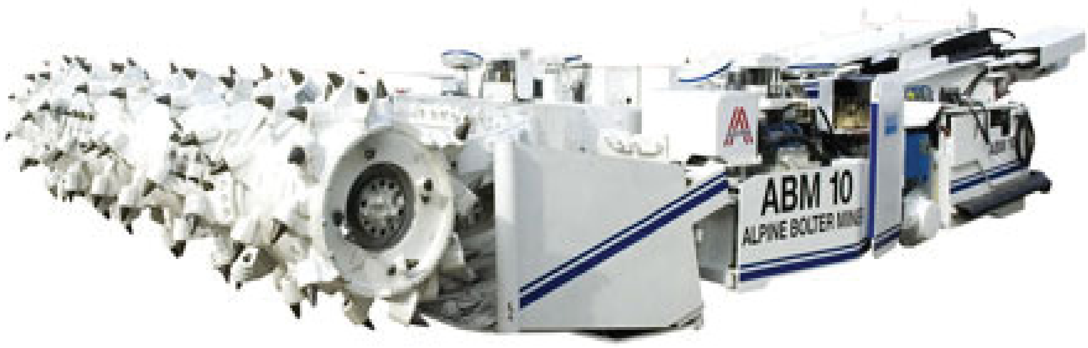
6. Detailed Engineering

6.1. Mining machine

The mining machine has been selected at the very beginning of the process to be able to determine whether it was possible to mine out the hard coal deposit in the available time frame. The mining machine was the first essential component to give the base for the mining system.
If the hard coal deposit has not been possible to be mined out in the available time frame the development of the remaining mining system would not have been necessary.

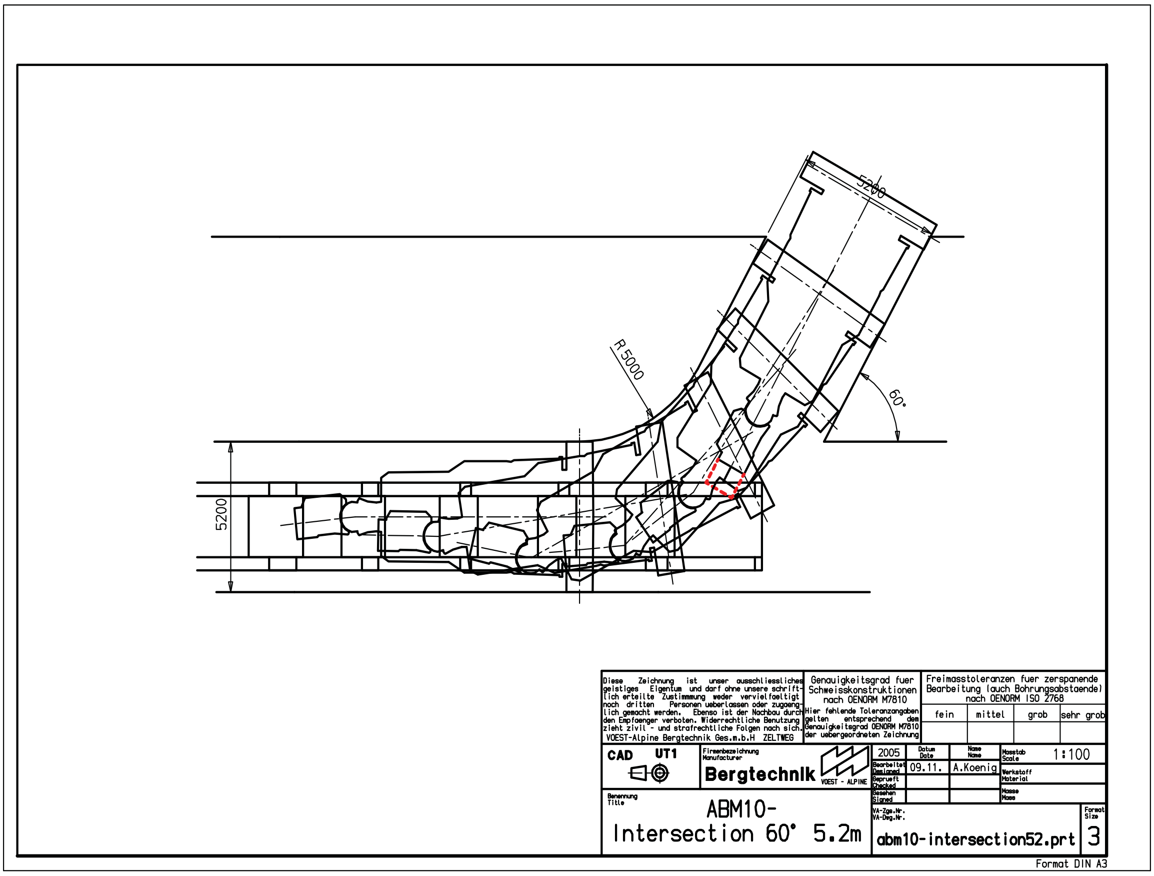
Therefore the Alpine Bolter Miner ABM10 (picture 76) from the mining machine supplier SANDVIK AB has been selected due to the fact that it will be able to mine out the hard coal deposit within the given parameters and time frame and is also able to install the primary roof support at the same time.

The exact and complete machine parameters and dimension have been presented in the previous chapter 5.4.1. and will therefore only be referenced to in the present chapter.



Picture 76: ABM10

The chain conveyor length will most likely be extended by about 900mm. The reason for this is to be seen in the necessity to cut the ventilation connections from one punch into the previous one. By doing so the overlap of the chain conveyor of the ABM10 and the hopper car will diminish to almost zero and the loading without spillage cannot be guaranteed anymore.
To prevent this from happening, the machine design will be altered to have the chain conveyor overhanging the machine rear end by additional 900mm (red extension piece in the following drawing, picture 77).



Picture 77: **Chain Conveyor Extension**

The ABM10 is a fully remote controlled machine. Only the roof drill rigs including the temporary roof support cylinders (one unit each side) are manually operated.

All functions necessary to stabilize the machine, to cut the coal, to gather and transport the cut material to the rear of the machine as well as all functions for traming are remote controlled.

6.2. Hopper Car

The hopper car, due to its special requirements (operating height, counter weight for the belt conveyor, pulling function for the pretensioned conveyor belt) and the task it has to perform (storage, release and retrieve function for conveyor structure) had to be engineered and designed completely new by the author. The biggest challenge for the hopper car is the extremely low operating height. As mentioned before the cutting height of the ABM10 in the available hard coal seam has to be within 1200mm.
The loading area of the hopper car has to fit below the chain conveyor of the ABM10 and has to cover a range of 3000mm overlap with the chain conveyor to guarantee a continuous loading process also when the ABM10 is advancing and the hopper car cannot follow immediately.

This can happen when, for example, problems occur with the installation of the conveyor structure and the time to install the stringers and idler stools takes longer than programmed. During this time the ABM10 can advance further without disturbing the material flow. The overlap of 3000mm gives the possibility to have the hopper car standing still and the ABM10 cutting for 3 full cutting sequences.

The chain conveyor of the ABM10 has got a height of approximately 450mm in the area where the conveyor drive motor, gearbox and return sprocket sit.
A space of around 100mm at the minimum has to be given from the top of the chain conveyor structure to the roof so the conveying of the material is not blocked.
This 100mm plus the 450mm give an overall dimension from the roof to the bottom of the chain conveyor of 550mm and a remaining height of 650 in which the hopper car has to fit in.
A certain clearance of the hopper car structure to the chain conveyor has to be given for going through tips and humps without the two steel structures of the machines to collide. 150mm are needed as a bare minimum.

This results in an available **hopper car height of 500mm**.

500mm available height is causing a serious problem when moving the hopper car around the underground environment where the ground and floor conditions are far away from favorable. Mud, spillage, rocks and coal lumps, only to mention some obstacles are making it necessary to give the hopper car enough ground clearance to not getting bogged and stuck.

A ground clearance of 250mm of all components of the hopper car was a value set by experience. The mining machine manufacturer Sandvik AB is using this ground clearance figure as a minimum with all their machine designs.

For our application the ground clearance of 250mm is absolutely not possible as inside the remaining 250mm hopper car height all necessary components would have to fit.

Those necessary components are:
- Return pulley
- Conveyor structure with carrying idlers and return idlers

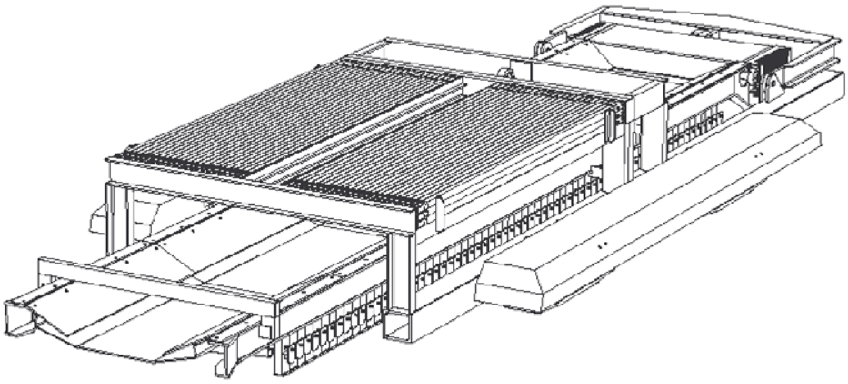
Also space for the mined out material has to be made available to be thrown from the chain conveyor onto the conveyor belt of the hopper car.

A way had to be found to get enough ground clearance whilst not wasting any height for the conveyor structure and the loading and filling height of the conveyor belt.

The only practical solution found was to design a main frame where the tracks are mounted onto and an additional frame inside which can be lifted for moving and lowered for the loading process.
This lifting frame would be able to utilize the space under the chain conveyor of the ABM10 which was normally used for operational clearance reasons (described before).

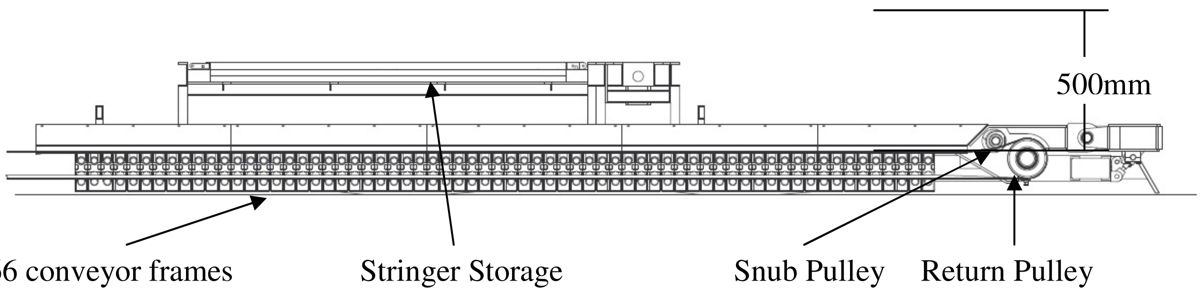
One additional advantage had to be utilized.

The Hopper car height is only supposed to be 500mm beneath the chain conveyor of the ABM10. This means only over a dimension of 3000mm. Behind the 3000mm loading section the hopper car steel structure can get up to around 1000mm without any problems with the clearance to the roof (see also picture 78 and 79).



Picture 78: **Hopper Car Lay-out**

66 conveyor structure frames have to be stored in the hopper car. Those conveyor structure frames have to slide in and out of the hopper car when it advances, retreates respectively.
They also have to move free and safe when the conveyor belt is running over them, as well as when the material is getting loaded onto the belt by the chain conveyor of the ABM10.



Picture 79: **Hopper Car Main Components**

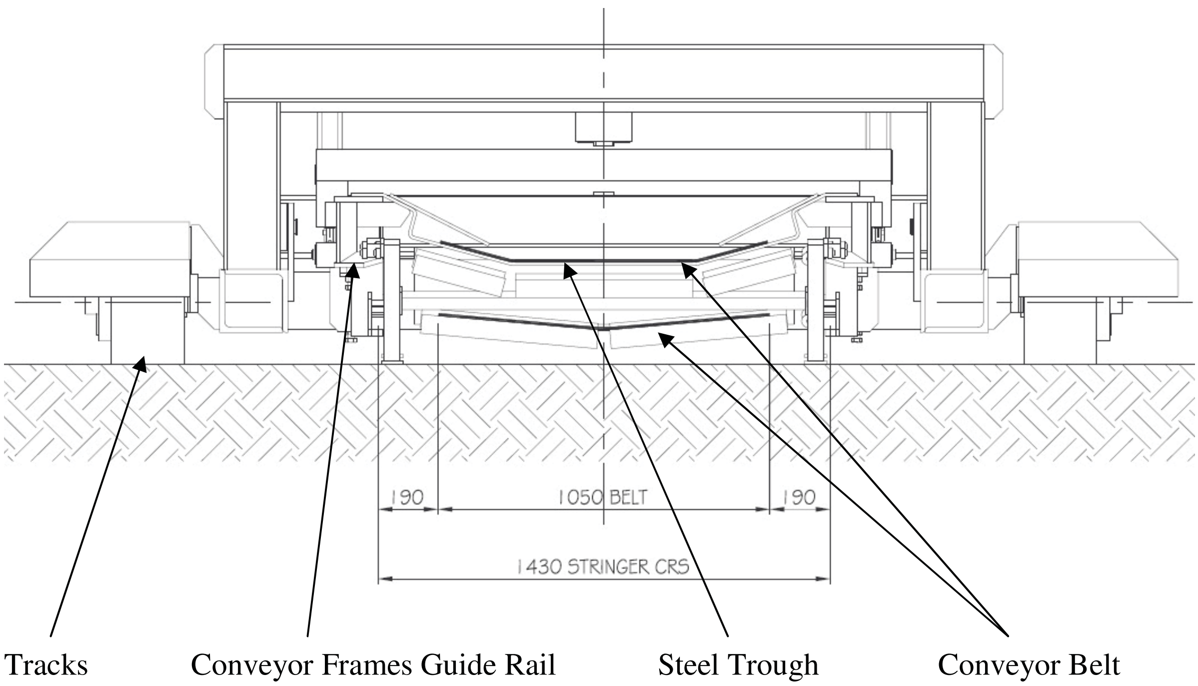
Due to the fact that the conveyor belt and the conveyor frames will move in the same direction when advancing behind the ABM10, the danger of the frames getting pulled out uncontrolled by the moving conveyor belt (due to the friction of the belt onto the carrying idlers) has to be avoided by all means.
This will be a safety concern for the authorities in the underground coal mine.

The running conveyor belt and the conveyor carrying idlers in the frames have to be separated to reduce the risk of such an uncontrolled action.
It was decided by the author that the conveyor belt and the idlers will have to be separated via a steel trough.

The advantage of a steel plate trough will be three fold:

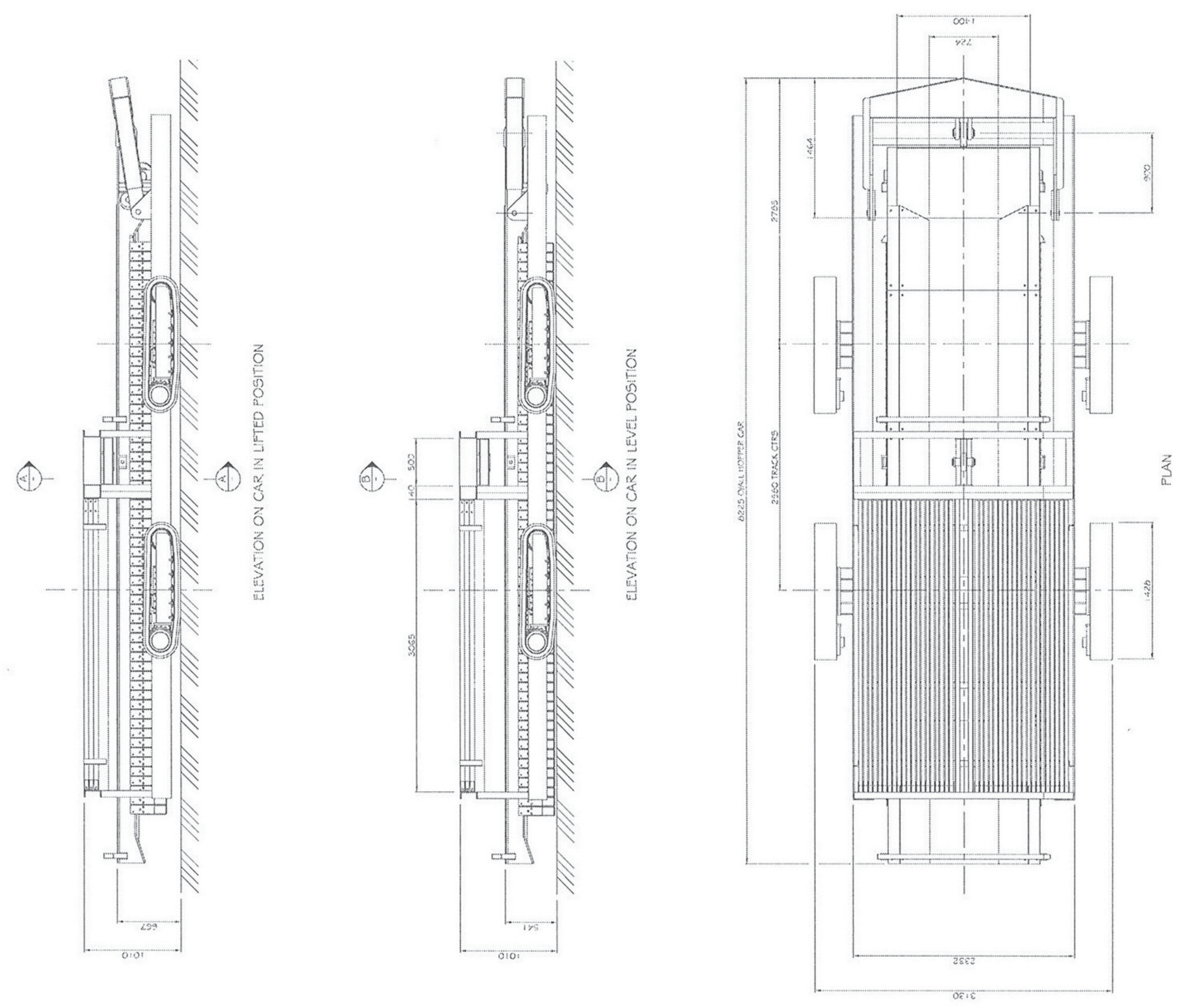
- No spillage through the trough onto the conveyor frame sliding mechanism
- No spillage through the loading area onto the floor
- No contact of the moving conveyor belt and the carrying idler which avoids contact with moving parts for the operators

With all the conveyor frames in the hopper car and the return pulley at the front end the hopper car will get to a length of around 8300mm which will be the top limit to get around corners.

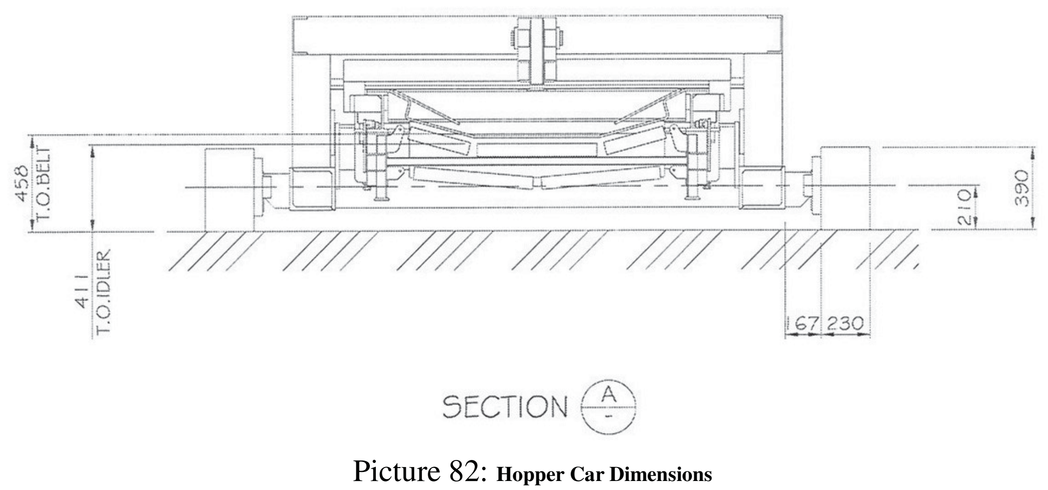


Picture 80: **Hopper Car Cross Section with Main Components**

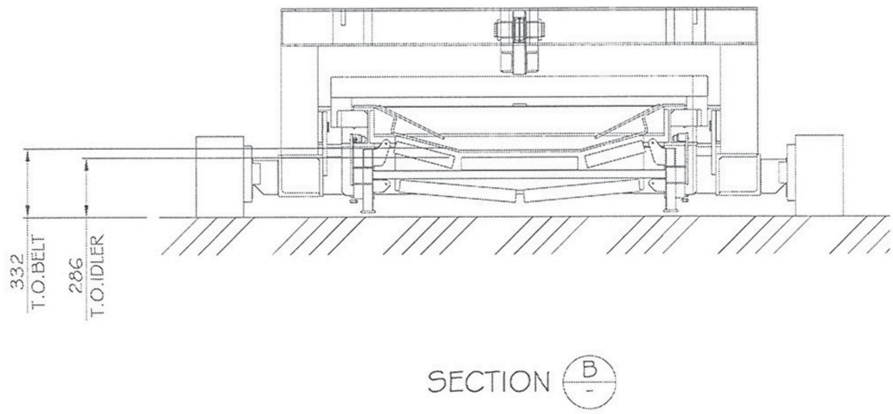
Picture 80, 81 and 82 shall visualize the hopper car with its main components as well as the main dimensions in the relevant characteristic drawings and sections through the main body of the hopper car.



Picture 81: **Hopper Car Dimensions**

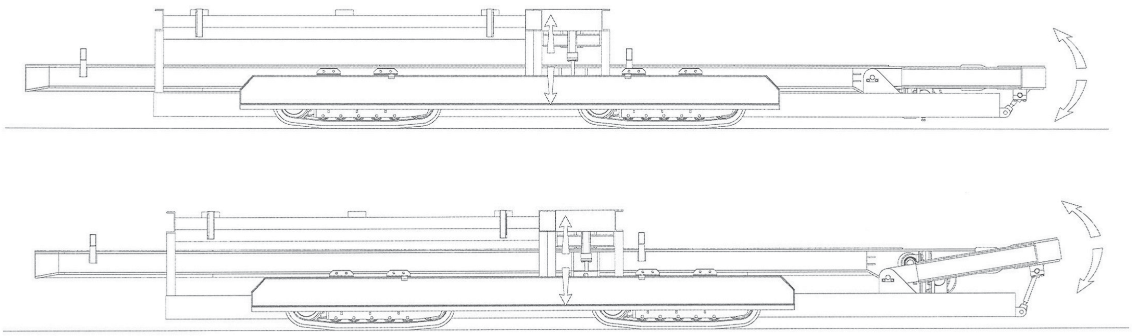


Picture 82: **Hopper Car Dimensions**



Picture 83: **Hopper Car Dimensions**

To use all possible width for the loading, the width of the hopper car was set to the ABM10 width of around 3100mm.
Lifting and lowering of the inner parts of the hopper car (picture 84) will be done via three hydraulic cylinders. One cylinder will be mounted at the front right in the center of the lifting frame and two after the loading hopper where the increased height can already be utilized for easier design.
The position of the cylinders is indicated with the arrows of the movement range.



Picture 84: **Hopper Car Inner Frame Lifting Function**

The hopper car has to be equipped with a track system, one each side.
The track design of the ABM10 had been used in the initial design due to the similarity of the components for the holding of spare parts afterwards.
The tracks from the ABM10 are having a multiple speed feature which is actually unnecessary for the hopper car. It requires a gearbox to run the multiple speed hydraulic motor (change of piston set angle will change the volume of the motor and therefore the rotation speed) and the hydraulic fluid volume needed to run each one of the motors is beyond the 100l/min range. This would mean a quite large hydraulic power pack to supply the necessary fluid power for running the tracks.

The maximum speed the hopper car will have to produce will be around the 5,0 to 6,0 meters/minute figure.
This is the anticipated/determined maximum speed necessary to have enough time to pull out conveyor frames, push conveyor structure back in or relocate the hopper car with the other material handling system.

Another design issue is the length of the hopper car. The ABM10 tracks are around 3500mm long, which would, when arranged in the middle of the hopper car frame, make the whole hopper car unstable when traming. The instability will come due to the too small foot print of the tracks and the over hanging loads on each side of the track frames.

During the early design stages of the hopper car it was found by the author that there was virtually no space for a hydraulic power pack to run the hydraulic tracks and to move the hydraulic cylinders, nor was there any space for an electric box to run electric motors for the tracks.
The consequence of all was that the ABM10 tracks were not suitable for our design (size wise and power wise).

The biggest advantage of the ABM10 track frames, the overall track height of around 450mm, has to be kept though, when looking for alternatives.

A further important decision had to be made by the author together with the manufacturer of the ABM10. The hydraulic power necessary to run the hopper car tracks and cylinders will be supplied by the ABM10.
The ABM10 has got a large hydraulic power pack which can supply the hydraulic power for traming with the only condition that the two units (the ABM10 and hopper car) will tram at different times. Together (at the same time) they cannot move as there would not be enough hydraulic power/fluid available. Because of the 3000mm overlap between the hopper car's loading area and the chain conveyor of the ABM10 this is not seen to be a problem.
The connection between the hydraulic power pack of the ABM10 and the hopper car will be done via hydraulic hoses.
This decision brought a big advantage from the design space point of view and will make the over all engineering much simpler.

Designs from different track frame suppliers were investigated to suit the low height application.

6.2.1 Track System

Various track system suppliers from around the world had been investigated and also already running mining equipment with such track frames had been looked at.

The research brought that the majority of mining and construction machines were using standard of the shelf track frames from companies like Caterpillar, Komatsu and Intertractor.

Whilst Caterpillar and Komatsu do not have very small track frames in their standard offering program, Intertractor had several small designs available (see picture 85 and table 19).

Picture 85: **Track Frames;** Source: www.intertractoramerica.com, brochure intertractor-america

Model	Link Pitch (mm)	A (mm)	B (mm)	H (mm)	D0 (mm)	D1 (mm)	G* (mm)	Output Torque**
PG-CS-B00	90	1070	1421	363	305.3	216	230	2.2
PG-CS-B00	90	1377	1750	386	305.3	216	230	2.2
PG-CS-B00	90	1495	1838	395	277.2	216	230	1
PG-CS-B01	101.6	1610	2091	483	376.6	300	300	4
PG-CS-B0	125	1780	2310	555	424.1	335	300	10
PG-CS-B21	135	1505	1984	515	373.7	269	300	11
PG-CS-B1	140	1505	2085	587	475	380	400	29.5
PG-CS-B1	140	1990	2580	670	475	380	450	18
PG-CS-B1	140	1990	2580	670	475	380	450	10
PG-CS-B1	140	2493	3010	524	415.8	330	400	9.5
PG-CS-B1	140	2560	3197	730	475	380	300	13
PG-CS-B1	140	2760	3347	680	475	380	500	20
PG-CS-B2	155.6	2185	2850	740	527.9	464	400	30
PG-CS-B2	155.6	2880	3535	756	527.8	464	450	30
PG-CS-B3	155.6	3670	4331	707	527.9	464	700	30
PG-CS-FL6	160	2913	3640	736	592.6	490	500	23
PG-CS-FL6	160	3310	4037	736	592.6	490	500	30
PG-CS-B4	171.1	2490	3290	847	634	536	500	30
PG-CS-B4	171.1	2946	3708	931.5	580.4	536	600	30
PG-CS-B4	171.1	3300	4085	818	634	507	400	44
PG-CS-B4	171.1	3610	4410	985	634	536	500	34
PG-CS-B4	171.1	4060	4850	910	634	507	550	55
PG-CS-B4	171.1	4095	4850	950	634	453	600	44
PG-CS-B5	175.4	3720	4522	833	650.5	510	500	60
PG-CS-B5	175.4	4980	5883	960	743	605	500	100
PG-CS-B60	190	3140	3968	881	704.2	510	400	60
PG-CS-B60	190	3020	4741	952	644.6	550	500	60
PG-CS-B60	190	4380	5211	878	644.6	550	500	60
PG-CS-B6	202.8	4280	5205	945	751.7	594	500	100
PG-CS-B7	215.9	4030	5036	1231	800.23	640	600	100
PG-CS-B7	215.9	4900	5906	1174	800.23	640	500	130

Table 19: **Track Frame Dimensions;** Source: www.intertractoramerica.com, brochure intertractor-america

Specifically the Model PG-CS-B00 came to the attention of the author with and overall height of around 400mm and a length of around 1800mm. Also the torque output of the tracks 2,2kNm seemed to be the right figure for our application.

After contacting the company Intertractor America (due to the close vicinity of the company to our mine site), the engineers of Intertractor came up with the following proposal:

DRIVE CALCULATION		Intertractor Passini Group	
Machine type:	*Conveyor*	Offer -No.:	*U060310*
Running Gear Size:	*B00*	Customer:	*Sandvik Minning VAMT*
Chain pitch:	*90,00*		
Project data			
G = total weight of machine [kg]:	4000	req. Machine datas at:	Vg max.
A = Axle distance [m]:	7,5	Pump volume [l/min] :	5,4
E = Gauge [m]:	2,27	System pressure [bar]:	200
μH = type of shoe :	0,8	Pump power [kW]:	2,04
b = shoe width [cm]:	23	speed [km/h] :	0,48
pm = med ground pressure [kg/cm²]:	0,12		
Rolling resistance μR = 0,1 - 0,15 :	0,1	Power and volume for each chain drive !!!!	
Efficiency ηK :	0,88		
ηK = 0,9 low ηK = 0,85 medium ηK = 0,8 high		μH 1bar =0,9 - 1,1 μH 2bar =0,8 - 0,9 μH 3bar = 0,7 - 0,8 rubber = 0,7 - 0,9 μH flat = 0,5 - 0,6	
Drive layout			
Tractive effort required		Tractive effort provided by drive	
		Gear size :	701C
		Ratio i = :	6,2
			Vmax
		Motor size :	100
		Sprocket z1 = :	21
KS slewing [daN]:	2643	Sprocket do [mm]:	305,3
KF driving [daN]:	200	Motor speed [min⁻¹]:	54
		Gear output [daNm]:	187
KT total tractive effort [daN]:	2.843	KT total available per drive [daN]:	1.081
		KT total available per side [daN]:	2.161
ability to climb [°]		26	

Sheet1 / Sheet2 / Sheet3 /

Table 20: **Calculation by Intertractor America**

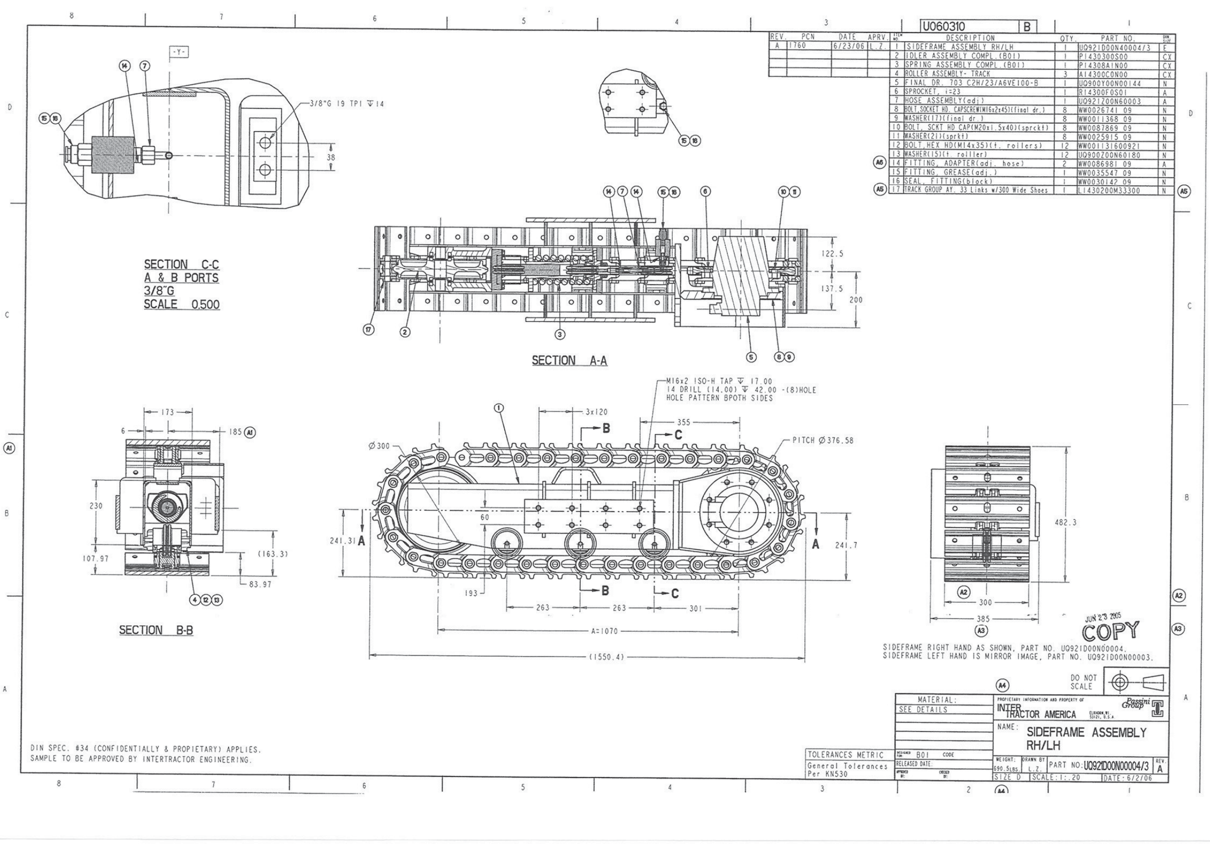
The basic information and parameters given to the engineers of Intertractor were:

- Maximum weight of the hopper car 4000kg
- Maximum pulling force of the hopper car 2000kg (belt tension x 2)
- Operation in mud and coal (lubrication effect of wet coal dust)

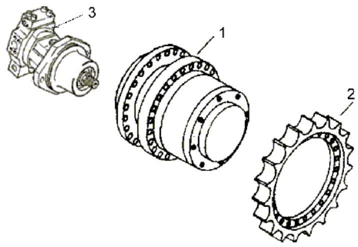
- Maximum operating speed 8m/min
- Maximum hydraulic pressure available 250bar
- Maximum hydraulic flow available at this pressure 200l/min
- Track frames as small as possible with a maximum height of 450mm

The result given back from Intertractor was more than satisfying.
Instead of two tracks in total, two each side (four in total) have to be used. This was a very good thing as the two track frames could be arranged in a way that the machine with its 8000mm length can run stable whilst still being able to move around corners with 5000mm corner radius.

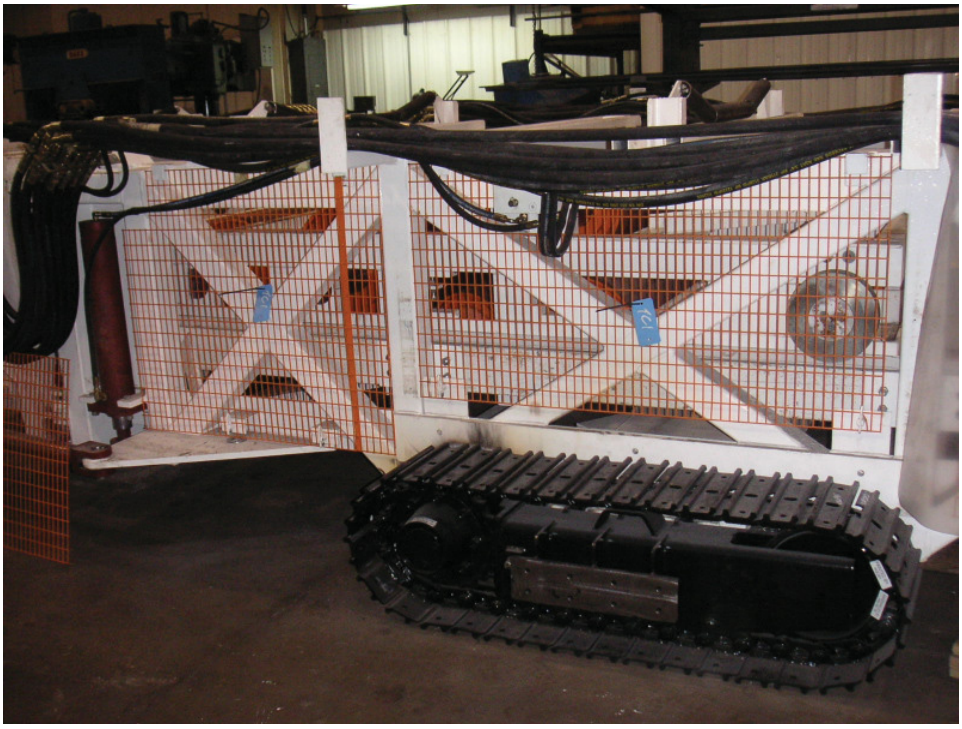
Not only were the costs of the tracks only a fraction of the tailor made heavy duty track frames of the ABM10 (those track frames are able to stand static forces of around 40 tons each and can bring up a pulling force of over 30 tons >> over kill for the application in the hopper car!!) but also the necessary hydraulic power was only 2,1kW (5,4l/min @ 200bar) per track and therefore a total of maximum 10kW (max 22l/min @ 200bar) when moving with a speed of 8m/min or 0,48km/h.
The following drawing (picture 86) is presenting the track as it was used in the final design. The overall height of 480mm and a length of 1550mm were still within the possibilities of the engineering of the hopper car. Picture 87 shows the track drive motor of the assembly.



Picture 86: **Track Frame Dimensions;** Source: www.intertracktoramerica.com, part of calculations by intertractor-america



Picture 87: **Track Drive Motors;** Source: www.intertracktoramerica.com, part of calculations from intertractor-america



Picture 88: **Track Frames on the finalized machine**

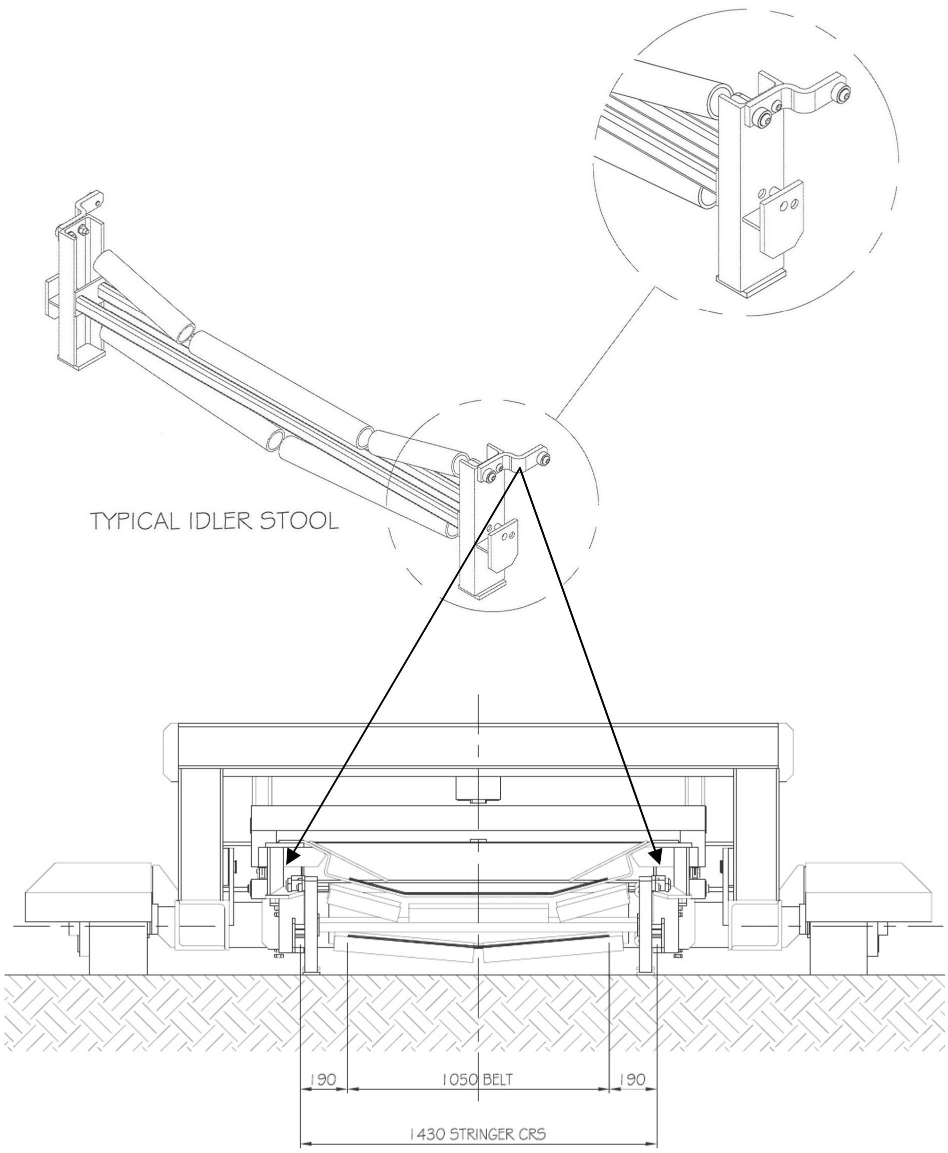
Due to simplicity reasons the proposed track frames will be used through out the whole machine. A photo of the finished product is shown above in picture 88.

6.2.2. Conveyor Structure Storage and Guiding Rail System

One of the most important things in the design of the hopper car is the storage of the conveyor frames inside the hopper car steel structure. The system had to be designed new by the author.

The main feature of the material handling system is the possibility to follow a coal cutting machine on its own and to advance forward and backward via the driven hopper car without

adding/taking out the conveyor structure from/into cradles brought by transporting machines to the extending conveyor.
The advantage and main feature of this new system shall be, that all necessary components for extending the belt conveyor, conveyor structure with rollers and the conveyor belt, are stored on or inside the material handling system so that nothing has to be transported to and from the system during the operation.

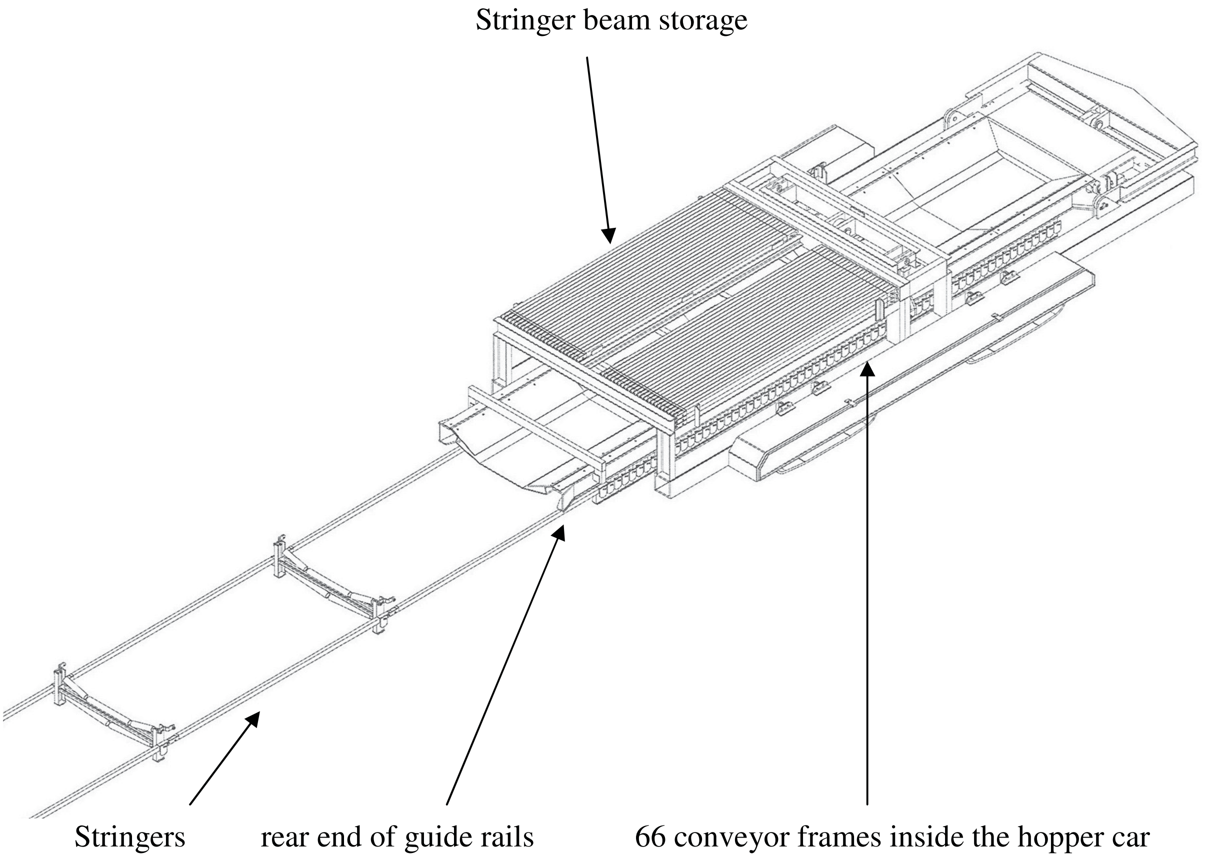


Picture 89: **Slide In/Out Devices in Hopper Car**

All 66 conveyor frames have to be stored inside the hopper car structure and have to be designed in a way that they move in and out in very reliable and easy way.
For that reason the conveyor structures had to be equipped with rollers or wheels so roll in and out on a rail (picture 89 and 91).

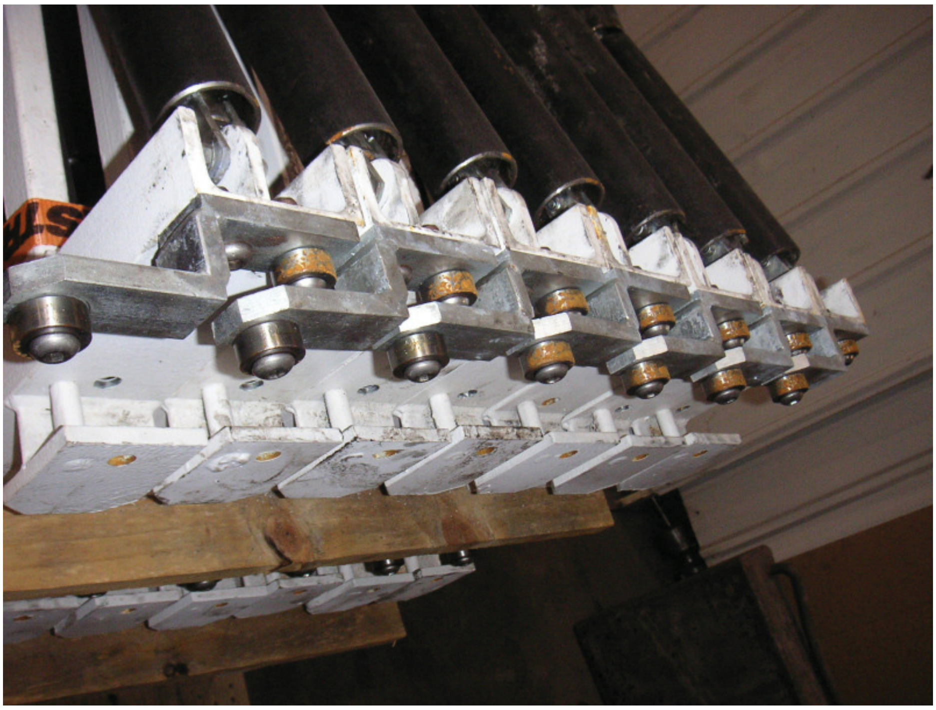
The roller or wheel arrangements on the conveyor frames for rolling in and out of the hopper car lifting frame rails have been designed to give the rollers enough spreading distance for a stable stance.
They also have to be accessed from the outside of the hopper car so that also the very last frame can be pulled forward by hand.

The rear end of the hopper car has to be designed in a way that the rails lower the frames onto the ground and also collect them back into the hopper car in a smooth way. This will be done by designing a sort of funnel for the collection when the frames are deviating from the centre line and also the rail was getting lowered at the end for the lowering to the ground (picture 90).



Picture 90: **Conveyor Extension System**

To safe space the vertical frame posts are arranged shoulder to shoulder whilst the wheels are arranged in the following way inside the hopper car guide rail.



Picture 91: **Sliding Devices on Conveyor Frames**

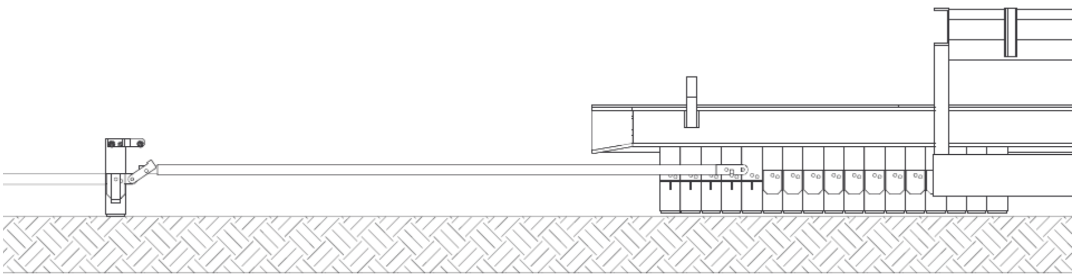
Stringers between the conveyor frames have to make sure that the structure will stand securely on the floor and will not drop over.

The storage of the stringer beams will be in the rear part of the hopper car where the height is not as critical as in the front part of the machine (compare to picture 90).

6.2.3. Conveyor Structure Pull-Out Sequence

The conveyor extension process will be explained in the following chapter

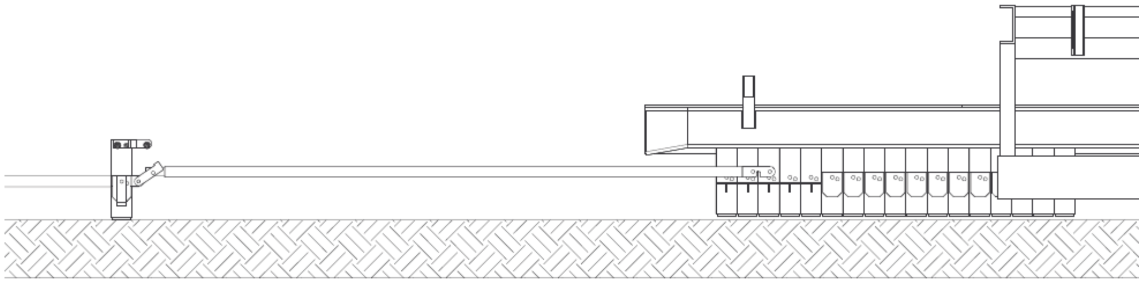
Step 1 (picture 92):



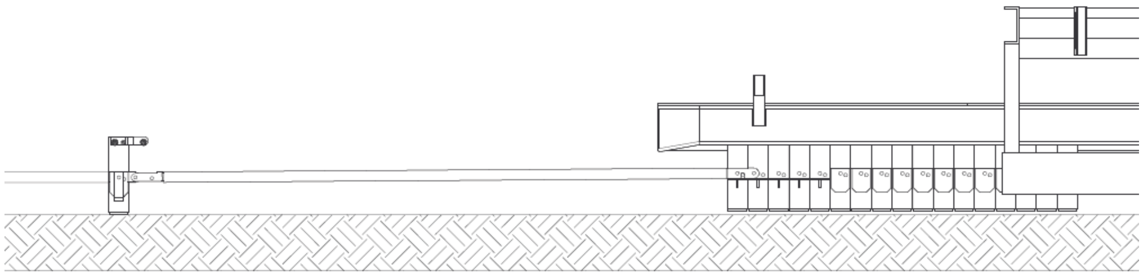
Picture 92: **Conveyor Structure Pull-out Sequence**

Conveyor frames inside the hopper car are all sitting inside the guide rail system suspended around 50mm above ground. The carrying side of the conveyor belt is getting loaded in the steel trough above the frames without touching the carrying idlers of the frames inside the hopper car. The return belt is running on the return roller of the frames inside the hopper car and keeping the frames inside due to the friction force of the running belt applied on them.
The stringer is attached to the last conveyor frame already outside the hopper car, standing on the floor and the conveyor belt running on both idler sets, the carrying set and the return set.

Step 2 (picture 93 and 94):



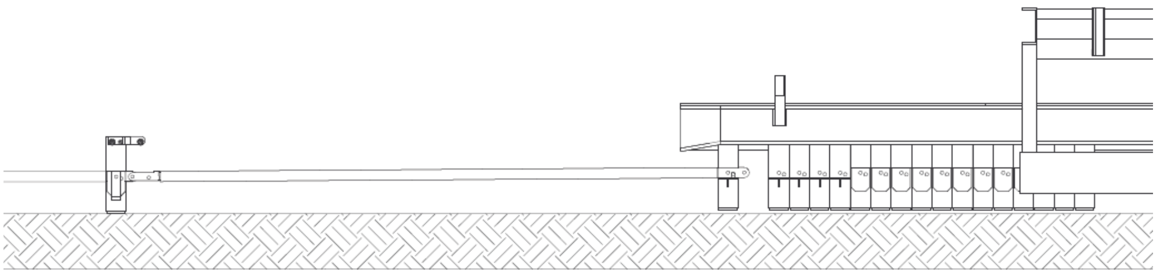
Picture 93: **Conveyor Structure Pull-out Sequence**



Picture 94: **Conveyor Structure Pull-out Sequence**

As the hopper car advances forward the attached stringer in getting dragged along the stored frames inside the hopper car to the rear end towards the last frame in the row.

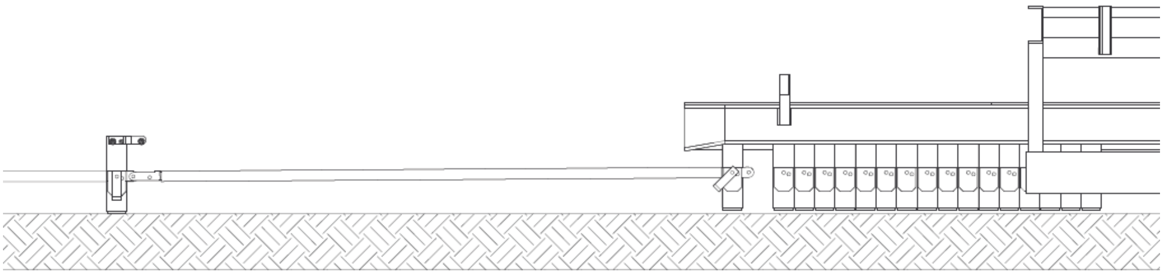
Step 3 (picture 95):



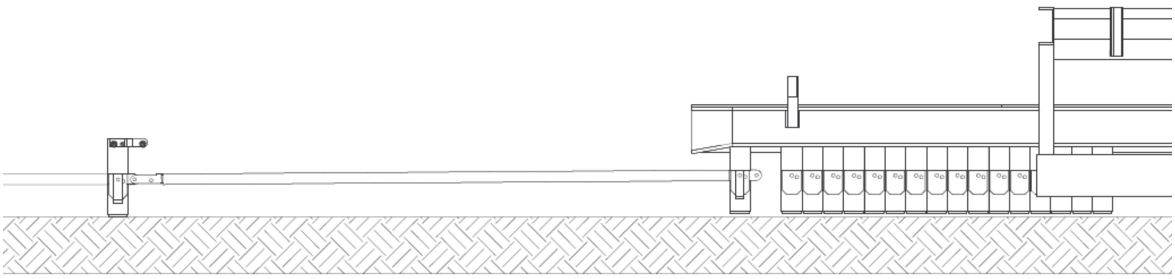
Picture 95: **Conveyor Structure Pull-out Sequence**

Arriving at the last frame in the row gravity lets the stringer fall onto the locking pin. The stringer end has got a slotted end which catches the last frame at the pin inside the flat stringer bracket (also see picture in chapter ‘Conveyor Frames and Stringers’).

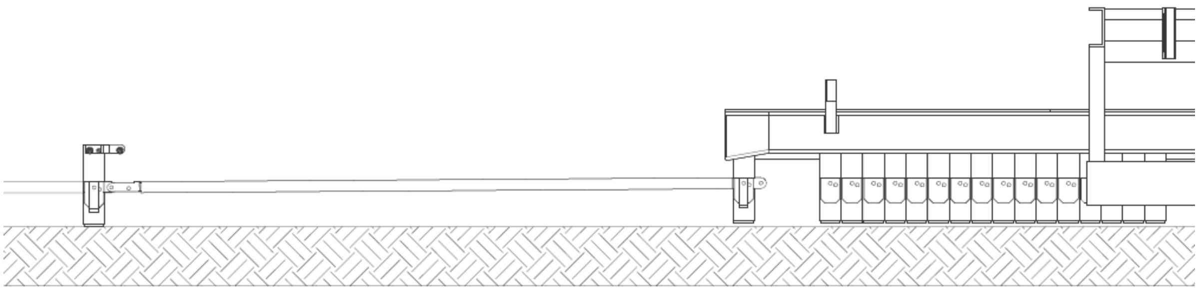
Step 4 (picture 96, 97, 98):



Picture 96: **Conveyor Structure Pull-out Sequence**



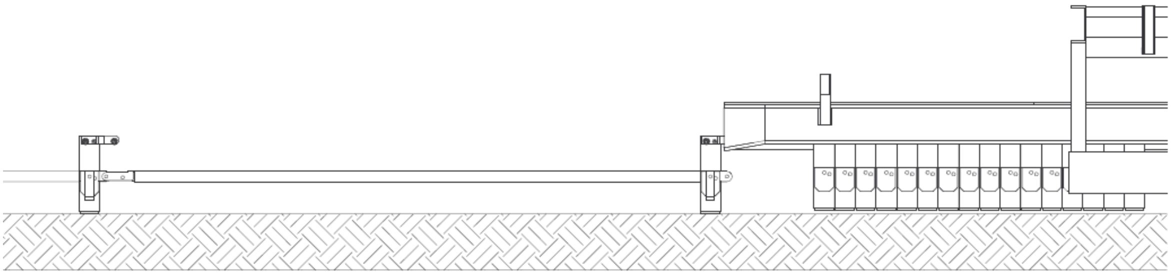
Picture 97: **Conveyor Structure Pull-out Sequence**



Picture 98: **Conveyor Structure Pull-out Sequence**

The stringer locked onto the frame via the pin/slot connection is getting pulled out when the hopper car advances till it reaches the end of the guide rails. A moving flap is designed to secure the stringer and frame so it cannot disengage outside the hopper car.

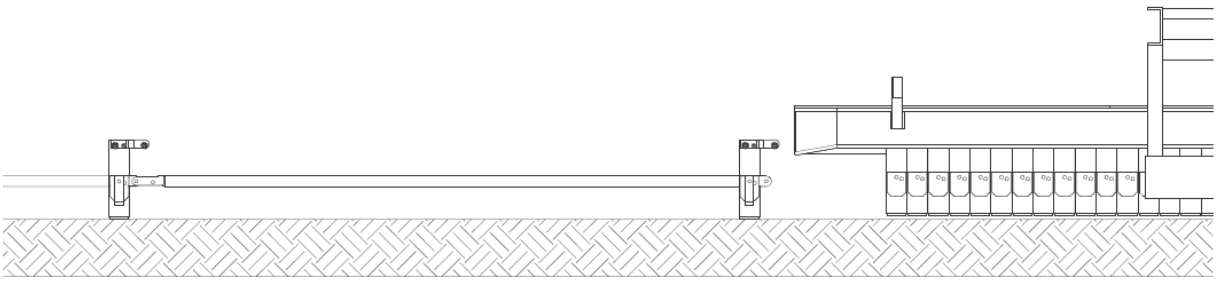
Step 5 (picture 99):



Picture 99: **Conveyor Structure Pull-out Sequence**

The end of the guide rail is bent downwards to smoothly lower the frame onto the ground

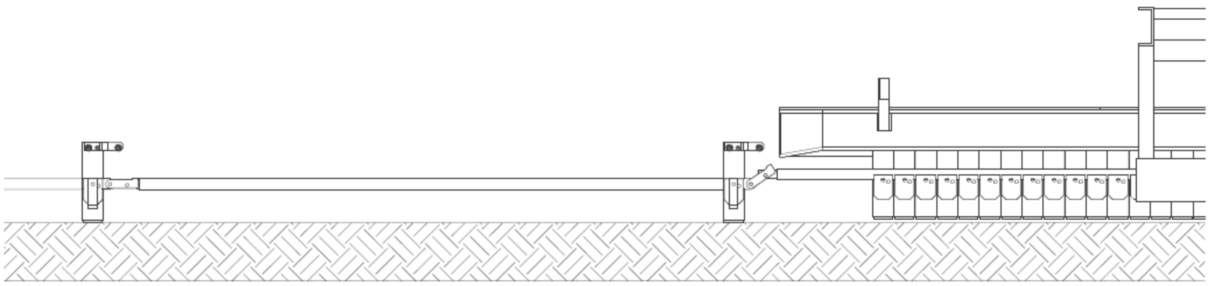
Step 6 (picture 100):



Picture 100: **Conveyor Structure Pull-out Sequence**

The stringer maintaining the idler spacing of 3000mm is installed, the conveyor frame is sitting on the floor. At this point the conveyor belt will leave the rear lip of the steel trough and is lowering itself onto the carrying idlers of the conveyor frames.

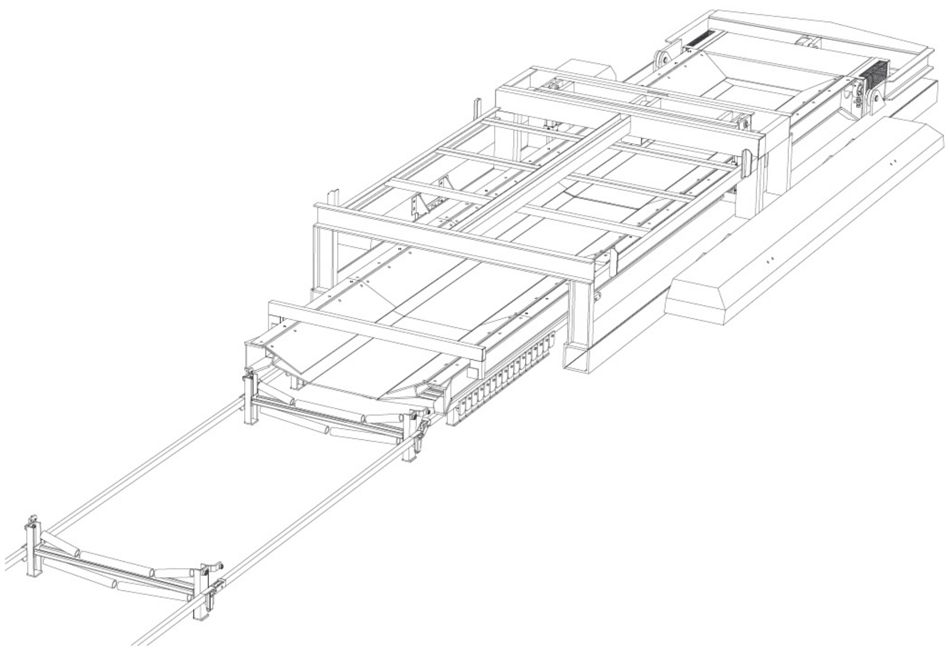
Step 7 (picture 101):



Picture 101: **Conveyor Structure Pull-out Sequence**

A new stringer is getting attached to the freshly installed conveyor frame and the sequence starts again. The front end of the stringer is hinged too to allow for the height differences of hopper car and installed frames.

All manual work to attach the stringers is outside the danger zone of the running belt and easy accessible from the outside of the hopper car (compare 102).



Picture 102: **Conveyor Structure Pull-out Sequence**

6.2.4. Hydraulics

As discussed before the hydraulic power to run the track system and to activate the hydraulic cylinders will be delivered from the ABM10 during the mining and retreat action. The connection will be done via free hanging and protected hoses.
Whilst the track system of the hopper car can only work in sequence with the ABM10 tracks (limited hydraulic flow potential), the hydraulic cylinders of the hopper car can be operated at whatever time necessary.

There need to be 6 hydraulic hoses connected to the hopper car:

- Left hand side track motors 2 hoses for pressure and return oil
- Right hand side track motors 2 hoses for pressure and return oil
- Manual directional control valves for the cylinders 1 pressure, 1 one tank line

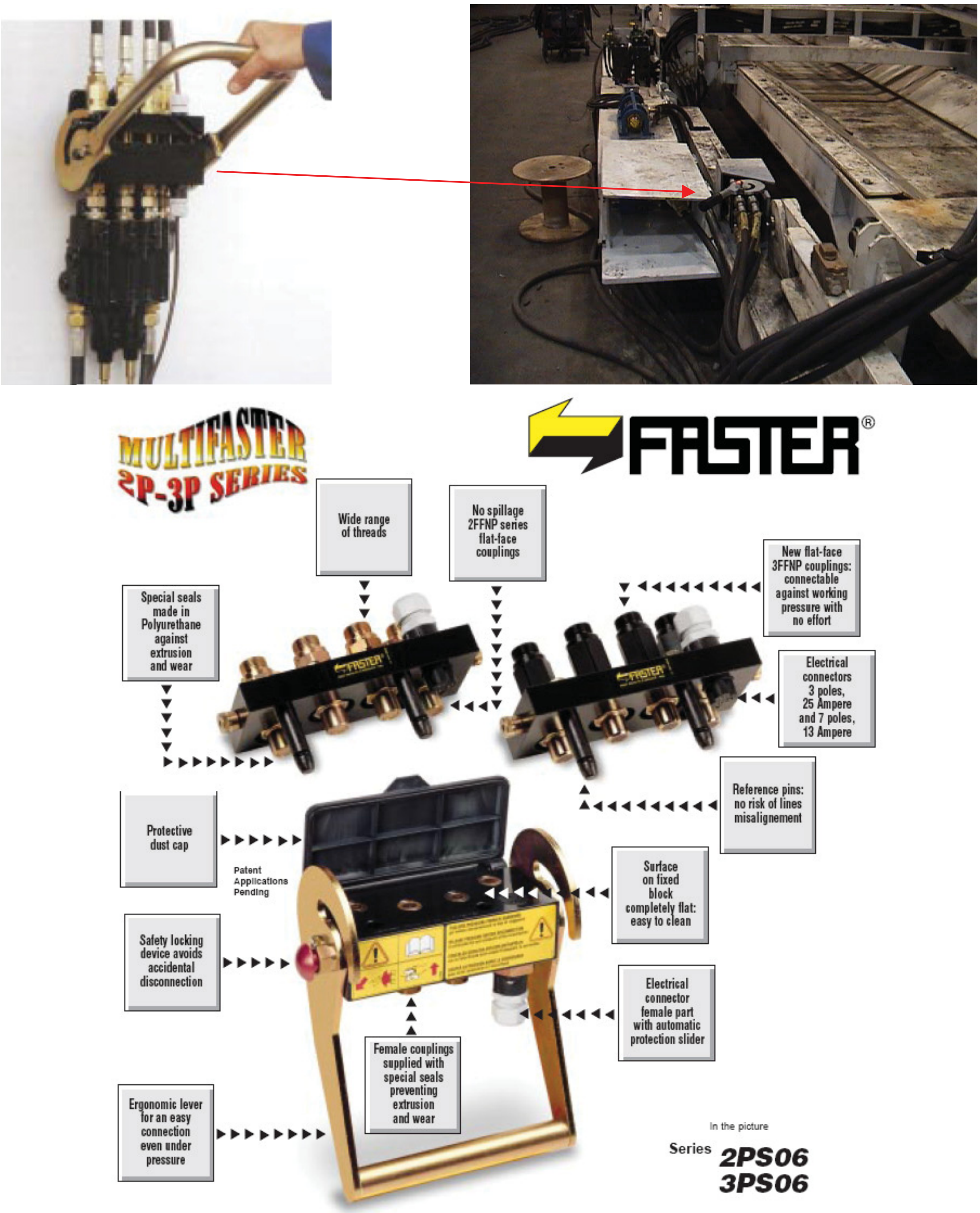
To make the manipulation of the hopper car easy for the operators the tracks of the hopper car will be activated via an electrical 'pendant remote control'. This is a semi-remote control box with electric push buttons for the track functions (independent forward/reverse functions).
Semi-remote control means the push button box is connected to the ABM10 Electric/Hydraulic via a cable. This gives the operator the freedom to move around during the manipulation of the hopper car to be able to manage obstacles and the alignment of the conveyor itself.

Only the three lifting cylinders to lift and lower the lifting frame will be controlled via manual directional control valves mounted on the hopper car side.

Not to rip the hoses apart in case the ABM10 proceeds further than the length of the hoses allows, a strong chain will be installed between the machines to act as a limitation of the independent movements of both machines.
The length of the chain will be determined by the length of the hoses and will therefore be just a little shorter than the hoses.

The remaining part of the materials handling system will also have a hydraulic system installed with an on board hydraulic power pack.
This power pack will supply all cylinders and tracks of the rear part of the materials handling system. Those hydraulic components will be discussed in the appropriate chapter.

This on-board hydraulic power pack will be used for the relocation of the materials handling system and therefore also for the hopper car.
By doing this, the hydraulic hose connection of the hopper car to the ABM10 will be disconnected via 'quick release couplings' and than re-connected to the on board hydraulic power pack of the materials handling system. This, again, will be done via hydraulic hoses and 'quick release couplings' (picture 102).

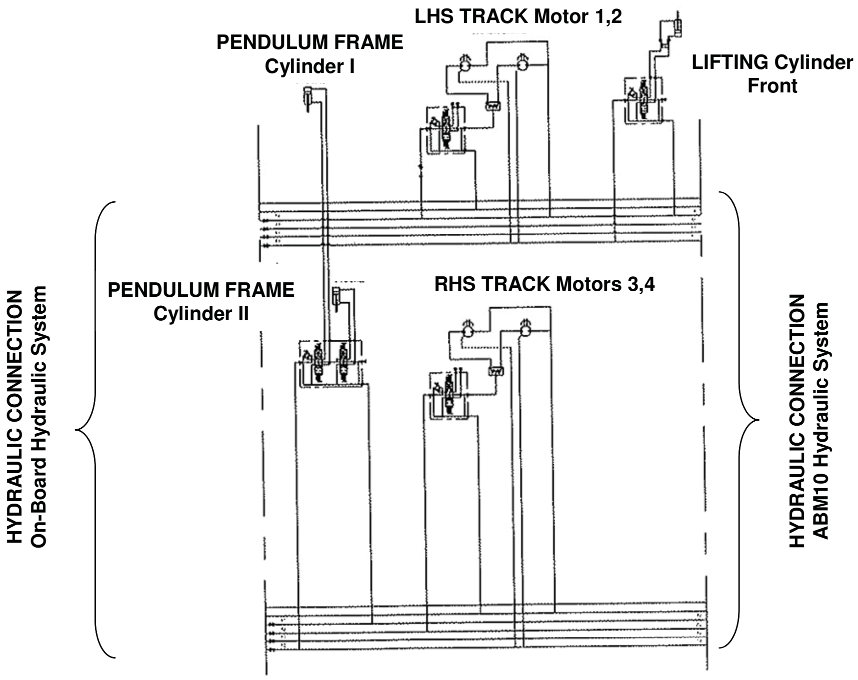


Picture 102: **quick release coupling;** Source: Brochure from company “Faster” based in the USA, photo: Sandvik

The USA based company ‘Faster’ is supplying such quick release couplings where all 6 hydraulic hoses can be connected and disconnected via one coupling and engaged or disengaged with a manual lever.

When operating the hopper car via the on-board hydraulic system, the track functions will be operated via manual directional control valves mounted at the side of the hopper car.
Picture 103 shows the hydraulic schematic of the hopper car.

The ABM10 and the materials handling system can then move completely independent.



Picture 103: **Hydraulic Schematic**

6.2.5. Electrics

Due to the mentioned space problems the electrical system on the hopper car will be limited to the absolute minimum.

The electrical system of the hopper car will be limited to the following parts:

- **Speed Sensor on the return pulley** to give information back to the ABM10 to stop the chain conveyor in case of a material handling system stop. This is to prevent the hopper of the hopper car to get over loaded or buried in coal. The speed sensor will be part of the ABM10 electric and only mounted onto the hopper car.
- **Area Lights** to illuminate the working area. Especially the area where the conveyor frames have to be handled manually the light is very important. Also the area where the chain conveyor feeds into the hopper the light is necessary. The electric power will be supplied from the ABM10 and is therefore part of the cutting machine.
- **Warning Lights and a Horn** will be mounted onto the hopper car to give the operators necessary information about the system. The horn will sound and a yellow light will blink

before the conveyor system of the materials handling system starts. When the conveyor runs and everything is all right a green light will be on all time. In case the loop take up unit will run out of conveyor belt the yellow light will start blinking again to give the operators the information to make themselves ready for the belt-feed-in-process within the next 10m of mining distance. The yellow light will be on permanently when the LTU unit ran out of belt and no advance movement is possible anymore. The red light will come on in case of danger and the stop of the belt conveyor.

- **Emergency Stop (push button system)** will be mounted along the hopper car which works together with the ABM10. It will stop the ABM10 conveyor system as well as the material handling system in case of problems or danger.
- **Communication line (intrinsically safe)** will connect the hopper car and the rest of the material handling system to transfer the information between the two remote units.

No other electrical system will be mounted on to the hopper car.

6.2.6. Level Control System

Underground mining sections and especially the roadways are far away from smooth operating surfaces. Due to the influence of water and the track systems as well as the broken up coal and rock the surface of the roadways is uneven with deep pot holes and mud on many occasions.
The low operating and mining height of the hard coal deposit on hand makes the roadway condition to a very important issue.

The hopper car accommodates the return pulley of the belt conveyor.

This return pulley has got a guiding function for the belt. Under normal conditions the pulleys will be used to ‘steer or guide’ the belt so it will run in the middle of the idlers along the center line of the belt conveyor.
In case the pulley is not perpendicular to the belt conveyor center line the belt will drift to one side till it starts rubbing on steels structures or in the worst case falls down the pulley shell. Rubbing of the belt on structures or running over the edges of the pulley causes premature wear on the belt.

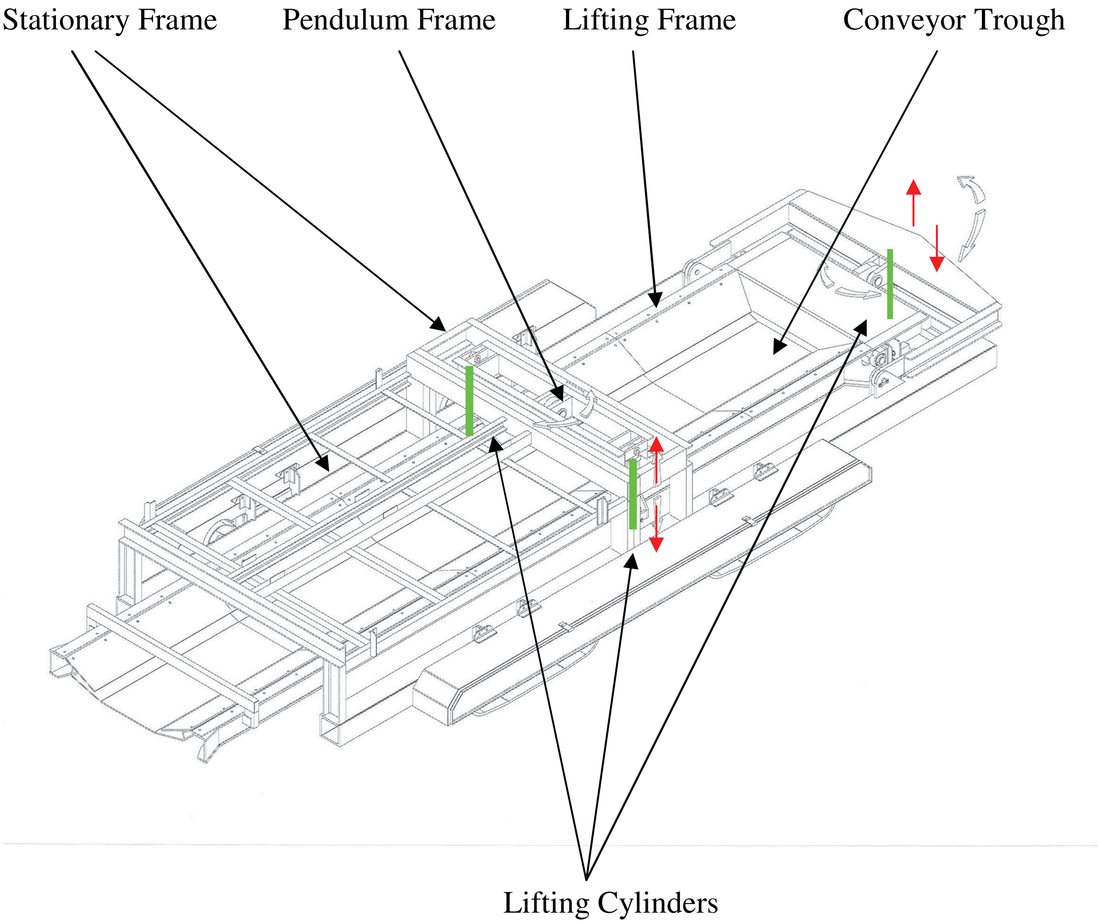
When advancing forward the hopper car will pull the conveyor belt out of the LTU unit. The hopper car will act as a counter weight to the belt tension via the return pulley.

During this advance process the belt conveyor will still operate which makes the whole system very vulnerable to misalignments between the hopper car centre line and the centre line of the remaining belt conveyor.

Based on the fact that the track mounted hopper car will run over the before mentioned underground roadways following the ABM10, it cannot be avoided that the hopper car will stop with one side of the track system lower than the other side.

The hopper car will come to a stop vertically tilted to one side along the centre line of the hopper car.
Also will it be impossible to have the two tracks on one side running exactly at the same speed than the other. Higher frictions and resistances on one side will cause the hopper car to deviate from the centre line in the horizontal way.
Both deviations from the centre line of the belt conveyor will cause the belt to run off its centered position.

The author was faced with the fact that the hopper car had to be equipped with possibilities to correct those deviations to maintain a proper running conveyor belt. A Level Control System with independent lifting cylinders, independent frames inside the hopper car and a Pendulum Frame had to be designed by the author (picture 104).

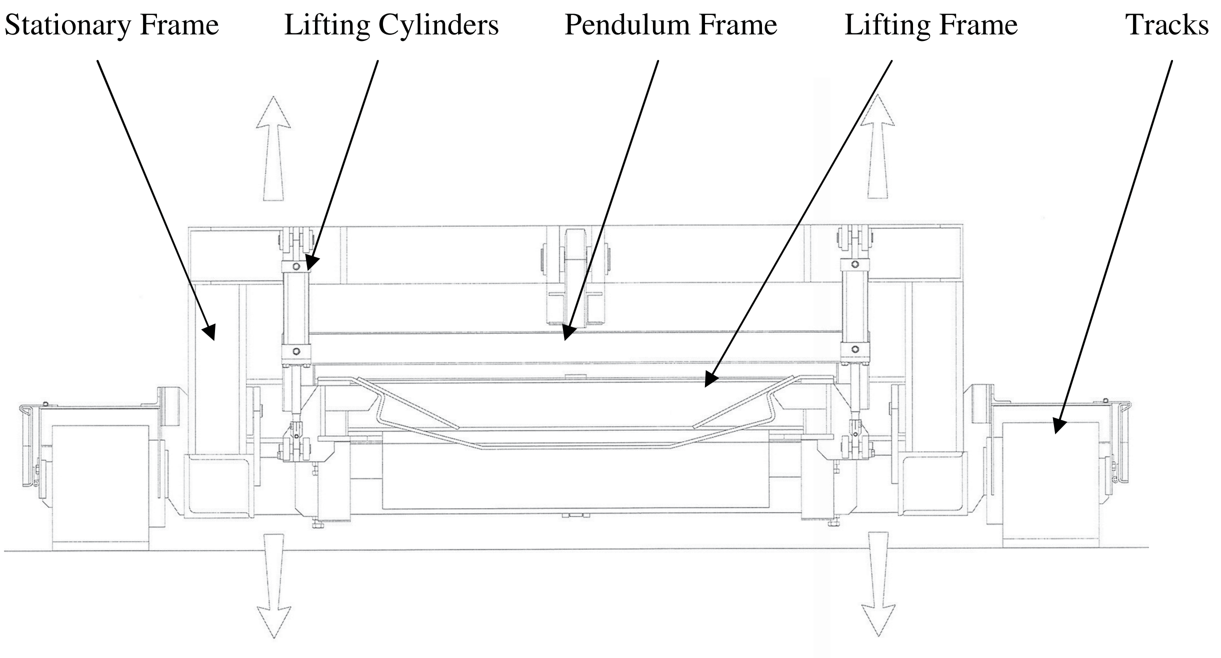


Picture 104: **Components of the Level Control System**

The lifting frame in the inside of the hopper car is a steel structure with a steel trough where the conveyor belt runs within through out the hopper car. At the end of the steel structure the belt will be lowered down onto the conveyor frames.
In the bottom area of the lifting frame are sitting the guide rails for the conveyor frames which is the storage area for those frames at the same time.
Also the return pulley is part of the lifting frame and is located at the very front.

The lifting frame is suspended inside the stationary hopper car frame on which the tracks are mounted to.

One hydraulic cylinder at the front and two at the back are acting as the lifting/lowering devices.
All three lifting cylinders can be addressed individually to be able to equalize uneven ground surface in both directions left to right and front to back (picture 105).



Picture 105: **Components of the Level Control System**

There was another problem encountered other than the height differences.

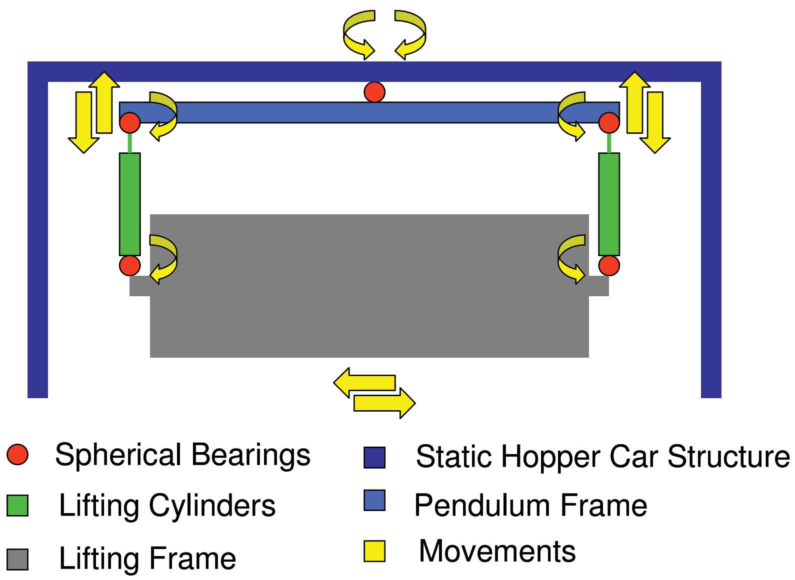
When moving the hopper car via the push-button-remote control and the hydraulic system of the ABM10 the car will start running by jerks, when the resistances vary over the driving distance and on both sides of the hopper car.
Those jerks will definitely upset the running behavior of the conveyor belt and subsequently cause a running failure.
A system had to be created by the author to lower the sensitivity of the conveyor to abrupt movements.
To give the lifting frame enough freedom to move freely all the connections, joints have been equipped with spherical bearings. Even though the spherical bearing will give the joints the

possibility to move slightly and to correct certain stresses caused by the jerks, it will not be able to react to larger movements in time and in the right manner.
The belt has to be kept inside the trough and coming out of the hopper car in a perpendicular way to be able to run the conveyor and to transport material without problems.

Additional to the spherical bearings, guide rollers will be installed to have fixed limits where the conveyor belt edges will be running against in case the riding off centre line becomes too excessive. That guide roller will be mounted in the rear part of the hopper car before the conveyor belt will be lowered onto the frames and inside the troughed carrying idlers.

The guide rollers are a security item only and should not be in contact with the belt edges on a permanent basis. This would destroy the belt edges.

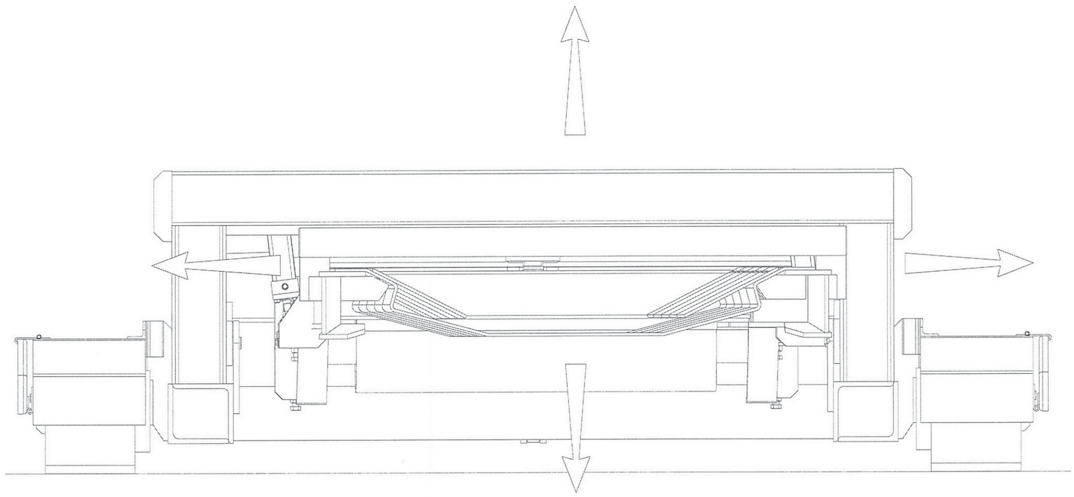
A 3D design study performed by the author showed that the distance necessary for the sideways movements have to be around 200mm to each side to be effective and able to react to the hopper car's movements.
This was the reason why a Pendulum Frame had to be invented and installed by the author (picture 106).
The pendulum frame in conjunction with the spherical bearings will give the lifting frame additional degrees of freedom.



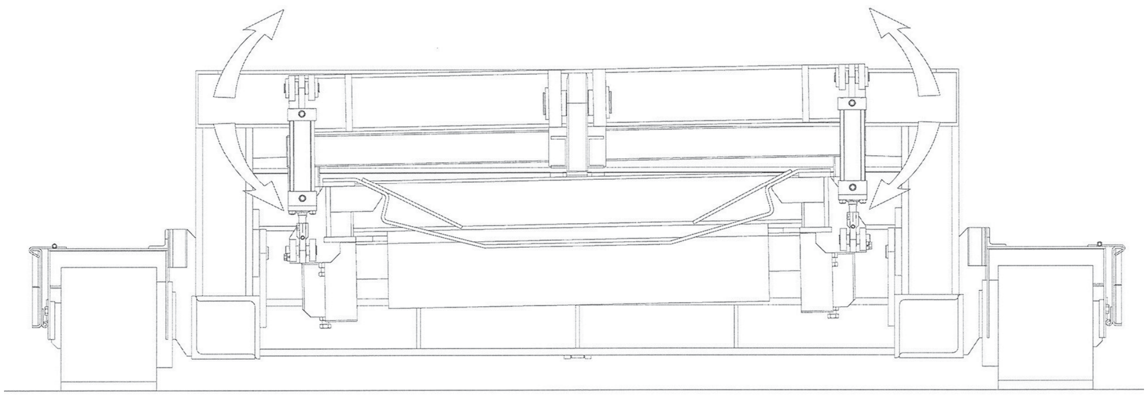
Picture 106: **Pendulum Frame, possible movements**

The pendulum frame allows the rear part of the lifting frame to move sideways whilst still keeping the lifting frame always parallel to the ground.

This pendulum frame will not only assist adjusting the lifting frame during the sideways correction, it will also assist in the height correction.
Compare to picture 107 and 108.



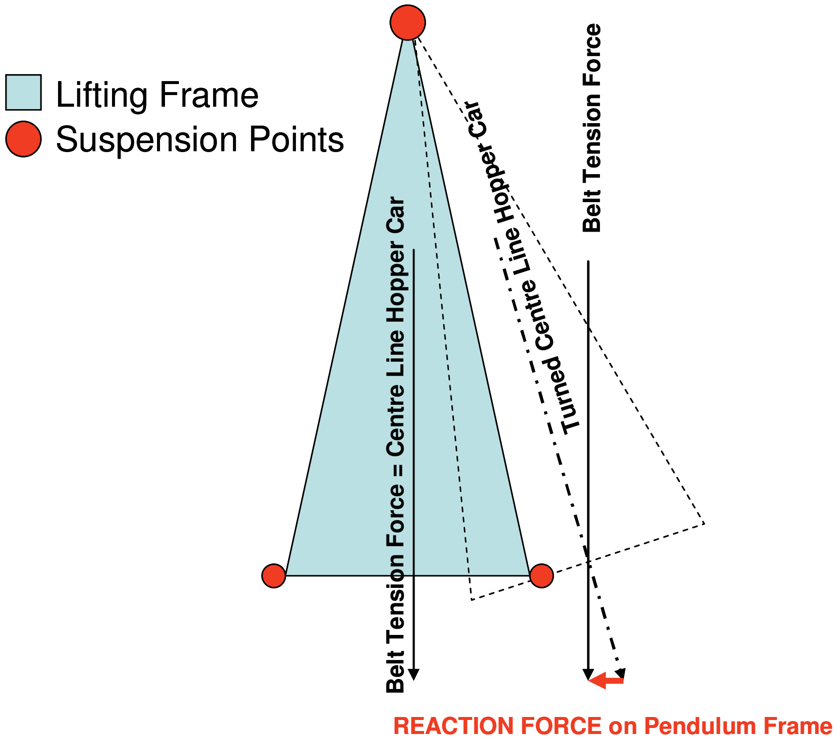
Picture 107: **Pendulum Frame, Degrees of Freedom**



Picture 108: **Pendulum Frame, Degrees of Freedom**

The conveyor belt tension and especially the direction of the tensioning force are correcting the direction of the lifting frame via the pendulum frame.

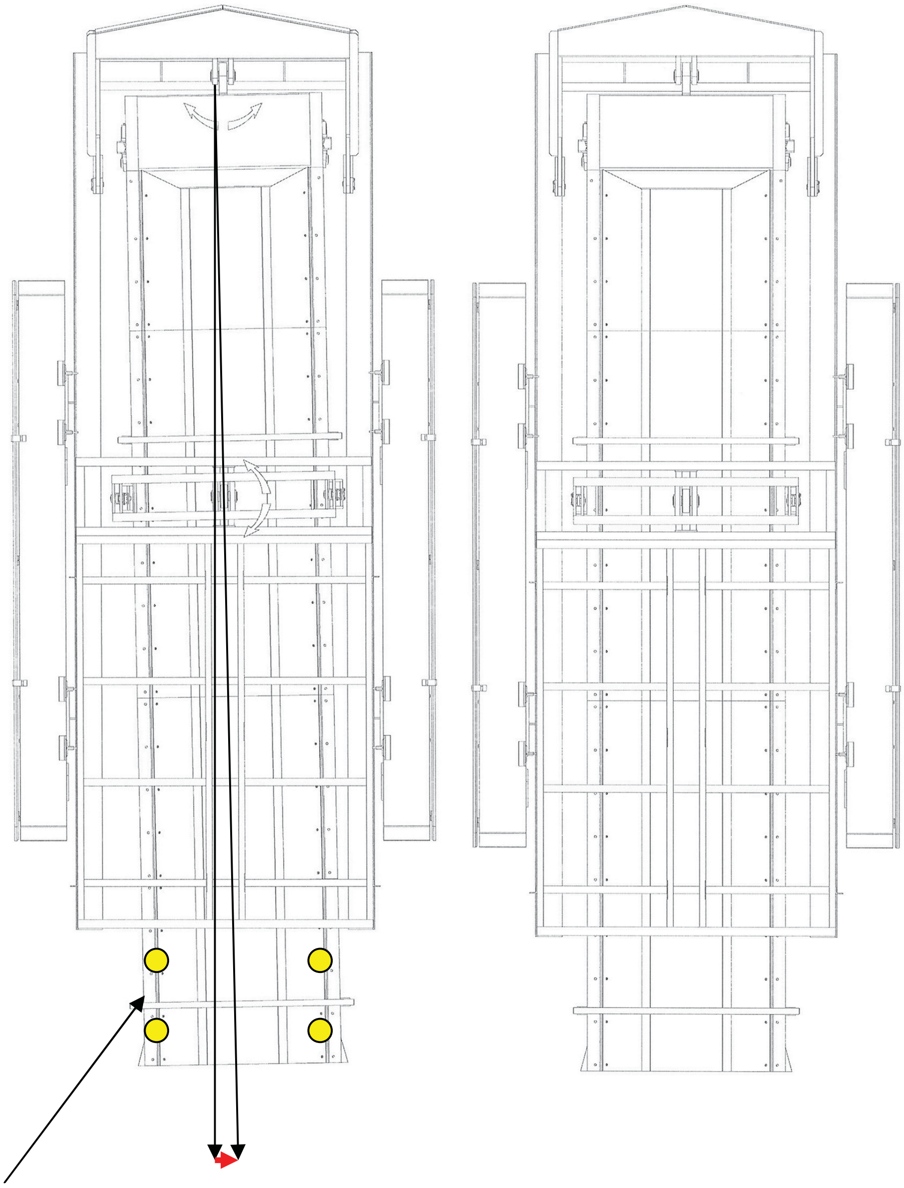
In case the hopper car centre line moves outside the centre line of the conveyor belt center line, a side force is getting created as a reaction to this.
This side force or reaction force is pulling the lifting frame via the pendulum frame into the direction of this force and will self-correct the misalignment (picture 109).



Picture 109: **Longitudinal Correction Schematic**

The return pulley as part of the lifting frame will rotate in the same way and the conveyor belt will still run perpendicular to the pulley, along the center line of the conveyor and therefore inside the carry idler trough.

As mentioned before the guide rollers (picture 110) at the end of the lifting frame will assist this reaction force to push the lifting frame in direction of the belt tensioning force.



Position of the Conveyor Belt Guide Rollers

Picture 110: **Longitudinal Correction with the help of Guide Rollers and Pendulum Frame**

Even though the pendulum system together with the guide rollers will be able to correct misalignments, it can do this only within an angle of +/- 15deg to the hopper car centre line. Outside this angle the belt edge will run permanently on the guide roller which has to be corrected via the hopper car tracks and the push buttons on the remote control.
The fact is that the rough correction will always have to be done via the tracks and the pendulum system will do the fine correction of the misalignment on its own.

6.3. Belt Conveyor Calculation

It was now at the time to get the conveyor parameter calculated and optimized.

Following parameters have to be determined:

- idler diameters to be minimized to reduce height
- trough angles maximized for loading and guiding
- conveyor drive power
- conveyor belt speed as around 2,0m/sec
- distance between idler sets to be maximized to reduce hopper car length
- conveyor belt thickness to be minimized to reduce height
- belt tension to be minimized to reduce steel structure strength and weight
- load to be 300t/hour
- bulk density around $800kg/m^3$
- maximum elevation 5,0m
- maximum length 600m

The main conveyor system parameters for the whole material handling system had to be fed into the calculation program called "Sidewinder V1.20" to calculate all the necessary conveyor data.

Those output and also the input parameters had to be optimized by multiple entering into the calculation program to find a compromise for all data. A specialist company was engaged to produce the requested figures together with the author.

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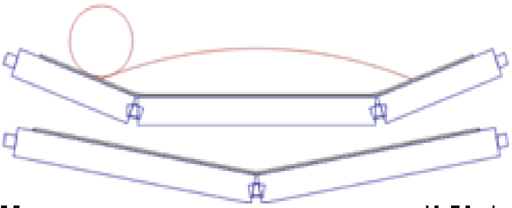
Several calculation steps, a manual iteration process, had to be performed to find the ideal combination of all parameters.

In the following the optimized calculation results are shown which will in fact influence the design of the material handling system (table 21). Due to the fact that the results influence the design of the overall system, special emphasis was given to those calculations by the author not to be forced to come back to this point and to repeat the calculations at a later stage.

Material Properties

Type .... Coal, Bituminous, Mined
Tonnage .... 300 TPH
Density .... 800 kg/m³
Maximum lump size .... 150 mm
Surcharge angle .... 20 deg
Percent lumps 1%



Mass .... 41.7 kg/m
%CEMA area .... 62 %
%Total area .... 41 %
Edge distance .... 152 mm
Bed depth .... 96 mm

Belt Properties

Type .... Fabric
Width .... 1050 mm
Rating .... 500 N/mm
Speed .... 2.00 m/s
Top cover thickness .... 0.5 mm
Bottom cover thickness .... 0.5 mm
Total thickness .... 5 mm
Mass .... 10.0 kg/m
Elastic modulus .... 5,149 kN/m
Tape length .... 647 m

3 Roll Carry Idler Set Input Data

Type .... 3 Roll
Carry Idler spacing (m) 3.00
Number of idler sets 87
Total drag (N) 7.5

Roll Data	*Center*	*Wing*
Diameter (mm)	60	60
Angle (deg)	0	20
Length (mm)	560	290
Shaft diameter (mm)	25.0	25.0
Series name	6205	6205
Rotating mass (kg)	2.7	1.5
Roll RPM	637	637
Min life (1000 Hr)	297	350
95% life [1] (1000 Hr)	297	350

[1] *L_{10} life above which 95% of idlers exceed*

2 Roll Return Idler Set Input Data

Type .... 2 Roll
Return Idler spacing (m) 3.00
Number of idler sets 127
Total drag (N) 5.0
Diameter (mm) 60
Angle (deg) 10
Length (mm) 560
Shaft diameter (mm) 25.0
Series name 6205
Rotating mass (kg) 2.7
Roll RPM 637
Min life (1000 Hr) 350
95% life [1] (1000 Hr) 350

[1] *L_{10} life above which 95% of idlers exceed*

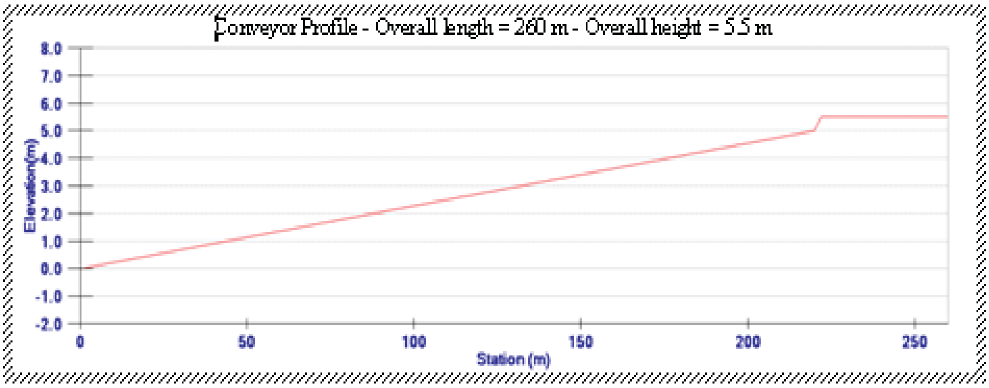


Table 21: **Belt Conveyor Calculations**

Take-up Data

Type .... Gravity
Location .... Tail
Tension .... 15 kN
Req'd steady state tension .... 7 kN
Req'd momentary tension .... 13 kN
Reqd tension for dynamic sag .... 7 kN
Max pulley force .... 30 kN
Running disp .... 0.00 to 0.05 m
Momentary disp .... -0.27 to 0.73 m

Take-up Displacement Summary

Total displacement (incl thermal) .... 1.00 m
Permanente elongation .... 4.20 m
Splice length (2 included) .... 1.60 m
Clearance .... 1.00 m
Required displacement .... 7.00 m

Maximum Belt Tensions

	Tension (kN)	Safety Factor
Steady State	26	20.54
Momentary	41	12.94

Minimum Belt Tensions

	Tension (kN)	Sag (%)
Steady State	15	1.06
Momentary	13	1.44

Starting and Stopping

Start control .... Constant torque
Start time .... 1.9 to 3.7 sec
O-Stop control .... Drift
O-Stop time .... 5.9 to 8.0 sec
E-Stop control .... Drift
E-Stop time .... 5.9 to 8.0 sec
E-Stop distance .... 5.9 to 8.0 m

Drive Station

Type ....
Total number of motors 1
Nameplate rating .... 45 kW
Total installed power .... 45 kW
Synchronous speed .... 1800 RPM
Reducer ratio 18.61
Motor inertia .... 0.7 kg-m^2
Flywheel inertia .... 0.0 kg-m^2
Total inertia .... 0.7 kg-m^2
Maximum starting torque 135%
Efficiency 95%
Pulley lagging type .... rubber
Backstop required .... No

Demand Power (kW)

Case	Demand Power (kW)	% Name-plate
Empty - Normal Frict.	12	26
Full - Normal Frict.	23	52

Din Factor and Total Equivalent Mass

Case	Din Factor	Belt Line Mass (kg)
Empty - Normal Frict.	0.0353	22,927
Full - Normal Frict.	0.0338	33,765

Stopping Times (sec)

Case	O-Stop	E-Stop
Empty - Normal Frict.	8.0	8.0
Full - Normal Frict.	5.9	5.9

Equivalent Belt Line Mass Summary
Not Including Motor Inertia

Pulley diameter for inertia .... 401 mm
Reducer ratio for inertia 18.610

Case	Belt Line Mass (kg)	Inertia at HS Shaft(kg-m^2)
Empty - Normal Frict.	17,041	2
Full - Normal Frict.	27,879	3

Maximum Case Belt Tensions (kN)

Case	Running	Dynamic
Empty - Normal Frict.	20	38
Full - Normal Frict.	26	41

Minimum Case Belt Tensions (kN)

Case	Running	Dynamic
Empty - Normal Frict.	15	14
Full - Normal Frict.	15	13

Table 21: **Belt Conveyor Calculations** (continued)

The main data are as following:

- Idler diameter 60mm
- Carry idler trough angle 20deg
- Return roller diameter 60mm
- Return roller trough angle 10deg
- Maximum idler spacing 3000mm
- Conveyor belt width 1050mm
- Conveyor belt thickness 5,0mm
- Conveyor belt tension maximum 15kN
- 23kW absorbed power

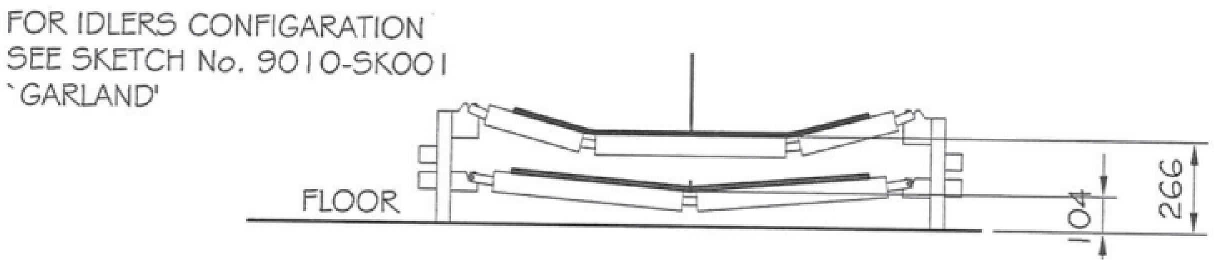
All other design parameters, dimensions and values for the belt conveyor system can be found in the input/output sheet above as well as in the sheets in the chapter 6.6. 'Loop Take Up Unit'.

6.4. Conveyor Structure and Stringers

The next emphasis has to be given to the conveyor frames and the storage of the frames inside the hopper car structure, the lifting frame.

Due to the fact that the hopper car had to be designed to be as short as possible, the conveyor frames will have to be stored in a way that they do not waste any space. They even have to be designed to be as low, as small and as light as possible.
300mm was the maximum height available inside the hopper car lifting frame to be able to fit the frames underneath the steel plate trough which was limited to the height of 500mm.

A conveyor structure had to be designed which was 300mm high, accommodating the carrying idlers and the return rollers, was stable enough to hold the load of the belt and the transported hard coal and which had an sliding arrangement engineered roll in and out of the hopper car.



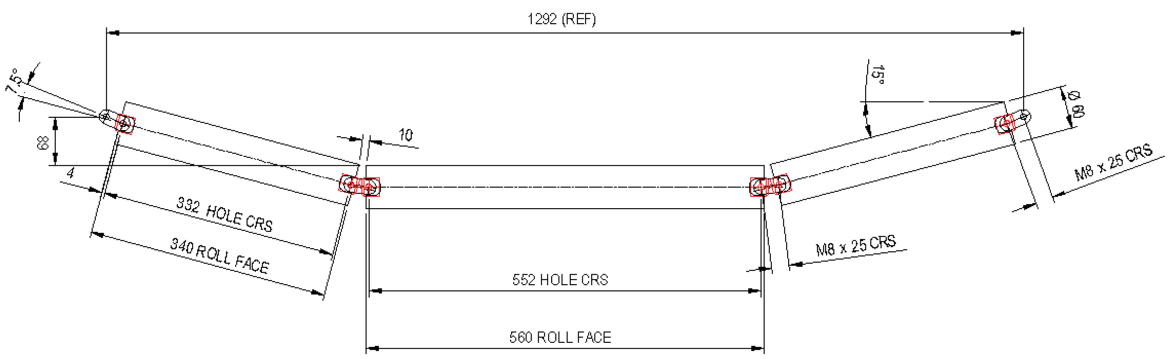
Picture 111: **Conveyor Structure design**

A rough design was put together (picture 111) to check all clearance parameters and to see what effect the frame do have onto the overall height of the hopper car and whether they fit underneath the loading trough of the hopper car.

The carrying idlers have to be troughed to maximize the loading and guiding capacity, the return rollers have to be troughed to get maximum guiding capability.

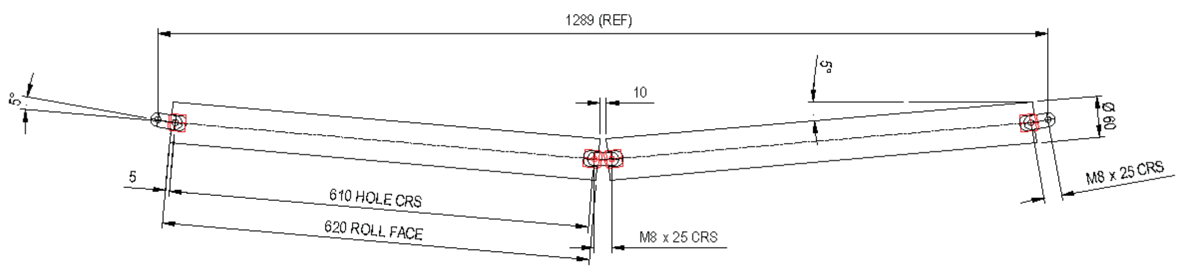
The carrying and return idler had to be designed by the author to have a diameter of only 60mm. Those idlers are far away from standard dimensions and have to be customized for this application.
Due to the fact that the structure frames have to be as light weighted as possible it was decided to use a garland design in which the rollers are connected by movable chain links and only connected to the frames at the outside links (picture 113 and 114).

Carrying Idler set:



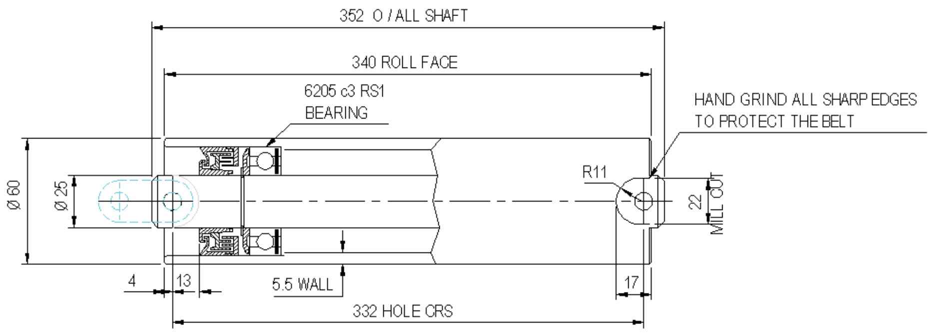
Picture 112: **Carry Idler Set**

Return Roller Set:



Picture 113: **Return Idler Set**

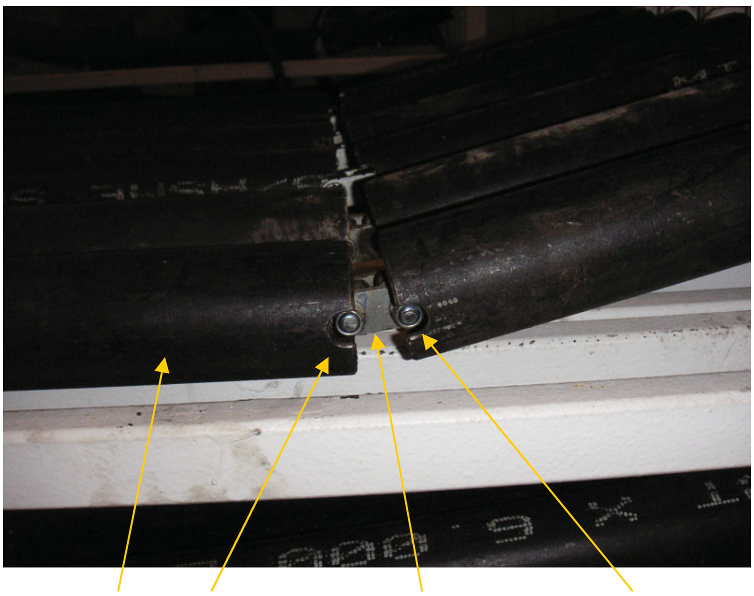
The **roller design** will look like the following (picture 114):



Picture 114: **Idler Design**

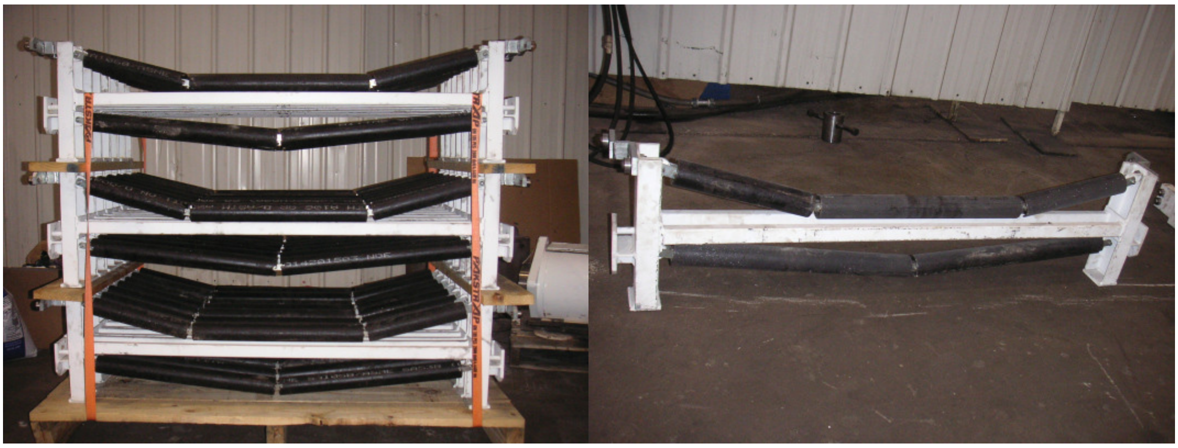
The special design required the use of tubular steel sections where the inside diameter got machines out to receive the bearings.
The axles which run through the whole idler roller have got connection eyes at each end to join the rollers together via two flat bars and bolts.

To keep the gaps between the rollers as small as possible the roller tube got extended and cut-outs arranged for the disassembly of the joining bolts (picture 115).



Idler tube cut-out flat bar joint connecting bolts
Picture 115: **Idler Connections**

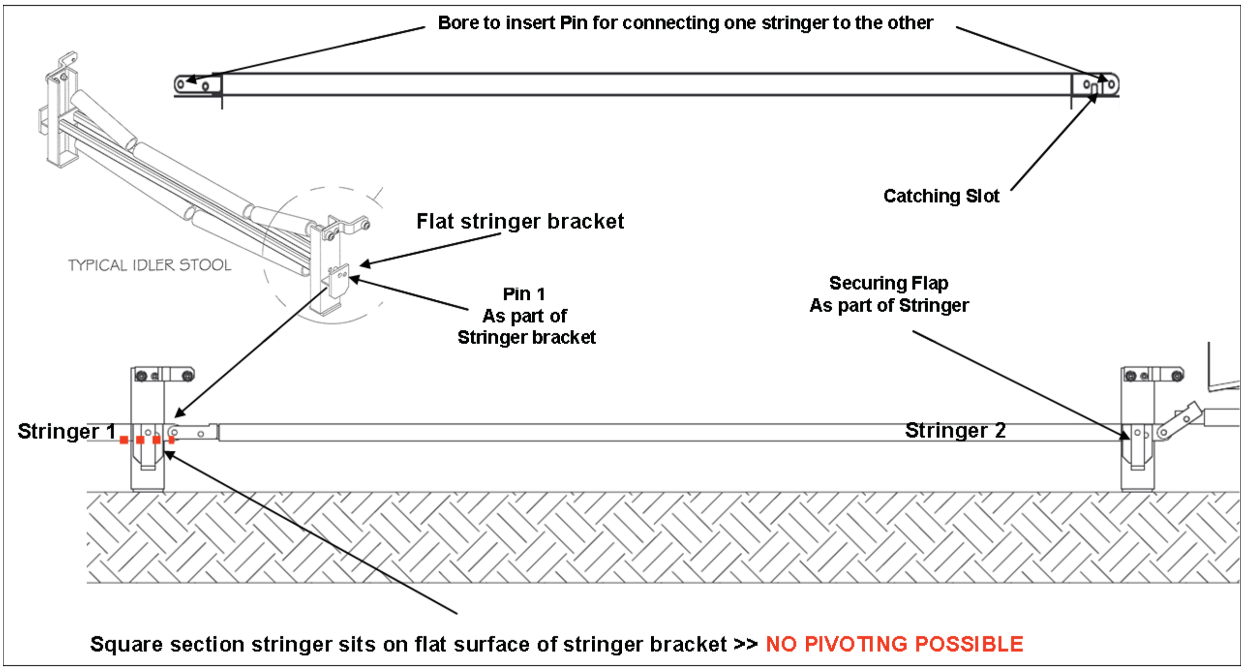
For the vertical columns of the frames, U-profiles were used. They are strong enough and the right size can be selected to have enough width to accommodate the horizontal connection beams as well as the right width to have the frames shoulder to shoulder inside the guide rails without the idlers colliding (also see picture 116).



Picture 116: **Conveyor Structure Design**

The stringers are the connecting rods between the conveyor structure frames. They have the task to keep the frames stable and in equal spacing to each other. The stringers also keep the individual frames parallel to each other so the conveyor belt can run over the idlers in a straight line.

Following sketch will demonstrate the principle of the stringer/frame connection.



Picture 117: **Conveyor Structure / Stringer Connection**

The key to a smooth operation in our application is that the stringers can be installed in a quick way by means of pins. At the same time the stringer has to keep the frames from falling over in cases of uneven floor or when the belt friction is pushing the frames to one side. This means that the pin connection between the stringer and the frame has to be more or less rigid. (also see the 'conveyor frame pull-out sequence' in chapter 6.2.3.).

This was achieved by the author by using a square tube as a stringer rod that will sit on a flat bracket on the frame. When the stringer is inserted, the frame and the stringer are locked in position without pivoting over the pin.

Principle shown in picture 117:

Stringer 1 finishes length wise past the stringer bracket for stability reason described in the picture. Stringer 2, the next stringer, will be connected to Stringer 1 via a manual inserted pin. The securing flap is needed to look the stringer and the frame to each other. The front side of the stringer is hinged to allow for the different heights of the frames on the floor and inside the hopper car.

For not loosing the pins they will be attached with chains onto the conveyor frames.

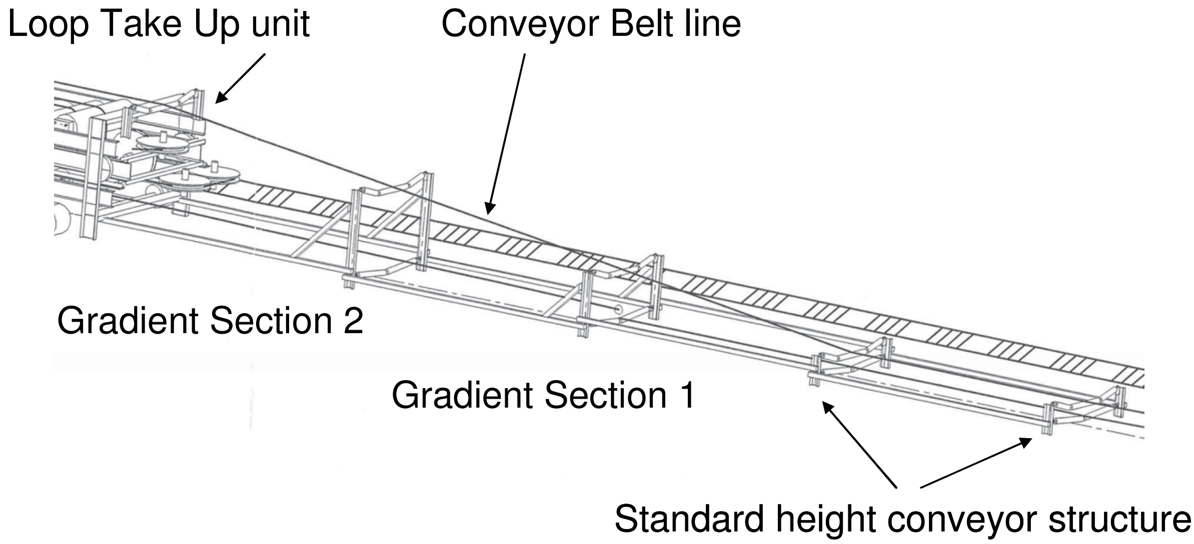
6.5. Gradient Structure

The gradient sections or gradient structures are the conveyor frames through which the conveyor belt when coming out of the mining tunnel gets lifted up onto the top of the Loop Take Up unit. The LTU unit is outside the mining tunnel and therefore outside the 1200mm cutting height area. The height available here is 1800mm out of which 1600mm can be used for the machines and equipment. The rest is needed for clearance to the roof.

Via an angle of maximum 17 degree, the carrying side of the conveyor belt is getting lifted from around 300mm up to 1600mm.

Two frames with increasing height had been used by the author and connected via stringers to give the necessary stability. The distance between the gradient structures will be the same as for the standard conveyor structure, therefore 3000mm.
The gradient structures are connected at the one side to the standard conveyor frames and on the other side to the LTU unit steel structure. The connection will be done via stringers.

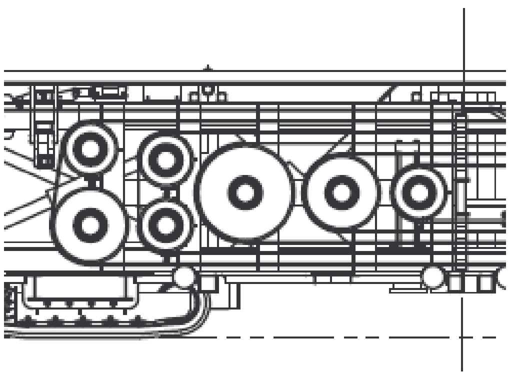
After the gradient sections the conveyor belt will be led over the top of the loop take up to the back where the material gets discharged via the discharge pulley onto the main section conveyor (picture 118).



Picture 118: **Gradient Structure**

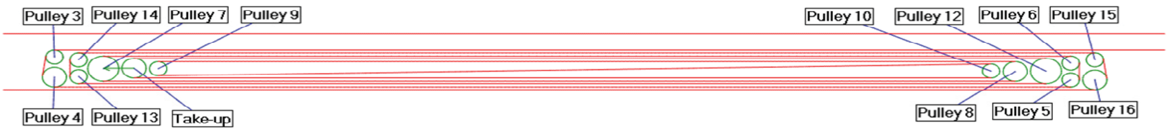
6.6. Loop Take Up unit (LTU unit)

The LTU unit is responsible for the storage of the conveyor belt surplus which is needed to extend the conveyor when the hopper car advances forward. Conveyor belt is getting pulled out of a loop system by reducing the length of the loops.
This is achieved by a stationary pulley system and a moveable pulley carriage which reduces the loop length by moving towards the stationary LTU pulleys.
The maximum amount of belt layers possible for the given height of 1600mm was used for the design which meant the use of 5 pulleys each side of the LTU unit.
To safe space the two largest pulleys from each side got replaced by the author by a “two pulley set” with the smallest possible diameter and arranged in the following way (picture 119).

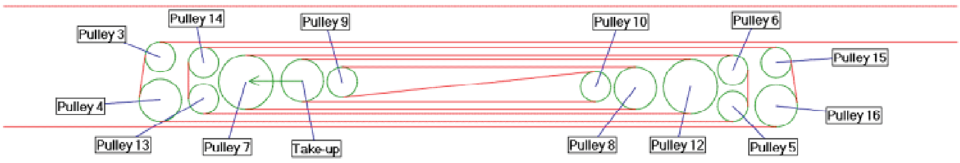


Picture 119: **“Two-Pulley” Design replacing one large Pulley in the LTU Unit**

The conveyor belt is looped in the following way (picture 120) and shown with the two pulley-sets in the full and empty stage.



Belt Storage Full

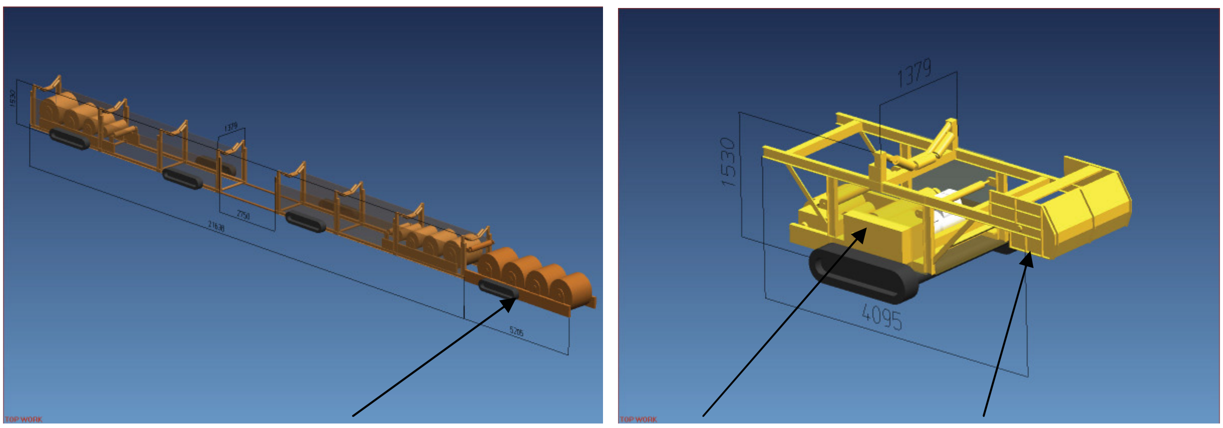


Belt Storage Empty

Picture 120: **Loop Take Up Unit**

The original design of the LTU unit had been taken as the base for the development.
Besides the change of the LTU pulley, the storage for the additional conveyor belt rolls had to be revisited as the original position of the belt rolls was in the way of the gradient structure.
Also the position to feed the belt in was difficult for the operators at the front of the LTU unit.

For keeping the overall length the same but, at the same time, gaining additional length of the active LTU unit length (to gain additional useable belt length in the LTU unit for advance distance reasons), the arrangement of the conveyor belt rolls and the conveyor drive had to be optimized (original design is shown in picture 121).



Additional Conveyor Belt Rolls Conveyor Drive Discharge Pulley

Picture 121: **original Design of the LTU Unit and the Drive Unit**

The additional belt rolls had to be looked at by the author as a first and most important measurement.

An investigation of the possibility to reduce the number of rolls as well as the optimization of the diameter of the rolls became necessary.
The necessity to reduce the 'dead length' of the LTU unit required the reduction of the 4 additional belt rolls down to maximum two!

In a second step, the belt rolls had to be arranged in the rear part of the LTU unit by combining the discharge pulley and the drive pulley together to one only driven discharge pulley. This will be the optimal position to feed in the additional conveyor belt as all working steps can be done in the operator's breast height.

The question the author was faced with was: How can the four belt rolls be reduced down to two rolls?

→ The first step was to increase the diameter of the rolls and to reduce the conveyor belt thickness to a minimum to be able to store more conveyor belt on the rolls.
→ The second step was to increase the 'active length' of the LTU unit to a maximum by not increasing the overall length of the unit so it was still possible to run the unit around the corners when relocating. The exact necessary length inside the LTU unit had to be calculated first. After that the maximum diameter, the rolls are allowed to have, could be determined.

The initial starting length of the mining system has to be determined first:

- ABM10 11,700mm
- Hopper Car 8,225mm
- Gradient Structure 9,000mm

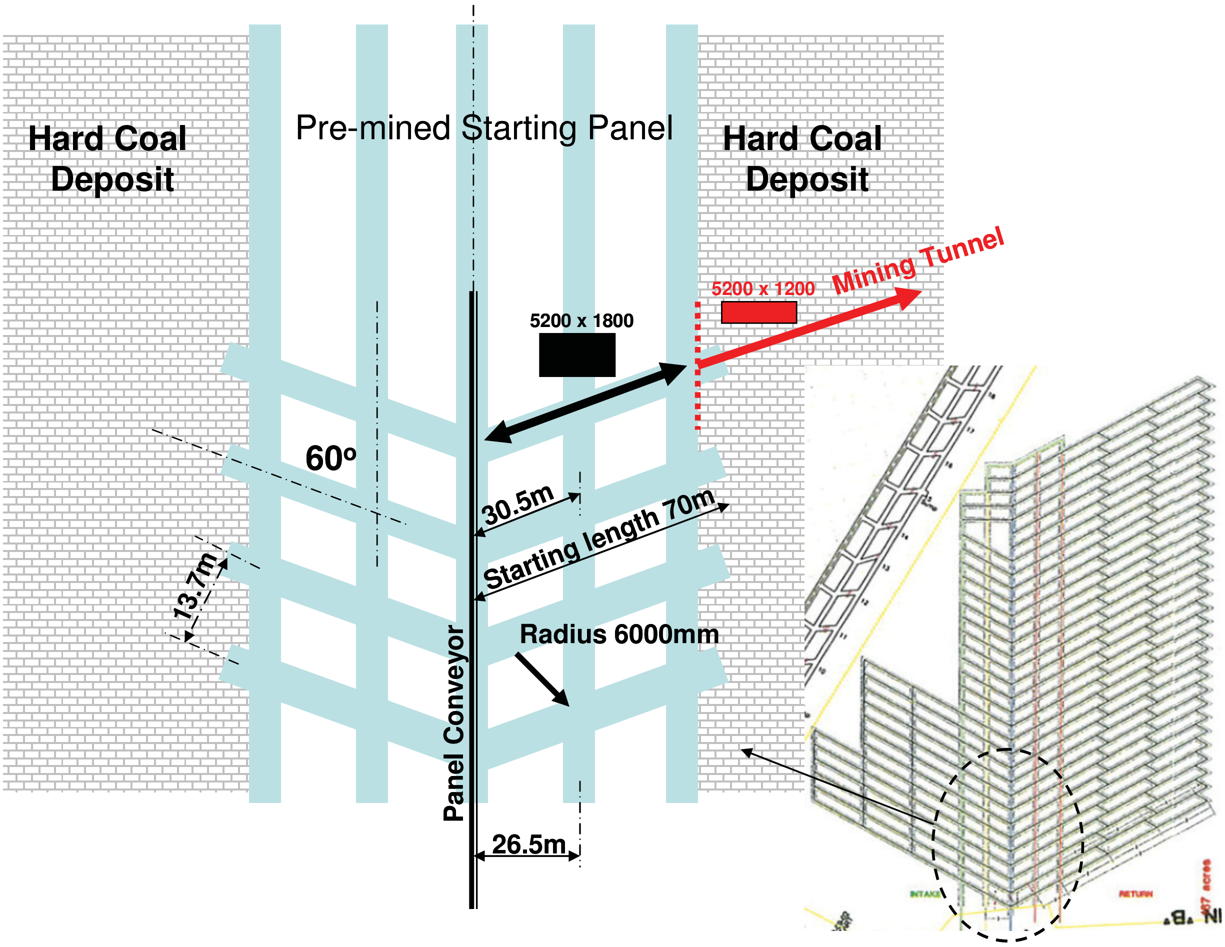
The overall minimum starting length of the front part of the mining system (ABM10, Hopper Car, and Gradient Structure) will be

11,700 (ABM10) -3,000 (overlap) + 8,300 (HC) + 9,000 (GStr.) = 26,000mm

Picture 122 was created to illustrate and simplify the estimations for the author.

The available distance from the outer side of the section conveyor to the starting face of the mining tunnel will be approximately 70m. This also includes a 10m initial starting length into the mining tunnel with the low cutting height of 1200mm (shown beyond the red dotted line).

70m over all start-up length minus the length of the front unit of the mining system which is 26m equals to 44m.



Picture 122: **Determination of the maximum Start-Up-Length of the System**

This gave the available space to fit in the Loop Take Up unit system including the drive and discharge unit and also the distance we need to expand the gradient structure as well as to pre-tension the fabric belting (taking out the majority of the stretching of around 6% in length difference between the relaxed and tensioned length).

44m will also include some distance to move the two units, the material handling system and the ABM10 apart from each other to be able to move them around individually and without too many problems.

The calculations have brought the author the fact that there is enough space available to fit in a LTU unit which stores enough conveyor belt length to reach the distance. The design criteria will now be the length of the combined LTU and drive/discharge unit which is long enough to store enough conveyor belt length to reach the 200m mining tunnel length with two on board storage belt rolls <u>and</u> (at the same time) to be able to drive the unit around corners without problems.

Further calculations had to be done by the author.

With a maximum storage belt roll diameter of 1200mm and a conveyor belt thickness of 7mm, 120m of conveyor belt can be spooled onto one storage roll.

Two times 120m means an achievable additional conveyor extension distance of 120m!

This means that at least an additional 160m of conveyor belt length have to be in the LTU unit as being actively available for belt extension (without the dead length needed for the length of belt needed to go around all pulleys of the LTU unit, two times the length of the LTU unit for carry and return purposes, hopper car).

To get around the corners without any problems, the author calculated, that the LTU/drive unit must not be no longer than 32 meters (see the simulations of previous chapters 5.4.2. and 5.5.5.)

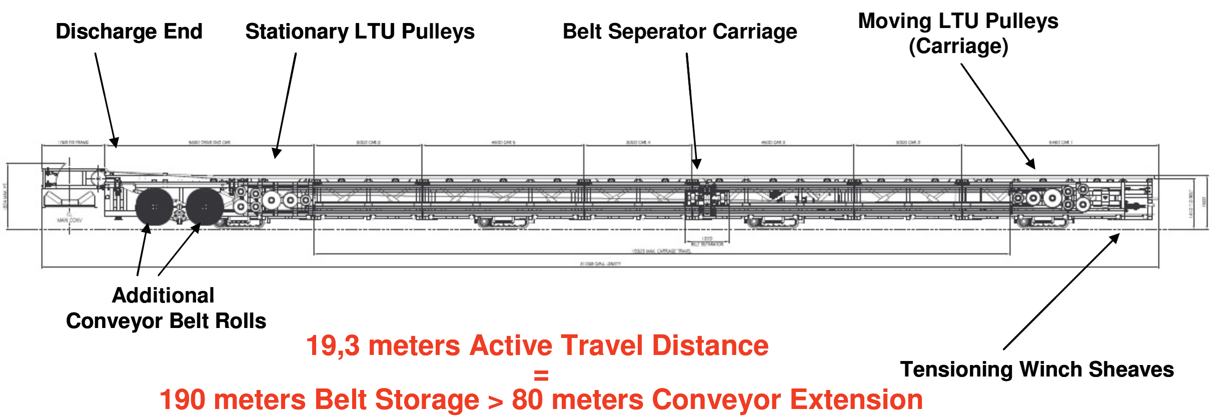
To reach the 200m mining tunnel length we have to calculate whether there is enough conveyor belt storage capability in the LTU unit.

- LTU/drive unit 32m
- Collapsed Gradient Str. 1m
- Hopper Car 8m
- ABM10 (w/o overlap) 12m

All up the length of the fully collapsed system is 53,0 meters.

To get from the collapsed position to the mining start face we need 17 meters.
This means that to get from the fully collapsed length to the end of the 200m mining tunnel length we need all up 2 x 217 meters = 434m of stored conveyor belt to extend the materials handling system to the maximum reach.

With the calculated parameters the author was able to put down the following design of the LTU and drive unit.



Picture 123: **Calculations of the stored Conveyor Belt Length**

Dimensional check (picture 123):

- Minimum 6% of the stored conveyor belt in the LTU unit and also the dead length sitting in the units will be used for stretching the belt only. This means that from the collapsed length to the starting face of the mining tunnel the fabric conveyor belt will be stretched only. Approximately 300m of belt will be needed to initially fill the LTU/drive unit including gradient structure and hopper car. 6% of 300m is already 18m of belt or 9m advancing distance.
- The initial length of the initial LTU storage brings us to the 80m – 17m + 9m = 72m mining distance
- Inserting the first storage roll of conveyor belt brings us to 72m + 60m + 3,5m*) =135,5m (*)belt stretch of 6% of 120m belt = 7,2m)
- Inserting the second roll brings us to 135,5m +60m + 3,5m*) = 199m with the 3m overlap of the ABM10 and the hopper car, the 200m tunnel length is reached.

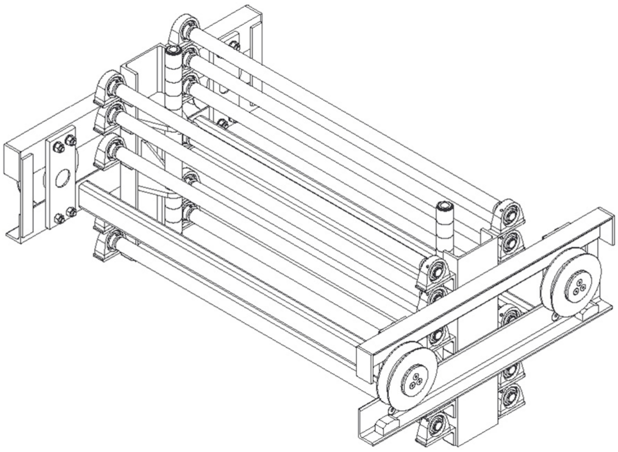
To reach the end the LTU unit has to <u>actively store 434 meters</u> of for extension available conveyor belt.
The author determined to choose the following belt pieces for the system:

- 2 x 120m for initial filling
- 1 x 60m for the initial filling
- 1 x 120m for the first belt storage roll
- 1 x 120m for the second belt storage roll

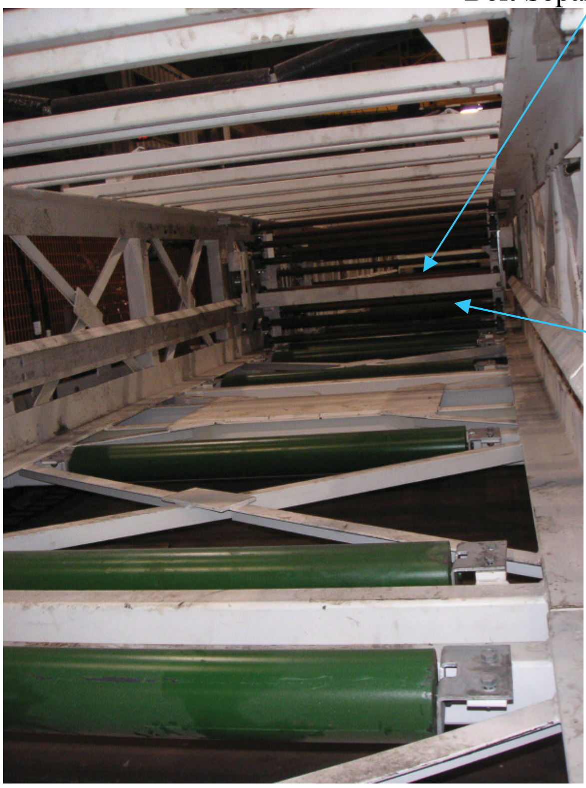
All up we will have 300m initial filling plus 240m belt storage = 540m overall conveyor belt length available for the operation.

4 times the equal length was chosen to not take out wrong belt pieces by mistake when the additional belt is getting taken out on the way back out of the tunnel. The 60m belt piece will be color coded to always stay in the system.

When the LTU unit is fully loaded the free loop belt length will be close to the 19 meter figure.
The conveyor belt weight will than have the belt hanging through resulting in rubbing of the individual belt layers.
This had to be avoided from happening by a belt separator carriage. The separator carriage consists of small rollers sitting in between the belt layers mounted onto a carriage with wheels.



Picture 124: **Belt Separator Carriage**

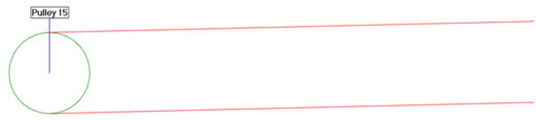


Picture 125: **Belt Separator Carriage** Picture 126: **Belt Separator Carriage**

The belt separator carriage in picture 124, 125, 126 will be arranged in the centre between the stationary LTU pulley set. It will always run with half the speed of the moving carriage so it will always stay in the middle of the two pulley sets.

The remaining necessary conveyor calculations are shown in the following table 22. They are the rest of the results from the previous calculations and are concentrated on the LTU unit and the conveyor belt inside with their reaction forces and tensions.

Pulleys at Tail



Belt storage

Pulleys at Head

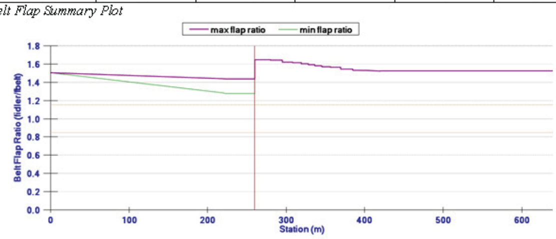
*Maximum Pulley Tensions (*Design tensions are 110% of Momentary Tensions)*

Pulley	Location (Element)	Type	Wrap (deg)	Maximum Resultant Tension (kN) (T1+T2)			Maximum Belt Line Tension (kN) (Max of T1 and T2)		
				Running	Momentary	Design	Running	Momentary	Design
Motor 1	Head (3)	1	180	40	55	61	26	41	45
Take-up	Head (5)	5	180	30	31	34	15	16	18
Pulley 3	Head (7)	2	90	21	23	26	15	17	19
Pulley 4	Head (9)	2	90	22	25	27	15	18	20
Pulley 5	Head (11)	4	180	31	37	41	16	19	21
Pulley 6	Head (13)	3	180	31	39	43	16	20	22
Pulley 7	Head (15)	2	182	32	41	45	16	21	23
Pulley 8	Head (17)	2	182	32	43	47	16	22	24
Pulley 9	Head (19)	3	180	33	45	49	16	23	25
Pulley 10	Head (21)	4	180	33	47	52	17	24	26
Pulley 11	Head (23)	2	90	24	35	38	17	25	28
Pulley 12	Head (25)	2	90	24	36	40	17	26	29
Pulley 13	Head (27)	2	82	22	35	38	17	27	30
Pulley 14	Head (29)	3	98	26	41	46	17	28	31
Pulley 15	Tail (34)	2	179	36	64	70	18	32	36

Pulley Geometry Details

Type	Diameter (mm)	Lagging type	Lagging thickness (mm)	Diameter with lagging (mm)	Face width (mm)
1	400	Rubber	0.0	400	1200
2	300	Rubber	0.0	300	1200
3	420	Rubber	0.0	420	1200
4	540	Rubber	0.0	540	1200
5	780	Rubber	0.0	780	1200

Belt Flap Summary Plot



* Belt flap resonance occurs at flap ratios 1, 2, 3, and 4

Belt Tension Summary Plot

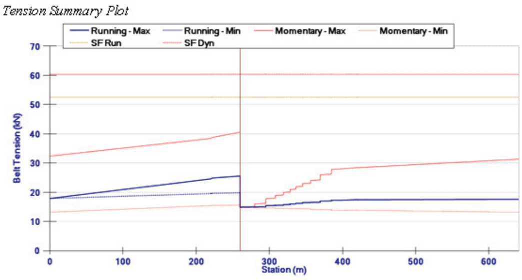


Table 22: **Belt Tensions and Pulley Reactions Forces**

Element Summary

Summary of Conveyor Elements

#	Name	Station (m)	Elevation (m)	Length (m)	Height (m)	Slope (deg)	Vertical Radius (m)	Idler Spacing (m)	Idler Type
1		0.0	0.00	220.1	5.00	1.30	0	3.00	1
2		220.0	5.00	2.1	0.50	14.04	0	3.00	1
3		222.0	5.50	38.0	0.00	0.00	0	3.00	1
4	Motor 1	260.0	5.50	0.4	-0.40	-90.00	0	0.00	0
5		260.0	5.10	20.0	0.00	0.00	0	3.00	2
6	Take-up	240.0	5.10	0.8	-0.78	-90.00	0	0.00	0
7		240.0	4.32	15.0	0.00	0.00	0	3.00	2
8	Pulley 3	255.0	4.32	0.2	0.15	45.00	0	0.00	0
9		255.2	4.47	0.4	0.36	90.00	0	3.00	2
10	Pulley 4	255.2	4.83	0.2	0.15	45.00	0	0.00	0
11		255.0	4.98	13.0	0.00	0.00	0	3.00	2
12	Pulley 5	242.0	4.98	0.5	-0.54	-90.00	0	0.00	0
13		242.0	4.44	11.0	0.00	0.00	0	3.00	2
14	Pulley 6	253.0	4.44	0.4	0.42	90.00	0	0.00	0
15		253.0	4.86	9.0	0.00	0.00	0	3.00	2
16	Pulley 7	244.0	4.86	0.3	-0.30	-89.11	0	0.00	0
17		244.0	4.56	8.0	0.25	1.79	0	3.00	2
18	Pulley 8	252.0	4.81	0.3	-0.30	-89.11	0	0.00	0
19		252.0	4.51	9.0	0.00	0.00	0	3.00	2
20	Pulley 9	243.0	4.51	0.4	0.42	90.00	0	0.00	0
21		243.0	4.93	11.0	0.00	0.00	0	3.00	2
22	Pulley 10	254.0	4.93	0.5	-0.54	-90.00	0	0.00	0
23		254.0	4.39	13.0	0.00	0.00	0	3.00	2
24	Pulley 11	241.0	4.39	0.2	0.15	45.00	0	0.00	0
25		240.9	4.54	0.4	0.36	90.00	0	3.00	2
26	Pulley 12	240.9	4.90	0.2	0.15	45.00	0	0.00	0
27		241.0	5.05	15.0	0.00	0.00	0	3.00	2
28	Pulley 13	256.0	5.05	0.2	-0.13	-40.93	0	0.00	0
29		256.2	4.92	0.4	-0.42	-81.87	0	3.00	2
30	Pulley 14	256.2	4.50	0.3	-0.24	-49.07	0	0.00	0
31		256.0	4.26	19.0	0.00	0.00	0	3.00	2
32		237.0	4.26	15.0	1.04	3.96	0	3.00	2
33		222.0	5.30	2.1	-0.50	-14.04	0	3.00	2
34		220.0	4.80	220.0	-5.10	-1.33	0	3.00	2
35	Pulley 15	0.0	-0.30	0.3	0.30	88.05	0	0.00	0

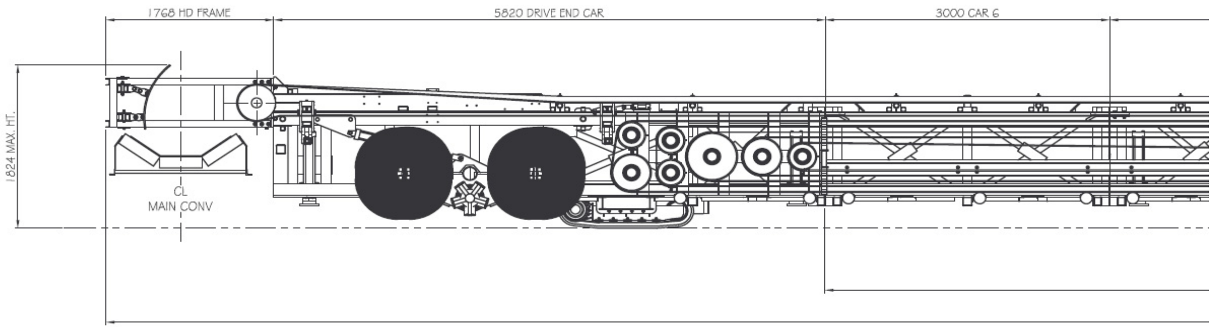
Element Tension Summary

Element Tension (kN)

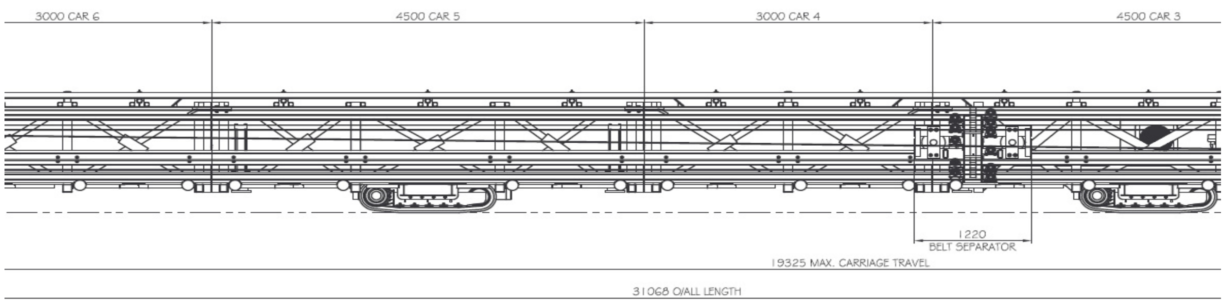
#	Name	Running			Momentary		
		Maximum	Minimum	%Sag	Maximum	Minimum	%Sag
1		18	18	1.06	32	13	1.44
2		25	20	0.77	38	15	1.20
3		25	20	0.76	39	15	1.18
4	Motor 1	26	20	0.00	41	16	0.00
5		15	15	0.25	15	15	0.25
6	Take-up	15	15	0.00	15	15	0.00
7		15	15	0.25	16	15	0.25
8	Pulley 3	15	15	0.00	16	15	0.00
9		15	15	0.24	17	15	0.25
10	Pulley 4	15	15	0.00	17	15	0.00
11		15	15	0.24	18	15	0.25
12	Pulley 5	15	15	0.00	18	15	0.00
13		16	16	0.24	19	14	0.25
14	Pulley 6	16	16	0.00	19	14	0.00
15		16	16	0.23	20	14	0.26
16	Pulley 7	16	16	0.00	20	14	0.00
17		16	16	0.23	21	14	0.26
18	Pulley 8	16	16	0.00	21	14	0.00
19		16	16	0.23	22	14	0.26
20	Pulley 9	16	16	0.00	22	14	0.00
21		16	16	0.22	23	14	0.26
22	Pulley 10	16	16	0.00	23	14	0.00
23		17	17	0.22	24	14	0.26
24	Pulley 11	17	17	0.00	24	14	0.00
25		17	17	0.22	25	14	0.26
26	Pulley 12	17	17	0.00	25	14	0.00
27		17	17	0.22	26	14	0.26
28	Pulley 13	17	17	0.00	26	14	0.00
29		17	17	0.21	27	14	0.26
30	Pulley 14	17	17	0.00	27	14	0.00
31		17	17	0.21	28	14	0.27
32		17	17	0.21	28	14	0.27
33		17	17	0.21	28	14	0.26
34		17	17	0.21	28	14	0.27
35	Pulley 15	18	18	0.00	31	13	0.00

Table 22: **Belt Tensions and Pulley Reactions Forces** (continued)

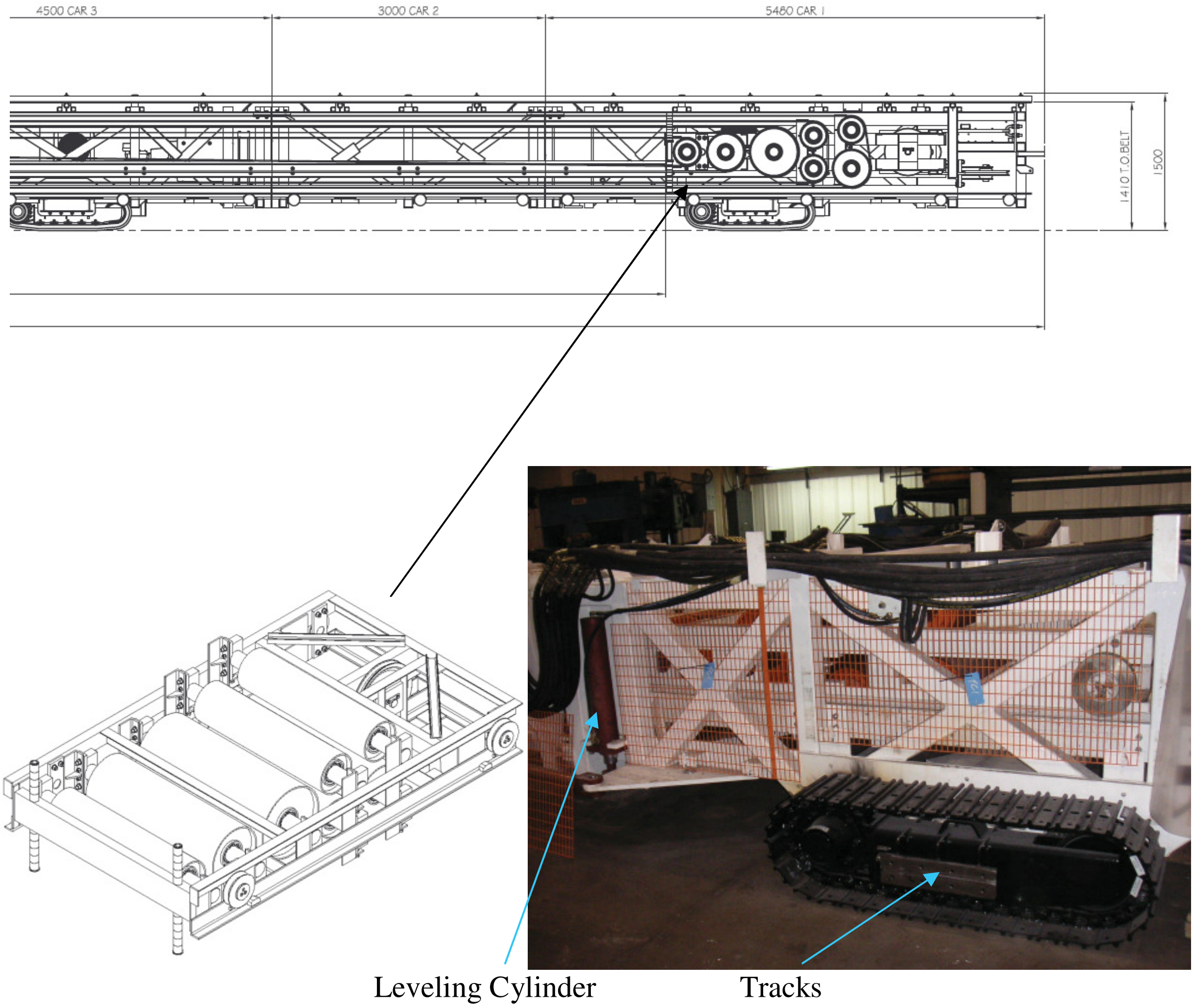
In the following pictures 127, 128 and 129 the whole LTU unit inclusive the new positioned drive/discharge unit as well as the new location for the Conveyor Belt Storage area is shown in detail with all relevant main dimensions.



Picture 127: **LTU Rear Section with Belt Storage and Drive Unit**



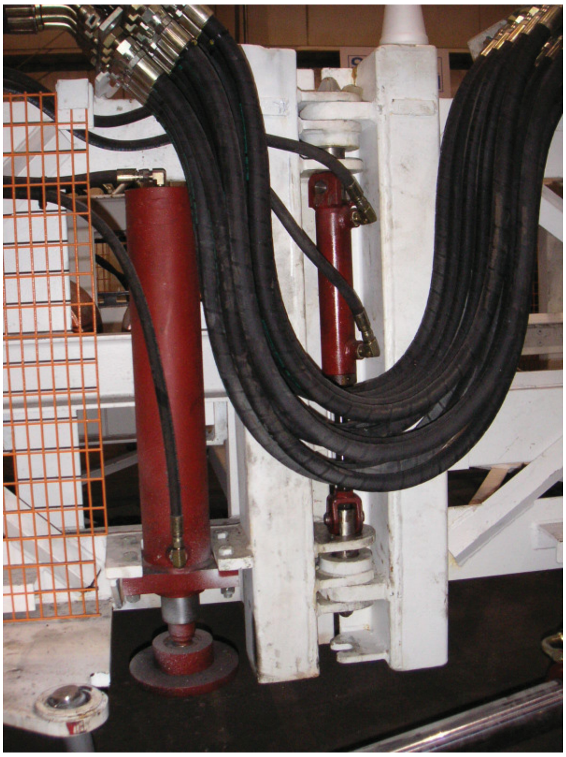
Picture 128: **LTU Middle Section**



Picture 129: **LTU Front Section with movable Pulley Carriage**

The most essential issue in the LTU unit is the necessity to run the belt loops without any problems and to be able to pull the belt out of the unit smoothly. The belt must run in the centre of the pulleys for most of the time. Also the material which is getting conveyed over the top of the LTU unit should not get spilled into the LTU unit when the belt runs off centre.
To keep the belt running in the centre of all the pulleys, also the drive pulley, the LTU unit should be almost horizontally in level.

Spirit level will have to be attached on several positions on the steel structure of the LTU unit to be able to check for the leveled position. The spirit levels will indicate both horizontal positions, right to left and front to back.
Whilst the front to back shall only be adjusted roughly to leveled position, the one essential for the proper run of the LTU/drive unit is the one from left to right position.
In case the LTU unit is not in level it has to be corrected. This can be done with the onboard leveling cylinders (picture 130) which can be activated individually.



Picture 130: **LTU Leveling Cylinders**

To be able to relocate the LTU/drive unit the whole system will be mounted on tracks. The same track frames as for the hopper car will be used for this, therefore please refer to the relevant chapter 6.2.1.).

6.7. Conveyor Belt

The main requirements for the conveyor belt beside that it withstands all system forces are to be as thin as possible and to be extremely durable.
A Good Year Monoply Coal Flo+ C630 RT 1x1 PVC Belt was selected by the author which is manufactured to the SANS 971:2003 standard which is also US-MSHA No. 28-83 as well as No. 28-83/1 approved.
The conveyor belt is a Class 630 Solid Woven (Wrap: Nylon and Cotton, Fill: Cotton) PVC belt with an overall thickness of 7mm. The top and the bottom PVC cover layer is around 0.8mm thick.
The breaking strength of the belt is 630kN/m and the operating strength 63kN/m.

All up 4 lengths with 120m each and 1 length with 60m will be used in the system. Together the material handling system will contain 540m of conveyor belt.
The shorter belt length will be marked so that it will not be mistaken with the other four 120m long belt pieces. This is essential when taking out the added conveyor belt pieces again and

winding them back onto the storage rolls when retrieving the system back out of the finished mining tunnel.
The four 120m long conveyor belt pieces have the same length for the reason, that it does not matter what mechanical link between them has to be removed and which one of the 4 pieces will be removed.

The connection of the belts will be achieved with mechanical joints as shown below in picture 131.

Picture 131: **Conveyor Belt Joint** Source: www.flexco.com

6.8. Belt Tensioning Winch

The tension of the conveyor belt is achieved with a variable frequency controlled winch, another essential part of the system.
The conveyor belt in the system will be tensioned to maximum 15kN.

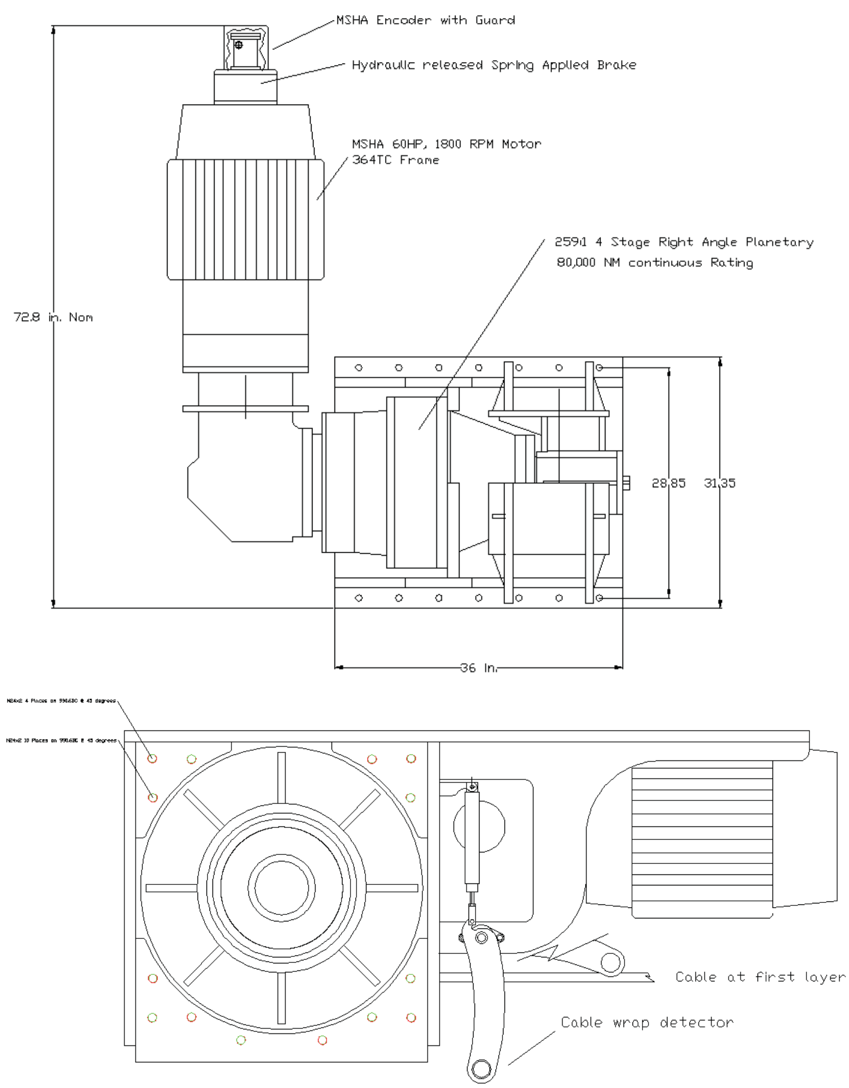
The electric winch consists of the following main components:

- Fully programmable Electronic
- Frequency converter
- Rotation encoder
- Electric AC motor, 450V, 45kW
- Hydraulic brake
- Bevel Gearbox
- Drum
- Steel wire rope
- Rope wrapper

Depending on the set frequency the Frequency Converter is supplying always the same AC frequency to the Electric motor. This determines the tension in the wire rope and further the tension force applied onto the moveable LTU carriage.

Due to the fact that the tension in the belt has to be 15kN and that there are 10 belt strands in the LTU unit, the winch has to apply a tension force of 150kN onto the moveable LTU carriage.
This force is applied via the steel wire rope which is spooled onto the drum and connected to the electric motor via the gearbox.

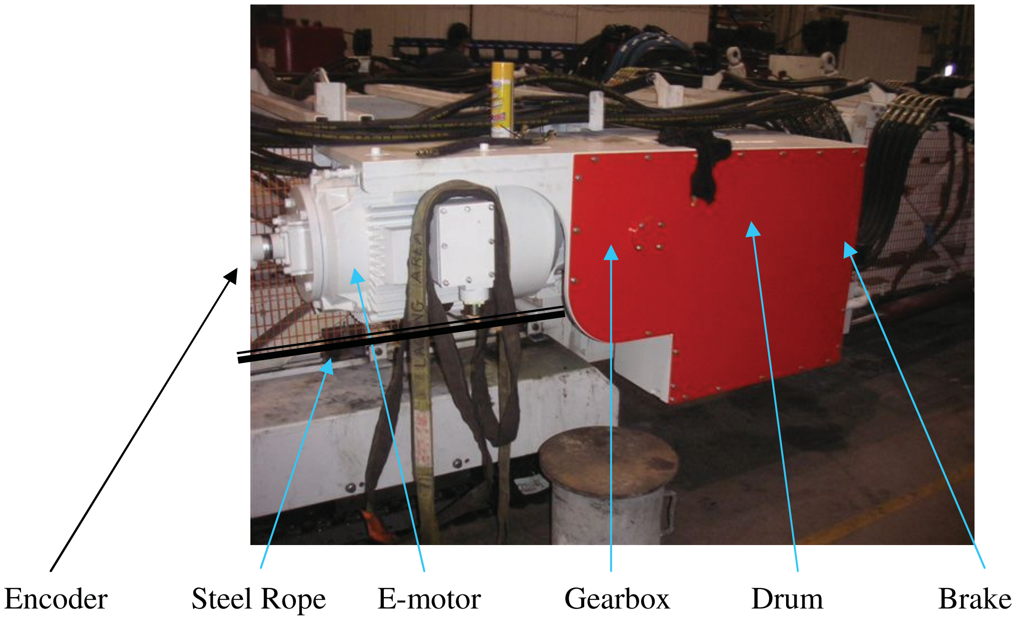
A hydraulic brake (spring loaded and released via hydraulic power) is maintaining the tension when no electric power is supplied to the winch. This brake is required as a safety device in case of emergency situations (picture 132 and 133).



Picture 132: **LTU Winch;** Source: Offering document SMC Electrical Products, Inc. (USA); Sun Source Inc. (USA)

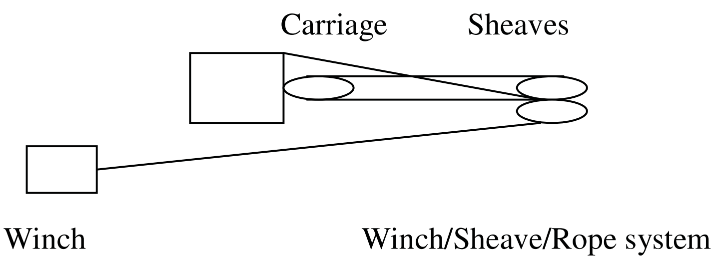
A rope wrapper and a wrap detector make sure that the steel wire rope is getting spooled onto the drum in the right way.

In case the hopper car is advancing and the conveyor is getting extended by pulling belt out of the LTU unit the winch is maintaining the maximum tension of 10kN in the conveyor belt by releasing the rope in a controlled way.
On the way out of the mining tunnel, the hopper car is reducing the belt tension due to the movement in the tensioning direction. The winch is also maintaining the tension in the conveyor belt by pulling in the steel wire rope onto the drum.
The reaction time of the winch in case of changing tension condition is in the millisecond area and can therefore react much quicker than the hopper car moves.
The just explained function of the system to tension the conveyor belt whilst it is operating and the conveyor getting extended and shortened at the same time is one of the key functions developed by the author for the Rapid Coal production System.



Picture 133: **LTU Winch Components**

The tension onto the moveable LTU carriage is applied via a sheave system (picture 134).



Picture 134: **LTU Belt Tensioning Principle**

The actual required power of the drive motor of the winch was calculated with around 30kW. A 45kW motor was chosen by the author so the created temperature in a stalling situation will not exceed critical coil temperatures inside the motor. Due to this over dimensioning of the motor size no forced cooling was necessary.

The reaction time of the frequency controlled winch to changes of the belt tension is in the area of 20 milliseconds and therefore quick enough to avoid belt slack in the system which would cause big problems in the extremely low height conveyor structure and the inside of the hopper car. Also the LTU unit would see the belt loops rubbing on each other and therefore premature wear on the belt as well spikes in the power requirement of the drive.
Belt slack would also cause the winch system to run instable as it might get into a situation where the drive power jumps and the winch tensioning action come into a resonance and make the system fail.

The tension of the conveyor belt will be programmed into the winch electronic a 'torque window'. The actual current tension force will be measured all times and put back into the electronic system for the control circuit. As long as the tension torque stays in between the set values the winch will not react.
As soon as it moves outside the window, the winch system will react. The reaction timing and speed will be tested and determined during the surface test of the system as too quick reaction will be as bad as a too slow one.
The winch electronics is fully programmable and all parameters can be set individually.

Following parameters will have to be set:

- Tensioning force in form of torque on the rope drum
- Reaction time
- Ramping up and down time for applying the torque
- Different torque settings for advancing and retrieving
- Different torque settings for the belt pull in and taking out process
- Start up torques
- Stopping torques
- Torque windows for safe operation of the conveyor system

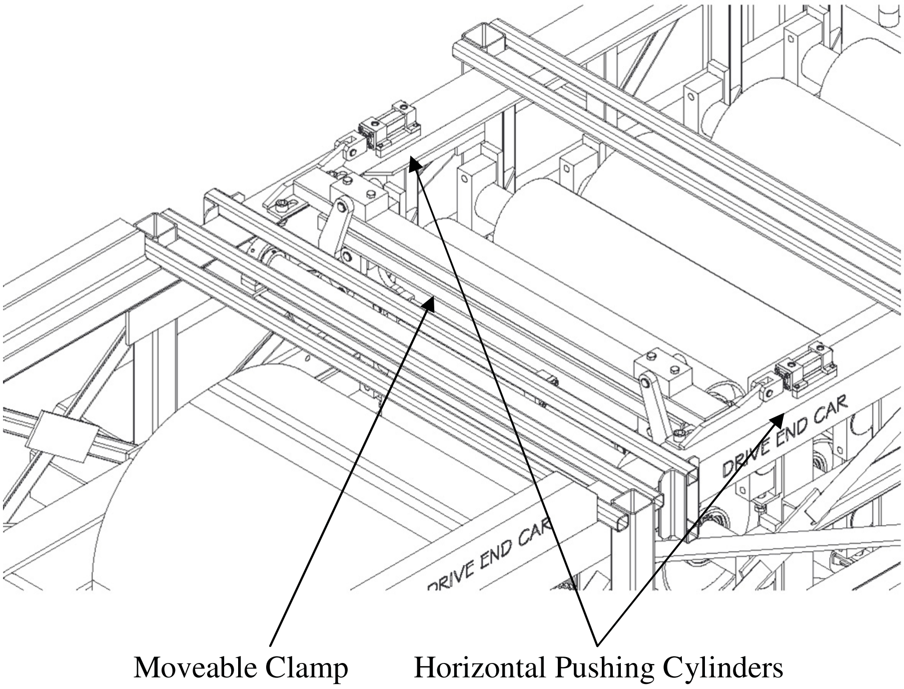
The operation and reaction of the winch in the running system and in combination with the hopper car pulling the conveyor belt or releasing the tension will be tested and finally set during the surface test and refined when in operation underground.

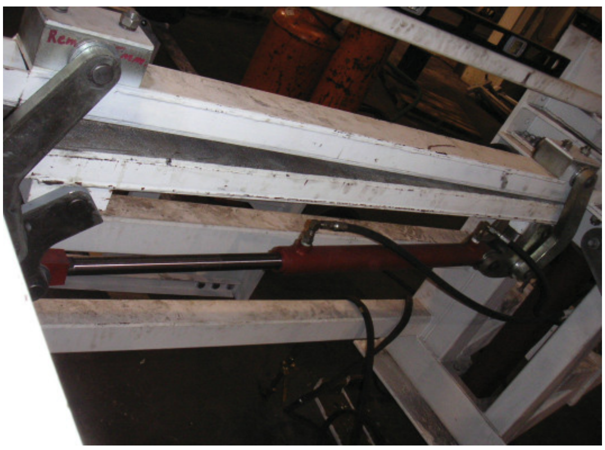
6.9. Conveyor Belt Storage / Extension / Retrieve

Two rolls with each120 meters of additional conveyor belt are stored inside the system (picture 140). They are stored in a way that they can be pulled into the LTU unit with onboard devices only. No additional equipment shall be used. This is another main feature of the system developed by the author to achieve the mining performance.

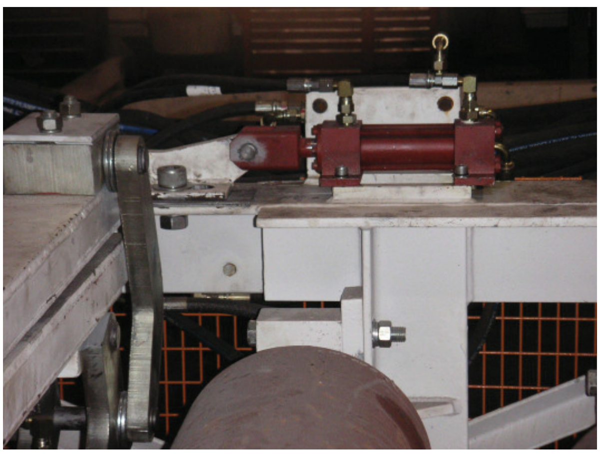
As soon as the LTU unit is empty an additional conveyor belt has to be fed into the system to load up the LTU unit to reach the 200m mining tunnel end.
240m of added belt will give the materials handling system a reach of additional 120m into the mining tunnel.
Two conveyor belt clamps are necessary to do this (pictures 135, 136, 137, 138, 139 and 140).
The first belt clamp, the stationary or fixed clamp (picture 138 and 139) clamps the belt and holds it in position.
The second clamp is moveable to clamp (picture 135, 136, and 137) it and move it against the driving direction to take out the belt tension and to create a slack to be able to take the mechanical belt joint apart.

Picture 135: **Conveyor Belt Clamping Device**



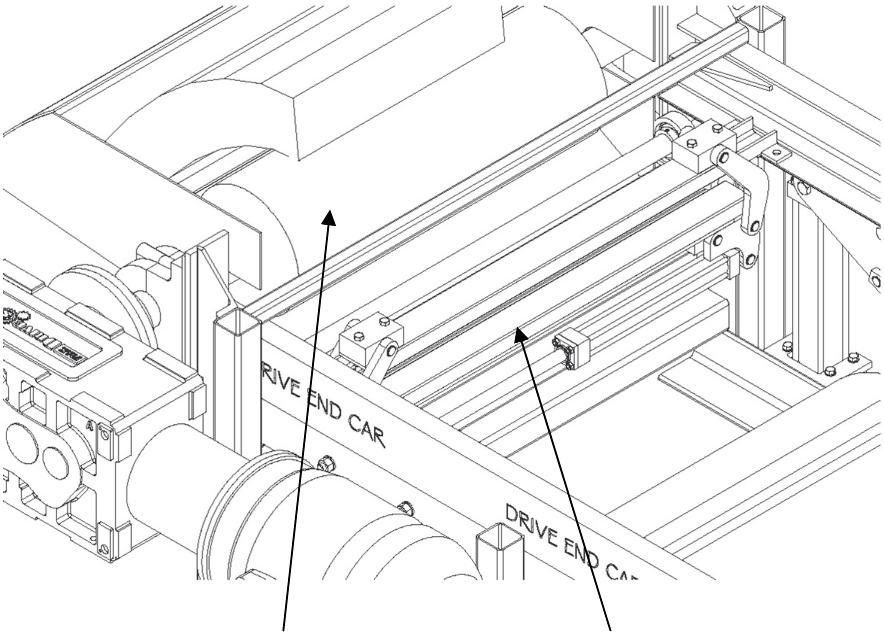


Picture 136: **Conveyor Belt Clamping Device**

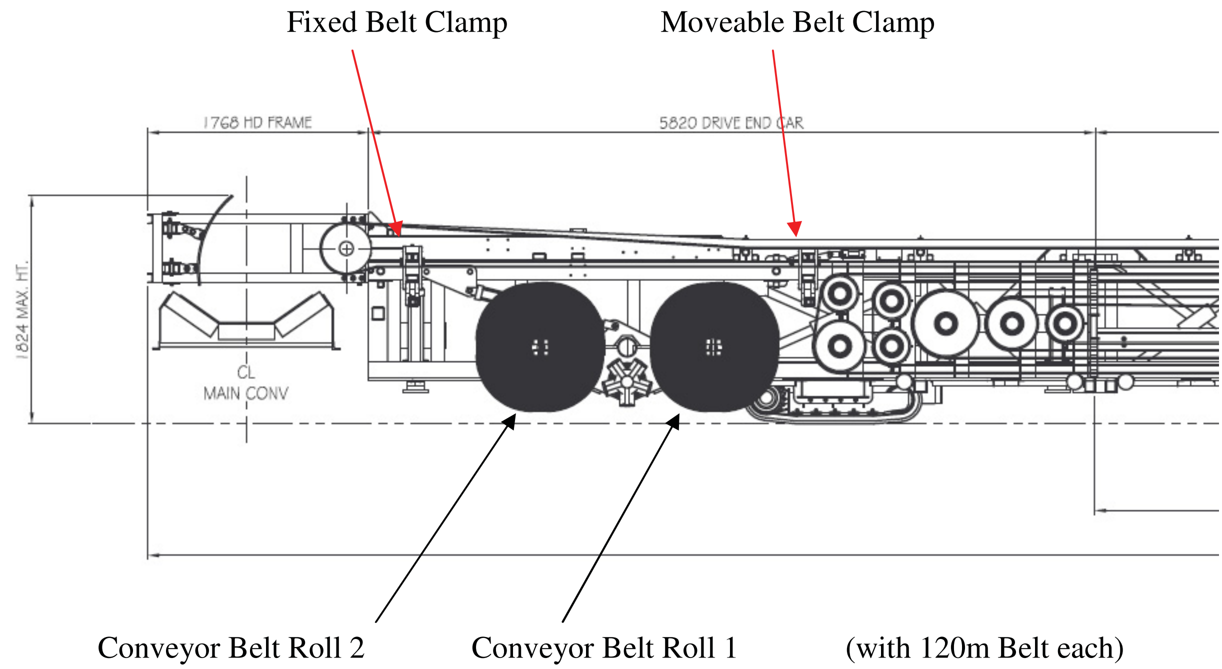


Picture 137: **Conveyor Belt Clamping Device**

Picture 138: **Conveyor Belt Clamping Device**



Picture 139: **Conveyor Belt Clamping Device**



Picture 140: **Conveyor Belt Clamping Device and Belt Storage**

When the LTU unit is empty and the movable carriage is at its minimum limit the conveyor has to be stopped for feeding the additional belt in.

An electric button has to be pushed and the ‘belt pull-in sequence’ will be started.

Three belt joint detectors are mounted in the system (picture 141). The first one will detect the position of a mechanical belt joint in front of the drive/discharge pulley, in the close proximity of the two storage rolls. After detecting the joint the electronic system will slow the conveyor belt down till it reaches the second belt joint detector right in front of the belt clamping system.
The third belt clamp detector will stop the conveyor belt with the mechanical joint right in between the two belt clamps.

The two belt clamps are getting activated and clamp the belt holding it in position.

The next step is to release the conveyor belt tension applied by the tensioning winch.

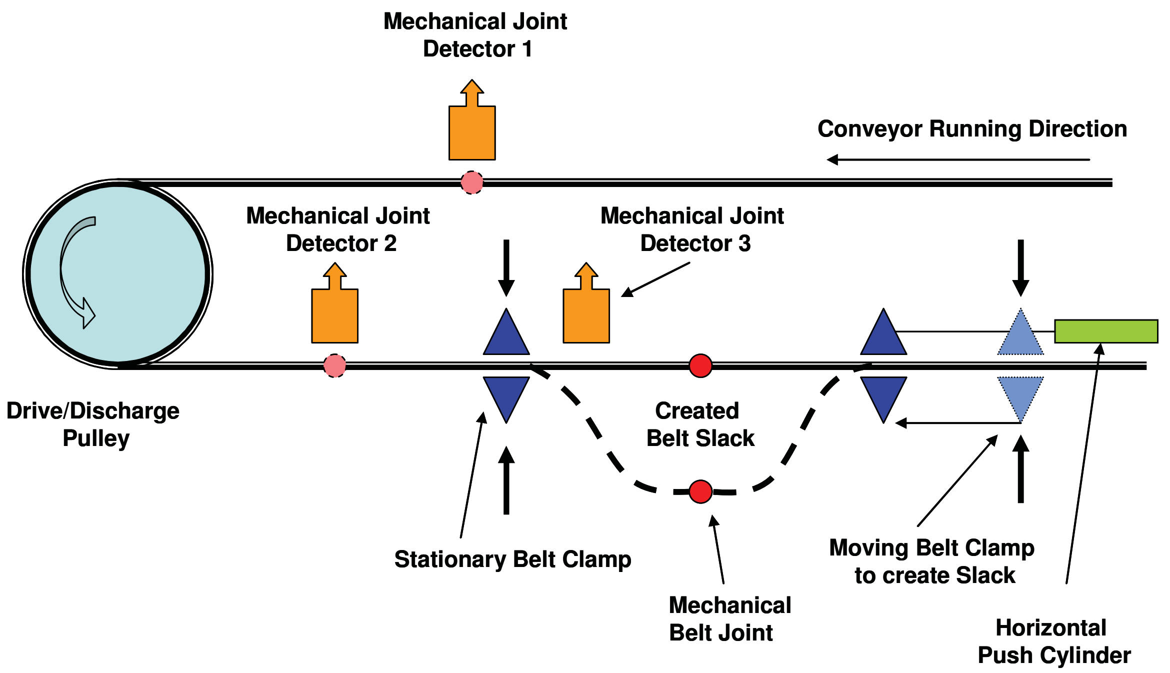
The moveable clamp will be pushed with two hydraulic cylinders in direction the fixed clamp to create a belt slack. This enables the operator to take apart the mechanical belt joint.

The loose conveyor belt end hanging out of the moveable belt clamp will be joined up with the loose end of the first conveyor belt stored on the belt roll 1. The moveable belt clamp will be released and the belt will be pulled into the LTU unit via the tension winch pulling the moveable LTU carriage back towards the maximum limit position.

As soon as roll 1 is empty the loose rear end of belt 1 will be connected to the front end of belt 2 and the winch pulls also belt 2 into the LTU unit.

When belt roll 2 is also empty the rear end of conveyor belt 2 will be joined to the loose end hanging out of the fixed belt clamp and the winch tensions the conveyor belt to the running tension setting.

The LTU unit is filled again and will reach to the end of the mining tunnel.



Picture 141: **Conveyor Belt Pull-In Sequence**

The same sequence only in reverse will be used to take out belt piece 1 and 2 when retrieving the system after finishing the mining process in the mining tunnel.

During the belt-taking-out process belt 1 and 2 are getting wound back onto their respective rolls. This winding up process is getting done with hydraulic motors activated manually when needed driving two spindles as shown below in the pictures 142 and 143.

Picture 142: **Belt Spindles**

Picture 143: **Hydraulic Drive Motors**

Due to the fact that we have reduced the quantity of storage belt rolls down to two, the diameter of each conveyor belt roll must be of a dimension of maximum 1200mm.

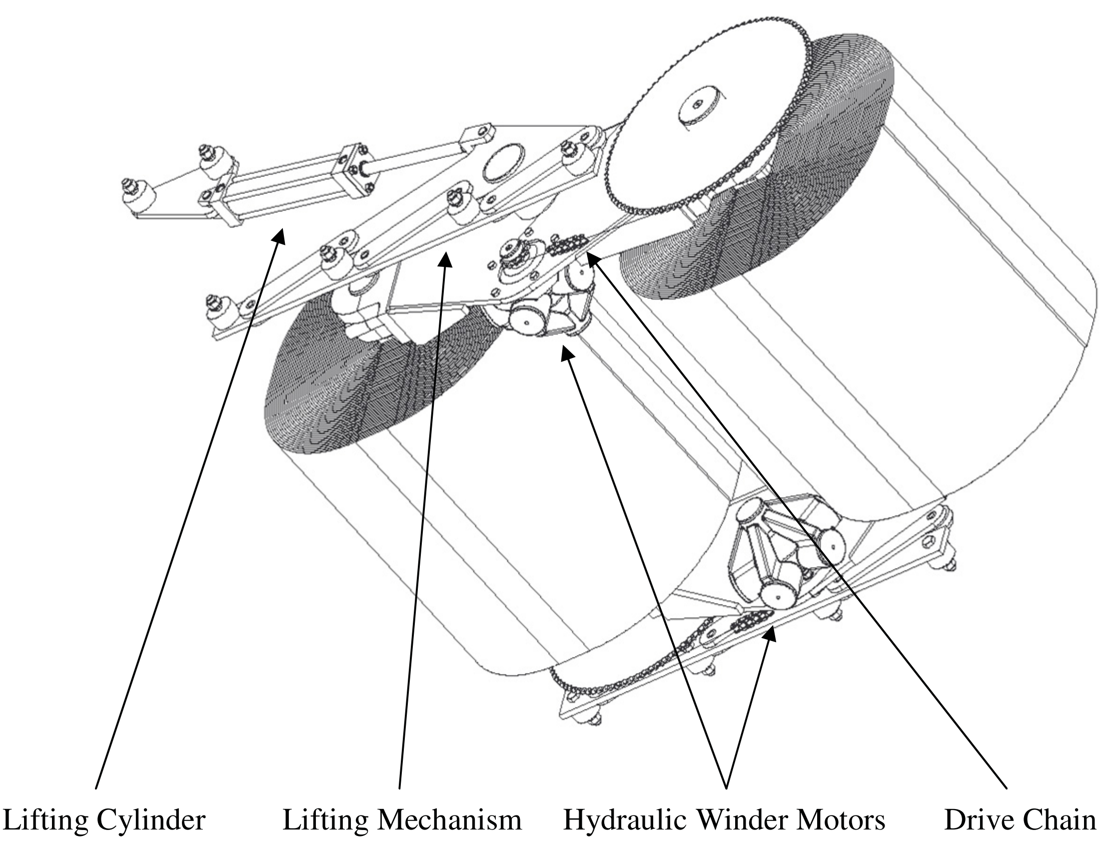
This creates a problem in the storage area!

For relocating the machine we need a ground clearance of at least 200mm. 200mm plus the 1200mm of the rolls is 1400mm in 1600mm allowed overall height.

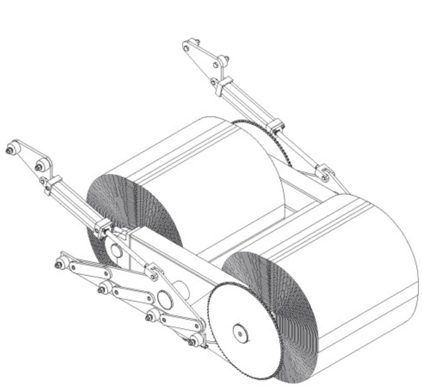
The top of the rolls would collide with and rub on the carrying strand of the conveyor.

A hydraulic lifting mechanism had to be considered and developed by the author which lowers the conveyor belt rolls onto the ground when the material handling system works and the conveyor belt is running (picture 144, 145 and 146).

In the situation of relocating the system the lifting mechanism will lift the conveyor belt rolls automatically when activating the tracks. It lifts the belt rolls to a ground clearance of 200mm.
In this situation the conveyor belt rolls will contact the carrying strand of the conveyor on the top but in this case the conveyor belt is not running, so no damage can occur.



Picture 144: **Roll Lifting Mechanism**



Picture 145: **Roll Lifting Mechanism**

Picture 146: **Roll Lifting Mechanism**

6.10. Conveyor Drive and Discharge Station

Due to the fact that the conveyor should run smoothly with sometimes different speeds a frequency controlled conveyor drive system had been selected.
A frequency controlled conveyor drive works with a variable frequency current supplied by a frequency converter in the electrical system was chosen by the author. The result was the ability to control the starting speed, the braking and stopping process as well as selecting different speeds of the conveyor for different operating situations in the likes of conveying, maintenance, belt feed in process, etc.
Revolutions of an AC electric motor are depending on the pole-pair-number of the motor and the frequency the AC (alternating current) gets supplied to the motor.

$$\text{Rev (rev/min)} = 60 * \text{Frequency (Hz)} / \text{N (Number of Pole-Pairs)}$$

Different speeds of the conveyor are needed for

- Conveying at 2,0m/sec
- Maintenance speed at around 0,5m/sec
- Approaching speed for the belt extension process 1,0m/sec, 0,5m/sec and 0,1m/sec
- Reverse speed at around 1,0m/sec

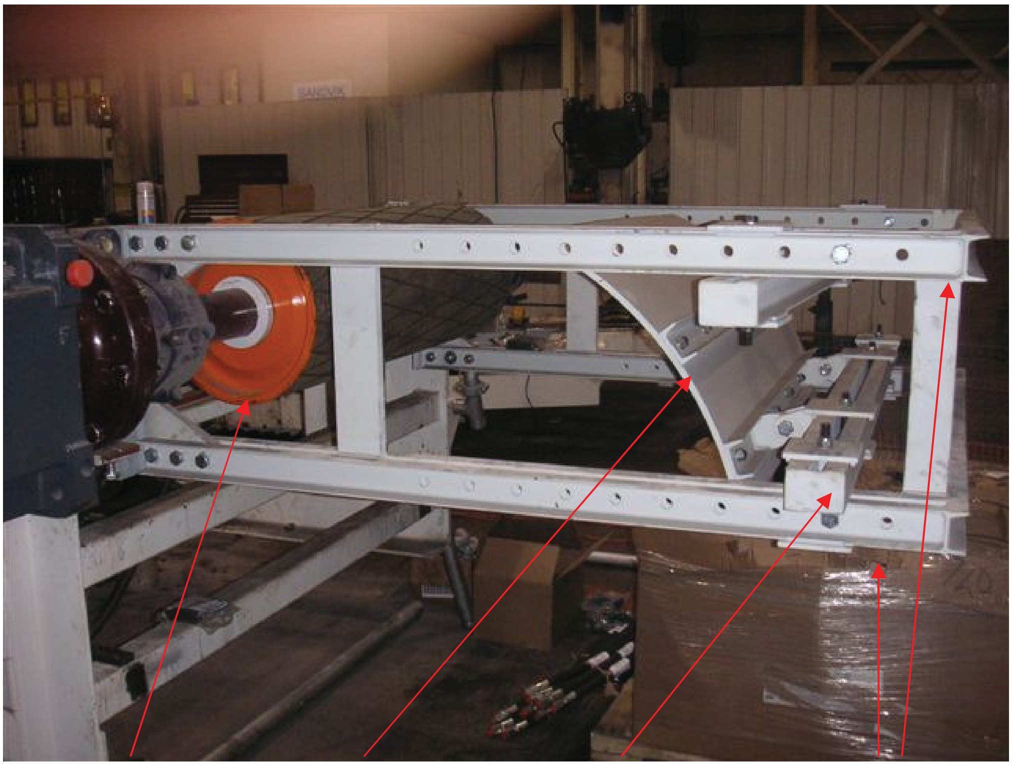
A standard 45kW underground approved squirrel cage 450V, 60Hz, AC E-Motor was selected (picture 147).

Picture 147: **Specification Plate of the 45kW E-Motor**

The motor (picture 148) will be mounted together with a bevel gear box to achieve the maximum conveyor speed. A gear ration of around i=24 had to be selected to achieve the 2,0m/sec conveyor belt speed.

Drive Motor Gearbox Coupling Drive Pulley

Picture 148: **Conveyor Drive Motor**



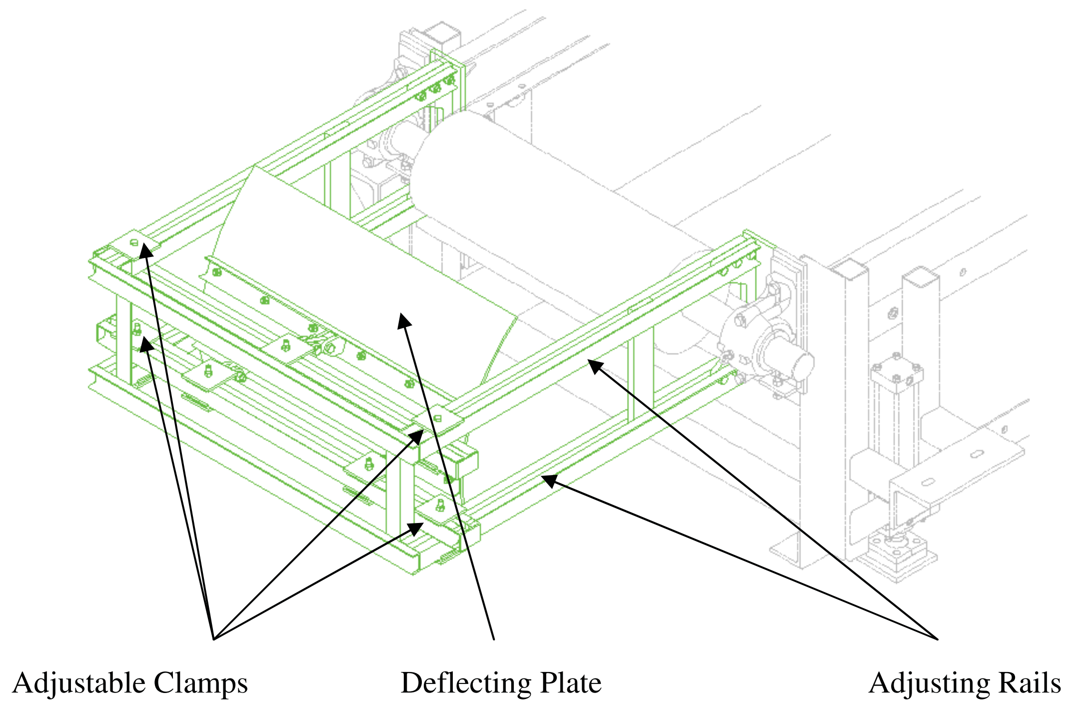
Drive Pulley Deflection Plate Adjustable Clamps Adjusting Rails

Picture 149: **Conveyor Discharge System**

The motor will be flange mounted to the gearbox with a rigid coupling. The gearbox mounted to the drive pulley shaft which is the discharge pulley at the same time. Due to the fact that the discharge pulley is the driving pulley the pulley shell will be rubber lagged.
The frequency converter will be part of the Electrical system and mounted inside an underground approved E-box.

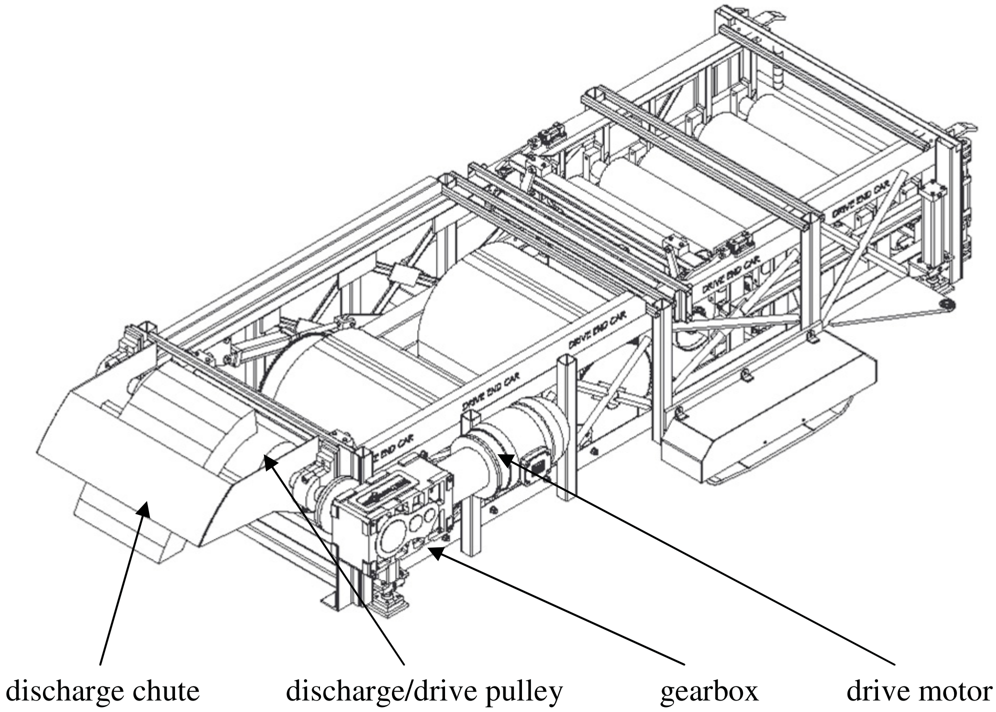
The discharge chute (picture 149 and 150) will be mounted with the discharge/drive pulley to transfer the conveyed material onto to main section conveyor. The section conveyor and the mining system are arranged in an angle of +60deg to each other. Also different angles are necessary to give the necessary flexibility of the system. To be able to do this the discharge chute has to be adjustable for angles between +/- 70 degrees in relation to the conveyor centre line.

The main frame will be mounted rigid; the chute deflection plates will be adjustable in the required angles (picture 150) to be able to optimize the loading onto the panel conveyor.



Picture 150: **Conveyor Discharge System**

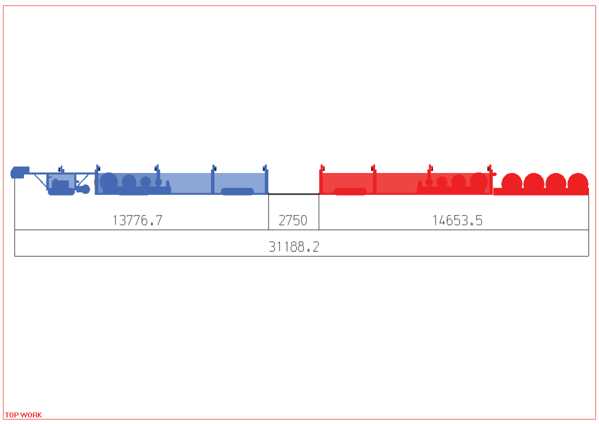
The complete drive/discharge structure looks like to following picture 151.
To optimize the holding of spares on stock, the conveyor drive is as well as the tension winch driven by a 45kW, 60HZ, 450V AC squirrel cage motor.



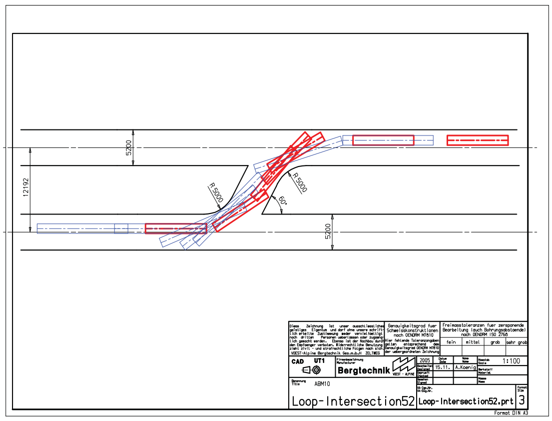
Picture 151: **Conveyor Drive and Discharge System**

6.11. Relocation System

The principle of relocation has been previously discussed for the basic lay out of the mining system.
The loop take up unit (LTU unit) with over 31m length has to be able to tram around corners to the new mining start position on its own.



Picture 152 (compare to picture 68)



Picture 153 (compare to picture 62)

In the mean time the design of the LTU unit got changed by the author in way that the 4 conveyor belt storage rolls had been reduced to two 1200mm diameter rolls and shifted to the rear of the LTU unit in the area where the drive motor and drive pulley had been initially.
To be able to do this the drive pulley was combined with the discharge pulley and therefore the drive unit shifted to the discharge pulley position. This gave space to arrange the belt storage rolls as well as the belt clamping system into the preferred position of the rear of the LTU unit.

Intensive discussions with the manufacturer of the PVC conveyor belt brought out a big problem. It resulted in the fact that the anticipated turning radius of around 5 to 6 meters was too much risk for the conveyor belt edges to get damaged.

The author had planned to have a 2750mm long piece taken out in the middle of the LTU and drive unit to open up an area where only flexible conveyor belt was exposed. This flexible behavior of the belt and the length of 2750mm were supposed to be used to create a steering area for the 31m long unit.
This flexible area should have given the split up LTU and drive unit the ability to go around corners by folding the belt loops in the opened up area.

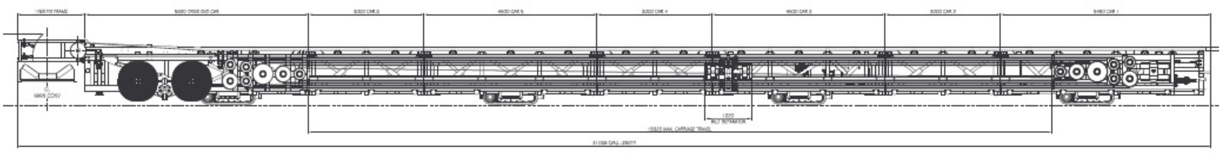
Talking to the conveyor belt manufacturer, the stretching of the conveyor belt edges in the areas of turning would have damaged the belt and a premature failure would have occurred. The angles between the two parts of the split up LTU and drive unit would have been around 45deg when going around the corners.

A maximum of 12deg angle was allowed in one spot for not overstressing the belt edges.

A return to the drawing board was necessary and the author had to design a system where the LTU unit, the drive/discharge unit and the belt storage area were split up into 7 individual carriages.
Every second one was mounted onto two track frames, the same which have been used for the hopper car already.

The individual carriages are connected via a pin system (picture 152).

Using 6 individual carriages or also called cars gave the ability to stay below the 12deg opening angle between two adjacent carriages when going around the corners.



Picture 152: **Final LTU Unit design**

4 track frame pairs have been used for the relocation process. The track frames manufactured by the company Intertractor are the same design as the frames on the hopper car.

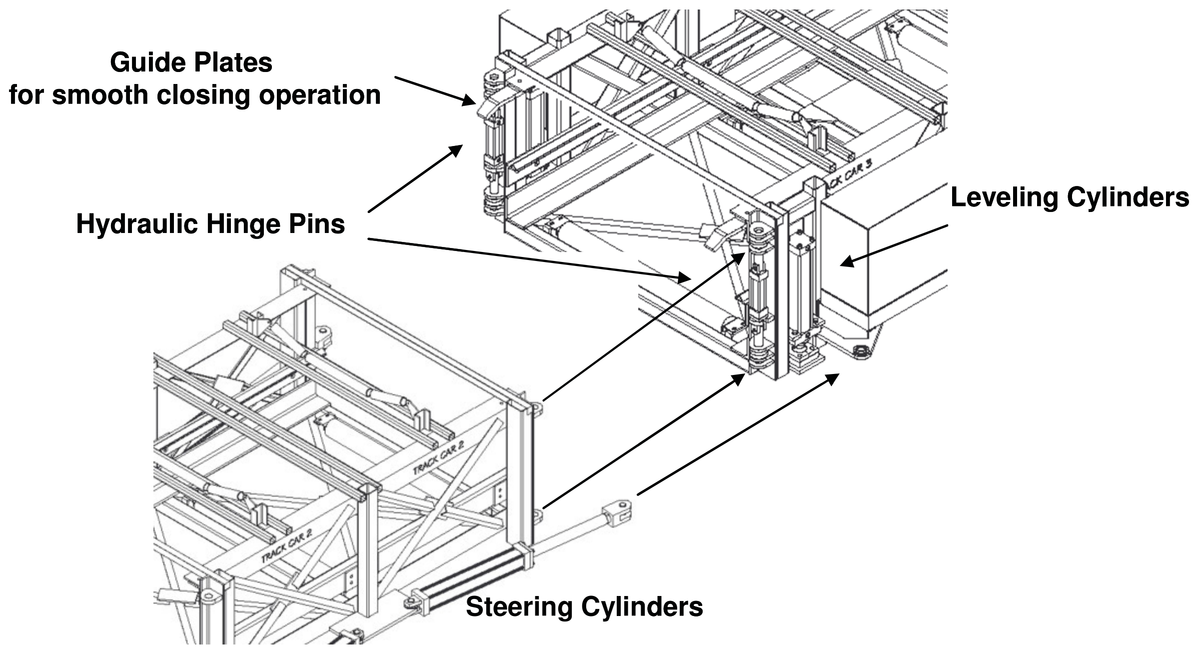
To steer the whole unit around a corner all connection hinges between the individual cars will be opened on one side whilst the hinge pins on other side remains closed to give the pivoting points.

The hinge pins are operated by a double side acting hydraulic cylinder to be able to open the hinges at the same time and from one location.
The opening action will be supported by hydraulic steering cylinders. The cylinders will act two fold.

On the one hand they are limiting the opening angle at 12 degrees to not overstress the belt edges.

The second and main task is to support the steering function when traming the unit around corners. The 6 cylinder pairs can be individually operated.

Guide plates are helping the opening and closing operation of the individual cars (see picture 153, 154 and 155).



Picture 153: **Steering Mechanism and Components**

Stabilizing/Leveling Cylinder Hinge Pin Steering Cylinder

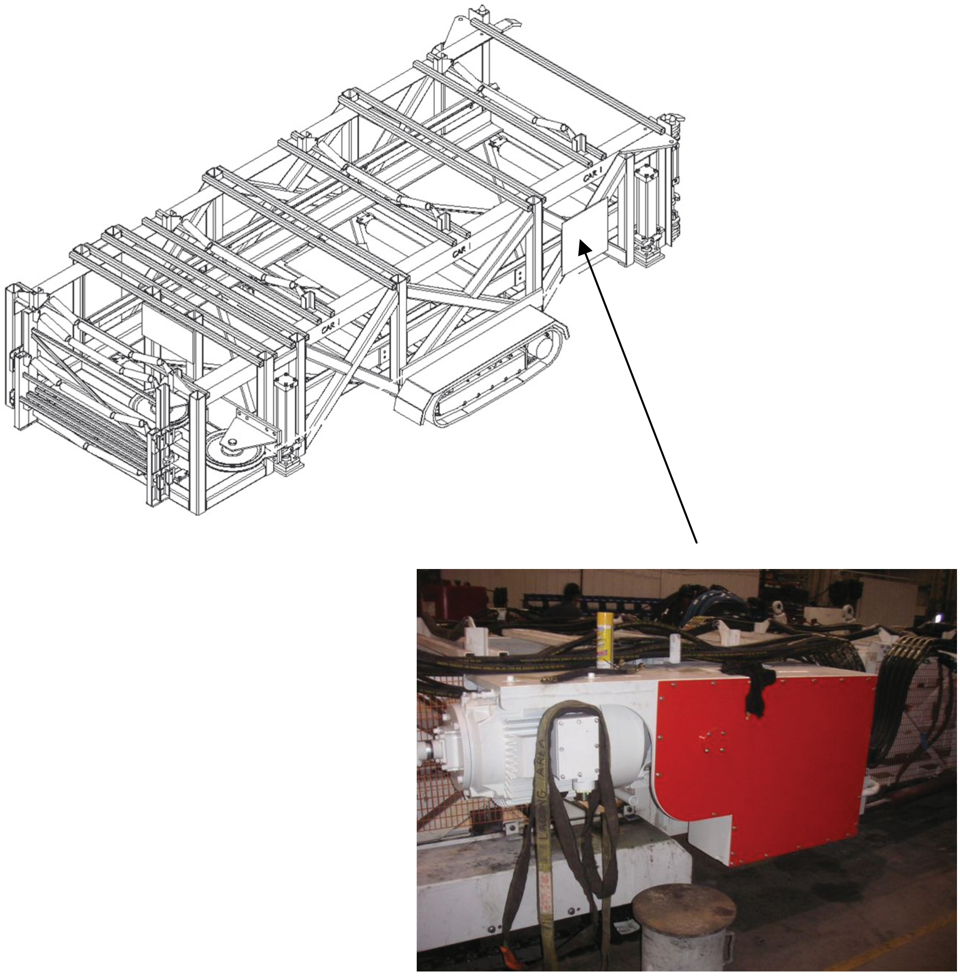
Picture 154: **Steering Mechanism and Components**

Picture 155: **Steering Mechanism and Components**

Individual carriages called cars were designed by the author and equipped to carry functional components to operate the mining system.
The LTU unit and drive/discharge unit was assembled in the following way, starting from the front of the LTU unit.

Car 1 (picture 156):

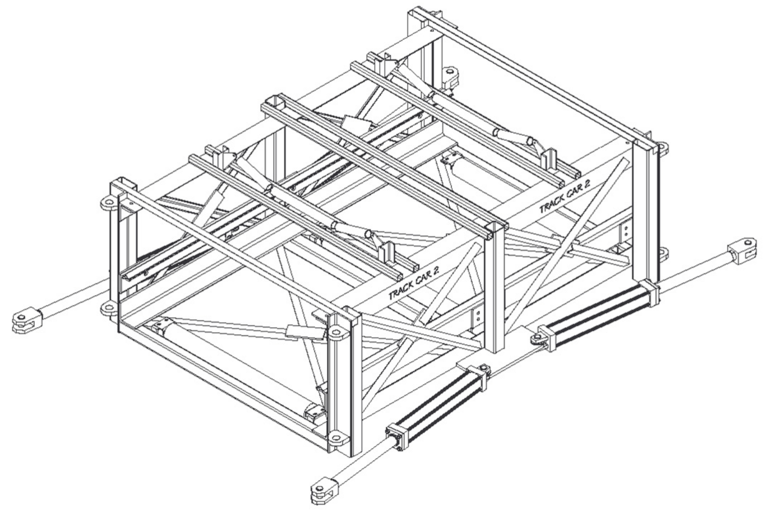
Car1 consists of the two telescopic gradient conveyor structures and also the sheaves for the steel wire robe coming from the tensioning winch and connected to the moveable LTU carriage.
The sheaves are turning the direction of the rope 180deg and guide it to the centre sheave of the moveable LTU carriage.
Also the winch itself will be mounted on the right hand side of this car right behind the track frame.
Car 1 is equipped with one track pair and also 4 off stabilizing/leveling cylinders.
Car 1 is also the maximum limit of the moveable LTU carriage.



Picture 156: **LTU Car No.1 carrying the Winch**

<u>Car 2, 4 and 6</u> (Picture 157):

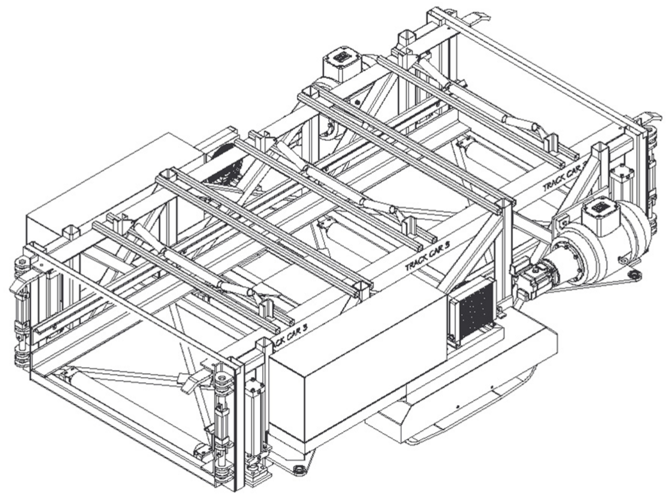
Car 2, 4 and 6 are equal design and are a connection car with 4 off steering cylinders and rails inside for the moveable LTU carriage. Those cars do not have a track system mounted.
They are suspended between the driven cars.



Picture 157: **LTU Car No.2, 4, 6, no tracks, with Steering Cylinders**

<u>Car 3</u> (picture 158):

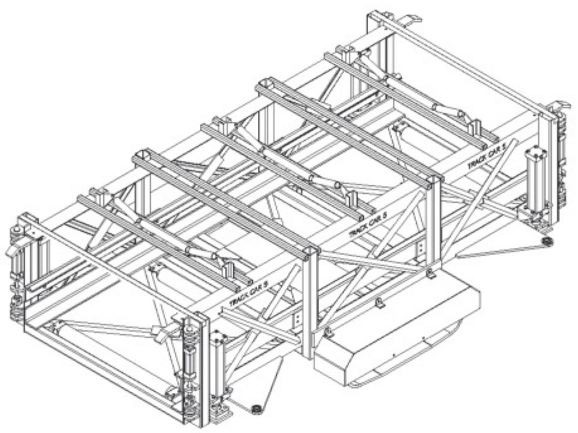
Car 3 will have the Hydraulic Power Pack mounted on both sides above the 2 track frames. Due to the size of the power pack and the actual hydraulic oil volume the power pack was split up into two equal systems each driven by a 45kW hydraulic motor.
Car 3 has got a pair of tracks mounted.



Picture 158: **LTU Car No.3 carrying the Hydraulic System**

Car 5 (picture 159):

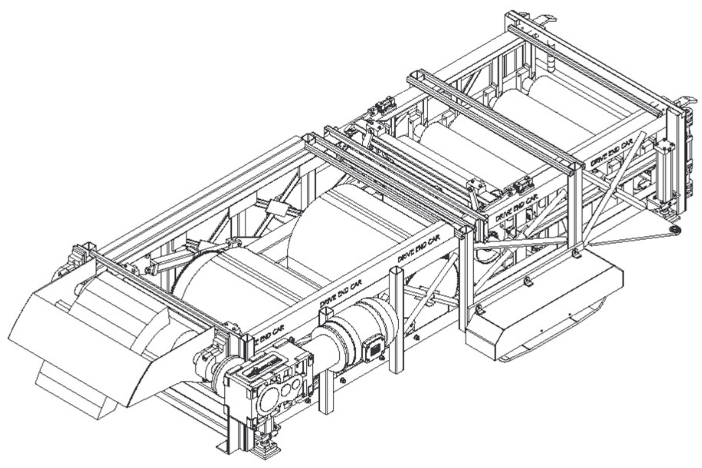
Car 5 is again a driven car and will have the electrical system sitting above the tracks.



Picture 159: **LTU Car No.5, no tracks**

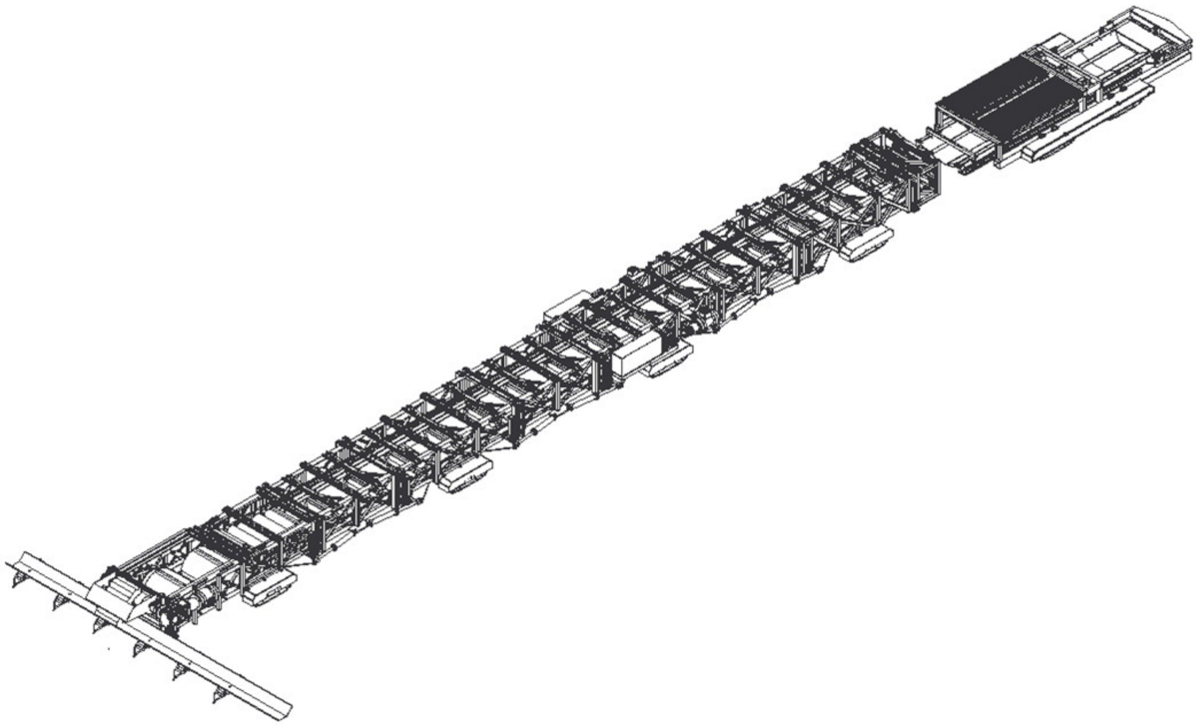
Car 7 (picture 160):

Car 7 is the already discussed car consisting of the discharge area with the chute, the conveyor drive, the additional conveyor belt storage system and the stationary LTU carriage.



Picture 160: **LTU Car No.7 with Discharge and Drive Station and Belt Storage**

The whole material handling system together with the hopper car standing in the fully retraced starting position will look like the following (picture 161).



Picture 161: **Rapid Coal Production System overall**

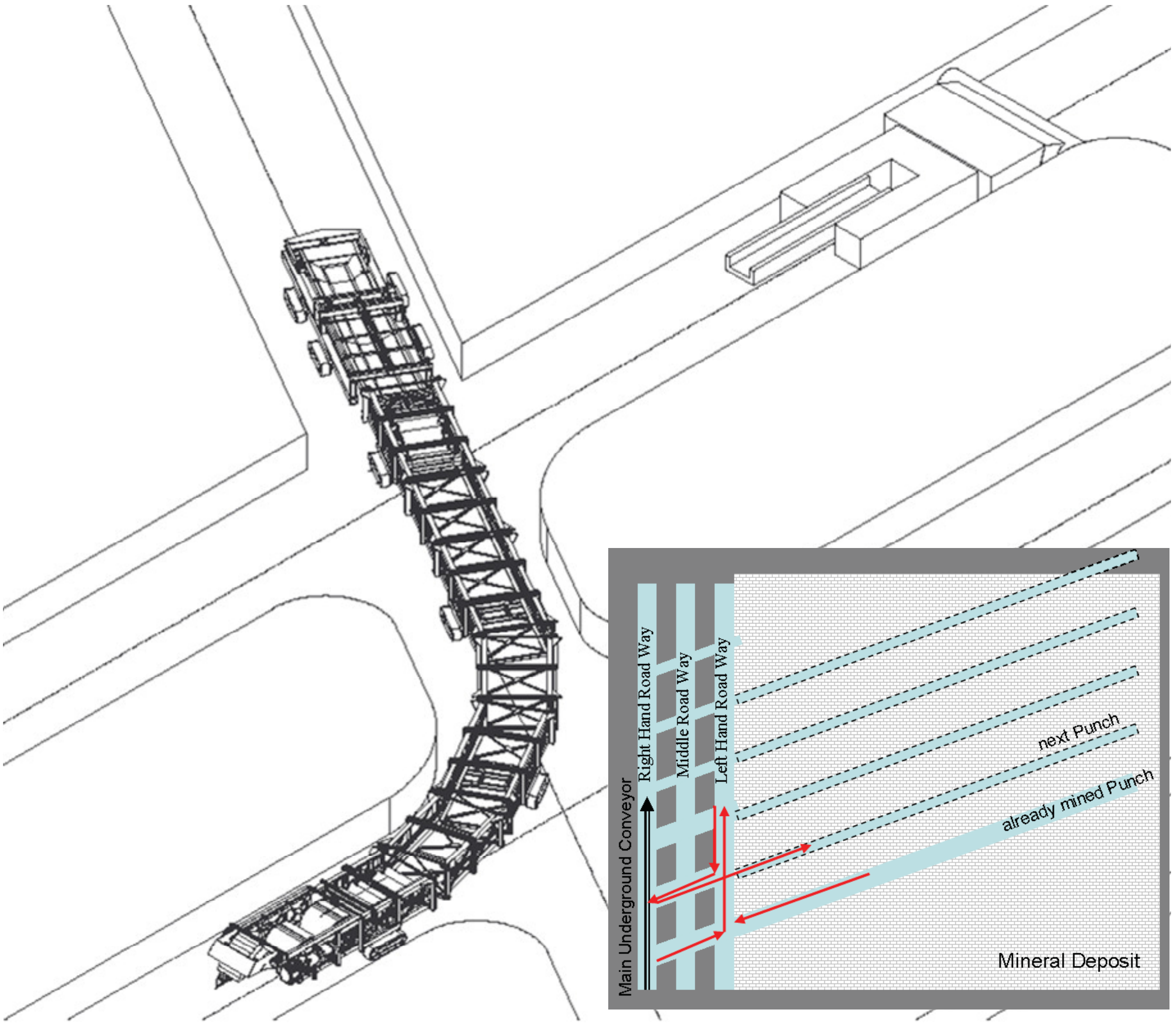
When relocating the system to the next starting position a corner with a radius of 6000mm has to be managed.

The whole relocation process will have to be done manually by three operators.
One operates on each end of the LTU and drive unit and one at the hopper car. The tracks will be operated via manual directional control valves which are interlocked with each other so they only operate together. This is a safety feature to stop the relocation process any time there is a reason for it at any part of the unit.
Each operator has got control over two pairs of tracks and three hinges inclusive the steering cylinders.
The system is also designed in a way that only one side of the hinge pins can be opened at the time.

The relocation process will work in the following steps (picture 162):

- Hopper car will push the telescopic gradient structures into the LTU unit
- Hopper Car will be disconnected from the ABM10 hydraulic
- Hopper Car will be connected to the onboard hydraulic power pack
- Leveling cylinders of LTU unit will be retracted
- Conveyor belt tension will be released

- Conveyor belt storage rolls will be lifted
- LTU unit operators will release all hinge pins on one right hand side of the system
- All operators will start moving the system via the tracks to the towards the turn
- Hopper Car will maneuver around corner
- Hopper Car follows with the support of the steering cylinders (see following picture)
- After finishing the relocation process the hinges will be pulled together
- All hinge pins will be engaged and locked
- Belt storage rolls will be lowered
- Belt will be tensioned
- Leveling cylinders will be used to bring the LTU unit in horizontal position (spirit levels)
- ABM10 moves to the front of the hopper car
- Hopper Car hydraulic will be connected to ABM10 hydraulic
- ABM10 and hopper car move to next starting position

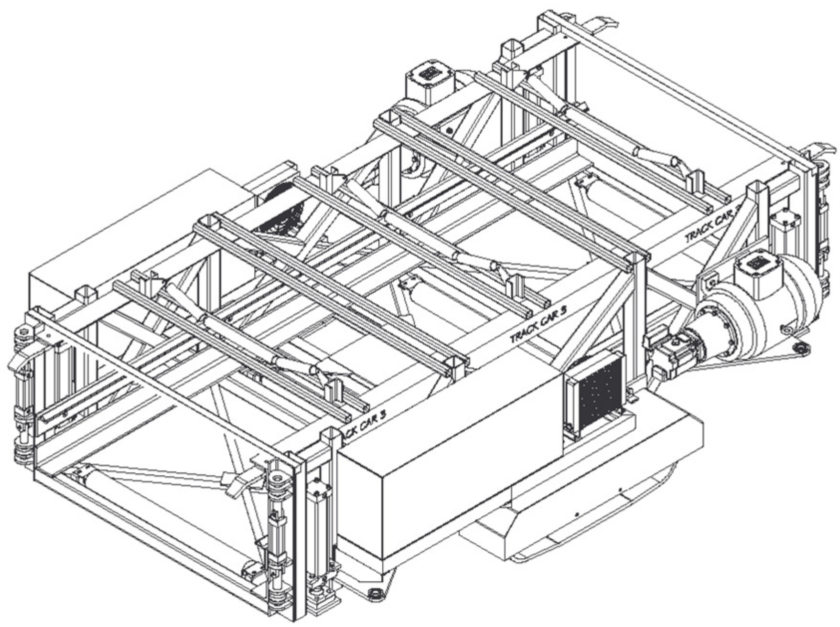


Picture 162 (compare to picture 75): **Rapid Coal Production System Relocation Sequence**

The relocation route is also shown following the arrows in the above shown picture 162.

6.12. Hydraulics

The large size of the hydraulic power pack made it necessary for the author to split up the system into two equal units.
Each side of the car number 3 (picture 158) one of the units will be mounted. Each unit will supply the relevant side of the material handling system. Even though the two power pack tanks are connected to communicating containers each side will run individually.



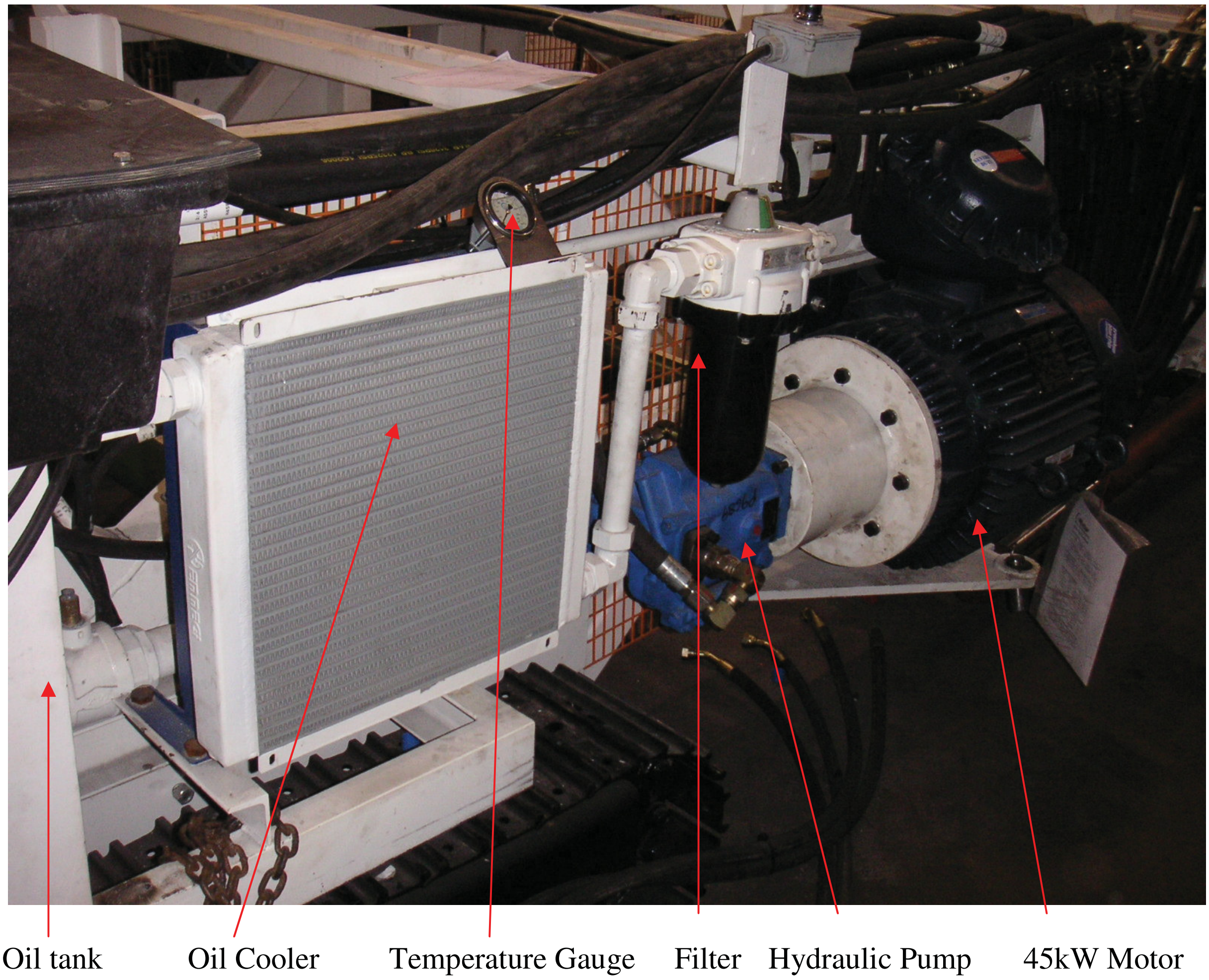
Compare to Picture 158: **Hydraulic Power Pack**

The relocation process will take more than 4 hours during which the hydraulic power pack will run permanently to supply oil to the tracks and the cylinders. The created temperature rise will be controlled by the two forced air coolers which again will be driven hydraulically.

The hydraulic power pack will consist of (picture 163):

- 2 x 45kW, 60Hz, 450V squirrel cage motors
- 2 x 45l/min, 210bar hydraulic variable volume piston pumps

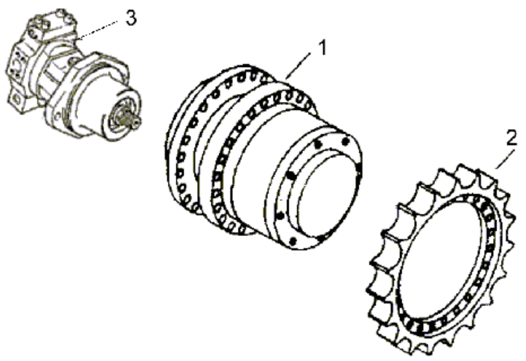
- 2 x 250l oil tanks
- 2 x 10 micron return oil filters
- 2 x forced air oil coolers
- 2 x oil filling filters
- 4 x pressure gauges
- 2 x temperature gauges
- 2 x oil level gauges
- 2 x dirt gate breather filters



Picture 163: **Hydraulic Components**

The hydraulic system consists of the following users:

- 12 x hydraulic motors of the track system (4 in hopper car, 8 in LTU unit)



1 Compact transmission

2 Sprocket

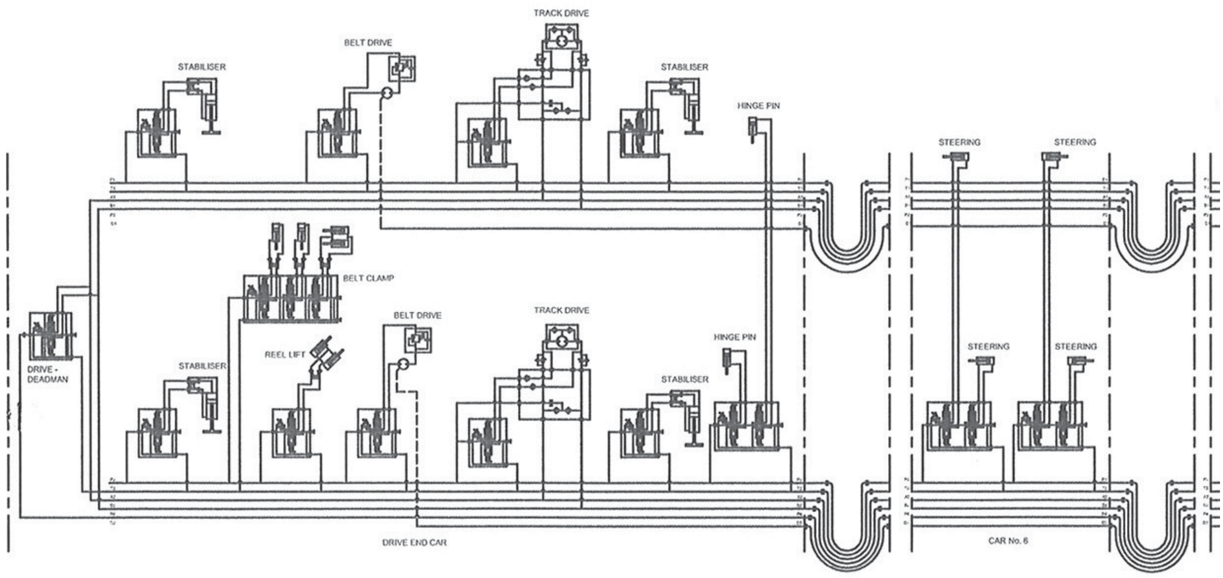
3 Diagram of a servo-hydraulic motor

Compare to Picture 87: **Hydraulic Track Drive Motors;** Source: www.intertracktoramerica.com,

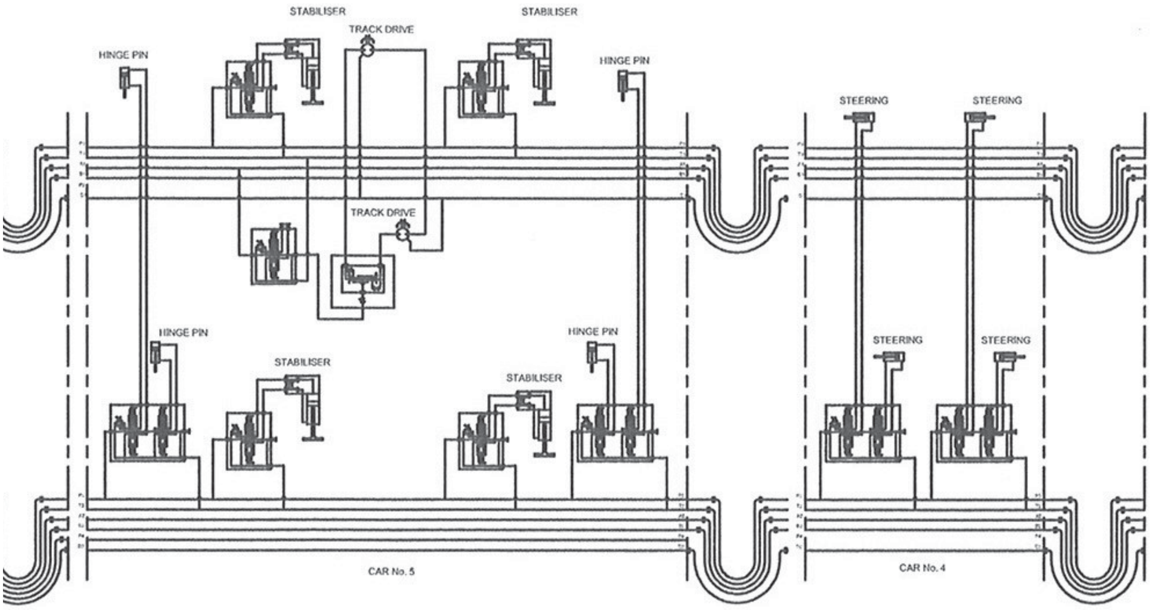
- 16 x leveling cylinders (LTU unit)
- 12 x steering cylinders (LTU unit)
- 2 x pendulum cylinders (hopper car)
- 1 x hopper lifting cylinder (hopper car)
- 3 x belt clamping cylinders (LTU unit)

All used hydraulic hoses will be 4 layer UG MSHA approved hoses.

Following the hydraulic schematic (picture 164, 165, 166 and 167) is shown starting from the drive unit end to the hopper car.

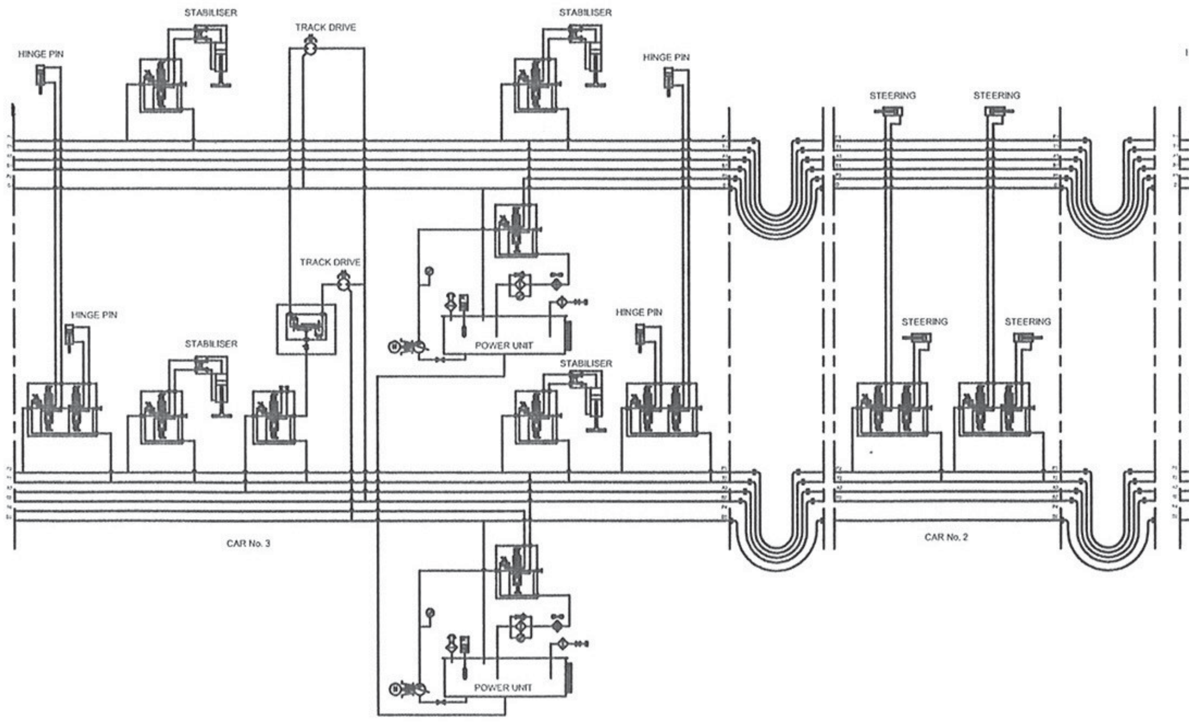


Picture 164: **Hydraulic Schematic incl. Belt Pull-In Devices**

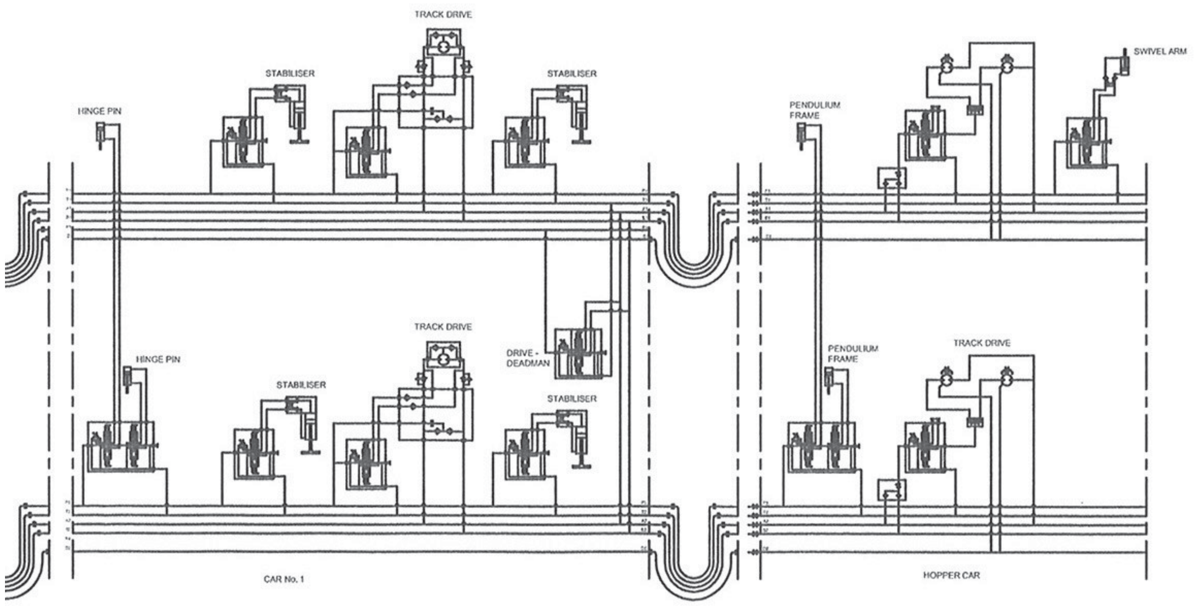


Picture 165: **Hydraulic Schematic**

The hydraulic hose loops shown on the schematics are the flexible hoses in between the individual cars giving the ability to open up the hinges over the 12 deg opening angle.



Picture 166: **Hydraulic Schematic incl. Hydr. Power Pack**



Picture 167: **Hydraulic Schematic incl. Hopper Car**

6.13. Electrics

Special emphasis had to be given to the electrical system and the approval for underground applications in coal mines.
Due to the fact that the electrical system also has to be MSHA approved, it is essential to consider those regulations diligently as changes to the system after it is finished and submitted for approval are very time consuming and delay the project unnecessarily.

Even though the system will not be considered permissible (flame and explosion proof) as it will be used out by the danger zone and in well ventilated environment, all components used will be engineered as such so the machine can be used in a gas environment when necessary.

Following basic parameters have to be considered and realized for the end product in cooperation with SMC Electrical Products, Inc. (USA) an expert in underground electrical systems:

Enclosure:

MSHA rated X/P, multiple boxes permitted if necessary.
Windows for viewing an Allen Bradley Panel view display and the belt visible disconnects
External operators as outlined
Painted white inside and out
All door panels hinged

Power Circuitry:

See attached one line diagram
Entrance cable #2 G-GC 2KV

Controls:

All interlocking and control via an Allen Bradley SLC 5/04
Fault displays etc. via an Allen Bradley Panel view 550T DH+
External DH+ output via a plug / receptacle
External devices via MSHA approved X/P receptacles preferred KH CXP-1
Inputs and outputs as listed on the attached unit specifications

Miscellaneous:

5 KVA Utility / control power transformer
Six (6) separate push buttons to operate panel view

Note: Great care must be taken to allow for heat dissipation due to the two frequency drives.

Belt Drive Electrics

Motor 60 HP Reliance 1800 RPM 480 VAC 3 Phase 60 HZ
VFD drive controlled by the panel view
Line reactors if required
Circuit Breaker Cutler Hammer MCP
Visible disconnect with lockout provisions
Current transducer, 150 amps full scale 4-20 ma output, or from VFD
MSHA approved continuity type ground monitor
Ground fault relay

Internal Connections:

Overload from VFD
VFD Fault
Local stop button*
Local start button
Spot belt button
Run auxiliary contact
Circuit Breaker auxiliary contact
Current 4-20 ma analog
Run Normal to VFD
Run Slow to VFD
VFD Fault reset
PLC to VFD speed reference analog
Ground Monitor relay contacts*
Ground fault relay contacts*

External Connections

Emergency Stop via an MSHA approved IS Relay*

Remote Stop via an MSHA approved IS Relay*
Sequence proximity switch
Head alignment switches via an MSHA approved IS Relay
Tail Alignment switches via an MSHA approved IS Relay
Speed detection high speed proximity switch
Fire Detection via an MSHA approved IS Relay
Chute Plug / Spillage via an MSHA approved IS Relay
Slow down point Proximity switch
Stop point Proximity switch
Audible warning horn output

* = Also hardwired

Hydraulic Pump Electrics (2 each)

Motor 60 HP Reliance 1800 RPM 480 VAC 3 Phase 60 HZ
FVNR Vacuum Contactor Allen Bradley Bulletin 1102-BOD93
Circuit Breaker Cutler Hammer MCP
Automatic reset overload relay Allen Bradley Bulletin 592 E1 Plus
Current transducer, 150 amps full scale 4-20 ma output
MSHA approved continuity type ground monitor
Ground fault relay

Internal Connections:
Circuit Breaker auxiliary contact
Overload contact*
Local stop button
Local start button
Starter auxiliary contact
Current 4-20 ma to PLC
Run output
Ground Monitor relay contacts*
Ground fault relay contacts*

External Connections:
Oil Level switch
Oil Temperature switch

Note: Also connected to Emergency stop circuit

Vector Winch Drive Electrics

Motor 60 HP Reliance 1800 RPM 480 VAC 3 Phase 60 HZ
VFD drive controlled by the panel view for the Drive motor
Line reactors if required
Dynamic Braking resistor

Automatic reset overloads for the blower motor
Circuit Breaker Cutler Hammer MCP
Current transducer, 150 amps full scale 4-20 ma output, or from VFD

Internal Connections:
- Circuit Breaker auxiliary contact
- Overload contact or from VFD
- Local stop button*
- Local start button
- Starter auxiliary contact or VFD run indicator
- Current 4-20 ma to PLC
- Run output
- Ground Monitor relay contacts*
- Ground fault relay contacts*
- Braking Resistor over temperature
- Brake Chopper status
- VFD Fault
- VFD fault reset
- % Torque
- Wind Speed

External Connections:
- Winch motor over temperature
- Position Encoder (Optional)
- Brake Release output
- Break Release limit switch
- Over travel limit switch #1 via IS relay
- Over travel limit switch #2 via IS relay
- Final over travel limit switch via IS relay

Note: Also connected to Emergency stop circuit

Vector Winch Blower Electrics

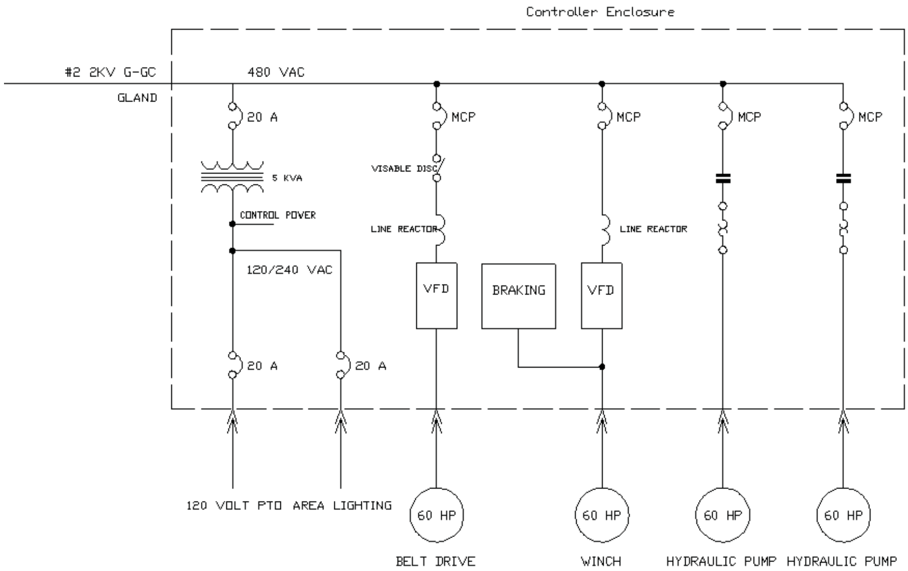
Motor 1.5 HP 1800 RPM 480 VAC 3 Phase 60 HZ
Contactor Allen Bradley Bulletin 505
Circuit Breaker Cutler Hammer MCP
Automatic reset overload relay Allen Bradley Bulletin 592 E1 Plus
MSHA approved continuity type ground monitor
Ground fault relay

Internal Connections:
- Circuit Breaker auxiliary contact
- Overload contact*
- Local stop button
- Local start button

Starter auxiliary contact
Run output
Ground Monitor relay contacts*
Ground fault relay contacts*

Note: Also connected to Emergency stop circuit

A rough one-line diagram in picture 168 shall illustrate the electric system of the machine:



Picture 168: **One-Line electrical Diagram;** Source: SMC Electrical Products, Inc.

The 45kW/60HP motors have the following specification (table 23):

DATA SHEET
Three-phase Induction Motor - Squirrel Cage

Customer	: PMS
Motor line	: Standard NEMA

Frame	: 364/5T	Service factor	: 1.15
Rated Output	: 60.0 HP (cv)	Duty cycle	: S1
Frequency	: 60 Hz	Ambient temperature	: 40 °C
Poles	: 4 poles	Altitude	: 1000 m.a.s.l
Full load speed	: 1775 rpm	Degree of protection	: IP55
Slip	: 1.39 %	Aprox. weight	: 380 kg
Voltage	: 208-230/460 V	Moment of inertia	: 0.6999 kgm²
Full load current	: 155-140/70.0 A	Noise level	: 75 dB(A)
Locked rotor amps	: 1115-1008/504 A		
Locked rotor current (Il/In)	: 7.20 - Code H		
No load current	: 52.9-47.8/23.9 A		
Full load torque	: 237 Nm		
Locked rotor torque	: 230 %		
Breakdown torque	: 270 %		
Design	: A		
Insulation class	: F		
Temperature rise	: 80 K		
Locked rotor time	: 20 s		

	D.E.	N.D.E.
Bearings	6314-C3	6314-C3
Regreasing int.	9789 h	9789 h
Grease amount	27 g	27 g

Performance under load

Load	cos ø	Efficiency(%)
100%	0.87	93.0
75%	0.84	92.2
50%	0.75	91.0

Table 23: **Data Sheet 45kW E-Motor;** Source: SMC Electrical Products, Inc.

The author has used the same E-motor for all applications in the material handling system.

Along the LTU and drive unit a Pull Wire Emergency Stop system will be installed and 4 Emergency Stop Buttons on each end of the hopper car. The Emergency Stop buttons on the hopper car will connected to the ABM10 electrics as are the area lights of the hopper car. The reason is to avoid the electric cable running along the conveyor system.
Only a communication line between the LTU unit, the hopper car and the ABM10 will be installed (intrinsically safe) to mange the material flow and stop the system when necessary.

Control sensors had also been installed by the author for system feed back. Following sensors were installed:

- Conveyor belt drift sensors (LTU unit, hopper car)
- Speed sensors (drive pulley, return pulley)
- Conveyor belt rip detector (LTU unit)

Warning lights and horns are used to communicate the operation of the conveyor system.

At the LTU unit, the drive/discharge unit and the hopper car a horn will be installed which will sound before the system starts up. At the same time the horn sounds a yellow light will blink to inform about the up coming conveyor start.

Further a red, a yellow and a green light will be installed with the following functions:

- Green light on > all operating systems are ok and running
- Yellow light blinking > LTU unit empty after 10 more meters advance distance
- Yellow light blinking > same time as horn sounds for up coming conveyor start
- Yellow light on > LTU unit empty
- Red light on > system stopped, danger or system failure

Area lighting will be installed at strategically points like along the LTU unit, at the discharge point, at the drive system and along the hopper car. The lighting will be installed on both sides of the units.

6.14. Finite Element Analysis

After all steel structures had been finalized the whole assembly was undergoing a Finite Element Analysis to check on the capability to withstand the applied and possible additional forces.
The normal and also some special extreme load cases had to be investigated by the author together with an FEA specialist and the reaction of the system in case of such stresses calculated.

3 load cases had been investigated.

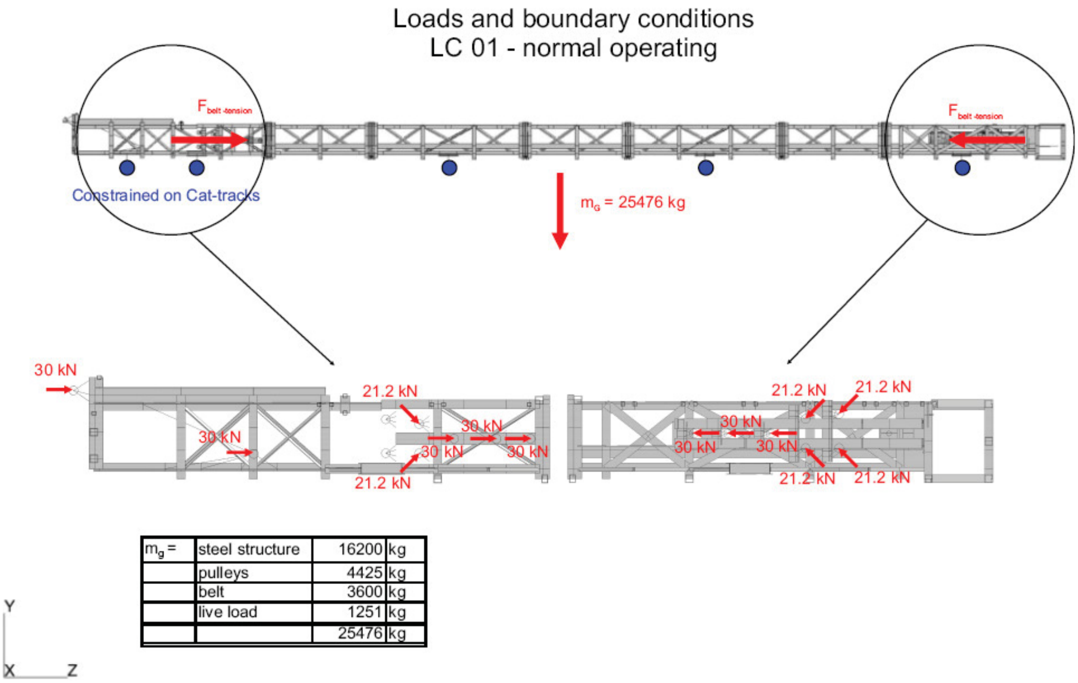
- Normal operating condition
- Normal travel of the unit around a curve
- Travel of the unit around a curve with one set of track frames lifted off the ground

Load case L01, normal operating condition will emphasis on the compression force. The 10-LTU-pulley system is introducing into the steel structure. This force is mainly created by the tension of the 10 loop belt storage and the tensioning winch.

Load case L02, normal travel around corners is specifically looking at the integrity of the hinges and the lug systems.

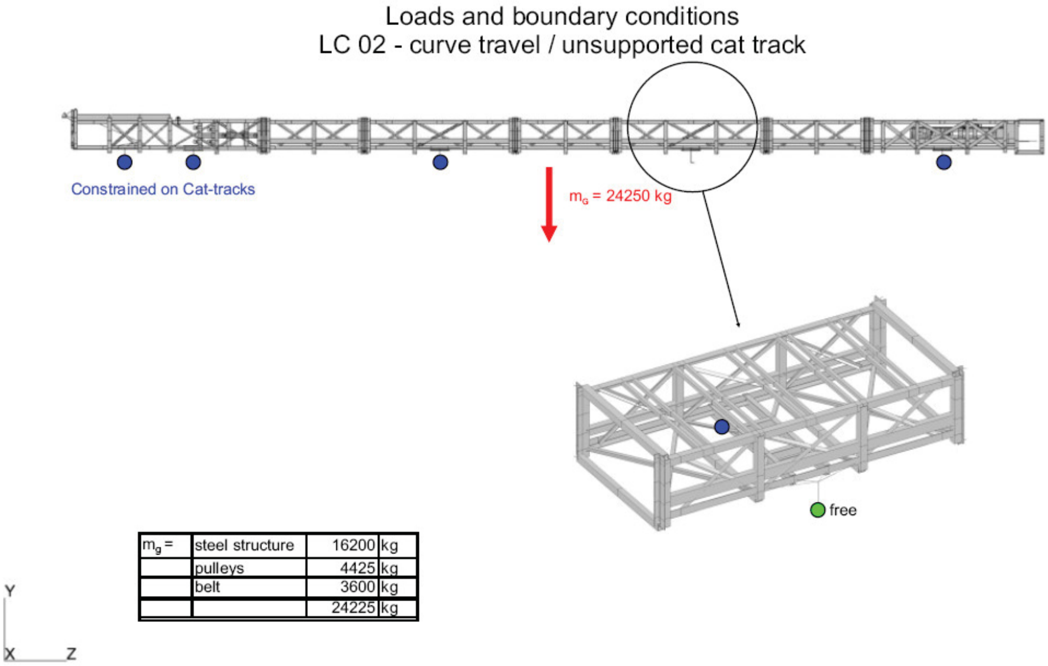
Load case L03, travel around curve with one track pair unsupported is looking at the situation when the long LTU unit runs over a dip and one pair of tracks (LHS and RHS of one carriage) is hanging in the air. The remaining cars have to take the weight as well as the hinges have to sustain higher stress levels and bending moments.
Following pictures are explaining the forces applied and used for the FEA.

Load case L01 (picture 169):



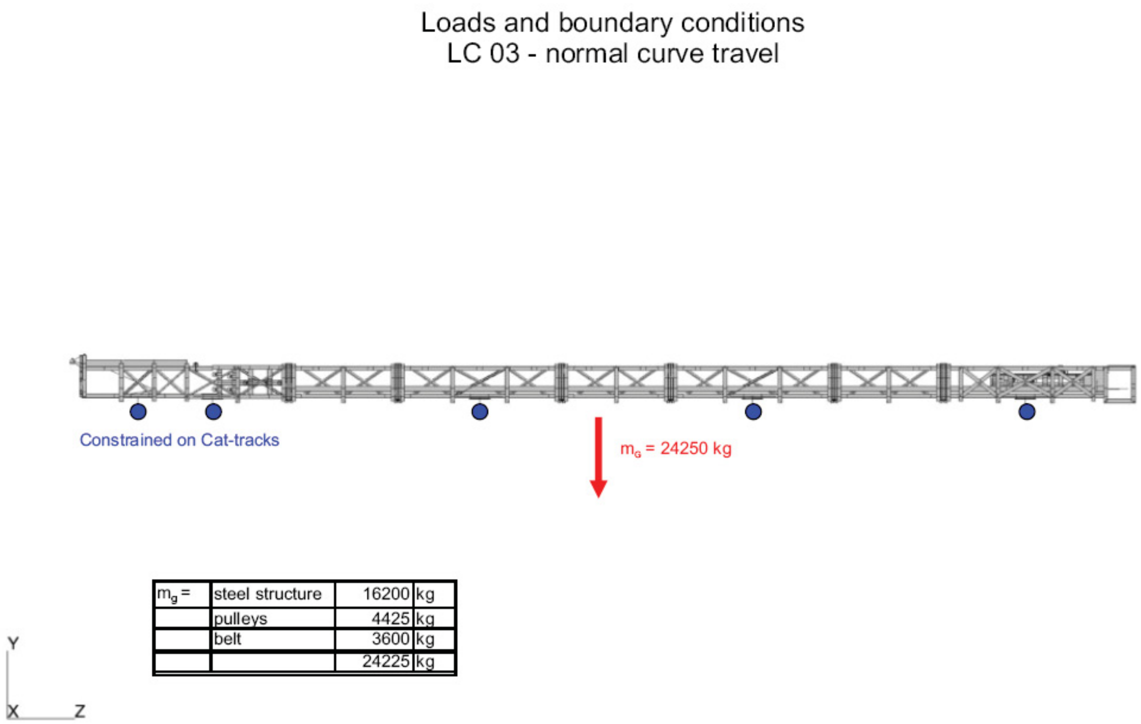
Picture 169: **FEA Load Cases**

Load case L02 (picture 170):



Picture 170: **FEA Load Cases**

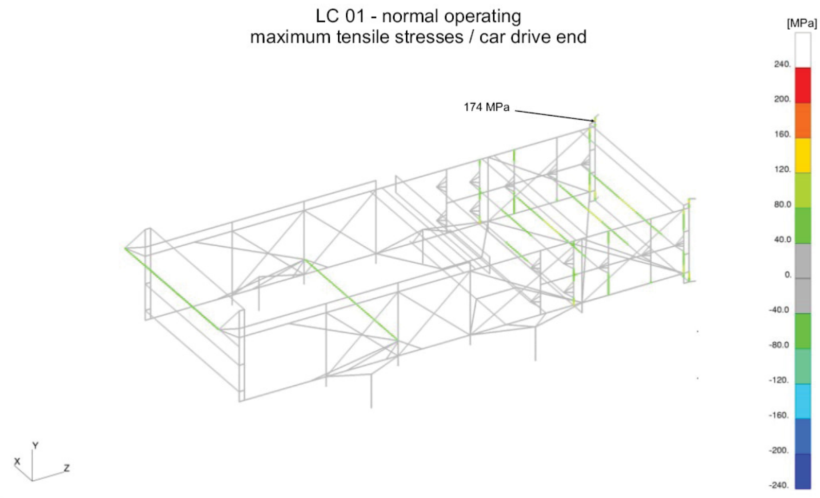
Load case L03 (picture 171):



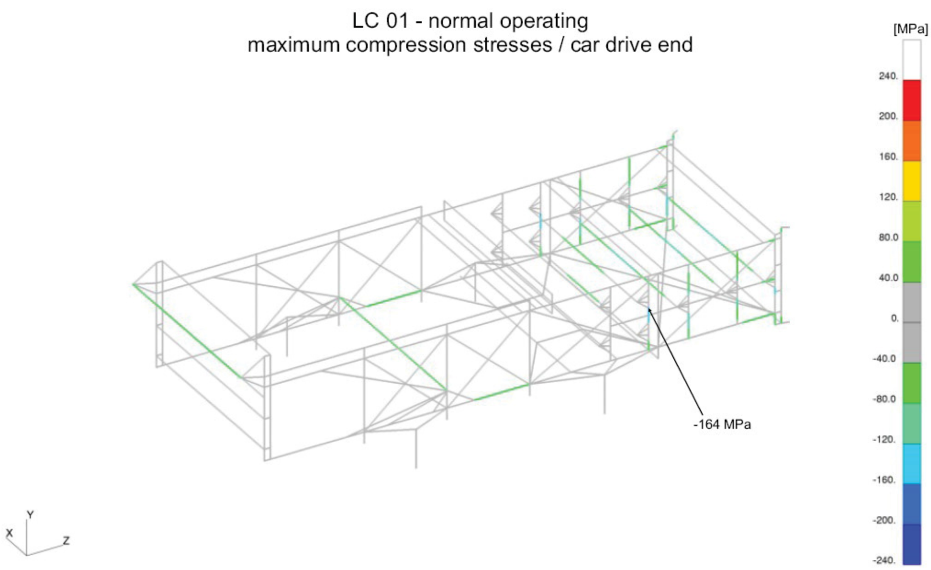
Picture 171: **FEA Load Cases**

Some selected FEA results and the effect they had onto the steel structure are shown in the following excerpt of the full FEA calculation:

Drive/Discharge Car 7 (picture 172 and 173):



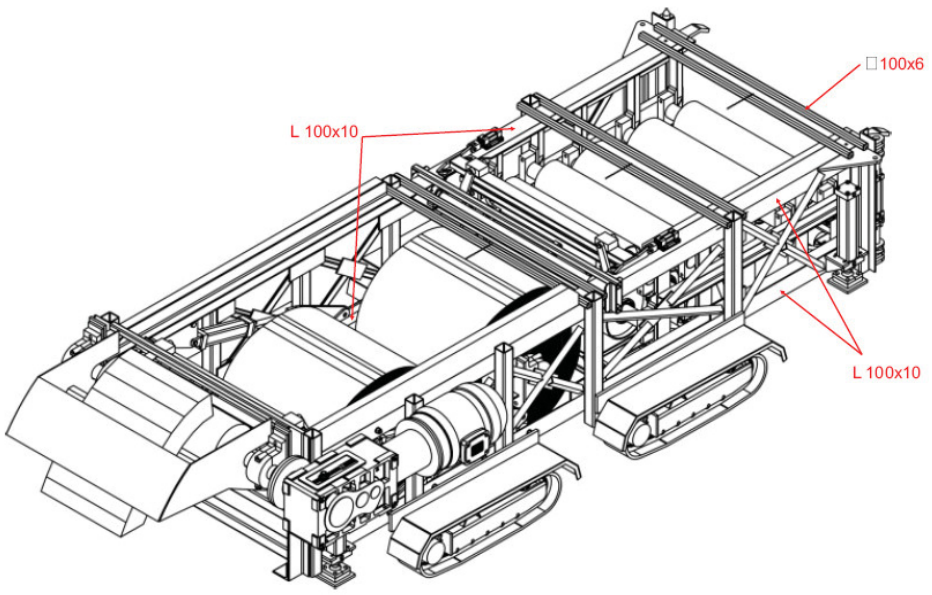
Picture 172: **FEA Results**



Picture 173: **FEA Results**

Based on the FEA results showing too high stresses in the above indicated areas, the steel profiles used along the drive car got changes from 80mm to 100mm and cross beams had to be inserted to stiffen up the structure (picture 174).

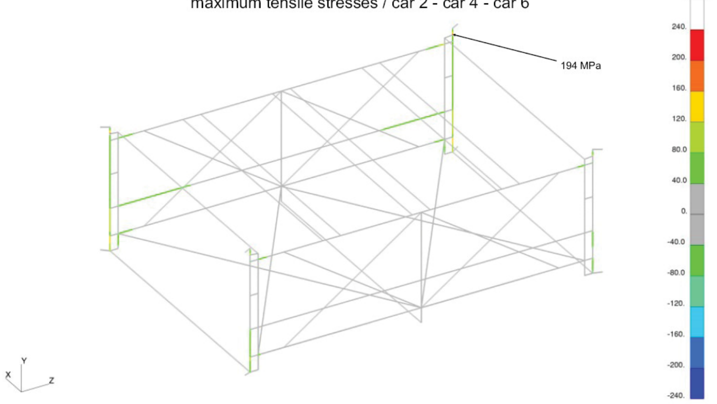
Proposals / car drive end



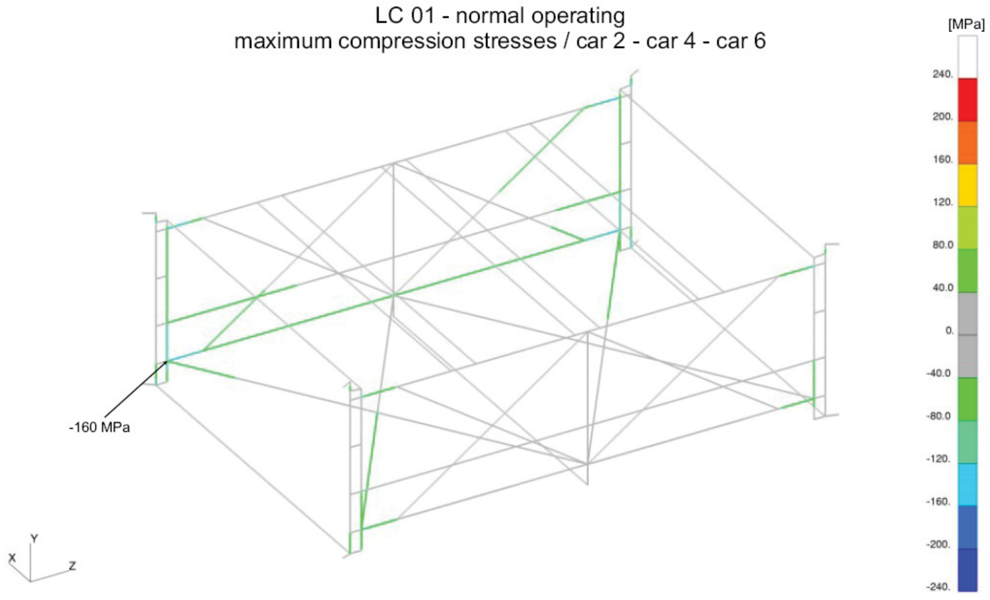
Picture 174: **FEA Results and Corrections**

Cars 2, 4 and 6 (picture 175, 176 and 177):

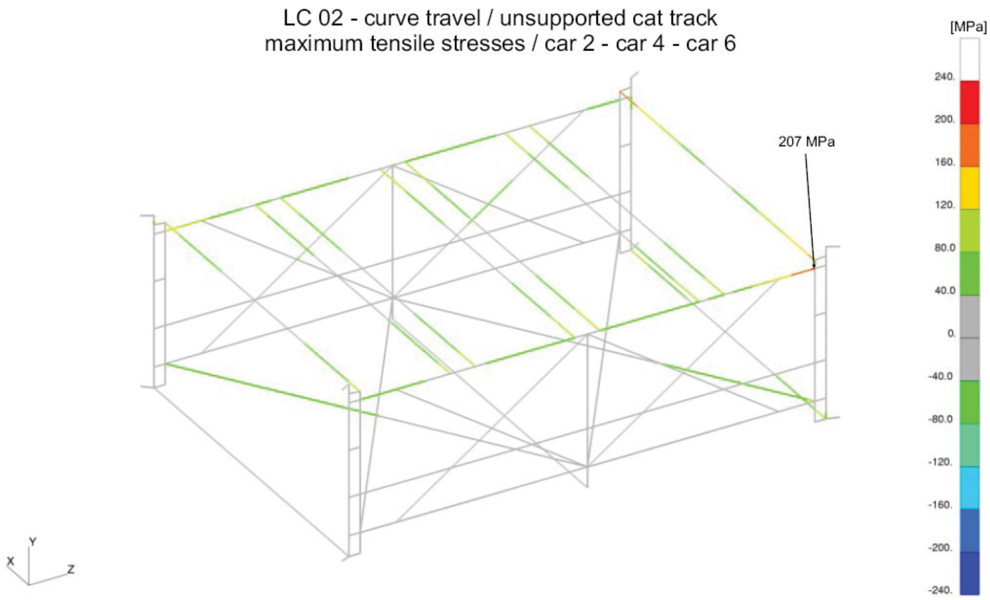
LC 01 - normal operating
maximum tensile stresses / car 2 - car 4 - car 6



Picture 175: **FEA Results**

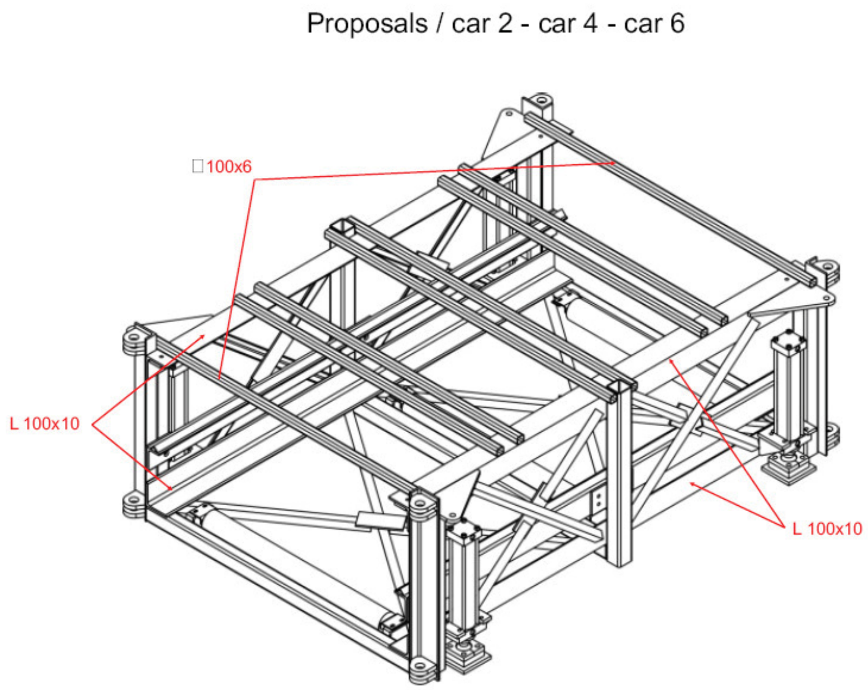


Picture 176: **FEA Results**



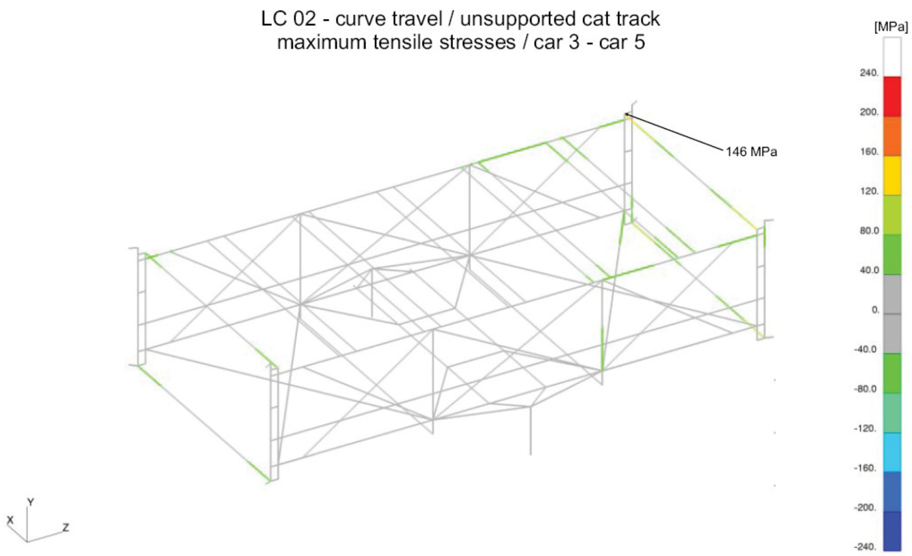
Picture 177: **FEA Results**

Also those cars showed the same result. Under the influence of the different load case the stresses would be outside the safe operating conditions and some beams had to be up graded to the next size (picture 178).

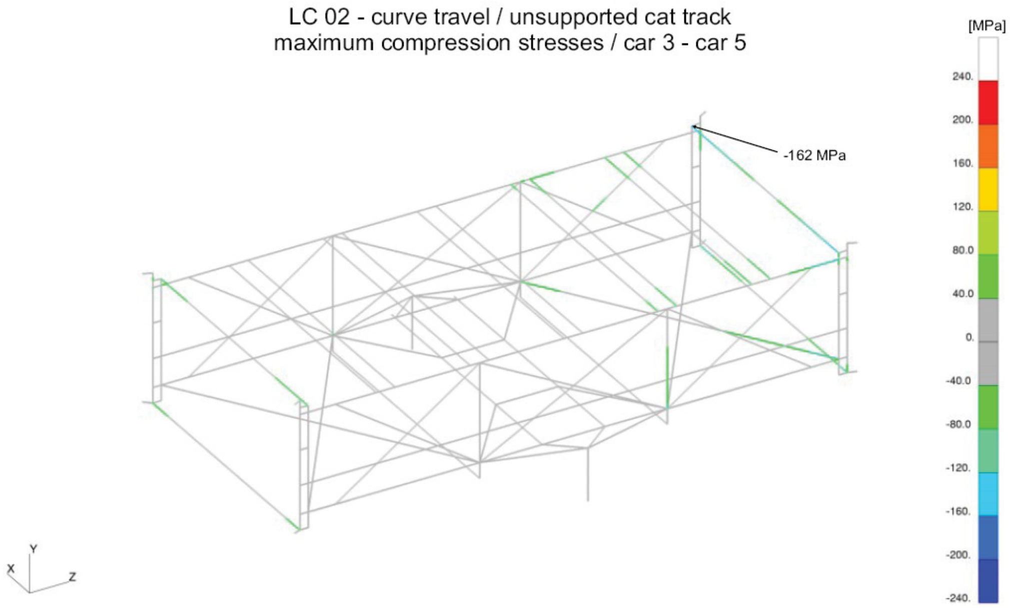


Picture 178: **FEA Results and Corrections**

Cars 3 and 5 (picture 179 and 180):

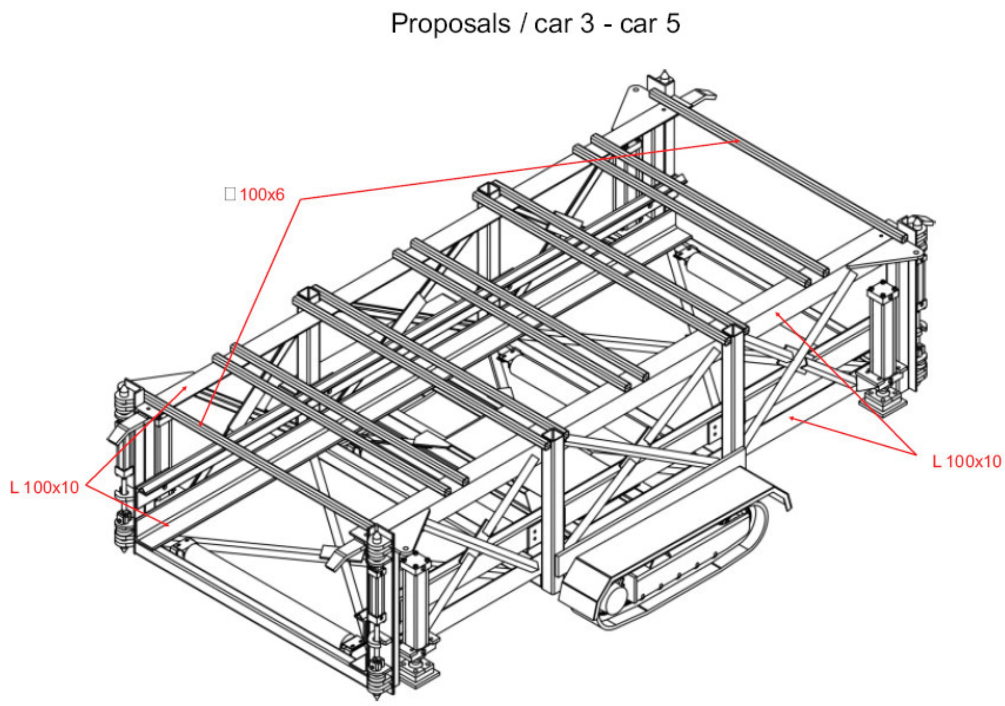


Picture 179: **FEA Results**



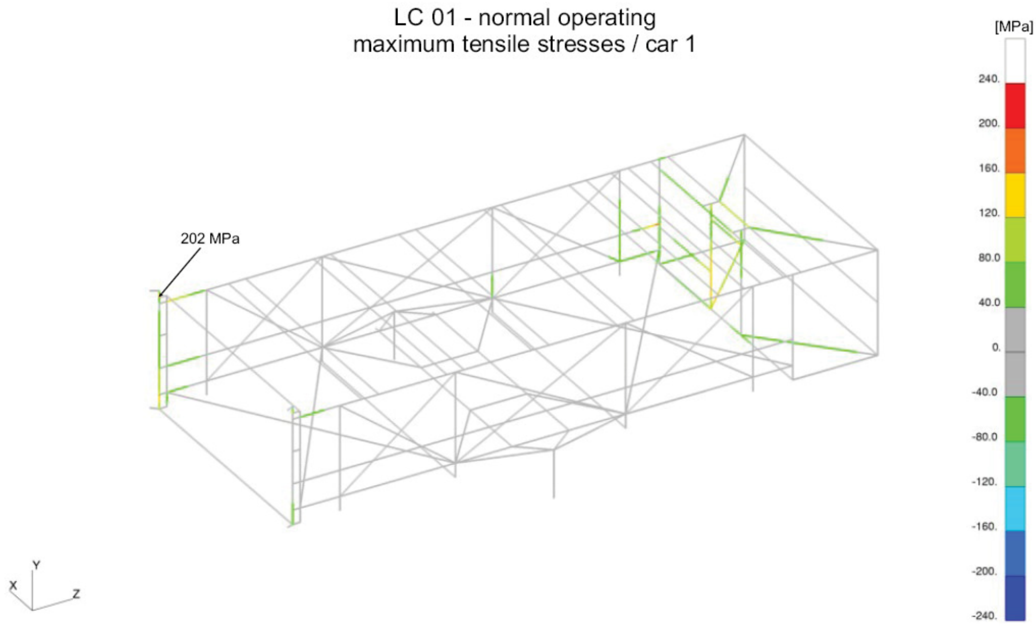
Picture 180: **FEA Results**

Same situation got presented at those cars (picture 181).

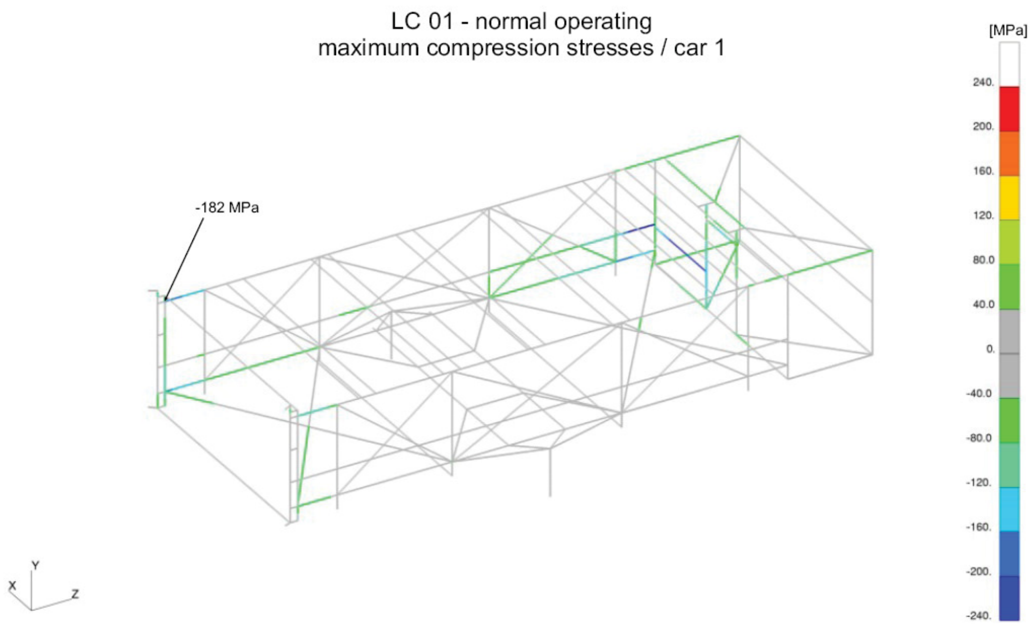


Picture 181: **FEA Results and Corrections**

Car 1 (picture 182 and 183):



Picture 182: **FEA Results**



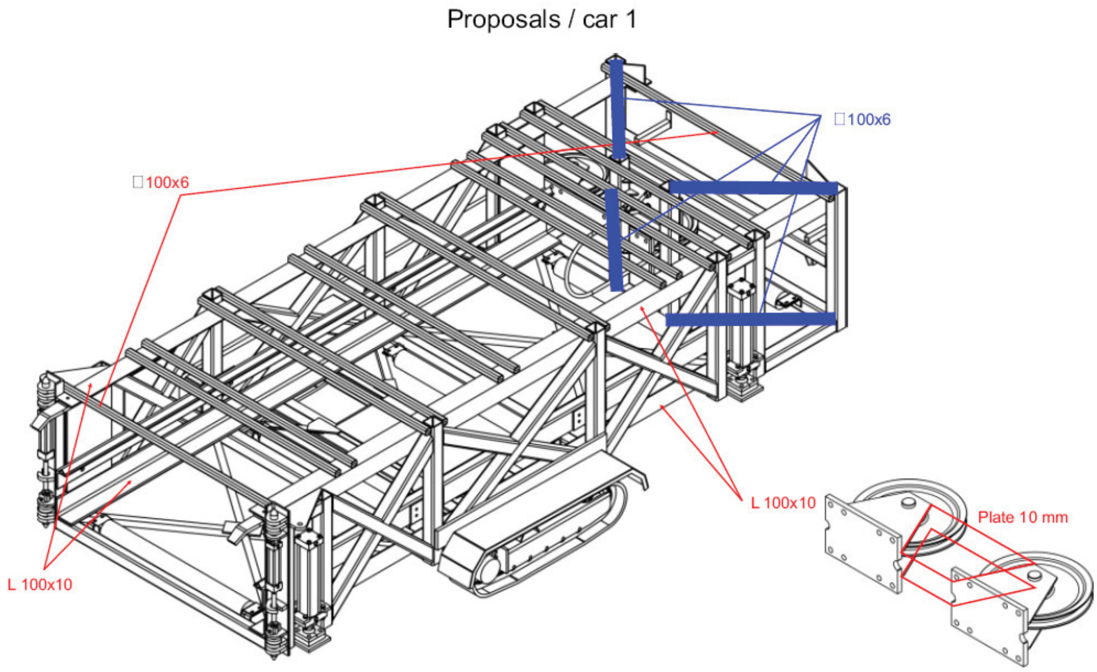
Picture 183: **FEA Results**

Car 1 showed a different picture.
The tensioning winch is introducing the full steel wire rope force into this car via the sheave system.

Due to the fact that the rope force is changing the direction twice around the front end of the car and the rope is also attached to this car the whole frame had to be strengthened (picture 184).

Braces taking the load of the rope force had to be implemented into the steel structure as well as the beams along the car had to be sized up as seen for the other cars of the LTU unit.

The holding plate of the sheaves themselves had to be enlarged and the attachment to the car redesigned.



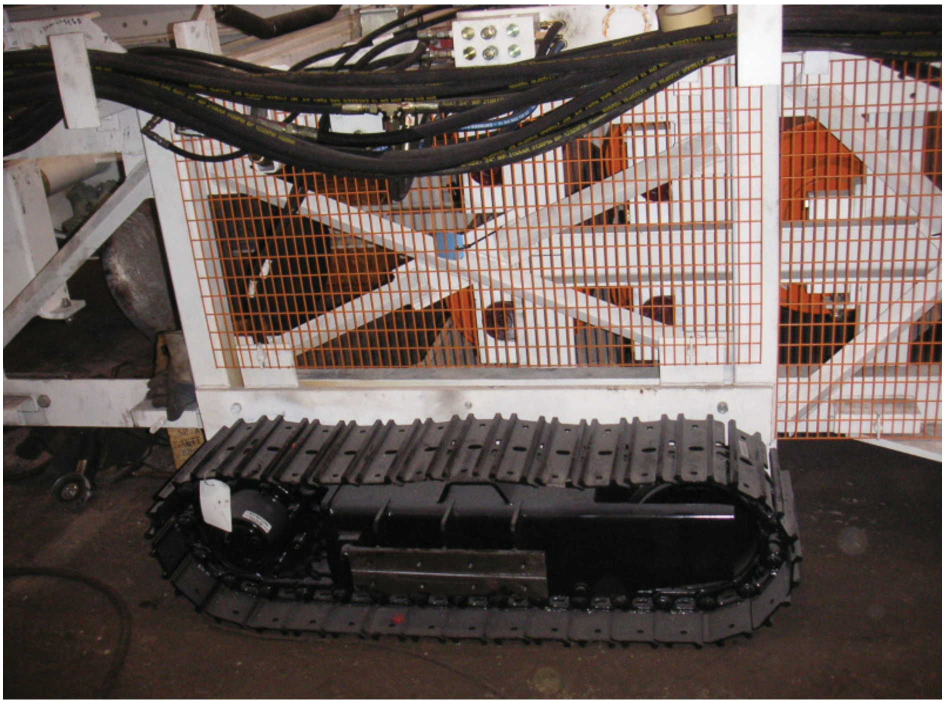
Picture 184: **FEA Results and Corrections**

All proposed changes to the steel structure shown in above pictures had been realized during the manufacturing of the system.

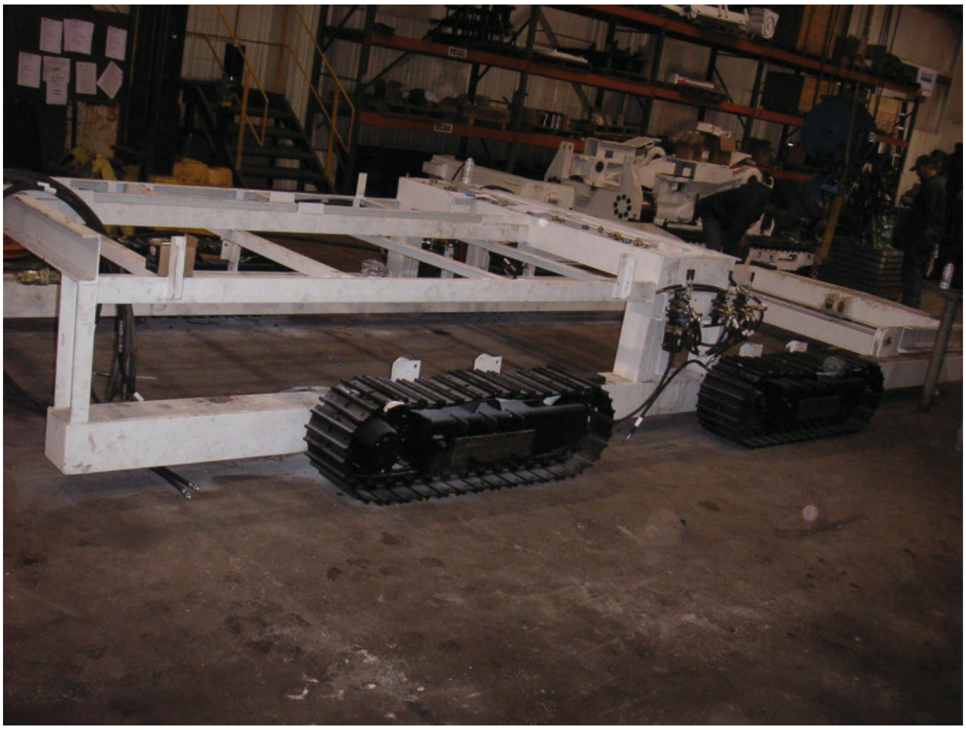
It has also to be mentioned that all hinge pins and clevises had to be enlarged in size to get the necessary strength for a situation where one side of the hinges is open and the other one is closed as well as this specific car is suspended over a dip.
The load getting applied to this hinge had to be overcome by upgrading the material to hardened and tempered 25CrMo4 steel with a tensile strength of around 1000N/mm^2.

All other connections and structures were strong enough for the applied load cases.

The overall engineering was finalized by the author at this point, all engineering data handed over to an in-house design department to create the manufacturing drawings and the mining system fabricated at a South African steel manufacturer based on them.



Picture 185: **Steel Structure of Car 7 including Protection Guards**



Picture 186: **Hopper Car Frame with Tracks**

The finished and assembled components of the machines had been put together to a complete and functioning system and a comprehensive workshop or surface test was started to make sure that all systems are functioning the anticipated way (also see pictures 185 and 186).


Picture 187: **View over top of the LTU unit**


Picture 188: **View into LTU unit with Return Rollers**

The picture 185, 186, 187 and 188 are shown to give an impression of the fabricated and assembled steel structures ready for the surface test.

The workshop and surface test was supposed to comprise tests for all functions and operational steps and sequences the mining system will have to perform when it is mining the left over hard coal deposit we have on hand.
Before not all those functions work satisfactory the system will not be installed in an underground environment as all work in the restrictions of the very low height seam will be enormous complicated compared to a surface area.

7. Surface Test

The reason for the surface test is to have all systems tested and accepted by the author and also some representatives of the underground mining company before the mining system goes underground.
The differences between correcting design and engineering problems in the surface workshop compared to an underground workshop in a coal mine are immense.

Also repairs to the system due to unforeseen circumstances are much easier done when a workshop with overhead cranes, welding machines, oxy set, etc, is available without any height and weight limitations.

Future operators of the system sent from the coal mining company as well as the author and Sandvik employees responsible for the development of the mining system will be assembled for the test team.
All conceivable and realistic test sequences and circumstances will be simulated to have problems surfaced as early as possible and before the mining system starts the production work underground.

When the mining system will be started to extract the left over hard coal deposit all involved people want to be sure that they have done everything possible to sort out the faults in the system.

7.1. Test location

The test location was decided to be the workshop area of the SANDVIK Mining and Construction division in Brier Hill near Pittsburgh, Pennsylvania, United States of America.

The reason for choosing this location was the fact that an R&D project always needs the proximity to a workshop where certain necessary corrections to the system can be done with not too many hazels

The location is also easy accessible by everybody invited for the tests and the area can be closed to the other people than the actual people from the mining company and the internal people working on the project.

150m of straight distance with a width of minimum 5m is required to perform the tests. This “test road” will be prepared in the backyard of the workshop at the Brier Hill company.

7.2. Simulation of underground conditions

The tests set out by the author together with the mining company will be done in six independent sequences:

- **<u>Pull out and retrieve sequence of the conveyor structure (SEQUENCE 1)</u>**: The ABM10 will be connected to the hopper car with all conveyor structure loaded into the storage area. The gradient structure will be connected to the LTU/drive unit. The hopper car and the ABM10 will advance forward and the gradient structure will be pulled out of the telescopic part of the LTU unit. Then the first conveyor structure will be connected to the gradient structure with the two stringers. The hopper car will be driven forward carefully to simulate the pull-out sequence. This will be done with about 10 conveyor frames and system will be checked and watched during every move. When finished the whole sequence will be performed backwards retrieving the conveyor frames back into the storage area of the hopper car. Before starting the sequence test the hydraulic system of the hopper car will be checked and the data recorded. This can be done after the hopper car is connected to the ABM10 hydraulic system.

- **<u>Conveyor Belt Pull in sequence (SEQUENCE 2)</u>**: 2 x 120m and 1 x 60m of conveyor belt will be pulled into the system and the previous test will be repeated. The tension winch will be deactivated at that time and no drive will be running.

- **<u>Test sequence of all electric and hydraulic functions (SEQUENCE 3)</u>**: All hydraulic and electrical functions will be tested and the pressures and currents set and recorded for reference reasons in the future.

- **<u>Conveyor Drive and LTU unit test (SEQUENCE 4)</u>**: The conveyor belt will be tensioned to the full tension force of 15kN with the winch and the conveyor drive motor will be tested. The conveyor belt will be running and the first test sequence will be repeated with the belt running and the tension winch working.

- **<u>Belt feed in and take out sequence (SEQUENCE 5)</u>**: One belt roll will be fed into the system and the sequence and all involved devices checked for functioning.

- **<u>Relocation sequence (SEQUENCE 6)</u>**: One full turn around 120deg with a radius of 6000mm will be driven through and after the turn driven further into a straight line for at least 10m beyond the distance when the full system had cleared the corner. The turn will than be driven in return to simulate the reverse ‘parking in’ sequence.

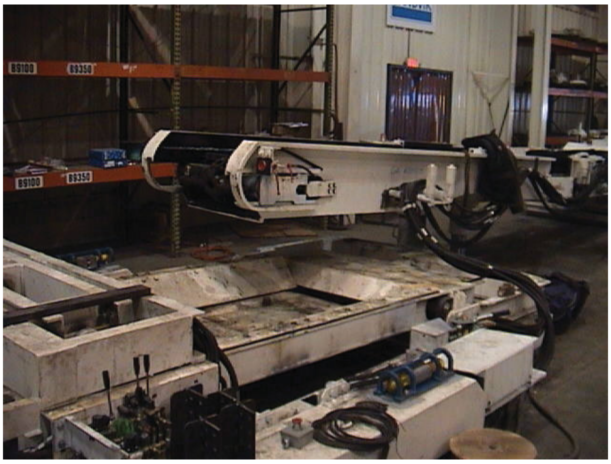
During the relocation sequence side walls with an inner distance of 5200mm will be installed to proof the underground suitability.

The height of 1800mm in the LTU/drive unit and the 1200mm in the mining tunnel will only be checked now and then as it would make the monitoring process of all components to difficult and the risk of damaging components would also be high.

Further to the tests a rough mining ground surface condition will be simulated to see the reaction of the system to those, a compared to underground coal mine more realistic, conditions.

7.3. Mining operation of the system

First step was to connect the two machines, the ABM10 and the hopper car together hydraulically (picture 189 and 190).



Picture 189: **Hydraulic Connections HC and ABM10**

Picture 190: **Hydraulic Connections HC and ABM10**

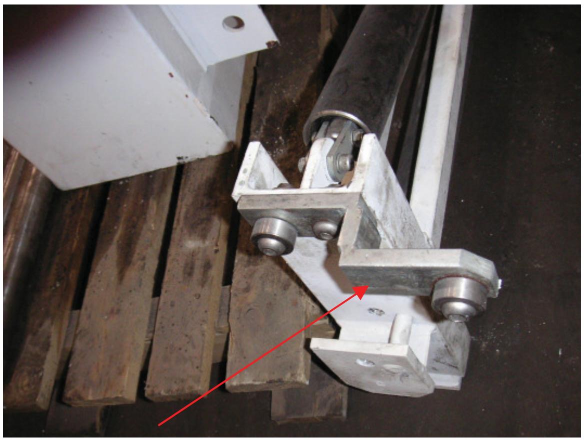
The ABM10 and hopper car got then brought outside the workshop to have more room to move for the testing. The ABM10 and the hopper car got connected up and the first test started (picture 191 and 192).

Picture 191: **Testing Area outside workshop**

Picture 192: **HC and ABM10**

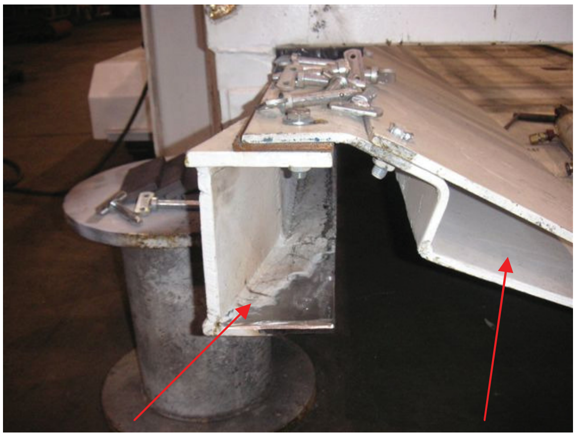
SEQUENCE 1 revealed the first problems (pictures 193 and 194):

Problem 1: During the pull-out and push-in action of the conveyor frames it was found that the structure did not run properly in the guide rails. The structures got jammed up inside the hopper car.



Original Roller design

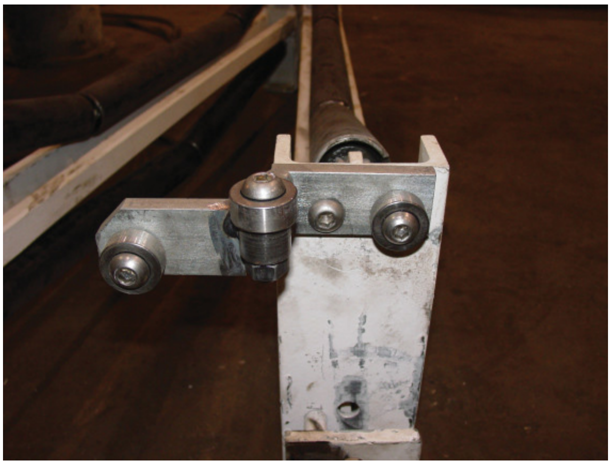
Picture 193: **Guide Roller design**



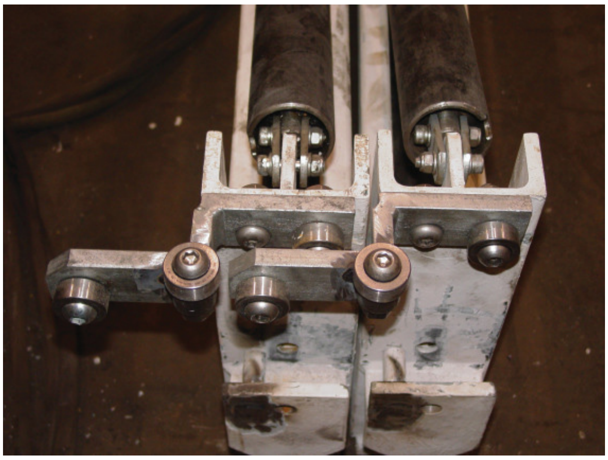
LHS Guide Rail in Hopper Car Steel Trough

Picture 194: **Guide Rail design**

The little brackets with the off-set rollers did not guide the frames in a straight line and therefore the frames jammed up inside the guide rail.
Close monitoring of the rolling behavior of the structures inside the rail suggested the need of a support to the side of the rails to avoid a skewing motion of the frames. An additional side guide roller was designed and mounted by the author and the trail repeated (pictures 195 and 196).



Picture 195: **New Guide Roller design**



Picture 196: **New Guide Roller design**

The result was not satisfying at all due to the fact that an anticipated exact side guidance of the frames was not given even though the additional rollers where offset the centre line of the frames and the frames started to skew in horizontal direction till they jammed up again.

The author decided to change the system completely by starting from scratch. Again the rolling action of the frames in the inside of the guide rails was studied closely and compared to other similar kind of working systems in other technical disciplines.
A system was found in and adopted by the author from the design of garage roller doors (picture 197).

Picture 197: **Single Roller design**

Only one single roller per side was used and in the shape illustrated in the picture.

The side walls of the rollers will be the protection against the riding up onto the guide rails, whilst the long rolling cylinder gives enough surface area to stand on and to keep the roller running in a straight line. Both rollers (one each side) were given plenty of side clearance at the axle to move freely in between the guide rails.

The diameter of the rolling cylindrical part was designed in a way so that the carrying idler set was running higher and almost touching the underside of the steel trough in the hopper car.
In the case that the frame wanted to shift or twist horizontally on the guide rails the higher outer idler of the three troughed carry idlers will get in contact with the underside of the steel trough bringing/guiding/rolling it back onto the normal track in the guide rails.
The conveyor belt itself when running over the return rollers (carrying side of the conveyor belt will run on top of the steel trough!) will assist in guiding the frames as well.
The shifting, skewing and jamming of the conveyor frames inside the guide rails of the hopper car got eliminated with this much simpler design.

Problem 2: As soon as the hopper car was moved forward via the hydraulic track system the next problem surfaced.

Although the ground clearance of the lifting frame with the conveyor frames stored inside could be increased with the lifting cylinders to give enough room between the floor surface and the frames, the main frame of the hopper car with the track frames attached to them was hitting ground when running over uneven floor. The big length of the main frame was too long to manage uneven floor conditions (pictures 198 and 199) without hitting the ground at the ends.

Picture 198: **Floor Contact of the Mainframe**

Picture 199: **Floor Contact of the Mainframe**

To achieve the necessary ground clearance of the main frame of the hopper car, the track frames got lifted by 100mm by attaching the mounting flanges higher on the steel frame. Checking the influence of such a modification onto the overall system showed no effect, the modification was therefore kept. The final ground clearance can be seen in the next picture (picture 200).

Picture 200: **Mainframe lifted 100mm via Track Frames**

Problem 3: The difficulties encountered during the previous test the hydraulic system around the track drives showed that the two track sets each side of the hopper car are never running in the same speed.
Close investigation of the system showed that the track with the lower resistance were running quicker than the others and the hopper car was hardly moving in a straight line. It became obvious that the hydraulic fluid was always going to the tracks with the lowest resistance whilst the ones with the higher residence and torque requirement were stalling at the same time.

This was causing the hopper car to run off centre and was pulling to one side only.

The author decided that it was necessary to install Flow Divider Valves in front of each hydraulic track motor to make sure that all tracks are turning with the same speed at all times and in the situation of differing resistances on both sides of the hopper car.

The installed flow divider valves made sure that the hydraulic oil was divided in the same quantity over the two track motor sets on both sides and making them turning with the same speed all time and therefore controllable.

SEQUENCE 1 could be finalized without any further problems.

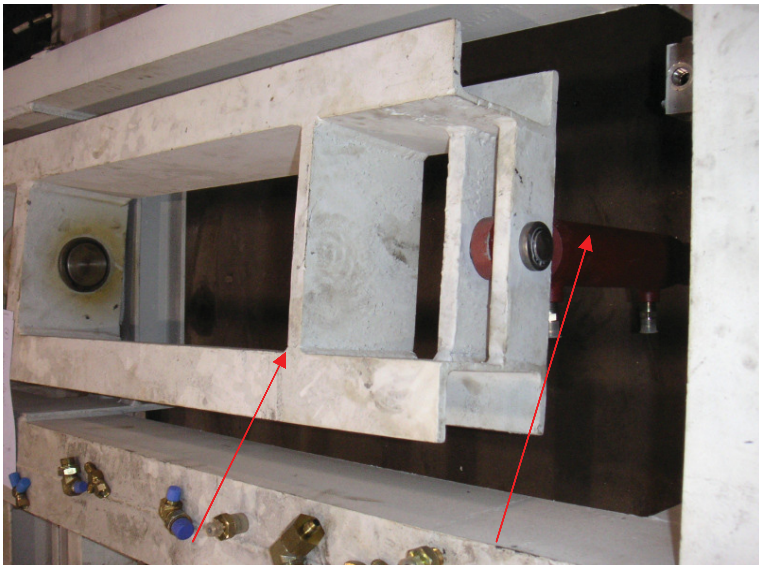
SEQUENCE 2 was started and also here some modification had to be done to the system.

Problem 1: The sequence showed that when starting to tension the installed conveyor belt with the hopper car by advancing slowly forward to take out the slack (no winch action yet) the steel trough in the hopper car was reacting in the anticipated way. The level control devices worked in the way the author designed them for.

However, when monitoring the process and specifically the reaction movement of the pendulum frame to the tensioning of the belt by its weight only, one thing became obvious right away.
The lifting cylinder set in the pendulum frame was definitely on the weak side and showed that it might not be able to stand the rough operating conditions underground for a long period of time (picture 202).
The calculations of the forces and stresses around those cylinders got repeated by the author by taking higher load cases including impact forces between the individual frames in the hopper car as well as temporary over-tensioning of the conveyor belt into consideration.

The calculation and a slightly higher safety factor made the author decide that the two lifting cylinders with 35mm diameter cylinder rods had to be upgraded to cylinders with 50mm diameter rods, the clevises reworked and the new cylinders installed (picture 201).

Picture 201 (cylinder new): **Modified Cylinder and Clevis**



Pendulum Frame Lifting Cylinder (old)

Picture 202: **Original Cylinder design**

Also the vertical rocking movement of the pendulum frame via its spherical bearing in the middle of the frame had to be limited to make sure that one side of the steel trough will not get slammed into the ground under any condition. This was also helpful during the installation of the conveyor belt into the hopper car as the pendulum frame was easier to set up properly in the situation where the conveyor belt is not tensioned yet.

Problem 2: Some corrections (flame cuts and grinding off sharp edges) to the steel work at several locations of the material handling system had to be done to clear the running belt in extreme positions so it does not get damaged during the mining operation.

Also a hold-down-roller for keeping the conveyor belt down when moving forward had to be installed (picture 203). During the tests with the moving hopper car the author found it necessary to keep the unsteady conveyor belt under control so it will not get damaged.
This hold-down-roller was becoming essential when moving forward and backward with the hopper car and the belt tension getting corrected by the tensioning winch. The conveyor belt started to move up and down and was hitting adjacent steel parts and might have been damaged under certain circumstances.

The tricky issue was to determine the right installation height of this roller so the coal which was loaded onto the belt would not be cleared off when passing the roller. The average height of the coal getting loaded onto the conveyor belt was calculated as well as the height of the steel parts the belt might collide with. Taking both measurements into consideration the installation height of the roller was set.

No further problem occurred during this sequence.

Hold down roller

Picture 203: **Hold Down Roller for Conveyor Belt**

SEQUENCE 3 was the next step to set all hydraulic pressures and hydraulic oil flows of the on board hydraulic system.
Every hydraulic function of the machine was tested and the pressures and flows of the idling as well as the loaded system recorded for future references and also for easy fault finding in case of malfunctions.
Some oil leaks had to be corrected.

SEQUENCE 4 was the first test where real operational situations had to be tested.
The tensioning winch was activated and the conveyor belt was tensioned to its maximum tension of 15kN.
The conveyor drive motor was started in a starting sequence beginning with only a very slow belt speed of around 0,5m/sec and stepped up to the 2,0m/sec via the frequency controlled drive. During the initial start the conveyor belt was closely monitored and corrective action taken to track the belt in the centre of the system by adjusting the position and orientation of the drive, discharge and return pulley. Those pulleys were designed adjustable via spindles and nuts so that small running directional correction can be done.

Also the LTU unit and the reaction of the running belt inside the unit was monitored closely for run off and belt rubbing on steel frames and other components.
The conveyor belt separator carriage was checked for functioning in the necessary way of always staying in the centre between the moving and the stationary LTU pulley set.
Also the alignment of the rails where the moving LTU carriage runs on was checked for steps and smooth running between the individual cars.

By moving the hopper car forward and reverse the reaction of the electronic winch got tested, set and adjusted accordingly. A smooth operation with not too many abrupt movements had to be achieved.
Every torque setting for each conceivable operation had been set and tested with the components running. The electronic winch reacted in the required time frame and in the anticipated manner.

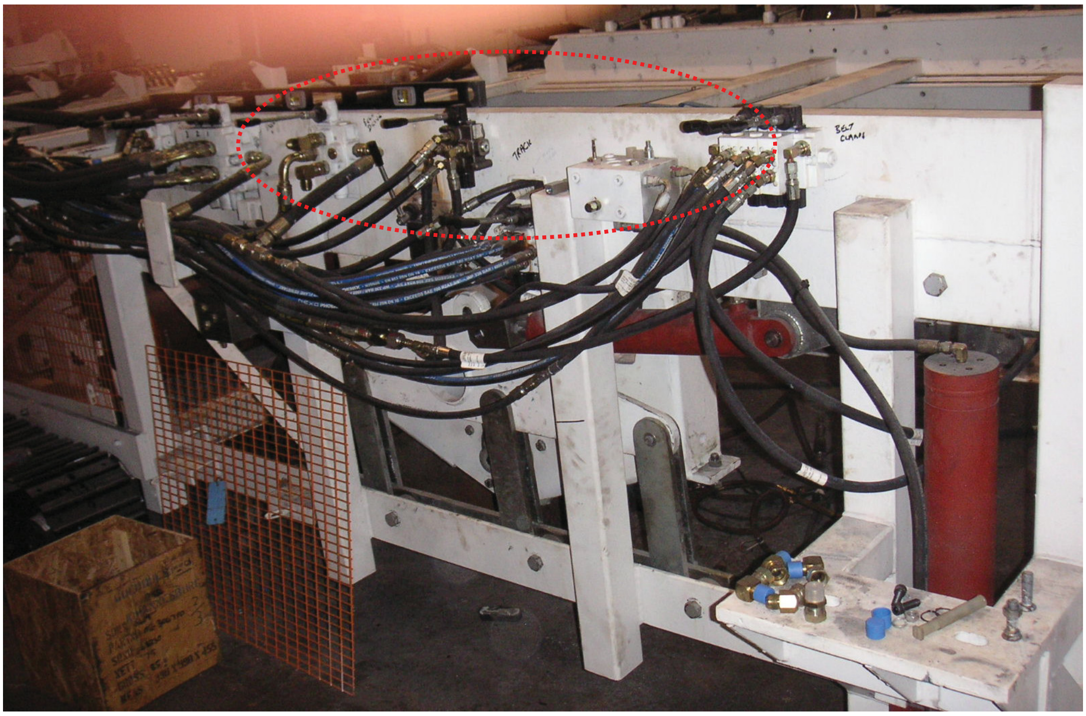
The drive gearbox was checked for running noise and the oil temperature got recorded over the time. Everything was found to be normal.

All idlers on the carrying side as well as the return side were checked for contact to the conveyor belt and also the running of the individual idlers was checked.
Clearances of the running belt and areas of close contact to sharp edges or other components were monitored. Also the leveling out of the LTU unit via the leveling cylinders and the spirit level got tested and found operationally acceptable.

The advancing/retrieving SEQUENCE 1 got repeated together with the ABM10 with no problems encountered.

SEQUENCE 5 was started right afterwards. Before starting to feed in a new conveyor belt length from one of the storage rolls some problems became obvious.

Problem 1: There was no reasonable access to the belt storage rolls (picture 204).



Picture 204: **Conveyor Belt Storage Area**

To be able to insert a new belt length into the LTU unit it is necessary to manually take the spindle out of the mechanical conveyor belt joint. At the time the spindle has to be taken out the mechanical belt joint will sit in between the belt clamps. The area where this has to happen sits right behind the shown steel structure.

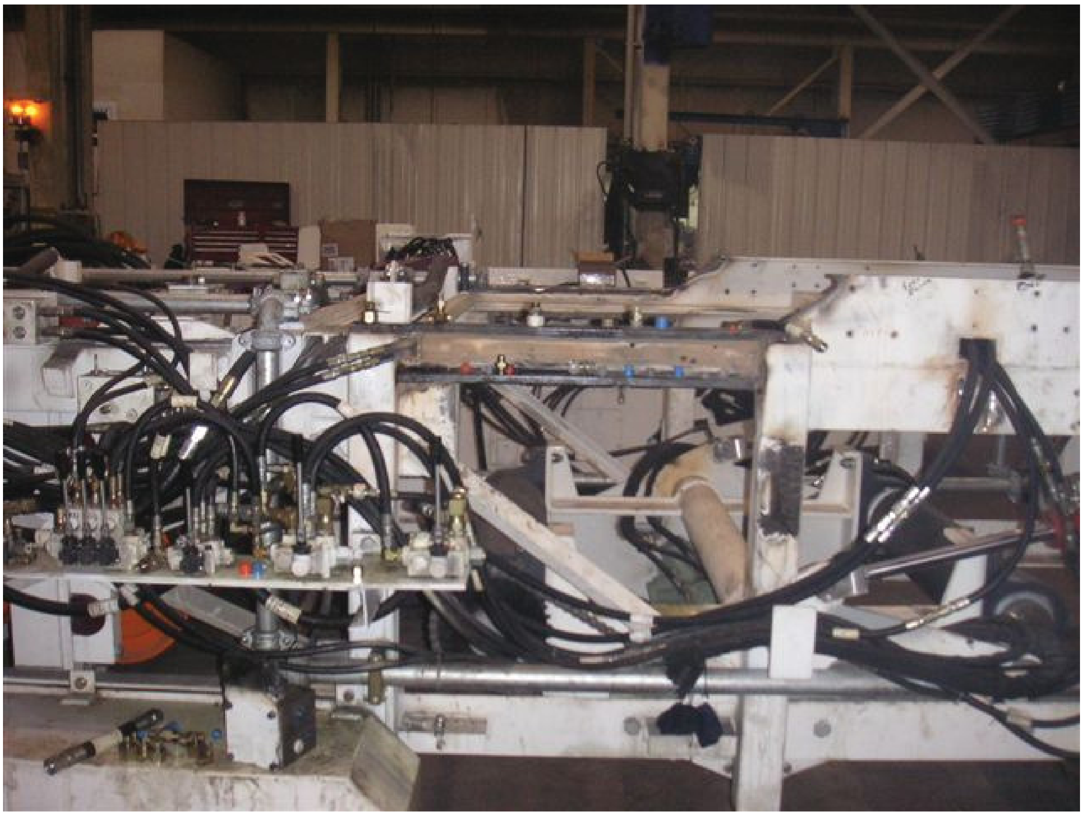
The whole highlighted area in the picture 204 had to be cleaned up to give access to the behind sitting belt storage rolls.

This was achieved by re-arranging all the hydraulic valves to a different place as well as cutting the top part of the steel structure off.
When cutting the steel structure great care has to be given the structural integrity of the steel frame of this car. All changes and cuts had to be cross checked with the FEA calculation so that the frames will not be overstressed. After recalculating the steel frames with different access areas cut out of the carriage frame the author decided on cutting out the top beams of the frames only to gain the necessary access area.

The following picture shows the rearranged area around the belt reelers and belt clamps (picture 205, 206 and 207).

Picture 205: **Carriage Frame modifications**

Picture 206: **Carriage Frame modifications**



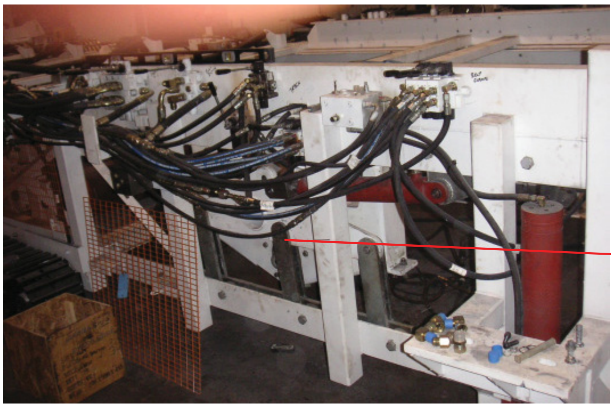
Picture 207: **Carriage Frame modifications with re-arranged hydraulic components**

Problem 2: After gaining proper access to the area between the conveyor belt clamps the next difficulty became obvious. To take out the spindle of the joint was rather easy to put it back in impossible without a flat surface where the belt ends can be put onto during the inserting of the spindle into the two conveyor-joint-ends.

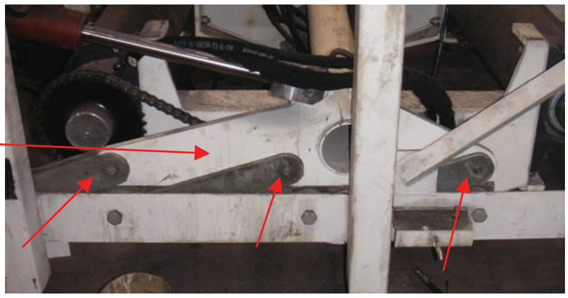
A sort of “table” was necessary to put in when joining the two belt ends together and the spindle was getting pushed in.
Performed investigation showed that there was practically no space inside the car to mount a table like this. Therefore the author and the mine operators decided that an independent plate was stored on the side of the car which could be placed in position by the two people inserting the belt.

Problem 3: After lifting and lowering the two belt rolls a couple of times another change had to be done.

During the lifting the securing rings and washers which were holding the pins of the belt roll lifting device in position started to break and the pins were working their way out of the joints (picture 208 and 209).



Picture 208: **Modified Lifting Device**



Picture 209: **Modified Lifting Device**

All pins in the lifting frame had to be taken out and a heavy duty securing device had to be designed by the author. At the end it was decided that a disk will be welded onto each side of each pin. The pivoting joins were permanently in position now. For disassembly of the lifting frame the steel disks would have to be ground off first. As we do not anticipate of damaging the device on a regular basis this solution was acceptable.

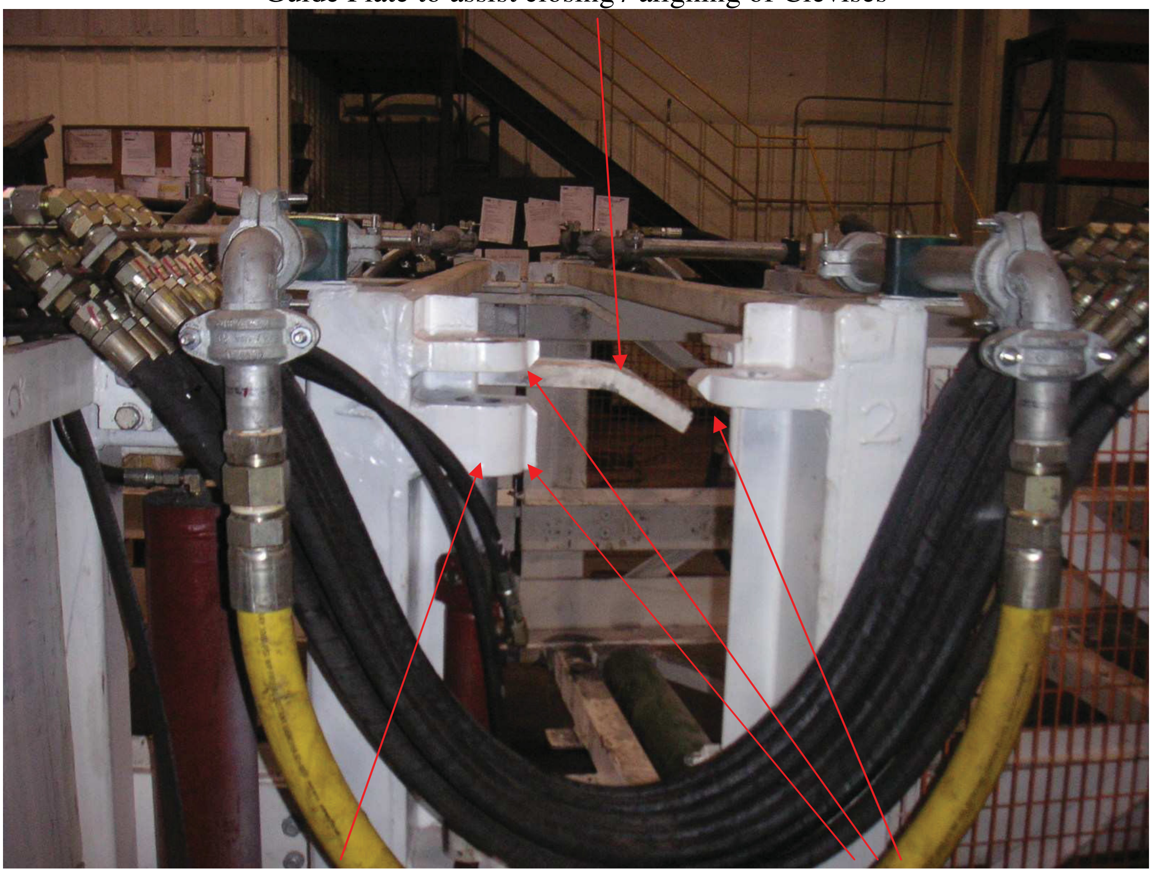
7.4. Relocation operation of the system

The relocation of the mining system is beside the mining operation the second essential part of the overall operation of the system.
Although the mining operation got simulated in its main steps and found acceptable for the start of the underground test, the relocation of the system was considered as important.
As said before only if the each and every operational step of the whole mining system was considered acceptable the machine will go underground to start the test operation. The relocation sequence is part of the operation and also very much responsible for the success of the project.

SEQUENCE 6 was started with the opening of all hydraulically activated hinge pins which are connecting the individual cars. The opening was working all right and also the spreading open of the hinges with the steering cylinders.

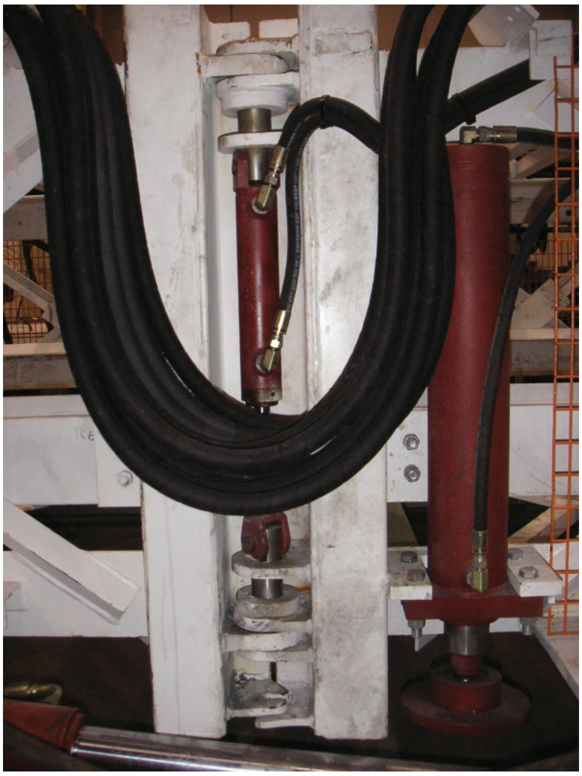
Problem 1: The closing of the hinges and the subsequent engaging of the hinge pins showed the necessity to have dedicated stop blocks inside the hinge clevises to align the clevis-bores properly for inserting the pins.
The thickness of the clevises (picture 211) had to be reinforced at the same time as it became obvious that the closing of the hinges requires much higher force than initially thought. This was due to the fact that the clevises when separated caused the carriages to twist and vertically misalign. The author had to recalculate the forces and stresses and increased the safety factor by two to guarantee a safe operation.
The closing of the hinges was only possible via the guide plates and the chamfer on the clevises together with the steering cylinders.
After one clevis got bent all clevises had to be reinforced in the following manner (picture 210).
The opening and closing action was repeated several times with the author monitoring the hinge parts closely and to examine the components for damage after every sequence.

Guide Plate to assist closing / aligning of Clevises



Reinforced Clevises Stop Blocks

Picture 210: **Reinforced Car Hinges**



Picture 211: **Initial design of Car Hinges**

With the modifications done and the stop blocks in place the hinges were shutting properly and the hinge pins engaged without any problems.

Problem 2: During the traming of the unit around the corner it became also obvious that a permanent communication of the operators was absolutely necessary.
Whilst the one operator encountered a problem and stopped the tracks the others did not know about that and kept traming forward. Collisions with the rib and subsequent damaging of guard plates were the consequence.
Beside communication devices (radios) the author had to engineer a technical protection system.
To connect all hydraulic circuits of the track with each other in a way that for the operation of the tracks one additional manual lever had to be pulled to run the tracks was found acceptable. This additional lever was connected into the hydraulic circuit so that in case one of the three operators was stopping his tracks all the others stopped automatically too.
For moving the tracks all three levers had to be engaged at the same time. This system helped protecting the unit against unwanted movements and therefore collisions.
Also the mentioned radios had been given to the operators so that also verbal communication was possible in case of troubles and also for easy operation.

Problem 3: The relocation of the ABM10 is considered easy and will be managed by the operators quickly. The material handling system is very different. Thorough and intensive training will be necessary to run the system smoothly through the relocation sequence.

Every one of the three operators running the relocation process will be extremely busy.

- The hopper car operator will be responsible for the hopper car as well as the conveyor belt in between the hopper car and the LTU unit. Angle and distance will be important whilst negotiating the hopper car around the corners.
- Two operators will each be responsible for three cars, three hinge sets and two sets of tracks. The negotiation of the LTU/drive unit around the corners will be complicated and will need intensive exercising before starting underground.

The whole relocation process was difficult but working fine after all relevant sequences had been done in the right order. Never the less the sequence would have to be repeated before taking the unit apart for transport.

7.5. Other necessary corrective adjustments

During the operational processes and the testing and simulation of all individual steps which the machine has to go through in the underground mining operation other corrective actions had to be done to the system.

- A fire suppression system had to be installed along the LTU/drive unit.

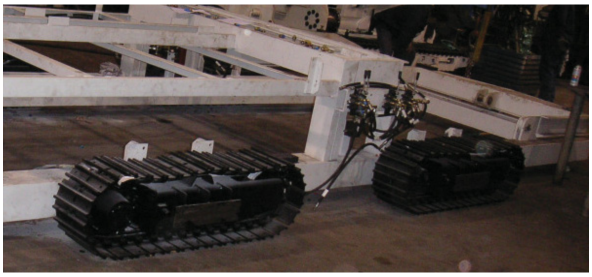
Picture 212: **Guards around moving Components**

Picture 213: **Guard Locks**

- All components mounted along the sides of the cars of the LTU/drive unit had to be protected with guard rails so that they did not get damaged when colliding with the rib during the relocation process. Components such as the hydraulic power pack, the electric box, the winch, hydraulic hoses, cables, etc. had to be protected.

- All tracks and other moving parts such as the internals of the LTU/drive unit got guarded with steel plates and mesh. All guards were designed to be removable with not too much effort (picture 212 and 213).

- All hydraulic valve blocks with the manual DCV attached had been arranged for easy ergonomically operation (picture 214 and 215).



Picture 214: **Original design**

Picture 215: **Rearranged with track guards**

- The hydraulic hose connections to the ABM10 got arranged with the conclusion that this will have to be re-arranged underground as soon as the real mining situation was clear (picture 216).

- All electrical components got also arranged in a way that the weekly inspection process which each undergrounds mining equipment has to go through can be done comfortably. For this the mine operators and the mine electricians had been consulted to find an acceptable arrangement for the people who will operate the unit in the future.

Picture 216: **ABM10 – Hopper Car Hydraulic Connection**

After the actual mining process with the conveyor extension procedure, the hopper car maneuvering and the retracting procedure were found acceptable to the author and the mining people only the relocation sequence of the mining system was not satisfying for anybody yet and had to be tested again from the beginning.

The turning radius of the underground tunnels was simulated with some wooden frames.

The shown height of the wooden frames was set to the 1800mm, the maximum height of the out-by part of the mining tunnels where the loop take up unit will be positioned (pictures 217, 218 and 219).

A turning radius of 5200mm was built and the whole mining system had to be maneuvered through the virtual mining tunnel to proof the operational fitness and suitability. The whole operation and test run was closely watched and monitored by the author and the representatives of the mining company to find and eliminate obvious problem zones.

Picture 217: **Start of the turn**

Picture 218: **Hopper car almost through the turn**

Picture 219: **Loop take up unit at the end of the turn**

At the very last the whole system got loaded up with coal which was brought to the test site by some trucks. The coal was skipped onto the floor right in front of the ABM10 which had to pick up the coal with its loading device and conveyed through the machine over into the hopper car through the conveying system to the discharge end of the Punch Mining System (picture 221 and 222).

Due to the fact that the reach of the ABM10 in relation to the Hopper Car was not satisfactory to the client the discharge conveyor of the ABM10 had to be extended by approximately 2000mm (picture 220).

This had to be done because the client wanted to gain enough overlap between the ABM10 and the hopper car in case the mining machine cuts a cross cut and would have to cut for one or two meter more to finish off the cross cut into the adjacent previous mining tunnel.

The additional 2000mm extension had to be seen as a safety factor for unforeseen situations and for not interrupting the mining process because of this.

Cutting fully through into the previous tunnel is necessary for ventilation purposes, to give the entry hole into the previous tunnel a proper shape and also to load away the coal completely not leaving excessive coal laying in the way of tunnel roadway.

Picture 220: Extended conveyor length of the ABM10

Picture 221: **Coal loaded on conveyor belt of mining system**


Picture 222: **Discharge Action**

Putting the height relations into perspective presents the difficulties the mine operators will have to go through during the actual mining sequences (picture 223).


Picture 223: **Hopper car at mining height with operator**

After the surface test was finalized, all findings, settings, records and experiences gained got summarized for the training of additional people on site of the coal mine.

8. Summarizing Facts and Findings

The permanent pressure from the world economy to increase the hard coal production and to maximize the profitability by reducing cost and by increasing efficiency is forcing the suppliers of mining equipment to develop new and innovative concepts for the underground coal mining industry.

The selling price for one ton of hard coal, thermal/coking coal of Northern Appalachian quality, in January 2007 of around USD 35,00 had a production cost of approximately USD 25,25 per ton of hard coal against. The price for one ton of this hard coal entering the European market was standing around Euro 61,00 including all transport cost and duties paid.

The mining system which was developed by the author and described in this work is following exactly this need of the mining economy and its players.

The developed mining system provides an extremely profitable way of extracting hard coal deposits in difficult geological conditions and even when present in quantities as small as 2.0 Mio tons overall.

After analyzing the world hard coal market, the achievable selling prices had been investigated as well as the cost of producing the hard coal through underground mining operations. Taking an average achievable selling price and deducting the necessary profit a mining house has to make, gives the maximum available cost per ton of coal figure. Deducting all side cost in the likes of transporting cost, import duties, taxes, etc., the initial maximum production cost for one ton of coal was estimated. With this $/t figure in mind the commercial basis for the mining system had been given.

The main objective of the author was to develop a mining system which was able to reduce the costs for the mining process to the absolute minimum by using only as few as possible personnel, using as few as possible auxiliary equipment, reducing the strata control measurements to the bare minimum and to utilize the equipment in the most efficient way.

Intensive and careful market research was performed by the author. It became necessary to visit different underground coal mines in different countries and to speak to the mining people in regards to what the strengths and more importantly, the shortcomings of the presently available machines were. With this information in mind, one specific underground coal mine was selected because it presented a high quality hard coal deposit with the right dimensions and a mine management willing to support the development. The available coal seam was as low as 1200mm, had an available coal quantity of around 2,0 million tons to mine out, an operational underground coal mine connected to it and most important, this deposit was not economical to mine with conventional mining methods.

All conventional mining methods were analyzed by the author to find the pros and contras of the presently available mining equipment on the market. Different coal cutting machines were investigated and compared as well as the material handling systems and the equally important strata control equipment. Keeping the underground coal mining regulations in mind all cost producing factors were taken into consideration and the possible cost saving areas determined.

None of the existing mining systems was able to do the job in the required way. Either the efficiency or the cost perspective failed the target. Even a combination of the machines was not efficient enough. It had to be a cutting machine, which has to set the bolts at the same time with a continuous material handling system switch behind the machine.
A complete new mining concept had to be developed to fulfill all necessary parameters needed to be able to mine out the hard coal deposit in a way to stay below the initially calculated maximum $/t production cost limit.

The author utilized state of the art technology used in various other successful mining application as basis for the development. The fastest Continuous Bolter Miner was picked to do the coal cutting job. The material handling system behind the miner was only available as a rough basis. The available technology had to be taken, re-engineered and adapted as well as complemented with completely new developments for the missing components and afterwards combined together again to the so called “Rapid Coal Production System”.

The ‘Rapid Coal Production System for Low Seam Applications’ invented and developed by the author came out of a Research and Development program of the ‘**SANDVIK Mining and Construction**’ sector of **Sandvik AB** and took around 18 months from the first concept of the author to the product ready for the surface test.

The state-of-the-art and available on the market principles of troughed belt conveyors, loop take up systems, track driven mining machines and conveyor belt splicing stations were combined with a number of innovative own new developments of the author into a completely new and innovative constellation..
Behind the extremely fast (cutting and bolting at the same time) Continuous Bolter Miner the material handling system was to follow on its own.
The miner and the material handling system were combined into a self supporting and completely independent running mining system being able to operate in coal seam heights as low as 1200mm.

The system consists (except of the miner) of a 500mm low hopper car having all necessary conveyor frames on board to mine a distance of over 200m. The track driven hopper car acts as the counter weight to the conveyor belt tension and is responsible for managing the conveyor belt during pull-out and retrieve motions. The level control of the hopper car manages all three dimensions through a new developed pendulum frame and via hydraulic cylinders. The hopper car is designed to pull the conveyor belt out of the loop take up unit against a dynamic operating winch system while the conveyor is operating.
The conveyor structure is only as high as 300mm with 60mm diameter idlers.
The Loop take up unit is designed to an optimized length to store as much as possible belt whilst being as short as possible at the same time. The Loop take up unit is track mounted and split up into 7 individual cars, connected through hinges, to be able to maneuver around corners while the conveyor belt stays inside the LTU unit.

The whole mining system is able to relocate through the underground mining tunnels and around 5200mm corners with the onboard power equipment and no other assistance than the operators. It also extends and retracts via this onboard hydraulic and electrical power pack.
All necessary material to extend a conveyor belt from 70 to 200m are stored onboard the mining system (60 off conveyor structures with stringers, two belt rolls inclusive belt pull in devices)
Only 5 operators are necessary to run the whole mining system.

The result of this R&D program was an extremely compact and very flexible universally useable mining system which deviates substantially from all other mining systems available on the international market.

After building the mining system prototype, a comprehensive surface test was performed to simulate and test all functions and also to run through all possible worst case scenarios.

Mining sequence, conveyor extension, conveyor retraction and relocation sequences got simulated behind the SANDVIK workshop in Brier Hill, Pennsylvania, USA. A mining tunnel height, width, turning radii had been built to produce real underground conditions. All initial ideas and concepts from the author were realized and refined during the engineering process and further developed into the final product during the assembly and the surface testing.

The mining system performed as anticipated and required by the mine operators.

During the surface tests it also became very clear, that, due to its absolutely new concepts, the mining system needs a fair amount of additional training for the operators to get the planned performance out of the system.
The machine, when going into operation underground for the first time, will be accompanied by at least two Service Engineers of the local SANDVIK company. This has to happen to make sure that eventual upcoming problems will be solved immediately before damage to the system occurs.

Two international patents have been registered for the mining system in the name of the author.
One for the mining system as a whole new assembly/operational concept and the second one for the hopper car with the conveyor frame storage including the self adjusting mechanism for the conveyor belt running direction (level control with the pendulum frame).
The initial registration had to be done for the whole system and will be split up after the patent had been accepted and a number allocated to it.

The Austrian first-registration number A 1537/2006, for a patent on the "Punch Mining system" was submitted on the 14.09.2006 as well as the corresponding US patent with the number 11/636132 on the 06.12.2006 (inventors: Manfred Fuchs for the overall mining system and Manfred Fuchs together with his assistant Simon Curry for the pendulum frame system)

Whether the 'Rapid Coal production System for Low Seam Applications' will be a commercial success or not will come out after being implemented in a real mining operation and after running for a reasonable time period.

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