Publication No. 01-144-173

ELECTROMAGNETIC TRANSPORT SYSTEMS FOR THE PHOSPHATE INDUSTRY

Prepared by

Magplane Technology, Inc.

under a grant sponsored by

FIPR
Florida Institute of Phosphate Research

March 2001

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ELECTROMAGNETIC TRANSPORT SYSTEMS FOR THE PHOSPHATE INDUSTRY

FINAL REPORT

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FIPR Contract #97-01-144

March 2001

DISCLAIMER

PERSPECTIVE

Due to the large volume of materials that are handled by the Florida phosphate industry during both the mining and manufacturing operations, transportation costs are a significant part of the cost of doing business. In addition to offering the possibility of reduced transportation costs, Magplane would be more energy-efficient than conventional ground transportation and would reduce roadway traffic congestion.

When this proposal was approved by the Board of Directors it was considered to have a very high potential to develop a new and decidedly improved method of solids transport. At the completion of this project it can be said that it still has a very high potential. The report presents what is probably best described as an overly optimistic evaluation of the program's accomplishments.

While the project accomplishments are significant, a review of the Project Goals as compared to the results achieved is revealing:

- **Scalable--operate as close to full-scale as possible, i.e. a 20 tph rate.** The original requirement was that the system would operate at a 40 mph speed. This speed was never achieved. This fact does not mean the system is unworkable but it does increase the capital cost and adversely affects the economics reported.

- **Able to load and unload cars on the move (desirable but not necessary).** Early on it was decided not to pursue this goal and nothing was done to demonstrate it was possible or practical.

- **Reliability--extended run time.** Not demonstrated.

- **Low-maintenance.** Not demonstrated.

- **No spillage or controlled spillage.** Not adequately demonstrated.

- **Involve the operation of at least two vehicles.** Not attempted. This is perhaps the most critical shortcoming of the project.

- **Modular guideway system that is easily maintained, replaced, or moved.** The question of easy maintenance was not answered but there are indications that this would not be possible with the system used. There is no question that the installation used could not be easily replaced or moved.

The basic conclusion that can be drawn is that much more testing must be done before the system as operated in this program can be considered as a candidate for a commercial application.

G. Michael Lloyd, Jr.
Research Director, Chemical Processing

ABSTRACT

The transport of bulk solids is a major effort, and a major cost, in the phosphate industry. An electromagnetic system that would propel capsules through an underground pipeline has been developed and promises to be a competitive alternative to truck, rail and slurry pipeline transport.

A demonstration project which uses a linear synchronous motor to move capsules has been constructed at IMC-Global, in Lakeland, FL. The demonstration project utilizes 700 feet of 24 inch diameter centrifugally cast fiberglass pipe, and contains a 200 foot long accelerator/decelerator section, a switch demonstration, and load and unload stations. The test vehicle can traverse back and forth at speeds up to 40 MPH. The six foot wheelbase vehicle uses six-wheel assemblies at each end of a rotating hopper, and has a maximum payload capacity of 660 pounds.

Electromagnetic Drives for pipeline systems are intended as a direct replacement for pneumatic drives. Pneumatic capsule pipelines have a long history, including the transport of limestone in a Japanese cement processing plant. However, various practical limits tend to constrain the throughput of pneumatic systems. The use of electromagnetic drives can greatly improve on the constraints and can result in cost effective systems able to compete with truck, rail and other transport systems. Underground pipe transport can also relieve the environmental impact of conventional transport, and result in faster delivery in overcrowded metropolitan regions.

Development of the Magplane electromagnetic capsule pipeline system was initiated by the desire of the Florida Phosphate Industry to find a cost effective way to reduce the environmental impact of conventional transportation of their very large quantities of material. Typical ore applications would use an underground pair of 24 inch diameter pipes for outbound and returning capsules, and would typically carry 10 millions tons per year over a distance of 3 to 30 miles. Preliminary economic studies have shown a satisfactory return on capital and have resulted in a willingness of the phosphate industry to undertake a significant R&D program.

This report gives the performance of the system during the test program carried out to date in which the basic feasibility of the design has been demonstrated. Plans are described for the follow-on test program intended to provide an indication of component lifetimes.

This report also gives results from an economic model that shows the potential for an attractive return on investment for future applications in the phosphate and other ore transport industries.

ACKNOWLEDGEMENTS

The project acknowledges the generous support of the Florida Institute of Phosphate Research, and the IMC-Global Company. We also acknowledge the financial support and technical contributions from Argila Enterprises and MTechnology, Inc.

We would also like to take the opportunity to thank specific individuals who have contributed so much to the project:

- Stephen Fairfax, Eric Taylor and Dexter Beals at MTechnology, Inc for their engineering work on the linear motors and for startup and operation of the system in Lakeland.

- The Engineeering Development Group at IMC-Global, for their contributions to the original concept and for installation in Lakeland. The group includes Viv Asley, Ron Hartung, John Keating, John Whitley and Terry Nelson.

- Bradford Smith at the Massachusetts Institute of Technology for his work on Economic Analysis and the design and costing algorithms for the load and unload stations.

- W. David Carrier III at Argila Enterprises, for his help in managing the activities in Florida.

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EXECUTIVE SUMMARY

The transport of bulk solids is a major effort, and a major cost, in the phosphate industry. An electromagnetic system that can propel capsules through an underground pipeline has been developed and economic studies indicate that it promises to be a competitive alternative to truck, rail and slurry pipeline transport. Underground pipe transport can also relieve the environmental impact of conventional transport and result in faster delivery in overcrowded metropolitan regions.

Development of the electromagnetic capsule pipeline system was initiated by the desire of the Florida phosphate industry to find a cost effective way to reduce the environmental impact of conventional transportation of their very large quantities of material. Typical ore applications would use an underground pair of 24-inch diameter pipes for outbound and returning capsules, and would typically carry 10 million tons per year over a distance of 3 to 30 miles. Economic studies have shown a satisfactory return on capital and have resulted in a willingness of the industry to undertake a significant R&D program to demonstrate the basic feasibility of the concept.

Electromagnetic drives for pipeline systems are intended as a direct replacement for pneumatic drives. Pneumatic capsule pipelines have a long history, including 20 years of experience with the transport of limestone in Japan. However, various practical limits tend to constrain the throughput of pneumatic systems and limit their competitiveness. The use of electromagnetic drives can greatly improve on the constraints and can result in cost effective systems able to compete with truck, rail and other transport systems.

A demonstration project was constructed to test the feasibility of the system concept and the various components. Seven hundred feet of 24-inch diameter pipeline were used. The facility is designed to allow the vehicle to be loaded and accelerated to 40 MPH, coast to a stop in climbing a 60 foot elevation hill; re-accelerate to 40 MPH in descending the hill, and decelerate to zero, unload, and then be recycled through the process. An electromagnetic switch demonstration is located between the accelerator and the hill. The six foot wheelbase vehicle uses six-wheel assemblies at each end of a rotating hopper, and has a maximum payload capacity of 660 pounds.

FIELD TESTING

During six weeks of final testing 5300 round trip cycles of the car were run at speeds up to 33 MPH. The total distance traveled by the car was approximately 700 miles, and the total travel time, about 40 hours, for an average speed of about 18 MPH.

A total of thirty-five complete load/unload cycles were run with a payload of 430 pounds. The operation of the system was under the total control of the Programmable Logic Controller (PLC).

The car carrying a maximum payload of 660 pounds was captured in the unload station where the payload was dumped. This demonstrated the ability of the magnetic coils to capture, rotate and dump a payload 30% higher than the nominal 500 pound payload. The hopper filled and dumped cleanly. Prior to dumping, the car was cycled 50 times at 25 MPH with 660 pound maximum payload. There was no appreciable loss of product from aerodynamic effects.

The tests which have been done to date establish the basic feasibility of the design. They have also resulted in several improvements to the design, primarily to increase the robustness of the motor control system against system electrical noise, and to alter the vehicle switching approach.

While no appreciable life testing has been accumulated, it is already possible to anticipate design details that will require improvement, most notably the joints between pipe sections. The fiberglass pipes used in the present demonstration were centrifugally cast against an OD mold, resulting in a close tolerance on the OD, but a significant variation in ID dimension. The wheels tend to shock load the dimensional steps at the joints resulting in early fatigue cracking. Use of pipes built from an ID mandrel would greatly reduce the dimensional variation. There has been little evidence of wheel or pipe surface wear to date. The wheel materials have been chosen to be soft relative to the pipe surface to assure that the wheels, which are replaceable, will wear rather than the pipe surface.

ECONOMIC STUDIES

An economic model has been developed which takes engineering and unit cost inputs and projects capital and operating costs for any prospective system. Major cost capital components include pipeline, vehicles, magnet assemblies, windings and load/unload stations. The elements of operating cost include power, material costs for maintenance, taken as a fixed percentage of capital cost, and labor costs for operating and maintaining the system.

The case studies show that pipeline diameters ranging from 18 to 24 inches and vehicle speeds of 20-40 mph are generally optimum for systems operating in the 3 to 30 mile, 1-10 Mt/y range. Slower speeds are more optimum at short distances where the load/unload station costs are a substantial fraction of the total cost. In nearly all cases, pipeline costs are the largest single component of capital cost, whereas the second-most expensive component depends on the distance and tonnage. Interestingly, the model shows that the incremental cost of adding a few of inches to the pipeline diameter is only a small percent, whereas the line limit capacity can be increased by tens of percentage points. Consequently, for systems where the upper limit capacity of the line is somewhat indefinite when it is time to commit to a pipeline size, it is better to pay the few percentage points in cost up front and have the capacity margin offered by the larger diameter line.

If capital costs are assumed to be annualized over 20 years at 20% and combined with operating costs, total costs range from a low of about \$0.02/ton-mile at 10 Mt/y and 30 mile hauls to about \$0.20/ton-mile at 2 Mt/y and 3 mile hauls.

The electromagnetic transport systems for phosphate rock can be competitive when compared with other methods of transport, for example, long-haul rail contracts at \$0.05/ton-mile, long-haul trucking at \$0.08/ton-mile, and short-haul trucking at 0.35/ton-mile. Against long-haul trucking, after tax rates of return of 45% are possible; against short-haul trucking, 30%; against rail contracts, 15%.

Economic studies have also been done for a typical 8 Mt/year "matrix" (ore, sand and clay mix) transport system. Results indicate that at a distance of 5 miles a rate of return of 45% can be achieved; at 2.5 miles the rate of return reduces to 22%. At a one mile distance, however, where the loading station dominates the cost, it is not possible to provide a positive rate of return. These matrix transport studies assumed that while of different design, the load and unload stations would be similar in cost to those assumed for the dry product.

CONCLUSIONS

Economic studies have shown that a 24-inch pipe diameter is near the optimum scale for applications in the phosphate industry. Tests carried out to date on the 700 foot long, 24-inch diameter pipeline have demonstrated the basic feasibility of the design. The parameters achieved demonstrate that electromagnetic capsule pumps have the potential to significantly reduce the throughput limitations of blower driven capsule pipelines. Economic studies of applications in the phosphate industry indicate that the electromagnetic capsule pipeline systems can be competitive with truck and rail transport.

While the field tests have demonstrated the basic feasibility, a follow-on project will be required before the technology can be made available commercially. One goal of such a project will be to demonstrate that the components can meet the lifetime requirements. A suitable follow-on project could be to replace truck traffic between two near-by processing plants.

INTRODUCTION

The transport of bulk solids is a major effort, and a major cost, in the phosphate industry. It is energy intensive, requires major capital investment and, in many cases has an unwanted environmental impact. Matrix is transported to the beneficiation plant, tailings are transported to disposal areas, clay is transported to the settling areas, and rock product is transported to the chemical plants, to local customers or to the port. The magnitude of solids handling is huge. The volume handled in the mining process alone is roughly equivalent to excavating the Panama Canal every year.

Matrix is transported to the beneficiation plant as a slurry pumped through pipelines several miles long. At 40% solids, more water than matrix must be pumped and the water recycled after the matrix is removed at the screens. The combined cost of electricity, high maintenance costs and frequent pump and guideway replacement as a result of wear makes this one of the most, if not the most, expensive item in the production of phosphate rock.

Underground electromagnetically driven pipelines have the potential to be competitive with conventional transportation methods used in the industry, and at the same time can provide a more environmentally friendly solution. Underground pipe transport can relieve the environmental impact of conventional transport and result in faster delivery in overcrowded metropolitan regions. Increased efficiency, together with the replacement of internal combustion engines with electrical energy, have a favorable impact on air and water quality. Displacement of truck traffic also reduces the wear and tear on roads and highways, reducing maintenance costs and relieving congestion.

Electromagnetic drives for pipeline systems are intended as a direct replacement for pneumatic drives. Pneumatic capsule pipelines have a long history, including the transport of limestone in a Japanese cement processing plant. However, various practical limits tend to constrain the throughput of pneumatic systems. The use of electromagnetic drives can greatly improve on the constraints and can result in cost effective systems able to compete with truck, rail and other transport systems. Underground pipe transport can also relieve the environmental impact of conventional transport, and result in faster delivery in overcrowded metropolitan regions.

A pneumatically driven capsule pipeline system has been in commercial use in Japan since 1983 (Kosugi and others 1992). It is used to transport limestone at a rate of 2 million metric tons per year between the Karasawa mine and the Sumitomo Cement plant, a distance of 3200 meters. Prior to construction, transport had been by rail. The parameters of the pipeline system are given in Table 1.

Table 1. Sumitomo Pneumatic System.

Material transported	Limestone
Pipe Diameter	1.0 meter
Transport Distance	3200 meters
Annual Throughput Capacity	2 million metric tons
Throughput capacity	346 metric tons/hr
Capsules per train	3
Payload per train	4,800 kg
Launch Interval	50 seconds
Average velocity	9 m/s
Pipeline Fill-Factor	2.7 percent

In a recent economic analysis of freight pipeline transport, Henry Liu points out that pneumatic pipelines have not found a large market because in most cases they can not compete with trucks on economic grounds (Liu 2000). He points out that pneumatic pipelines are limited to a rather small linefill because of the limitations on the blowers. The Sumitomo system in Table 1 has a fill factor of 2.7%. A relatively large pipeline is therefore required for the throughput capacity achieved. He points out the throughput is further limited by the conventional use of inline loading and unloading. Liu concludes that substituting an electromagnetic drive for the external blowers can achieve as much as four times larger linefills, resulting in factors of two reductions in capital cost. William Vandersteel had also pointed out the fundamental gains of switching to electromagnetic drives, and has a basic early patent on the idea (Vandersteel 1984).

Both Liu and Vandersteel suggested the use of Linear Induction Motors (LIM) for the electromagnetic drives. Magplane has utilized Linear Synchronous Motors (LSM) for the drive because they are tolerant of larger operating gaps. This is a feature important to the construction of realistic pipeline systems.

The electromagnetic system we have constructed has a vehicle design speed of 40 MPH (18 m/s), twice as high as those in Table 1, further increasing capacity for a given pipeline diameter. Electromagnetic switching has been utilized, allowing multiple off-line stations accessed by “no-moving parts” switches.

A demonstration project has been constructed to test the feasibility of the system concept and the various components (Montgomery and others 1999, 2000).The facility is designed to allow the vehicle to be loaded and accelerated to 40 MPH, coast to a stop in climbing a 60 foot elevation hill; reaccelerate to 40 MPH in descending the hill, and decelerate to zero, unload, and then be recycled through the process. An electromagnetic switch demonstration is located between the accelerator and the hill. The six-foot wheelbase vehicle uses six-wheel assemblies at each end of a rotating hopper, and has a payload capacity of 660 pounds.

This report presents an economic model suitable for optimization of parameters and to project the competitiveness of the concept against conventional transportation methods.

The project was initiated in August, 1997. Installation at the IMC-Global Lonesome Mine site began in May, 1999. Stage 1 field tests were competed in November, 2000. Stage 2 and 3 testing will be undertaken in the future when anticipated resources become available.

METHODOLOGY

A preliminary economic model was first developed to explore the system parameters most likely to be of interest to the phosphate industry. It was determined that a pipeline of 24-inch diameter would serve a variety of applications, and that size was therefore picked for the scale of the demonstration project.

It was recognized at the outset that one of the most important applications to address was the transport of matrix. It was also recognized, however, that materials handling of the wet and clay-like matrix would equally represent a special challenge, and one somewhat unrelated to demonstrating the basic feasibility of the electromagnetic drives. It was therefore decided to limit the project scope to the transport of phosphate rock product. The economic studies, however, were extended to typical matrix transport parameters.

The length and layout of the demonstration project was chosen to address the core feasibility issues. There was, however, some consolidation and compromise between the original layout proposed and that constructed, driven by the desire to keep the project within budget. In spite of the compromises, the project believes that the basic feasibility of the concept has been demonstrated. The degree to which the goals of the original proposal were addressed is discussed in more detail in the section on Assessment of Test Results.

The economic model has been refined throughout the project, and has been used to project the likely competitiveness of the concept against conventional transportation methods.

RESULTS

DEMONSTRATION PROJECT

Prototype System Description

A prototype to demonstrate the feasibility of electromagnetic propulsion of capsules in pipeline applications has been constructed at the IMC-Global Lonesome Mine site (Montgomery 1999, 2000). The demonstration project utilizes 700 feet of 24-inch diameter centrifugally cast fiberglass pipe, and contains a 200 foot long accelerator/decelerator linear synchronous motor section, a switch demonstration, and load and unload stations. The test vehicle traverses back and forth, and is capable of a peak speed of 40 MPH. The six-foot wheelbase vehicle uses six-wheel assemblies at each end of a rotating hopper, and has a maximum payload capacity of 660 pounds.

A cross section of the pipe containing a typical vehicle is shown in Figure 1, and the vehicle is shown separately in Figure 2. The linear synchronous motor “stator” winding is mounted on the outside of the tube leaving the inside of the tube free of obstructions. The permanent magnet assembly mounted on the vehicle consists of four poles, alternately north and south.

Pipe

A cast fiberglass “waste water” pipe product is used for the straight sections, and is supplied in 20 foot lengths with a 5/8-inch wall. Because the winding is on the exterior of the tube, the tube must be made from a non-conducting material. The curved sections are also fiberglass, but are built on an interior removable mandrel. The sections are joined by standard sealed couplings. The pipe can be run at ground level, in elevated sections, or underground.

Vehicle

The vehicle consists of a cylindrical open-top hopper 20 inches in diameter by 48 inches long, attached to wheel carriers at each end through pivot bearings. This allows the hopper and the wheel assemblies to rotate independently around the pipe line central axis. Each wheel carrier has six wheels spaced at equal 60 degree angles. The wheels are 6-inch diameter polyurethane coated standard industrial units with sealed ball-bearings. The overall length of the vehicle is 84 inches. The magnet assembly occupies a 90 degree by 48-inches long sector at the bottom of the vehicle, and in the version of the vehicle shown, has the ability to rotate around the central axis independently from the hopper. A photograph of the vehicle is shown in Figure 3.

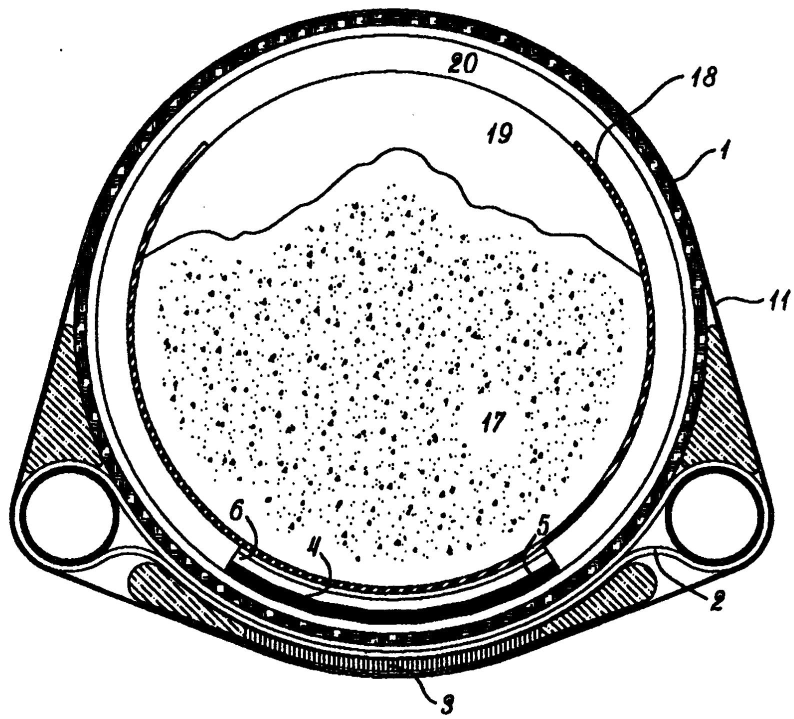

Figure 1. Cross-Section through Pipe and Capsule Mid-Section: (1) Fiberglass Tube; (2) LSM Winding; (3) Laminated Winding Back Iron; (4,5) Permanent Magnet Assembly; (6) Magnet Back Iron; (11) LSM Winding Support Strap; (17) 270 Kg Payload; (18) Payload Hopper Shell; (19) Payload Hopper Back Wall; (20) Wheel Carrier Flange.

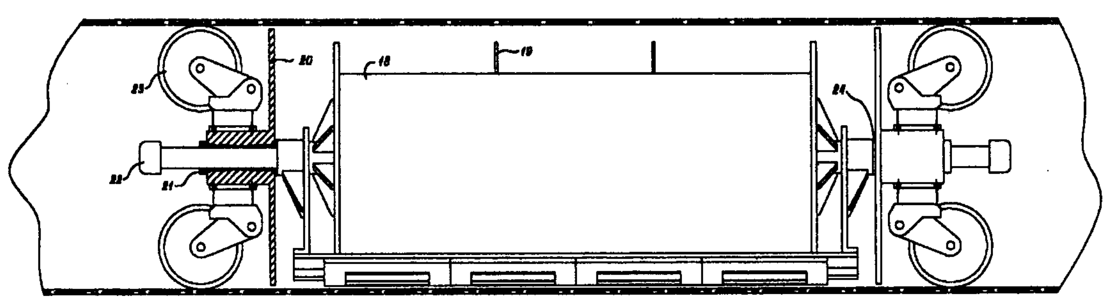

Figure 2. Elevation View of Pipe and Capsule. The 4-Pole Permanent Magnet Structure Is Below the Payload Capsule. In this Design Configuration, the Wheel Assemblies, the Payload Hopper and the Magnet Assembly Can All Rotate Independently from Each Other.

The fully loaded capsule weighs 1420 pounds, of which 500 pounds is the nominal payload. The ratio of payload to overall weight (0.35) is lower than one might have postulated from conventional capsule systems. This is largely a consequence of the need to carry an on-board magnet system, which weighs 200 pounds. Additional tare weight reductions may be possible as the project moves beyond the prototype stage.

Magnet Assembly

The magnet assembly consists of an array of individual blocks 2-inch x 2-inch by 0.5-inch deep, magnetized parallel to the 0.5-inch dimension. They are located on a curved back-iron plate 24-inch by 48-inch long by 0.5-inch thick. The 112 individual magnet blocks are arranged in sets of 28 to form four poles, two north and two south. The poles have a “pole pitch” of 12 inches, and a repeat pitch of 24 inches. The magnets blocks are magnetized prior to mounting on the back iron. A photograph of the magnet assembly is shown in Figure 4.

Linear Synchronous Motor Winding

A linear synchronous motor concept was chosen over a linear induction motor concept because it retains reasonable efficiency at large operating gaps. The gap between the magnet face and the effective centerline of the winding is 1.25 inches.

The linear motor windings are wound in single modules attached to the outside of individual 20 foot pipe sections. Two motor modules are shown in Figure 5. The motor winding was designed to use a continuous length of electric wire rated for direct burial. While this choice complicated the winding process, it resulted in the complete elimination of internal connections between motor conductors. Only the six ends of the three-phase windings need to be connected to the motor drive system.

Each length of wire forms one phase of the three-phase winding, and is wound back and forth 14 times using special tooling. A single phase of the winding (artificially foreshortened) is illustrated in Figure 6.

A laminated iron backing of 0.5-inch thickness is included outside the winding to double the effective permanent magnet field at the winding, reducing the power requirement by a factor of four. The winding and back iron mounted on the tube are pictured in Figure 7.

In a freight transport pipeline there is no need for the capsules to maintain a constant velocity, and therefore no need to cover the entire length of the line with motor windings. Rather the capsules can coast between periodically spaced motor modules which boost the speed lost to wheel friction and moving air in the pipe.

Figure 3. Car Loaded with 660 Pounds of Phosphate Rock Parked in the Opening Above Motor Number 3.

Figure 4. Magnet Assembly Rotated 180 Degrees for Inspection Purposes.

Figure 5. Lakeland Demonstration Project Hill Climb. Motors Number 9 and 10 Are Shown in the Foreground.

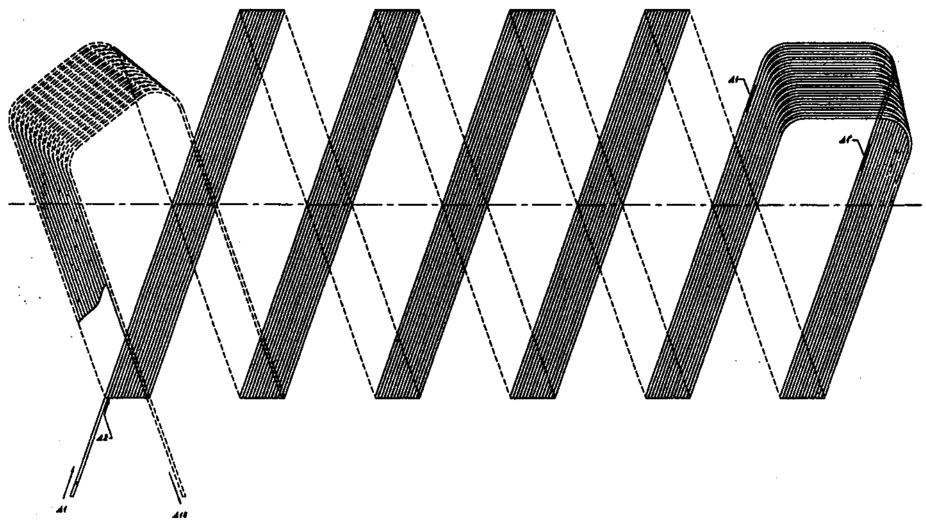

Figure 6. Single Phase of the Three-Phase Linear Synchronous Motor Module (Not Shown Full Length).

Figure 7. Close-Up of LSM Winding Showing End Turns and Back Iron.

The fraction of motor coverage required is a function of the allowed loss of speed between motors, the capability of the motors, the pipeline and capsule characteristics and the velocity and spacing between capsules. For the prototype 24-inch pipeline and 1580 pound loaded capsule, the minimum percent coverage required on flat ground is 5% if capsules are traveling at 1 second intervals and are permitted to lose 10% of their 40 MPH speed before re-acceleration.

While limiting motor coverage to small percentages of the total pipeline has a beneficial economic effect, it presents a potential problem of system restart after a loss of power. In the above examples (assuming level ground) the 1 second spaced capsules would coast to a stop in 158 seconds, traveling a distance of 1700 m (4320 feet). The relatively long coast times allow normal recovery from the most common power failures which are only a few seconds in duration. The capsules would simply coast to a somewhat lower velocity before being automatically re-accelerated when the power was restored. However for long-term power outages, a restart strategy is required.

The restart strategy chosen would depend on an assessment of the expected frequency of long duration power outages. If it were once a year, a slow recovery could be tolerated; for example, motorized "recovery" capsules could clear the pipeline. If the expectation were for much more frequent long-term outages, a more pro-active system would be required. The use of low-cost, "recovery" windings between fully powered motor section is a reasonable option.

The above discussion is based on a hypothetical level-ground installation. In cases where significant altitude changes must be accommodated, motor coverage would need to be further increased.

Magnetic Switch

In our early switch concepts, a set of excitation coils would have been placed on the outside of the tube ahead of the switch to rotate the on-board magnet from its natural 0 degree vertical position to the 90 degree horizontal orientation. The attraction between the magnets and an external iron insert in the tube wall would then provide sufficient force to balance the centripetal force (and the overturning moment on the car) in rounding the curved switch path . These excitation coils would need to be synchronously driven to follow the field pattern from the magnets on the moving car.

In the current design, the external switch coils interact with simple iron pieces on the hopper side wall rather than with the swinging permanent magnets. A set of electromagnets external to the pipe is used to magnetize and attract the iron added to the hopper. In this concept, the external magnets are located at the 90 degree orientation along the length of the switch branch and importantly, do not need to be synchronously driven. A DC current is applied to the magnets on the side of the switch branch to be followed.

Field test experience to date suggests that eliminating the requirement to synchronize the phase and frequency of the switch coils greatly increases the switch reliability. One can afford an occasional missed synchronization in the propulsion system because the next motor can make up the difference. Faulty operation of the switch synchronization, however, could have resulted in a failure to negotiate the appropriate branch, leading to possible mechanical damage.

A second benefit accrues to the new design in that the on-board magnet assembly no longer needs to rotate independently from the hopper, allowing the permanent magnets to be attached directly to the bottom of the hopper.

The four external switch magnets are each six foot long, and are used to hold the car against the inside wall of the 200 foot radius curved branch of the switch. A static test has been performed using the external magnets and the re-built car to demonstrate that the magnets exert sufficient force on the fully loaded car to hold it against the inner wall at full velocity.

The switch magnets are installed on the 30 foot long horizontal curve of the pipeline as shown in Figure 8. Sensors can be used to confirm that the car rides on the inside wall of the tube. When the switch magnets are not energized, the vehicle will naturally follow the outside wall. Testing in a curved tube (as opposed to a real "Y section" switch) is a necessary initial step, and demonstrates the critical element of the switch at minimum expense.

The car was modified to reflect the new switch concept. Some additional iron was added to the sides of the hopper, and the on-board magnet assembly was pinned to the hopper, thus omitting the original rotating feature. When the next generation vehicle is designed, omission of the rotation requirement will simplify the design and remove unnecessary weight.

Load and Unload Stations

In the demonstration project the load station consists of an accumulation hopper with a control slide valve which dumps through a chute into the at-rest hopper section. The unload station has the ability to rotate the hopper 180 degrees, and leave a clear path for the load to gravity dump into a collection hopper. A transfer conveyor then returns the dumped load back to the accumulation hopper. The load/unload stations are pictured in Figure 9.

It had been the original intent to demonstrate the feasibility of on-the-fly load and unload operations. However, after studying other installations (for example, the Sumitomo pneumatic pipe line installations) it was determined that *loading* on-the-fly was not practical at the vehicle velocities needed. A conceptual design for *unloading* on-the-fly was designed, and the magnetic systems necessary to demonstrate the essential

Figure 8. Switch Demonstration Magnets in the Process of Installation on the 30-Foot Long, 200-Foot Radius Horizontal Curve.

Figure 9. Load-Unload Stations and Transfer Conveyor.

features of the manipulation were installed. That was done by mechanically rotating the external magnets around the stationary car to dump the load without mechanical contact. In an actual on-the-fly unloading station, a helical magnetic path would have been constructed along the vehicle path. The stationary vehicle approach allowed the unload and load stations to be integrated and allowed the use of a short transfer conveyor to return the dumped load to the loading hopper.

Economic analysis has confirmed that unloading on the fly is desirable but not essential. Depending on system length, reduction in capital investment range from 5% to 15% if on-the-fly unloading is introduced.

Power Conversion and Control

A standard 100 HP commercial four-quadrant motor drive is used to drive the synchronous motor modules. The drives are outfitted with proprietary control systems to enable them to automatically synchronize the LSM, and to interface with the global control system. An output frequency of 30 Hz is synchronous with 40 MPH (18 m/s.) Ten modules in series are required to accelerate a fully loaded vehicle to 40 MPH. Five 100 HP drives, pictured in Figure 10, are time-shared between the ten motor sections.

Motor Control. The drive system uses current-source inverter drives rather than PWM units. Current-source inverters utilize SCRs, which provide performance benefits in pulsed duty applications.

The motor, motor drive, and control systems must operate in a harsh mine environment with high reliability and minimal maintenance. This requirement led us to reject solutions that required power sources on each vehicle to transmit position and thrust angle information to the motor control system.

The motor control system instead uses open-loop control of the linear synchronous motor (LSM) thrust angle rather than feedback control. Feedback control is often described as essential for stable LSM operation. Since the majority of LSM studies were conducted for transportation of people this basic assumption was based on the premise that ride quality is an important consideration. That premise is invalid when designing an ore transportation system.

The test facility includes several features that will not be installed in a production system. The vehicles operate in both forward and reverse directions in the same pipeline, while production systems will have a dedicated supply and return line. There are five motor drives shared between the 10 motors for reasons of economy. (Motor drive 1 controls motor 1 and 6, motor drive 2, controls motors 2 and 7, etc.). In a production system, the rating of the motor drive is matched to the power requirements of the motor,

Figure 10. Five 100 HP Motor Control Units Shared Between Ten Motors.

typically on the order of 100 kW, and the complexity and failure modes associated with multiplexing of motor drives is not warranted.

Control Overview. The control system functions are divided between the Current Source Inverter (CSI) control card and the Programmable Logic Controller (PLC) that coordinates the operation of individual motor drives, load and unload stations, and the magnetic switch. The CSI control card is responsible for sensing the position and speed of the car, synchronizing the magnetic field produced by the motor winding to the vehicle, and determining appropriate acceleration or deceleration forces as a function of vehicle velocity. The PLC enables each motor drive, sets target velocity and direction for each motor drive, operates the load, unload, and switch mechanisms, and collects data from various diagnostic instruments used for development. A personal computer provides development tools and a graphical user interface for the operator.

Magnetic Velocity Sensor. A simple pickup coil with an iron core is located on the joint between motor sections. The changing magnetic field produced when the vehicle field assembly passes over the coil produces a voltage on the sensor. The control card processes the sensor signal to determine car velocity, which is proportional to the frequency of the induced voltage. This simple system is electrically isolated from the motor winding and has no moving parts. In the test facility, the vehicle is operated in both directions in the pipeline, necessitating two sensors, one at each end of each motor. The

PLC control system selects which sensor is to be used, depending on the direction the vehicle is traveling.

Phase Delay Control. The CSI drive must energize the motor winding after all four poles of the vehicle field assembly are engaged in the motor winding. This requires a delay, inversely proportional to vehicle velocity, from the time the magnetic field sensor is activated. This delay is computed digitally on the CSI control card.

The delay is determined by the distance between the sensor and the motor winding, divided by the vehicle velocity. This distance is programmed into the control card after the motor sections and sensors are installed in the field. The distance between the sensor and the winding are then measured and programmed into each control card. Vehicle acceleration is then measured with independent photo diode diagnostics and small changes in the distance value made until acceleration is maximum. The adjustment range is 0-40 cm, with resolution of 0.5 cm. Adjustments of +/- 1 cm produce very little change in the performance of the system.

The test facility requires that the vehicle be operated in both forward and reverse directions, requiring two magnetic velocity sensors and two phase delays. The sharing of one drive between 2 motors then requires a total of four phase delays. The PLC signals the control card which phase delay is to be selected.

Acceleration/Deceleration Control. The vehicle is typically accelerated as rapidly as possible, but in some modes of operation the vehicle may be decelerated or operated at constant velocity. Acceleration is a complex function of vehicle mass and speed. The thrust available from the motor is determined by the current in the motor winding. Low vehicle velocity results in longer motor operating time. The motor winding current is limited at low vehicle velocities by heating of the motor conductors. High vehicle velocity results in enough induced EMF in the motor winding so that drive voltage limits result in reduced currents. Finally, the vehicle is considerably heavier loaded than unloaded, and the acceleration must be adjusted accordingly.

The acceleration available as a function of velocity is calculated for both loaded and empty vehicles. The results of the calculation are converted to digital values that are stored in an EPROM on the control card. The EPROM is used as a look-up table; car velocity (motor drive frequency) is presented to the EPROM, and the corresponding increment in velocity is returned on the EPROM output bus. Different tables are constructed for empty and loaded vehicles, and with different values of acceleration for testing purposes. The EPROM has sufficient memory for 4 independent look-up tables.

Emergency Braking Control. The PLC can close a contactor to place a short across any motor winding, which provides a braking force on any vehicle moving in the winding. Loss of AC power disables the PLC, so a normally closed contactor is included in each motor control. The contactor is held open whenever AC power is available, but loss of power results in all contactors closing and all motors braking the vehicle. This arrangement is specific to the demonstration, and would be handled differently in a production system.

PLC Controller. The Allen-Bradley SLC-50 series of programmable logic controls is used for overall system control and supervision. The PLC sets the direction and acceleration values for each motor, enables the motor drives, operates the contactors used to connect a single motor drive to two motors, and collects data from some diagnostic instruments used during development. Ethernet is used to link the PLC with the operator's computer and any software development computers. One diagnostic package particularly useful during startup is composed of a separate PLC that is powered from batteries which can be placed in the hopper of the vehicle. This PLC is connected to a radio modem that provides an ethernet link between the moving PLC and the stationary controls. The moving PLC records values of magnetic field, thrust, air pressure, temperature, or other desired information and transmits the results to the main PLC. The data is then transferred to the operator's computer for analysis, plotting, and printing.

Motor Performance. The thrust that can be provided by the modules at a given speed is related to the phase angle between the winding drive and the magnet poles on the capsule; it is maximum when the angle is 90 degrees. If feedback control between the vehicle position and the drive phase is employed, angles approaching 90 degrees can be utilized. If the angle is reduced to 60 degrees, the system will operate stably without feedback control. The thrust at 60 degrees drops to 87% of the maximum available, but is a reasonable tradeoff against the complexity of a feedback loop requiring continuous and accurate position sensing, and the need for on-board transducers.

The calculated performance of a pair of boost modules sufficient to restore a 10% drop in velocity is given below, and is based on the prototype design. A commercial 100 HP conventional synchronous motor drive unit is sufficient to power a module. The performance of the two motors can be seen to be somewhat different, reflecting the speed dependent characteristics. The efficiency increases with speed, but the increased back EMF developed by the vehicle motion cuts into the maximum voltage limit on the drive, decreasing the current and thrust available.

Table 2. 1st Boost Motor and Drive at 60 Degree Phase Angle.

Velocity in (m/s)	15.9	Efficiency (%)	55
Velocity out (m/s)	17	Power factor(%)	44
Time (s)	0.26	traction power (kW)	37
Acceleration (m/s^2)	4.5	input power (kW)	69
Thrust (N)	2242	input power (HP)	93

Table 3. 2nd Boost Motor and Drive at 60 Degree Phase Angle.

Velocity in (m/s)	17	Efficiency (%)	58
Velocity out (m/s)	18	Power factor(%)	44
Time (s)	0.24	Traction power (kW)	37
Acceleration (m/s^2)	4.2	Input power (kW)	65
Thrust (N)	2090	Input power (HP)	88

Field Testing

Stage 1 - Demonstration of Basic System Feasibility

The demonstration project was constructed to test the feasibility of the system concept and the various components. The test allows the vehicle to be loaded and accelerated to 40 MPH, coast to a stop in climbing a 60 foot elevation hill; re-accelerate to 40 MPH in descending the hill, and decelerate to zero, unload, and then recycled through the process. The switch demonstration is located between the accelerator and the hill.

During six weeks of final testing 5300 round trip cycles of the car were run at speeds between 18 and 33 MPH. The total distance traveled by the car was approximately 700 miles, and the total travel time about 40 hours.

Load and Unload. A total of thirty-five complete load/unload cycles were run with a payload of 430 pounds under total automatic PLC control.

The car was captured and dumped from the unload station carrying a maximum payload of 660 pounds, thus also demonstrating the ability of the magnetic coils to capture, rotate and dump a payload 30% higher than the nominal 500 pound payload. Prior to dumping, the loaded car was cycled 50 times at 25 MPH with the maximum payload.

The hopper filled and dumps cleanly. There is a minor problem in the loading station chute where some rock is trapped by internal support structure. The trapped material has a tendency to vibrate loose and fall into the bottom of the pipe after the car leaves the loading station. This minor spillage was dealt with by cutting discharge holes in the bottom of the pipe; on the next cycle the car sweeps the spillage out the holes.

Motor Phasing. The initial phasing of the ten motors have been tuned to achieve design levels of acceleration in the individual motors. The task of initial phasing is greatly assisted by having acceleration diagnostics which use photo diode sensors at the exit of each motor to measure exit velocities. The PLC uses the signals to present the car velocity at the exit of each motor and the calculated acceleration achieved by that motor. The diagnostic photo diode sensors are independent of the electromagnetic sensors at the motor exits which are used only for control.

Maximum Velocity Tests. In preparation for the highest speed testing a number of component tests were performed and several system operation changes made:

- The 3-phase connections of the ten motor sections were rewired from “delta” to “Y”. This was done to avoid any possibility that induced circulating currents in the delta connected windings might be unduly influencing the vehicle speed. The Y connection avoids induced currents, but is subject to higher induced voltages when the car passes over the winding.

- A passive safety feature was incorporated to guarantee that the car could not return through the motor sections at a velocity high enough to cause it to exit the pipe above the load/unload stations. Motors 10, 9 and 8 are now automatically shorted by the PLC control after the car leaves to climb the hill. The shorted windings passively brake the car’s speed as it returns from the hill to a value below that which could exit from the end of the pipe. The remaining motors (7 through 1) then re-establish the proper speed to bring the vehicle into the load or unload station.

- Pressure measurements were made in the pipe ahead of the car. There had been some concern that the relatively tight fit of the wheel plates in the pipe was producing excess aerodynamic drag. A pressure pulse of 0.2 psi appears when a 25 MPH vehicle first closes off the opening in motor 3 and compresses the air in the rest of the pipe. After the air column has fully accelerated, however, the pressure drops below 0.05 psi. At 0.05 psi, the drag is about 20 pounds and thus unlikely to be a significant contributor to retarding acceleration in the speed range of interest.

The maximum velocity achieved during final testing was 33 MPH and is 83% of the design goals of 40 MPH. The maximum speed achieved is currently being limited by the voltage protection circuits on the motor drives which are tripping out at only 60% of the voltage capability of the motor drive. When proper high-power filter capacitors are installed on the motor drives, the car should be capable of acceleration beyond the current velocity limit.

Ultimate achievement of the design speed of 40 MPH, and demonstration of motor drive operation at full capacity will be technically important. We note, however, that the 40 MPH milestone goal was established at the start of the program when we assumed that the operating speed would have a more significant impact on system cost. Later modeling showed that for long hauls (a distance of 30 miles), 40 MPH is the optimum speed. However, only a 6% cost increase results from a choice of 30 MPH. At shorter hauls (3 to 10 miles), the minimum cost is actually achieved at 30 MPH.

Measured Motor Performance. The acceleration achieved by each motor on a typical run is given in Table 4. The calculated acceleration is based on the measured values of input and output velocities at the ends of the tubes on which each motor is mounted, and an assumption about the effective length of each winding. The individual accelerations range from a low of 1.86 m/s^2 to a high of 3.07 m/s^2. The variations depend on a combination of initial phasing of the wave relative to the car, and the efficiency of the particular motor.

The average acceleration measured over the ten motors was 2.42 m/s^2, approximately half that calculated in Table 2. This is consistent with the limitations on the output voltage of the motor drives, which limited the delivered power to half or less of the drive capacity.

Table 4. Typical Exit Velocities and Acceleration of Individual Motors.$^+$

Sensor Location	Velocity (m/s)	Acceleration (m/s^2) ***
Motor 1 in	5.62 *	
Motor 1 out	7.01	2.62
Motor 2 out	8.35	3.07
Motor 3 out	9.14	2.06
Motor 4 out	9.95	2.30
Motor 5 out	10.92	3.02
Motor 6 out	11.7	2.63
Motor 7 out	12.24	1.93
Motor 8 out	12.75	1.90
Motor 9 out	13.23	1.86
Motor 10 out	13.92 **	2.79
average		2.42

+ empty car @ 920 pounds (418 kg)
* entrance to motor 1 @ 12.5 MPH
** exit from motor 10 @ 31.1 MPH
*** assumed fully engaged winding length @ 11 feet

The measured losses of the car traveling in the pipe were larger than the original calculations which assumed a coefficient of wheel friction of 0.01, and an air friction loss at 40 MPH and 3 second vehicle intervals, approximately equal to wheel friction. Losses were measured at 25 MPH for the vehicle during field testing. The measurements were made by allowing the vehicle to coast across the different unenergized motor sections and measuring the change in velocity as the car slowed down.

Losses occur from wheel friction, air friction, and electromagnetic losses from eddy currents induced in motor windings as the car moves past. While theoretically possible to separate the loss components by varying the speed at which the measurements were taken, it did not prove practical. The variation from motor to motor (probably due to variations in diameter and wheel compression) were larger than the speed variations, masking the dependencies.

The motor-to-motor losses varied from an effective friction loss of 0.04 to as high as 0.15. That is, approximately 2 to 8 times that assumed in the original calculations. This increase is a result of some combination of undersize tubes which caused considerable compression of the wheels; attraction between the car magnets and the winding back iron, which substantially increases the apparent weight of the car; induced eddy currents in the delta connected motor windings, not previously considered; and possibly larger than anticipated local turbulent air friction losses.

Switch Magnet Demonstration. The switch magnetic system was energized during the final testing. The interaction between the magnets and the car could be observed through a small observation port, but we were not able to convincingly demonstrate that the car was forced to follow the inside radius. We had previously carried out a factory test in which the switch magnetics were shown to be capable of balancing the overturning moment on the car, which is the dominant load. We have reason to believe, therefore, that the magnets are actually holding the car against the inner wall. However, our hope that we would be able to hear a difference in the wheel noise during the passage with and without the magnets energized, was not realized. In future testing we will need to install more definitive diagnostics.

Stage 2 - Provide Sufficient Operation to Support a Decision to Deploy

At the completion of the stage 1 tests we have demonstrated to our satisfaction the basic feasibility of the overall concept. However, we believe that significantly more testing will be required before a decision could be made to deploy the system. In addition to demonstrating additional components, for example, a fully operational switch and alternate unload options, we propose that approximately 300 hours of operations be carried out during this stage. The overall objective for this stage is to provide the information necessary to support a decision to deploy the electromagnetic technology in a commercial operation.

Three hundred hours of operation would provide approximately 30,000 cycles and a travel distance of 5000 miles. This should be adequate to project failure rates for all components and to provide information for design improvements that might extend those lifetimes.

Accumulation of 300 hours of operation over a one year period requires an average of 40 hours of operation a month for eight months.

Stage 3 -- Confirmation of Commercial Components Designs

After a decision is made to deploy a commercial system, additional testing would be undertaken to confirm design details. By way of example, the 30,000 cycles accumulated in stage 2 still represents a small fraction (0.01%) of the cycles that a given pipeline joint would experience over a 20 year life, assuming passage of a vehicle every 6 seconds.

Unlike replaceable components (wheels, for example) the pipe joints would be expected to be lifetime components. It would therefore be prudent to carryout true life tests on joints. A test stand where fully loaded wheels were cycled back and forth over a test joint would be one cost effective approach.

Pipe surface wear and in situ re-coating strategies would represent another area amenable to a test stand approach. During this period various components in the Lonesome mine installation would also be upgraded and additional operation of the system carried out.

Assessment of Test Results

The tests which have been done to date establish the basic feasibility of the design. They have also resulted in several iterations of the design, primarily to improve the robustness of the motor control system against system electrical noise, and to alter the vehicle switching approach.

While no appreciable life testing has been accumulated, it is already possible to anticipate one design detail that will require improvement, namely the joints between pipe sections, which will require a special design to avoid fatigue failure of the pipe ends. The fiberglass pipes used in the present demonstration were centrifugally cast against an OD mold, resulting in a close tolerance on the OD, but a significant variation in ID dimension. The wheels tend to shock load the dimensional steps at the joints, resulting in early fatigue cracking. Use of pipes built from an ID mandrel would greatly reduce the dimensional variation, but we believe that more basic design changes will eventually be necessary to meet the long life fatigue strength requirements. One possible solution would be to use metallic elements in the pipe-to-pipe joints by molding metal end rings into each pipe.

There has been little evidence of wheel or pipe surface wear to date. The wheel materials have been chosen to be soft relative to the pipe surface to assure that the wheels, which are replaceable, will wear rather than the pipe surface. We can anticipate, however, that it will be necessary to develop a technique for in situ re-coating of the pipe surface.

Contract Goals

At the end of the FIPR contract, the project has met a number of the most important proposed goals; some proposed goals have been partially met by demonstration of one or more essential aspects; and some proposed goals have not been met under this contract, largely due to budget constraints. A follow-on project funded by Magplane will address the most important missing goals.

Proposed Goals Accomplished

- The project has demonstrated the control and propulsion of an electromagnetically driven vehicle with a scale and operating parameters potentially useful for deployment in the phosphate industry.

- The project has demonstrated economic projections based on the design which suggest an attractive rate of return against competitive modes of transport for both short and long-haul applications.

Goals Partially Accomplished

- The project has demonstrated a modular pipeline design with straight and curved sections which can be assembled and disassembled.

    - The project has **not** demonstrated that the pipeline modules are sufficiently rugged or sufficiently readily assembled and disassembled to be qualified for deployment in the mining environment.

- The project has demonstrated the magnetic system necessary to manipulate a vehicle in an on-the-fly load or unload system.

    - The project has **not** demonstrated the proposed goal of on-the-fly loading or unloading. The vehicle is brought to a stop for these operations.

- The project has demonstrated the magnetic system necessary to manipulate a vehicle through a two branch switch

    - The project has **not** demonstrated operation of an actual switch. The system simulates one curved branch of a switch by using magnetics along a curved section of pipe but does not actually contain a two branch "wye" element.

Goals Not Accomplished

- The project has **not** demonstrated the simultaneous operation of two vehicles. Only a single vehicle has been operated. However, the control and power system architecture would permit operation of a second vehicle

- The project has **not** demonstrated system operability over an extended period. A total of 5300 round trip cycles have been run; however, this is not sufficient to establish the estimated component lifetimes and maintenance requirements.

Background. Throughout the contract period, project decisions have been resource driven. In particular, it has proved more expensive and time consuming to develop the propulsion and control system than anticipated, and that has forced continual prioritization of the funds. It has also proved necessary to use substantial company funds to supplement FIPR and IMC project resources in order to bring the project to the point where the present accomplishments have been possible.

Demonstration of On-the-Fly Load and Unload. The proposal stated the desirability of on-the-fly load and unload operations, and proposed to establish their feasibility. By studying other installations we determined quite early in the project that *loading* on-the-fly was not practical at the vehicle velocities that we needed. We did, however, develop a conceptual design for *unloading* on-the-fly, and installed the manipulative magnetic systems necessary for that operation. That essential manipulation has been demonstrated by mechanically rotating the magnets around the stationary car to dump the load without mechanical contact.

Experiments were done early in the program to determine the minimum time necessary for a load to dump as a function of moisture content. It was determined that a minimum time of 1.5 seconds should be allowed.

The proposal stated that the trade-off between multiple parallel stations and on-the fly operations would be examined. We have done that analysis and confirmed the proposal supposition that unloading on the fly is desirable but not essential. Reduction in capital investment ranges from 5% to 15% if on-the-fly unloading is introduced.

Demonstration of Switching between Parallel Load Paths. The need to routinely switch between multiple paths is essential to the realization of significant throughput in economically sized pipelines. The proposal suggested the use of a mechanical switch, but recognized the high desirability of a no-moving-parts magnetic concept.

The project has partially demonstrated the magnetic manipulation necessary for switching by installing a magnetic structure along the inside radius of a 30 foot long curve. This is a significantly less expensive demonstration than installing a bifurcated “wye” section, which requires a magnetic structure and a run-out hill climb on both branches.

Simultaneous Operation of Two Vehicles. The simultaneous operation of two vehicles was proposed because it was recognized that more than one vehicle would need to occupy the accelerator at one time to accommodate the necessary short launch intervals. The power and control architecture was therefore chosen to accommodate more than one vehicle in the accelerator, requiring that each motor module have a separate power drive. The cost of the power units proved more expensive than anticipated, however, and it was later decided to “time share” five power units between ten motors. While the control system is capable of dealing with two simultaneous vehicles, the time share decision ruled out the ability to actually deal with more than one vehicle.

Shortage of resources also influenced the decision not to build a second vehicle. However, the first vehicle has been modified several times, reflecting the evolution of the design.

Extended Operation. It was proposed that the system would be run for an extended period. It was the intention that this operation would address the robustness of the system, and identify problematic design details.

During about six weeks of final testing 5300 automatic cycles of the car were run at speeds between 18 and 33 MPH. The total distance traveled by the car was approximately 700 miles, and the total travel time about 40 hours. However, this is not sufficient to establish the estimated component lifetimes and maintenance requirements.

Some test stand work was also done early in the program to give an indication of wheel and pipe surface wear under more extended operation. Two hundred hours of operation of a wheel set rotating against the pipe surface were accomplished without noticeable wear on either surface.

Follow-on Project. We believe that a sufficient number of goals have been met to establish the basic engineering feasibility of electromagnetic transport, and to provide input to the economic projections. Additional testing, system demonstrations and design improvements will be necessary, however, before a client such as IMC-Agrico could undertake deployment of a system with confidence.

Magplane anticipates financial backing to undertake these next technical steps with the collaboration of IMC-Global personnel. As a first step, extended operations will be demonstrated. As a second step, a two-branch switch will be installed and demonstrated. As a third step, on-the-fly unloading could be demonstrated if warranted. Given the marginal impact on economic projections, however, that decision will be assessed at a later time.

The operational test results from the follow-on project will be shared with FIPR as they become available.

ECONOMIC STUDIES

This section will discuss a method for economic modeling of a new bulk material transport system that can be flexibly configured to meet a wide variety of system needs, including transport distance, terrain, and throughput. The system employs a linear synchronous motor, and is given the name of an Electronic Transport System, or ETS for short. The economic model which captures the system cost relative to the haul distance and throughput needs. The variety of case studies evaluated here demonstrate the responsiveness of the model to a wide range of system requirements, and show that the new ETS system can be a competitive alternative to truck rail and slurry pipeline-based ore transport.

Supporting rationale for the cost bases used in this memo are presented in Appendix A. An approach to the design and costing of the load/unload stations is presented in Appendix B.

General trends are discussed in the “Parametric Sensitivity” section through the use of several hypothetical combinations of haul distance and throughput. In some instances cases use a generic rate of return on capital, in others they use specific competitive targets to calculate the actual rate of return. Six out of seven cases in the parametric section use a common set of cost basis assumptions. In the second section, “Cost Basis Sensitivity”, several cases are compared with alternate cost basis assumptions.

Cost Model Description

The system configuration is adaptable to most economically meet a wide range of system requirements, including transport distance, mass flow rates, and terrain. Pipeline diameter, and vehicle velocity are selected to minimize total system cost (annualized capital plus operating) as a function of transport distance and throughput. Nominal velocities are limited to 40 mph.

The cost for acceleration and deceleration of the vehicles is not include in the main line costs, but rather is accounted for in the load/unload station model, as described in Appendix B. This approach forms the basis for both engineering and cost calculations on the end stations for this economic model.

The cost model is currently set up as a spreadsheet which takes as input both engineering parameters which describe the system (Table 5) and unit cost values for the system components (Table 6). Unit costs are associated with a specific sizes and are derived either from purchased-equipment experience or from the literature. Unit costs are then scaled within the spreadsheet for required size in accordance with the rules given in Appendix A. A brief summary of the scaling rules and assumptions are given in Table 7.

Within the spreadsheet, multiple input parameters may be varied as part of any case study, although for systematic study, it is usually best to vary a single parameter at a time. This will result in a single curve of unit cost, for example, as a function of a single parameter, while other inputs are held constant. Families of curves may be created by duplicating the spreadsheet, changing a second parameter, and then linking the multiple spreadsheet outputs to a single plot.

Table 5. Engineering Inputs.

Input	Description
Distance	Length of the line
Average velocity	Average container velocity over the length of the line
Pipe diameter	Nominal inside diameter of the pipeline
Hopper diameter	Nominal inside diameter of the ore container
Hopper length	Length of the ore container
Fill fraction	Fraction of the ore container volume filled
Ore density	Density of the payload, source to destination
Number of coupled cars	Number of cars coupled together for each launch, run and dump
Yearly tonnage target	Payload delivered from source to destination
Return mass payload	Payload carried from the destination to the source
Fraction time utilization	Average fraction of time during which the line is operational
Motor coverage	Fractional length of the line covered by the motor winding
Minimum load time	Minimum time required to load and launch a single vehicle
Acceleration distance	Length of acceleration reference winding to reach velocity
End station length	Minimum length required for switching, loading and unloading

Table 6. Unit and Other Cost Inputs.

Unit cost input	Description
Vehicle	Cost per vehicle includes wheels, frame, hopper and bearings
Magnet assembly	Cost for magnet assembly per vehicle
Winding	Cost for a single section of LSM winding attached to pipeline
Back iron	Cost for iron laminations used with the single winding section
Power	Cost per watt for ac power converters
Block	Cost for controls to each powered winding section
Central control	Cost (lump sum) for central control station
Pipeline	Cost for materials, trench digging, installation and backfill
Load/unload stations	Cost for load/unload station components (see Appendix A2)
Power cost	Cost for operating power
Maintenance cost	Cost to maintain equipment
Labor cost	Personnel cost for operating and maintaining the system

Table 7. Cost Basis Summary.

	Scaling	Cost Assumptions Option B *
Capital Cost		
Vehicles	# vehicles x $(\text{payload})^{1/2}$	$2,000/vehicle
Magnet Assemblies	vehicle diam x length	$1,000/vehicle
Motor Windings	pipeline diameter x coverage	$2,000/module
Rescue Windings	pipeline diameter x coverage	option to include @ 0.33 x motor winding module
Back Iron	pipeline diameter x coverage	$1,400/ module
Motor Drives	$P = mv(n\_v) + Ldv^3/Re^{0.2}$ power scales with wheel friction and air drag	$1.00/watt
Block Control Units	fixed	$2,500
Central Control	fixed	$500,000
Pipeline (installed)	outer diameter^2	$360/m; 30% reduction in pipe material
Right-of-Way		$30,000/acre developed land $10,000/acre undeveloped 50 foot ROW
Load/Unload Stations	tonnage; number of parallel stations; % acceleration deceleration in mainline and branches [ref]	Option to use on-the-fly or stationary unloading
Cost per ton		Includes only operating cost; the rate of return is calculated from projected savings relative to specific competition; computation includes taxes, depreciation, and future discount rate.

- Appendix A and Table 31 explain the difference between Option A and Option B Cost Basis Assumptions. Option B represents more sophisticated assumptions introduced later in the parametric studies.

Table 7. Cost Basis Summary (Cont.).

Operating Cost		
Cost of Electricity		$0.04/kWh; the rate paid by IMC-A in Florida
Maintenance		3% of capital cost annually
Labor		long haul - Fixed Operating Crew @ $700,000; additional techs for maintenance unchanged short haul - $500,000 of operating crew; no additional techs for maintenance
Insurance/property tax		1.5% of capital

Parametric Sensitivity Case Studies

To evaluate the cost performance of the ETS, a number of different systems have been studied. These systems are characterized in Table 8 by source-to-destination (haul) distance and annual tonnage rate (throughput).

Table 8. Characteristics of Hypothetical Case Studies.

Case	Distance (mi)	Annual Tonnage Mt/yr	Assumptions
1	30	5 and 10	Cost Basis: Option A; generic rate of return
2	3	1 and 2	Cost Basis: Option A; generic rate of return
3	10	3 to 5	Cost Basis: Option A; generic rate of return
4	100	20 to 50	Cost Basis: Option A; generic rate of return
5	27	3 and 8	Cost Basis: Option A; specific competition
6	34	8.3 Mt/yr	Cost Basis: Option A; specific competition
7	1 to 5	8 Mt/yr	Cost Basis: Option B; generic rate of return and specific competition

For these studies, some parameters were held constant, although such constraints are not a restriction on the model in general. The common items include:

- hopper inner diameter which is 4 inches less than the pipeline inner diameter
- hopper length of 4 feet
- hopper fill-fraction of 70%
- ore density of 100 lb/ft^3
- no return payload (cars return empty)
- time utilization fraction of 80%
- fraction of motor coverage of 6%

- number of cars coupled in each launch, transit and dump group equals 3
- minimum load/unload time of 2 seconds per group
- flat terrain is assumed
- cost of competing system: \$3/t (explained under Case 1 below)

Results of the studies are presented on a case by case basis, in their Table 8 order below. Each case builds on the previous results, so only the observations which are advanced by the successively presented systems are discussed.

Case 1—30 Mile, 5 and 10 Mt/y System

Figure 11 presents the *capital* cost of the 30 mi, 5 Mt/y system as a function of the pipeline diameter at vehicle velocities of 20, 30, 40 and 60 mph. The figure shows that a capital cost minimum occurs in the vicinity of pipeline diameters of 16 to 20 inches, depending on velocity. The system that minimizes capital cost operates at 40 mph, with an 18-inch pipeline. Smaller pipelines tend to be more optimal at higher speeds because, for this combination of haul distance and throughput, the pipeline costs are the largest fraction of capital costs (see Figure 12 which shows the relative component costs of the system). Since pipeline costs are a strong function of the pipeline diameter, smaller diameters are desirable, and these are made possible by increasing the vehicle velocity to keep the throughput constant. Note that the second most costly component is the vehicles.

Figure 13 presents the *operating* costs using the same pipeline diameter and vehicle velocity ranges as were used to evaluate the capital costs. Here, the operating costs exhibit minima at slightly larger pipeline diameters when compared with the capital cost minimum. Also, the plots show that the 20 and 30 mph systems have the lowest operating costs at their optimal pipeline diameters. This is in contrast to the capital cost minimum, where, for pipeline diameters above 16-inches, the 40 mph system was the best choice.

Because the pipeline diameter that minimizes the capital cost is somewhat different than the diameter that minimizes the operating cost, a criterion is needed to evaluate the overall system economics that would be most desirable to a potential investor. The criterion used here minimizes the *total system cost*, which is defined as the sum of the annualized capital cost plus the operating cost. Calculation of the annualized capital cost requires a choice of a minimum attractive rate of return and a time over which the return will be realized. Throughout the evaluations in this section, these values are fixed at 20% and 20 years, respectively. Although the choice of these values is not critical to the generalized analyses here, these values are of great importance to a potential investor. Attractive rates of return will vary in different economic environments, and the period over which the return is realized must obviously be matched with product of the yearly tonnage target and the projected total tonnage over the lifetime of the operation. A different rate of return and lifetime will result in a different relative weight being applied to the capital cost before it is added to the operating cost, so not all choices of attractive rate of return and lifetime will necessarily give the same total cost minimum.

With the optimization criterion defined as minimizing the total cost, an optimum operating point for the 5 Mt/y system can be determined. This is accomplished using Figure 14, where the total system cost is plotted as a function of pipe diameter. The figure shows the minimum total system cost is $0.091 per ton-mile, and it occurs with a pipeline diameter of 18 inches and a vehicle velocity of 40 mph. A total of 7,188 vehicles (with 14-inch diameter, 4-foot long hoppers) are required round trip and a total of 6 parallel, load/unload-station branches are required at each end of the line to handle the 5 Mt/y throughput. The cost elements at this optimal point are summarized in Table 9.

Table 9. Cost Summary for Optimum 30 Mi., 5 Mt/y System.

Capital costs	($M)
vehicles	10.1
magnet assemblies	5.0
motor winding assem.	5.3
rescue winding system	0.0
back iron	1.0
power units (outgoing)	2.1
power units (return)	1.2
block control units	0.4
central control	0.5
pipeline	12.6
load/unload stations	5.0
Total capital cost	43.2

Operating costs	($M/y)
Power	2.08
Maintenance	1.30
Labor	1.38
Total Operating Cost	4.75
$/ton	0.95

Table 10. Cost Summary for Optimum 30 Mi., 10 Mt/y System.

Capital costs	($M)
vehicles	15.6
magnet assemblies	7.8
motor winding assem.	6.5
rescue winding system	0.0
back iron	1.2
power units (outgoing)	3.3
power units (return)	1.6
block control units	0.4
central control	0.5
pipeline	18.8
load/unload stations	5.5
Total capital cost	61.4

Operating costs	($M/y)
Power	3.28
Maintenance	1.84
Labor	1.53
Total Operating Cost	6.64
$/ton	0.66

If the distance is kept at 30 miles, but the yearly haul is increased to 10 Mt/y, the optimum pipeline diameter increases from 18 to 22 inches (Figure 15). The optimum vehicle velocity remains at 40 mph. The vehicle count has now increased to 8,694, and the number of parallel load/unload stations at each end has increased to 7. The cost results for the 10 Mt/y, 30 mile haul are summarized in Table 10. The total system cost for this higher haul rate has decreased from $0.091 to $0.064 per ton-mile. Higher haul rates over the same distance always decrease the total system cost (in $/ton-mile) because not all costs scale with load. Although the actual payback period minimum occurs at a pipeline diameter of 22 inches, the minimum is quite broad over the 18- to 26-inch diameter range. This observation points out an important point: *If the desirable annual tonnage* *rate is not known exactly before it is time to build a system, it is often better to err slightly on the side of a larger pipe diameter than what may be the optimum at the somewhat-prematurely-estimated target load*. If, with this target system for example, the pipeline diameter were increased to 24 inches, a size that better accommodates potential future throughput increases, the total system cost increases to only $0.065/ton-mile. This represents a total cost increase of only 1.6% over the 22-inch system, but the line limit capacity (see definition in Appendix A3) has increased by 23%. Remember that the hopper diameter is 4 inches less than the pipeline diameter, so the relative increase in the line limit capacity is proportional to $[(24-4)/(22-4)]^2 = 1.23$.

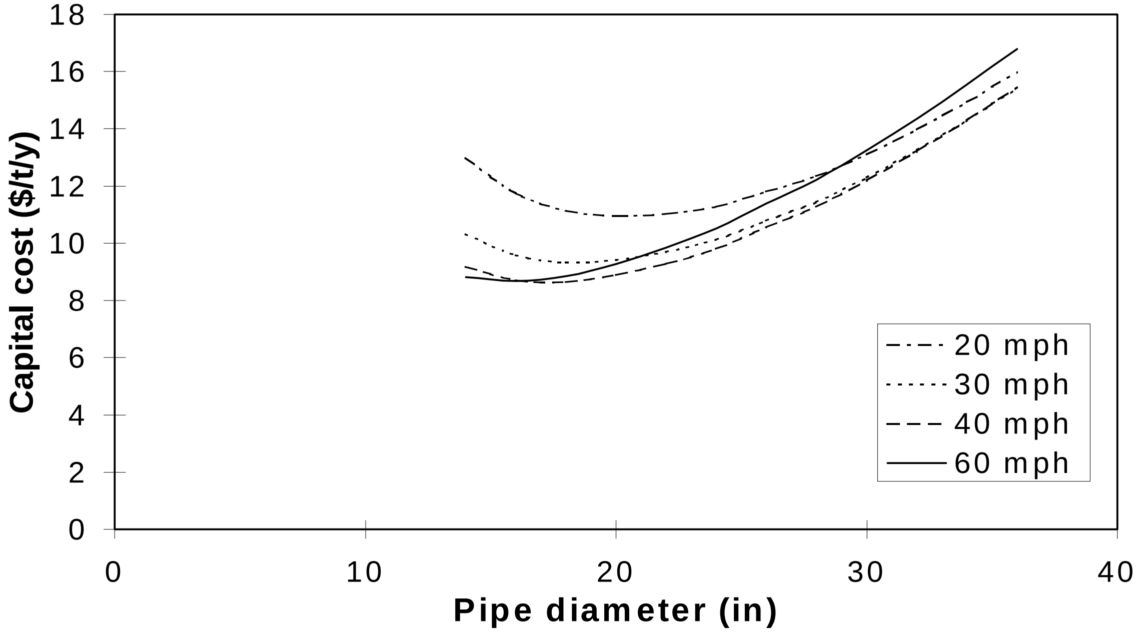

Figure 11. Capital Cost vs. Pipe Diameter, 30 Mi., 5 Mt/y System.

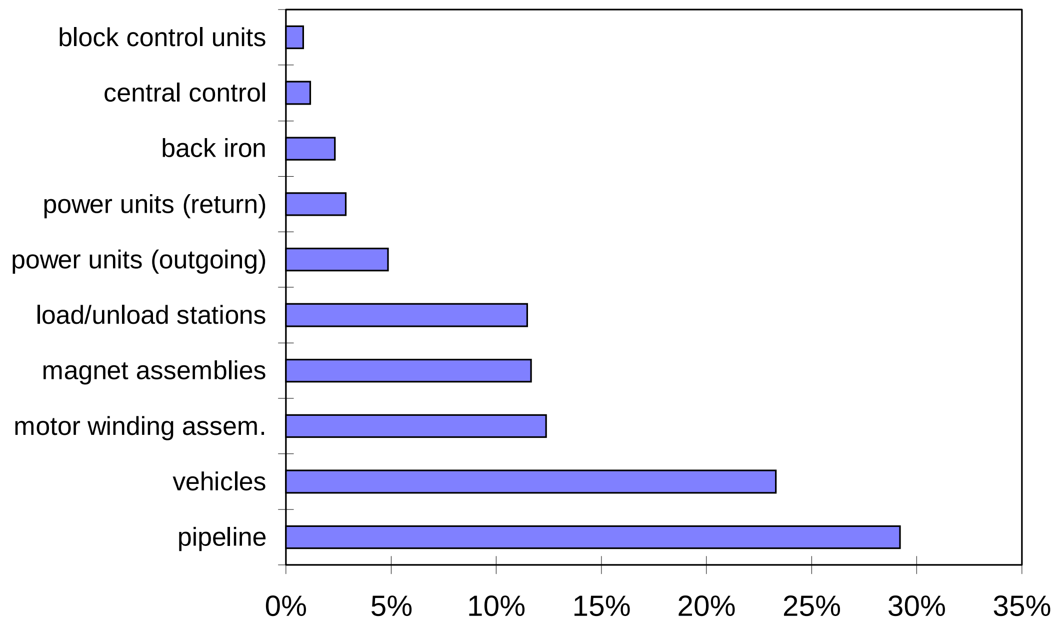

Figure 12. Histogram of Relative Costs for 5 Mt/y, 30 Mi. System.

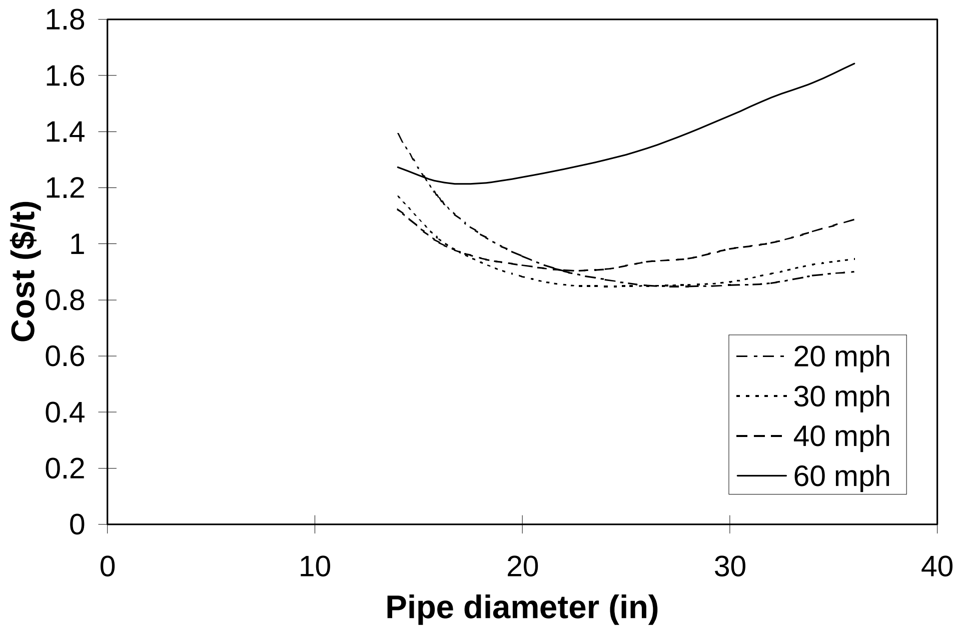

Figure 13. Operating Cost vs. Pipe Diameter, 5 Mt/y, 30 Mi. System.

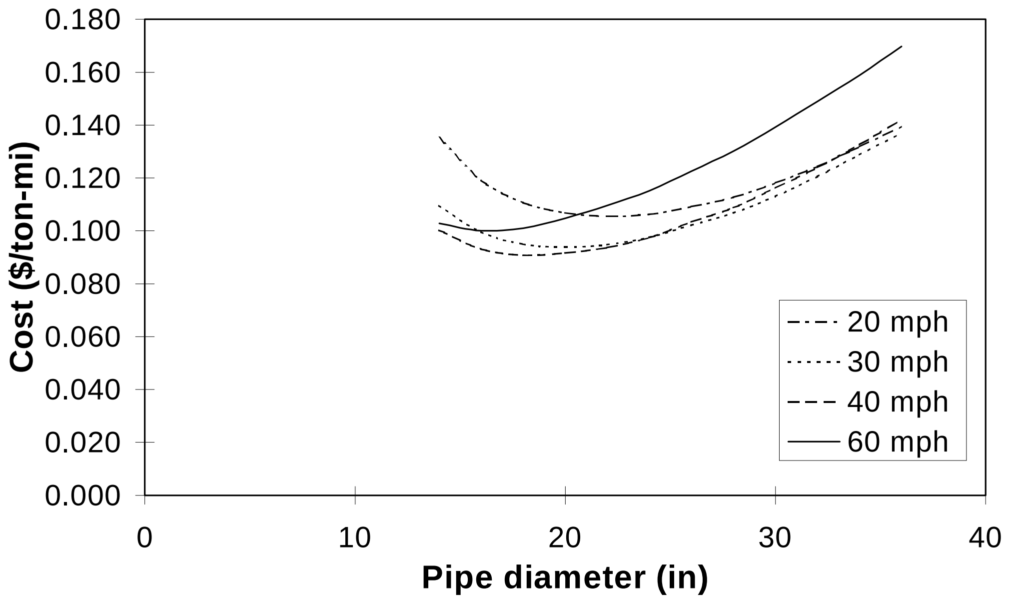

Figure 14. Total System Cost, 5 Mt/y, 30 Mi. System.

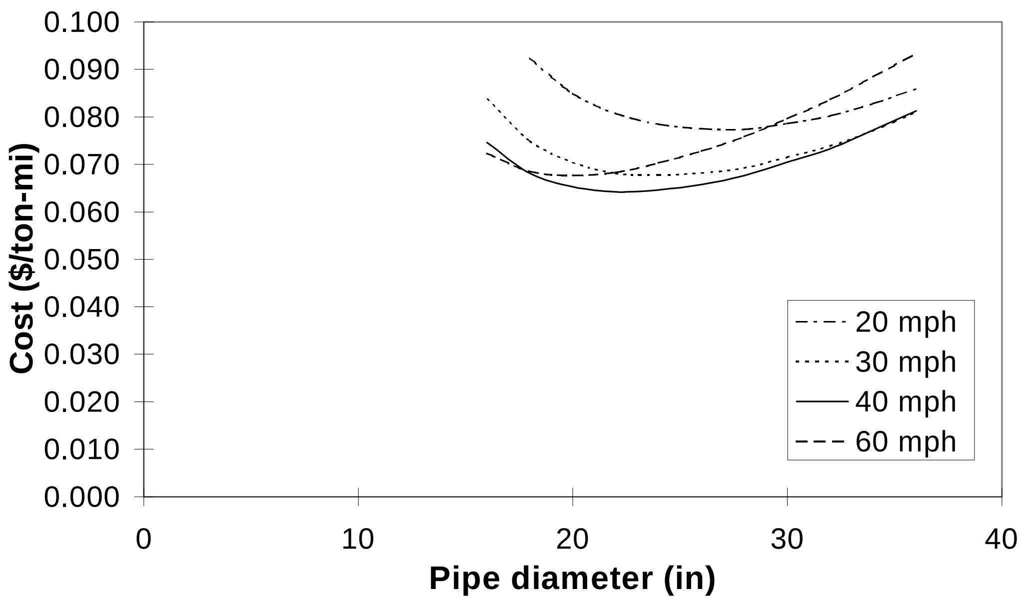

Figure 15. Total System Cost, 10 Mt/y, 30 Mi. System.

Case 2—3 Mile, 1 and 2 Mt/y System

The results for this case show that a 20-inch pipeline at vehicle speeds of 20 mph are the optimum system at 1 Mt/y over 3 miles. This system operates at a total system cost of $0.57 per ton-mile. This rate is substantially higher than that found for the 30 mile, 5 Mt/y system. Because the annual tonnage rate is a relatively low 1 Mt/y, only a 226 vehicles are required for the round trip, and only one load/unload station branch is required at each end of the line. Despite the single branch at each end, the load/unload station costs have become the dominate cost as shown in Table 11. The load/unload station cost is a major reason why shorter haul distances are generally more expensive per ton-mile. The load/unload station costs are minimized at slower vehicle speeds, and thus, the optimum speed is 20 mph. Since 20 mph is the optimum speed in the 20-60 mph range, 10 mph was also evaluated. The results show, however that the 10 mph curve lies between the 20 and 30 mph curves, and 20 mph results in the lowest capital cost. To explain this, note that at 10 mph, the vehicle count would double relative to 20 mph, and this cost increase, together with the accompanying magnet cost increase, is sufficient to more than offset the decreased cost of the load/unload stations. Capital and operating costs for the optimum system, designed around the 20-inch pipeline and 20 mph speed, are summarized in Table 11.

Table 11. Cost Summary for Optimum 3 Mi., 1 Mt/y System.

Capital costs	($M)
vehicles	0.2
magnet assemblies	0.1
motor winding assem.	0.6
rescue winding system	0.0
back iron	0.1
power units (outgoing)	0.1
power units (return)	0.1
block control units	0.0
central control	0.5
pipeline	1.6
load/unload stations	2.2
Total capital cost	5.5

Operating costs	($M/y)
Power	0.11
Maintenance	0.17
Labor	0.70
Total Operating Cost	0.97
$/ton	0.97

Table 12. Cost Summary for Optimum 3 Mi., 2 Mt/y System.

Capital costs	($M)
vehicles	0.4
magnet assemblies	0.2
motor winding assem.	0.6
rescue winding system	0.0
back iron	0.1
power units (outgoing)	0.1
power units (return)	0.1
block control units	0.0
central control	0.5
pipeline	1.6
load/unload stations	2.8
Total capital cost	6.4

Operating costs	($M/y)
Power	0.13
Maintenance	0.19
Labor	0.70
Total Operating Cost	1.02
$/ton	0.51

As a matter of interest, if the 1 Mt/yr system were designed around a 24” pipeline rather than the optimum 20-inch size, the optimum speed is still 20 mph, and the total system cost increases by 11%, but the system limit capacity increases by 54%.

If the annual tonnage is raised to 2 Mt/y, the optimum pipeline diameter remains at 20 inches, the optimum speed remains at 20 mph, but the payback period drops to 1.1 years. Vehicle count is increased to 450, and the number of parallel load/unload station branches has increased to two per end. Total system cost drops to $0.35/ton-mile. Capital and operating costs are summarized in Table 12.

Case 3—10 Mile, 3 and 6 Mt/y System

For the 10 mile, 3 Mt/y case, the results show the optimum system at a pipeline diameter of 18 inches, a speed of 30 mph and a total system cost of $0.16/ton-mile. Although the optimum speed is 30 mph, operating at 40 mph is almost identical in capital and operating cost. The 30 mph system requires 1,932 round trip vehicles and 4 parallel load/unload station branches per end. Costs are summarized in Table 13.

Table 13. Cost Summary for Optimum 10 Mi., 3 Mt/y System.

Capital costs	($M)
vehicles	2.1
magnet assemblies	1.1
motor winding assem.	2.2
rescue winding system	0.0
back iron	0.4
power units	0.4
power units (return)	0.2
block control units	0.1
central control	0.5
pipeline	6.3
load/unload stations	2.6
Total capital cost	15.7

Operating costs	($M/y)
Power	0.37
Maintenance	0.47
Labor	0.78
Total Operating Cost	1.62
$/ton	0.54

Table 14. Cost Summary for Optimum 10 Mi., 6 Mt/y System.

Capital costs	($M)
vehicles	3.8
magnet assemblies	1.9
motor winding assem.	2.4
rescue winding system	0.0
back iron	0.4
power units (outgoing)	0.6
power units (return)	0.3
block control units	0.1
central control	0.5
pipeline	7.5
load/unload stations	3.1
Total capital cost	20.6

Operating costs	($M/y)
Power	0.61
Maintenance	0.62
Labor	0.85
Total Operating Cost	2.07
$/ton	0.35

If the throughput is doubled to 6 Mt/y, the pipeline diameter of the optimum increases to 24 inches, the optimum vehicle speed is 30 mph, and the total system cost is $0.10/ton-mile. The 6 Mt/y system requires 1,426 vehicles and a total of 4 parallel load/unload station branches per end. Costs are summarized in Table 14.

Case 4—100 Mile, 25 and 50 Mt/y System

The optimized 100 mile, 25 Mt/y system, requires 39,018 vehicles operating in a 24-inch pipeline at 60 mph. A total of 14 parallel end-station branches are required at each end to load/unload the throughput. Total system cost is $0.040/ton-mile. With these results, there is a word of caution: the capital investment of $280 M represents a major undertaking, and the production of the 39,000 vehicles in an efficient and timely manner would require both planning and serious negotiation with any potential fabricator. In fact, it is perhaps correct to question the validity of the model for such a huge scale-up from the production levels at which the cost bases have been established. Should such system be seriously considered, new cost bases should be obtained after at least some preliminary price discussions with potential fabricators. This work is outside the scope of this memo. The costs are simply presented in Tables 15 (25 Mt/y) and 16 (50 Mt/y), as the model projects them without further justification.

Table 15. Cost Summary for Optimum 100 Mi., 25 Mt/y System.

Capital costs	($M)
vehicles	78.0
magnet assemblies	39.0
motor winding assem.	23.8
rescue winding system	0.0
back iron	4.5
power units (outgoing)	29.9
power units (return)	15.5
block control units	2.0
central control	0.5
pipeline	74.8
load/unload stations	11.8
Total capital cost	279.8

Operating costs	($M/y)
Power	28.81
Maintenance	8.39
Labor	4.38
Total Operating Cost	41.58
$/ton	1.66

Table 16. Cost Summary for Optimum 100 Mi., 50 Mt/y System.

Capital costs	($M)
vehicles	120.1
magnet assemblies	60.0
motor winding assem.	29.7
rescue winding system	0.0
back iron	5.6
power units (outgoing)	48.9
power units (return)	20.0
block control units	2.3
central control	0.5
pipeline	116.8
load/unload stations	15.6
Total capital cost	419.6

Operating costs	($M/y)
Power	46.7
Maintenance	12.6
Labor	5.1
Total Operating Cost	64.3
$/ton	1.29

If the throughput is doubled to 50 Mt/y, the system, requires 46,182 vehicles operating in a 30-inch pipeline at 60 mph.. A total of 18 parallel end-station branches are required to load/unload the throughput. Total system cost is $0.03/to

Case 5 – 27 Mile, 3 and 8 Mt/y System

At any design point, costs may be compared with those of a competitor, which is often a railroad.

This case addresses ore transport for Client A over a distance of 27 miles using the Electromagnetic Transport System (ETS). Two yearly tonnage options were evaluated, one at 3 Mt/y, and a second at 8 Mt/y, which would address a potential cooperative expansion of the system with Client B. This study indicates that ETS is an attractive alternative to the railroad at 8 Mt/y, with a payback period of only 3.6 years, but is probably unattractive at the lower annual tonnage of 3 Mt/y, with a payback period of 10.5 years.

This study addresses an initial yearly haul of 3 Mt/y, with a potential for future expansion to 8 Mt/y in a possible cooperative effort with Client B. Consequently, this cost evaluation examines ETS transport over the 27 mile run at both ends of the payload range. Costs are compared with an estimate from the railroad for $7 M at the 3 Mt/y rate, or $2.33/ton. In all cases, ETS vehicle speeds are assumed to be 40 mph, hopper length is 4 feet, and pipelines are two-way. In addition, considering the client's relatively near-term need (operational in 2004), we only evaluate costs for our baseline system which operates with pipeline and hopper diameters of 24" and 20", respectively, where much of the engineering is already complete.

Whether the haul is by ETS or railroad, loading and unloading facilities must be provided at the ends of the run. These costs are assumed to be common and are therefore not included in the cost comparison. The ETS costing algorithm does, however, include a line item for the load/unload station costs, and these costs are included for reference.

A total of 1906 vehicles are required, excluding spares, to transport 3 Mt/y over the 27 mile distance. Capital cost for this system, excluding load/unload stations, is $36.8 M. Operating costs are estimated at $3.6 M, or $1.20/ton. Should the system capacity be expanded in the future to 8 Mt/y, an additional 3170 vehicles would be added for a total vehicle count of 5076. Capital costs would increase to $47.7 M. Operating costs for the higher capacity are $5.3 M or $0.66/ton.

Details of these costs are summarized in Tables 17 and 18. Table 17 also shows the costs that would be charged by the railroad for comparison. These costs were provided at $7 M/y for 3 Mt/y or $2.33/t. The same cost per ton is applied to the 8 Mt/y case for the railroad.

Table 17. Operating Cost.

Cost Element	Capacity		Railroad	
	3 Mt/y	8 Mt/y	3 Mt/y	8 Mt/y
Power	1.4	2.5		
Labor	0.9	1.2		
Maintenance	1.2	1.6		
Total	3.5	5.3	7.0	18.64
Total tonnage per year	3.0	8.0	3.0	8.0
$/ton	1.16	0.66	2.33	2.33

Table 18. Capital Cost.

Cost Element	Capacity	
	3 Mt/y	8 Mt/y
Vehicles	3.8	10.2
Pipeline/track	20.2	20.2
Magnets	1.9	5.1
Windings+iron	7.6	7.6
Power units	2.7	3.9
Block control	0.1	0.3
Central control	0.5	0.5
Total	36.8	47.7
Load/unload*	2.8	4.4

*Not included in capital cost for this comparison.

Although the load and unload station costs are not included in the cost comparison, they are noted below Table 18 for reference.

Since the operating costs are well below the costs that would be charged by the railroad, we are left with room for capital recovery. For the 3 Mt/y case, we are left with $3.5 M/y to recover our capital investment of $36.8 M. This corresponds to a payback period of 10.5 years and a capital recovery rate of 7.1% over 20 years. At 8 Mt/y, ETS is far more competitive, with a payback period of 3.6 years and a capital recovery rate of 27.8%.

In conclusion, at 3 Mt/y over 27 miles with a payback period longer than 10 years, the ETS is probably not competitive with the railroad. However, at the higher annual tonnage of 8 Mt/y, ETS is extremely competitive, offering a payback period of only 3.6 years and a 20 year capital recovery rate of 28%.

Case 6 – 34 Mile, 8.3 Mt/y System

This case presents what might be a typical railroad charge of $3.62 per ton over a 34 mile distance for an 8.3 Mt/y haul. Since this tonnage is close to the 10 Mt/y case, it will be used for comparison. The Electromagnetic Transport System (ETS) economic model shows that a 34 mile, 10 Mt/y system using a 24-inch pipeline and operating at 40 mph has a capital cost of $70.4 M, and an operating cost of $7.2 M/y. Note that the capital cost for this ETS converts to $1.45/ton if annualized over 20 years at 20%, and the operating cost is $0.72/ton. Together, these amount to $0.064/ton-mile. The ETS cost of $0.064 compares quite favorably with the railroad charges in this case, which amount to $0.106/ton-mile. The ETS is even more attractive when one realizes that the $0.064/ton-mile cost has already applied a 20% capital recovery factor to the capital cost. A more fair comparison would simply calculate the payback period for the ETS as if it were to replace the railroad as the transport system. The payback period is calculated by dividing the ETS capital cost ($70.4 M) by the difference in the annual cost of operating the railroad, ($3.62/ton)(10 Mt/y) = $36.20 M/y, vs. the ETS ($0.72)(10 Mt/y)=$7.20 M/y, or a net annual savings of $29 M/y. This savings will payback the capital cost of $70.4 M in only 2.43 years, implying that a capital recovery rate of 41% over 20 years could be realized and still achieve breakeven with the railroad. This result is even more remarkable when one considers that the ETS capital cost includes $5 M for load/unload stations, at least some of which would also have to be incurred for the rail-based system, but is probably not included in the $3.62/ton rate.

ETS costs are based on the reference design, which employs a 24” diameter pipeline and a 48” long, 20” diameter hopper in each vehicle. A total of 6624 vehicles are required to haul 8.3 Mt/y at a launch interval of one 3-vehicle cluster every 2.8 s. The capital costs for this system are summarized in Table 19, with the load/unload station costs listed separately. The total ETS cost of $60.8 M, excluding load/unload stations, is compared with a 1993 proposal for a new client-owned railroad which was estimated to cost $68.1 M. Using Bureau of Labor and Statistics inflation (CPI) data, this cost escalates by 13% to $77.0 M in 1998. The client-owned railroad estimate included costs for land acquisition, permitting, support equipment and contingency. These items were assumed to be unnecessary for the ETS transport system, and are not included in the ETS capital cost estimate.

Table 20 presents the estimated operating cost for the ETS, excluding amortization of capital costs. These costs are compared with those estimated for the client-owned railroad, and with the charges per ton that were being paid at the time to the CSX railroad for transport. Again, both client and CSX costs are escalated by 13% to get to 1998 dollars.

Although the operating cost ($0.78/ton) for the ETS is somewhat greater than the operating cost ($0.69/ton) for the client-owned railroad, the ETS capital costs are lower, so these two systems could be considered as approximate economic equals. The client-owned railroad option would pay itself off against the ETS after 23 years of operation. If the operations to port were to be continued for less than 23 years, then the ETS would be the slightly more economical choice.

Table 19. Capital Cost Comparison, New Client-Owned Railroad.

Capital cost ($M)	ETS	IMC-owned RR
Vehicles	13.2	
Pipeline/track	25.4	
Magnets	6.6	
Windings+iron	9.6	
Power units	5.1	
Block control	0.3	
Central control	0.5	
Total	60.8	77.0

Load/unload	4.4

Table 20. Operating Cost Comparison, Client-Owned and CSX Railroads.

Operating cost excluding capital recovery ($M/y)			
	ETS	IMC-owned RR	CSX RR
Power	3.2		
Fuel		0.6	
Labor	1.3	0.8	
Maintenance	2.0		
Cars		1.4	
Track		3.1	
Total	6.5	5.8	30.0
Total tonnage per year	8.3	8.3	8.3
$/ton	0.78	0.69	3.62

The ETS competes very favorably with the CSX Railroad. The difference in annual operating costs are substantial at ($30 M - $6.5 M)=$23.5 M. This implies that the $60.8 M capital cost for the ETS, excluding load/unload stations, could be paid back in only 2.6 years, which gives a very attractive capital recovery rate of 38% over 20 years. If the load/unload station costs are included in the capital costs, the \$65.2 M capital costs are paid back after 2.8 years, and the capital recovery rate drops slightly to 36%, which is still very attractive.

In conclusion, if the ETS is compared against the client-owned railroad, the two potential systems are judged to be approximate economic equals, with the ETS being slightly less expensive if mining operations last fewer than 16 years. On the other hand, the ETS is extremely attractive when compared against the CSX railroad.

Case 7 – 1-5 Miles; 8 Mt/yr System

In this case a parametric study is done of a typical phosphate matrix transport distance and annual tonnage application. This case was done to explore if the matrix application would be economic under various assumptions, and therefore possibly justify future R&D. We use the simplifying assumption that the load/unload stations would be similar in cost to the dry product stations, while recognizing that they would need to be of a different concept. To the extent that they were to be more expensive in order to handle the more difficult product, the economics would be less favorable.

Unlike the previous six cases which used the Option A unit cost assumptions, this case uses the Option B assumptions. The matrix product is assumed to have a density of 125 lbs/ft^3.

The change from Option A to Option B unit cost assumptions are summarized below:

- *Motor winding assemblies*—cost is reduced to a factor of 0.3 times the previous cost based on the assumption of using automated winding equipment which will reduce the net winding cost when spread over larger production quantities.
- *Pipeline*—cost is reduced to a factor of 0.77 times the previous cost based on discussions with the pipe vendor who provided cost buying pipe in larger quantities.
- *Load-unload stations*—cost is reduced by a factor of 0.5 times the previous cost based on unloading the vehicles on the fly. Load station cost basis is unchanged.
- *Labor*—The fixed portion of the labor cost is reduced from \$0.7M/year to \$0.5M/year based on lower cost of living and generally lower average salaries at the installation location, together with an assumption of lower fixed management cost for a relatively short-length system.
- *Power*—Cost of power is reduced from \$0.12/kwh to \$0.04/kwh, which is the generally prevailing rate at the installation location.

Optimizations were performed at the 1, 2.5 and 5 mile distances. The component cost breakdowns for the optimum in each case as well as for the reference 24-inch pipe are given in the following tables for the three distances.

Table 21. Cost Summary for 1 Mile, 8 Mt/y Matrix System.

	24" Pipe		30" Pipe	
	Cost ($M)	%	Cost ($M)	%
Capital cost				
Vehicles	0.65	19	0.49	16
Magnet assemblies	0.29	8	0.22	7
Motor winding assem.	0.07	2	0.09	3
Rescue winding system	0.00	0	0.00	0
Back iron (optional)	0.04	1	0.06	2
Power units (outgoing)	0.06	2	0.06	2
Power units (return)	0.02	1	0.02	1
Block control units	0.02	0	0.01	0
Central control	0.50	15	0.50	17
Pipeline	0.58	17	0.90	30
Load/unload stations	1.21	35	0.66	22
	3.43		3.01	

	24" Pipe	30" Pipe
	($M/y)	($M/y)
Annualized capital cost at 20%, 20 y	0.71	0.62
Operating cost		
Power	0.03	0.03
Maintenance	0.10	0.09
Labor	0.50	0.50
	0.63	0.62

Table 22. System Characteristics for 1 Mile, 8 Mt/y Matrix System.

Number of vehicles (two-way)	170
Vehicle cluster size	3
Cluster launch interval(s)	6.87
Average power requirement (kw)	77
Payload per vehicle (lb)	1290
Time utilization	0.9
Line fill factor	10%
Number of parallel loading stations	2

Table 23. Cost Summary for 2.5 Mile, 8 Mt/y Matrix System.

	24" Pipe, 30 mph		26" Pipe, 30 mph	
	Cost ($M)	%	Cost ($M)	%
Capital cost				
Vehicles	1.05	19	0.95	18
Magnet assemblies	0.47	9	0.42	8
Motor winding assem.	0.18	3	0.19	4
Rescue winding system	0.00	0	0.00	0
Back iron (optional)	0.11	2	0.12	2
Power units (outgoing)	0.18	3	0.18	3
Power units (return)	0.07	1	0.07	1
Block control units	0.03	0	0.02	0
Central control	0.50	9	0.50	9
Pipeline	1.44	27	1.69	31
Load/unload stations	1.55	29	1.28	24
Total	5.57		5.42	

	24" Pipe, 30 mph	26" Pipe, 30 mph
	($M/y)	($M/y)
Annualized capital cost at 20%, 20 y	1.14	1.11
Operating cost		
Power	0.08	0.08
Maintenance	0.17	0.16
Labor	0.58	0.50
Total	0.82	0.74

Table 24. System Characteristics for 2.5 Mile, 8 Mt/y Matrix System.

Number of vehicles (two-way)	386
Vehicle cluster size	3
Cluster launch interval(s)	4.92
Average power requirement (kw)	217
Payload per vehicle (lb)	924
Time utilization	0.9
Line fill factor	9%
Number of parallel loading stations	3

Table 25. Cost Summary for 5 Mile, 8 Mt/y Matrix System.

	22" Pipe, 40 mph		24" Pipe, 30 mph	
	Cost ($M)	%	Cost ($M)	%
Capital cost				
Vehicles	1.70	20	2.03	23
Magnet assemblies	0.76	9	0.91	10
Motor winding assem.	0.33	4	0.36	4
Rescue winding system	0.00	0	0.00	0
Back iron (optional)	0.21	2	0.22	2
Power units (outgoing)	0.45	5	0.35	4
Power units (return)	0.24	3	0.14	2
Block control units	0.05	1	0.05	1
Central control	0.50	6	0.50	6
Pipeline	2.42	28	2.88	32
Load/unload stations	1.93	23	1.55	17
Total	8.58		9.00	

	22" Pipe, 40 mph	24" Pipe, 30 mph
	($M/y)	($M/y)
Annualized capital cost at 20%, 20 y	1.76	1.85
Operating cost		
Power	0.18	0.14
Maintenance	0.26	0.27
Labor	0.58	0.58
Total	1.01	0.99

Table 26. System Characteristics for 5 Mile, 8 Mt/y Matrix System.

Number of vehicles (two-way)	846
Vehicle cluster size	3
Cluster launch interval(s)	3.29
Average power requirement (kw)	519
Payload per vehicle (lb)	619
Time utilization	0.9
Line fill factor	10%
Number of parallel loading stations	4

The 1-mile system has a new cost minimum of $0.154/ton-mile, and this requires a 30” diameter pipeline, a total of 170 vehicles operating at an average speed of 20 mph, and 2 parallel loading stations. The 5-mile system has a cost minimum of $0.069/ton-mile and requires a 22” diameter pipeline, at total of 846 vehicles operating at an average speed of 40 mph, and 4 parallel load/unload stations at each end. For comparison, the same distances were run for our current 24” pipeline design. Total cost (annualized capital + operating) using the currently designed 24” pipeline run about 8% higher at 1 mile, but only about 3% higher at 5 miles. Capital costs are annualized at 20% over 20 years.

The electromagnetic pipeline matrix transport system can also be compared against competitive costs of the slurry pipeline now widely used in the industry. IMC Agrico considers the slurry pipe line operating costs to be $0.11/ton-mile. Against this competitive target, our results indicate that at a distance of 2.5 miles a rate of return of 22% can be achieved, and at 5 miles, 45%. At a one mile distance, where the loading station dominates the cost, it is not possible to provide a positive rate of return. The operating cost without capital recovery, however, would be at breakeven with the slurry pipe approach.

These results are clearly dependent on the simplifying assumptions made which not only assume that the load/unload stations are similar in cost to the dry product stations, but that matrix loading and unloading would in fact prove practical. To the extent the stations are substantial more expensive, the breakeven distances would become longer.

Consolidation of Case Results

In this section we provide an overview of the previous cases by a consolidation of cost per ton mile versus distance, and of optimum speed.

Cost per Ton-Mile

Figure 16 consolidates the cost per ton-mile costs versus distance, in three discrete yearly tonnage amounts of 2, 5 and 10 Mt/y for a fixed pipeline diameter of 24 inches. The 24-inch pipeline size is close to the optimum in many cases, and, as has already been pointed out, it allows room for economically increasing throughput capability. At each yearly tonnage plotted, the optimum vehicle speed for that tonnage is assumed and is noted in the plot legend. At each tonnage, the total system cost in $/ton-mile starts at a high level at short distances, and then flattens out to nearly constant cost per ton-mile at distances greater than about 15-25 miles. The “knee” in all the curves is below 10 miles.

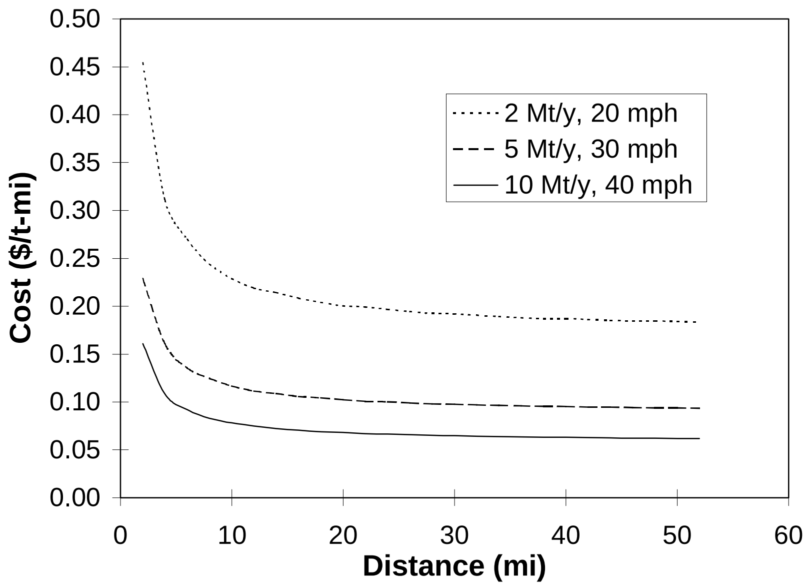

Figure 16. Total System Cost vs. Distance at Different Tonnage Rates.

Optimum Speed

The economic model will identify the optimum combination of pipe diameter and vehicle speed for any given set of capacity, length of travel and cost basis assumptions. There is generally a fairly broad minimum. For example, Table 27 gives the relative total system cost as a function of speed for 20, 30 and 40 MPH. For the high-tonnage, long-haul case 1, the minimum speed occurs at 40 MPH, but the cost at 30 MPH is only 6% higher. For the low-tonnage, short-haul case 2, the minimum cost is reached at 30 MPH, but the cost at 20 MPH is only 6% higher. In both cases 3 and 7, the minimum occurs at 30 MPH, but the penalty of operation at 20 MPH is less than 8%.

Table 27. Optimum Speed.

Case	Capacity (Mt/yr)	Distance (miles)	Speed (MPH)	Cost Relative to Minimum Cost ($/ton-mile)
Case 1	10	30	20	1.17
			30	1.06
			40	1.0
Case 2	2	3	20	1.06
			30	1.0
			40	1.0
Case 3	3	10	20	1.03
			30	1.0
			40	1.07
Case 7	8	2.5	20	1.08
			30	1.0
			40	1.03

Optimum Line Fill

The fill factor (the percent of the tube occupied by vehicles) varies for the different cases. In short hauls, where the cost of handling the vehicles at the end stations is an important component of the cost, the optimum fill factor is lower than in long-haul cases. The 3 mile short-haul Case 2, for example, has an optimum fill factor of 7%. Case 1 at 30 miles, on the other hand, has an optimum fill factors of 18%. We would not expect to find fill factors above 20% for any cases of interest to the industry.

Cost Basis Sensitivity

Introduction

Parametric studies made in the previous section used common cost basis assumptions which are identified as "Option A" in Appendix A. In this section we fix the pipeline parameters at the prototype scale (24-inch diameter pipe), and instead vary the cost basis assumptions. Major capital cost components include pipeline, vehicles, magnet assemblies, windings and load/unload stations.

The elements of operating cost include insurance and property taxes, power, material costs for maintenance and labor costs for operating and maintaining the system. The model minimizes total system cost, which we define as the sum of the annualized capital cost plus the operating cost, by varying system parameters such as pipe diameter and vehicle speed.

Calculation of the annualized capital cost requires a choice of a minimum attractive rate of return and a time over which the return will be realized. In Option A we fixed these at 20% and 20 years on a before-tax basis as illustrative.

In this section we compare our results against specific competitive targets, and have calculated the after-tax rate of return that would be realized over a ten year period if the new system were installed. This memo also introduces various modifications to the Option A cost basis. The after-tax rate of return for several of the cases discussed below are listed in Table 28.

Table 28. After-Tax Rate of Return.

	Competition	Rate of Return
Long haul with Option A unit cost assumptions	Long-haul trucks	29.2%
Long haul with Option B unit cost assumptions	Long-haul trucks	44.4%
Long haul with Option B unit cost assumptions	Long-term RR contracts	15.7%
Short-haul with Option A unit cost assumptions	Short-haul trucks	11.5%
Short-haul with Option B unit cost assumptions	Short-haul trucks	28.9%
Short-haul combined with long-haul project, both with Option B unit cost assumptions	Short- and long-haul trucks	42.9%

Examples from the Parametric Studies

Cases 1 and 2 in the previous section treated a generic long-haul, high-tonnage and a generic short-haul, low-tonnage installation . The former could represent transport of finished product from Lakeland to the Port of Tampa, and the latter, in-process material from Kingsford to New Wales. The tables below are taken from the capital and operating cost numbers in the earlier examples, which were derived from the unit costs and assumptions in Option A. The next section will examine the impact of changes in those assumptions.

Table 29 summarizes the Case 1 long-haul, high-tonnage base case with the Option A unit cost assumptions. The capital cost is $61.2M, with the pipeline and the vehicles being the dominant costs. The annual operating cost, exclusive of capital recovery, is $6.7M, with the dominant cost being electric power.

The $6.7M annual operating cost on a ton-mile basis is $0.022/ton-mile. A typical competitive long-haul truck cost is estimated by IMC-Agrico to be $0.08/ton-mile, leaving a reasonable margin for capital recovery. The after-tax rate of return shown in Table 29 is 29.2%. This would probably be considered attractive for a long-term infrastructure investment of this nature. An example illustrating the calculation of after-tax rate of return is given in Appendix A.

Under the Option A unit cost assumptions used in the model, competition with a typical existing railroad long-term contract would be more difficult. Such a contract is estimated by IMC Agrico to be in the $0.05-0.07/ton-mile range; at the lower end 0.$05/ton-mile rate, the rate of return on the capital would be only 6.2%.

Table 29. Case 1 from Parametric Sensitivity.

Distance (miles)	30	
Tonnage (Mtons/yr)	10	
Capital Cost	$M	% of Total
Pipeline	18.8	30.7
Vehicles	15.6	25.5
Magnet assemblies	7.8	12.7
Motor windings	7.7	12.6
Load/unload stations	5.5	9.0
Power units & control	5.8	9.5
Subtotal	61.2	100.0
Annual Operating Cost	$M	% of Total
Power	3.3	49.3
Maintenance	1.8	27.7
Labor	1.5	23.0
Subtotal	6.7	100.0

	$M	$/ton-mile
Annual Operating Cost	6.7	0.022
Competitive Target *	24.0	0.08
Savings available	17.4	
Pre-Tax Rate of Return	37.1%	
After-Tax Rate of Return	29.2%	
Payback period	2.95 years	

Table 29. Case 1 from Parametric Sensitivity (Cont.).

	$M	$/ton-mile
Annual Operating Cost	6.7	0.022
Competitive Target +	15.0	0.05
Savings available	8.4	
Pre-Tax Rate of Return	7.2%	
After-Tax Rate of Return	6.2%	
Payback period	6.5 years	

* long-haul truck transport + long-term RR contract transport

Table 30 summarizes the short-haul, low-tonnage Case 2 from the parametric studies. Here the capital cost is $6.4 M, with the dominant cost being the load/unload stations. The annual operating cost, exclusive of capital recovery, is $1.02M, with the dominant cost being labor costs. This operating cost on a ton-mile basis is $0.17/ton-mile, considerably higher than that for the long-haul case.

Table 30. Case 2 from Parametric Sensitivity.

Distance (miles)	3	
Tonnage (Mtons/yr)	2	
Capital Cost	$M	% of Total
Pipeline	1.6	25.0
Vehicles	0.4	6.3
Magnet assemblies	0.2	3.1
Motor windings	0.7	10.9
Load/unload stations	2.8	43.8
Power units & control	0.7	10.9
Subtotal	6.4	100.0

Annual Operating Cost	$M	% of Total
Power	0.13	12.7
Maintenance	0.19	18.6
Labor	0.70	68.6
Subtotal	1.02	100.0

Table 30. Case 2 from Parametric Sensitivity (Cont.).

	$M	$/ton-mile
Annual Operating Cost	1.02	0.17
Competitive Target **	2.10	0.35
Savings available	1.08	
Pre-Tax Rate of Return	14.1%	
After-Tax Rate of Return	11.5%	
Payback period	5.2 years	

** short-haul truck transport

A typical competitive short-haul truck cost is estimated by IMC-Agrico to be $0.35/ton-mile, also larger than a long-haul truck cost. The available margin results in an after-tax rate of return of 11.5%. While not as attractive as the long-haul case, it might be acceptable when the project is partially justified as a development phase of the long-haul system. We consider a case in Table 34 where the short-haul demonstration project is actually combined with the financial considerations of the long-haul project.

Impact of the Option B Unit Cost Basis Assumptions

The rates of return for both the long-haul and short-haul cases using the Option A cost basis assumptions can be improved by use of the Option B unit cost assumptions given in Appendix A. The differences in the cost basis between the two options are summarized in Table 31 for a base case, 24-inch diameter pipe system.

Table 31. Cost Basis Option Summary.

	Option A Unit Cost Assumptions	Option B Unit Cost Assumptions
1.0 Vehicles	$2,000/vehicle	unchanged
2.0 Magnet Assemblies	$1,000/vehicle	unchanged
3.0 Motor Windings	$6,750/module	$2,000/module
4.0 Rescue Windings	not used	option to include @ 0.33 x motor winding module
5.0 Back Iron	$1,400/ module	unchanged
6.0 Power Units	$1.00/watt	unchanged
7.0 Block Control Units	$2,500	unchanged
8.0 Central Control	$500,000	unchanged
9.0 Pipeline		
9.1 Right-of-Way	not included	$30,000/acre developed land $10,000/acre undeveloped 50 foot ROW
9.2 Pipe Cost	$109/two-way foot	30% reduction
10.0 Load/Unload Stations	Stationary load and unload	Option to use on-the-fly unload
11.0 Cost per ton	Included both operating cost and capital recovery cost at a fixed 20% rate of return for 20 years; computation on a before-tax basis.	Includes only operating cost; the rate of return is calculated from projected savings; computation includes taxes, depreciation, and future discount rate.
12.0 Operating Cost		
12.1 Cost of Electricity	$0.12/kWh	$0.04/kWh
12.2 Maintenance	3% of capital annually	unchanged
12.3 Labor	Fixed Operating Crew @ $700,000; additional techs for maintenance	long haul - unchanged short haul - $500,000 of operating crew; no additional techs for maintenance
12.4 Insurance/property tax	not included	1.5% of capital

The long-haul and short-haul cases' baseline are repeated below using combinations of the Option B assumptions. The pipe diameter and vehicle speed, however, is held constant rather than re-optimized in the cost model as would be done if one were “fine tuning” the design.

Table 32. Long-Haul Case 1 from Parametric Sensitivity, with Option B Assumptions.

Distance (miles)	30	
Tonnage (Mtons/yr)	10	
Capital Cost	$M	Savings
Right of way	2.7	-2.7
Pipeline	13.2	5.6
Vehicles	15.6	0
Magnet assemblies	7.8	0
Motor windings	2.3	5.4
Load/unload stations	2.8	2.7
Power units & control	5.8	0
Subtotal	50.1	11.1

Annual Operating Cost	$M	Savings
Insurance and property tax	0.8	-0.8
Power	1.1	2.2
Maintenance	1.8	0
Labor	1.5	0
Subtotal	5.2	1.4

	$M	$/ton-mile
Annual Operating Cost	5.2	0.017
Competitive Target *	24.0	0.08
Savings available	18.8	
Pre-Tax Rate of Return	59.1%	
After-Tax Rate of Return	44.4%	
Payback period	2.1 years	

	$M	$/ton-mile
Annual Operating Cost	5.2	0.017
Competitive Target +	15.0	0.05
Savings available	9.8	
Pre-Tax Rate of Return	19.4%	
After-Tax Rate of Return	15.7%	
Payback period	4.5 years	

* long-haul truck transport + long-term RR contract transport

As summarized in Table 32, the inclusion of all the Option B assumptions reduces the capital cost of the long-haul, high-tonnage case from $61.2M to $50.1M, and the operating cost from $6.7M to $5.2M. This combination of reductions increases the rate of return from 29.2% to 44.4% against long-haul truck competition. It also predicts a more reasonable rate of return of 15.7% against existing railroad long-term contract competition. The dominant capital cost savings come from the pipeline and motor winding components of cost. Inclusion of the on-the-fly unload option contributes about 5% to the overall reduction. The dominant operating cost reduction comes from the reduction in the cost of electricity.

Table 33. Short-Haul Case 2 from Parametric Sensitivity for Option B.

Distance (miles)	3	
Tonnage (Mtons/yr)	2	
Capital Cost	$M	Savings
Right of way	0.00	0
Pipeline	1.12	0.48
Vehicles	0.40	0
Magnet assemblies	0.20	0
Motor windings	0.21	0.49
Load/unload stations	2.03	0.77
Power units & control	0.70	0
Subtotal	4.66	1.74

Annual Operating Cost	$M	Savings
Insurance and property tax	0.07	-0.07
Power	0.04	0.09
Maintenance	0.19	0
Labor	0.49	0.21
Subtotal	0.79	0.23

	$M	$/ton-mile
Annual Operating Cost	0.79	0.13
Competitive Target **	2.10	0.35
Savings available	1.31	
Pre-Tax Rate of Return	36.8%	
After-Tax Rate of Return	28.9%	
Payback period	3.0 years	

** short-haul truck transport

As summarized in Table 33, the inclusion of all the Option B assumptions reduces the capital cost of the short-haul, low-tonnage case from $6.4M to $4.7M, and the operating cost from $1.02M to $0.79M. The rate of return increases from 11.5% to 28.9%. The most significant capital cost savings come from inclusion of the on-the-fly unload option which contributes about 15% to the cost reduction. In the short-haul case, the load and unload stations are a more significant fraction of the cost than they are for the long-haul case. The most significant reduction in operating cost derives from the reduction in labor cost.

If the short-haul installation is undertaken in the context of a development project to support the long-haul project, a more favorable economic picture can result from combining the two projects as shown in Table 38. The rate of return for the combined projects is 42.9%, only slightly less than the 44.4% from a stand-alone long-haul project, but substantially higher than the 28.9% for the stand-alone short-haul project.

Table 34. Combined Projects.

	M$
Capital Cost	
Long Haul table 4.0	50.1
Short Haul table 5.0	4.7
total	54.8
Operating Cost	
Long Haul table 4.0	5.2
Short Haul table 5.0	0.8
total	6.0
Savings Available	
Long Haul table 4.0	18.8
Short Haul table 5.0	1.3
total	20.1
Pre-Tax Rate of Return	56.9%
After-Tax Rate of Return	42.9%
Payback period	3.2 years

Cost Sensitivity Issues

Unload-on-the-Fly Option. The results in Tables 32 and 33 were based on including all of the Option B cost basis assumptions including the option of on-the-fly unloading. In order to help assess whether further development of the on-the-fly unload option is useful, we repeat two of the previous cases without the on-the fly option. Table

35 shows the long-haul case, where the after tax rate of return is reduced from 44.4% to 41% against the long-haul trucking competition.

Table 35. Case 1 Without On-the-Fly Unloading.

Distance (miles)	30	
Tonnage (Mtons/yr)	10	
Capital Cost	$M	Savings
Right of way	2.7	-2.7
Pipeline	13.2	5.6
Vehicles	15.6	0
Magnet assemblies	7.8	0
Motor windings	2.3	5.4
Load/unload stations	5.5	0
Power units & control	5.8	0
Subtotal	52.9	8.3
Annual Operating Cost	$M	Savings
Insurance and property tax	0.8	-0.8
Power	1.1	2.2
Maintenance	1.8	0
Labor	1.5	0
Subtotal	5.2	1.4
	$M	$/ton-mile
Annual Operating Cost	5.2	0.017
Competitive Target *	24.0	0.08
Savings available	18.8	
Pre-Tax Rate of Return	54.0%	
After-Tax Rate of Return	41.0%	
Payback period	3.2 years	

Table 36 shows the short-haul case where the rate of return is reduced from 28.9% to 22.7% when stationary rather than on-the-fly unloading is not used.

Table 36. Case 2 Without On-the-Fly Unloading.

Short-haul with all Option B assumptions except on-the fly unloading		
Distance (miles)	3	
Tonnage (Mtons/yr)	2	
Capital Cost	$M	Savings
Right of way	0.00	0
Pipeline	1.12	0.48
Vehicles	0.40	0
Magnet assemblies	0.20	0
Motor windings	0.21	0.49
Load/unload stations	2.80	0
Power units & control	0.70	0
Subtotal	5.43	0.97

Annual Operating Cost	$M	Savings
Insurance and property tax	0.07	-0.07
Power	0.04	0.09
Maintenance	0.19	0
Labor	0.49	0.21
Subtotal	0.79	0.23

	$M	$/ton-mile
Annual Operating Cost	0.79	0.13
Competitive Target **	2.10	0.35
Savings available	1.31	
Pre-Tax Rate of Return	28.5%	
After-Tax Rate of Return	22.7%	
Payback period	3.6 years	

** short-haul truck transport

Rescue Winding Option. None of the cases treated in this section have included the option of rescue windings. These windings would be used to recover from a long time power outage where the vehicles would have coasted to a stop, not necessarily over an existing motor. The system would recover automatically from a short time power outages as the vehicles would still be coasting when the power were restored. Lower-cost options exist for clearing the pipeline after a sustained power outage including motorized “mule” vehicles, and we therefore do not generally include a cost for rescue windings. Table 37, however, shows the impact in the event they were included along with the other Option B cost basis assumptions. The rate of return for the long-haul case is reduced from 44.4% to 32.1%, and of the short haul. from 28.9% to 20.4%.

Table 37. Impact of Rescue Winding Added to Tables 32 and 33.

Long Haul	M$
Capital Cost	61.7
Operating Cost	5.4
Competitive Cost	24.0
Savings Available	18.6
Pre-Tax Rate of Return	41.2%
After-Tax Rate of Return	32.1%
Payback period	3.8 years
Short Haul	M$
Capital Cost	5.71
Operating Cost	0.81
Competitive Cost	2.10
Savings Available	1.29
Pre-Tax Rate of Return	25.4%
After-Tax Rate of Return	20.4%
Payback period	4.8 years

CONCLUSIONS AND RECOMMENDATIONS

Economic studies have shown that a 24-inch pipe diameter is near the optimum scale for applications in the phosphate industry. Economic studies of applications in the phosphate industry indicate that the electromagnetic capsule pipeline systems can be competitive with truck and rail transport.

The tests which have been done to date have establish the basic feasibility of the design. They have also resulted in several iterations of the design, primarily to improve the robustness of the motor control system against system electrical noise, and to alter the vehicle switching approach. The parameters achieved demonstrate that electromagnetic capsule pumps have the potential to significantly reduce the throughput limitations of blower driven capsule pipelines.

While no appreciable life testing has been accumulated, we can anticipate that the joints between pipe sections will require an improved design to avoid fatigue failure of the pipe ends.

While there has been little evidence of wheel or pipe surface wear to date, we can anticipate that it will be necessary to develop a technique for in situ re-coating of the pipe surface to achieve multi-year lifetimes.

While the demonstration tests have demonstrated the basic feasibility, a follow-on project will be required before the technology can be commercially available. One goal of such a project will be to demonstrate that the components can meet the lifetime requirements. A suitable follow-on project could be to replace truck traffic between two near-by processing plants.

REFERENCES

Kosugi S and others. 1992. A capsule pipeline system for limestone transportation. 7th International Symposium on Freight Pipelines.

Liu H. 2000. Improving economics of existing pneumatic capsule pipeline system for transporting general cargos. 2nd International Symposium on Underground Freight Transportation, Delft, The Netherlands.

Montgomery B and others. 1999. Electromagnetic pipeline transport system for the phosphate industry. 1st International Symposium on Underground Freight Transportation, Columbia, Missouri.

Montgomery B and others. 2000. Electromagnetic pipeline demonstration project. 2nd International Symposium on Underground Freight Transportation, Delft, The Netherlands.

Vandersteel W. 1984. Pneumatic pipeline transport system. US Patent # 4,458,602, July 10, 1984.

Appendix A

ORE CAR COST SCALING BASIS

Appendix A

ORE CAR COST SCALING BASIS

VEHICLES

- Assume baseline 20 inch x 48 inch hopper vehicle cost at $2,000.

  The prototype vehicle cost approximately $10K. The vendor who fabricated and assembled the prototype vehicle estimated that if he were to make 250 vehicles they would cost $4.4K each with the tooling amortized over that quantity. We have allowed $2K per vehicle on the assumption that the vehicle could be simplified and designed for ease of manufacture. The smallest capacity system examined (3 megatons/year over 2 miles) would require 500 vehicles. The largest, 10 Megatons/year over 30 miles, would require 8, 600 vehicles.

- Assume vehicle cost scales in proportion to $(\text{payload})^{1/2}$. A quantitative rationale for this scaling is presented for material costs based on a simple assumption of a bending stress limitation in the hopper design, while the rationale for this same scaling relative to the labor costs for vehicle manufacture is only intuitive for now.

  For the material, we assume that the sizing of structural material for the main hopper is governed by bending stress limits in the material. For a cylindrical tank, the payload of the tank is proportional to the volume:

$$M = Ar^2 l$$

where M is the mass of the payload, r is the radius and l is the length of the cylindrical hopper, and A is a proportionality constant. The bending stress of a thin wall tube about an axis perpendicular to its own is given by the standard equation:

$$\sigma_b = \frac{M_b c}{I} = \frac{M_b (r/2))}{tr^3 \pi} = \frac{M_b}{2tr^2 \pi}$$

The bending moment, M_b, is proportional to the payload, however, so we can conclude that the required wall thickness, t, can remain constant as we increase the radius to handle larger and larger payloads in each vehicle. If we assume that the material cost for the hopper are proportional to the cylindrical volume, then material cost is given by

$$C_m = Brlt$$

where B is a second proportionality constant. Thus, we have cost scaling as the first power of radius and payload scaling as the second, so at constant wall thickness and hopper length, material cost scales as the square root of payload.

Although we have no such simple logic for the other costs in the vehicle manufacturing process (i.e., labor), it does seem intuitive that it might not take twice the labor assemble a vehicle with diameter 2d compared that for a vehicle with diameter d.

- Total vehicle cost is vehicle cost x (number of vehicles going and returning plus vehicles in each station).

MAGNET ASSEMBLIES

- Baseline 20 inch x 48 inch hopper vehicle utilizes approximately 80 magnets, each 2" x 2" x 0.5", and each valued at $10, resulting in a cost of $800/vehicle, which has been rounded up to $1,000/vehicle to cover import and shipping costs.

  The cost of the neodymium-iron-boron permanent magnets has been substantially reduced over the last several years by entry of several large Chinese manufacturers who now sell the material in large quantity. For large quantities the manufacturers cost the material at $40/kg. On this basis the 2"x2"x 0.5" basic building block units are $9, rounded up to $10 in our estimate. Eighty blocks are $800, which we round up to $1,000 to include shipping and import duties.

- Assume scaling would be by surface area of the vehicle, hence proportional to vehicle diameter and length.

MOTOR WINDING ASSEMBLIES

- Baseline windings are 18' long and are for a 20" diameter hopper. Vendor cost for a single prototype winding is $13,500.

Option A: Assume that volume productions can reduce the prototype cost by a factor of 2, so use $6,750 per 18' winding.

Option B: Assume that the cost could be reduced substantially by using an automatic winding machine. The prototype motor modules were wound using an entirely hand-layup method on a tooling form. We estimate that in large quantity, the cost using automatic machines could be as low as $2K/module.

Table A-1. Option B Motor Module Cost Breakdown.

Cost Element	Cost ($)
Wire cost (750 pounds)	750
Winding parts (joints, forms)	500
Technician time (10 hours @$50/hr)	500
Amortize automatic winding machine @$250K over 1000 windings	250
Total unit cost	2000

The motor windings are assumed to cover 6% of the total length of the winding. The vehicles coast between motors, where there speed is boosted back to the system peak velocity.

- Assume the cost scales as the surface area, that is with the pipeline diameter (i.e., the winding has constant depth.) The spread sheet allows for less than 100% length coverage.

- The required percent coverage for the baseline vehicles obtained from a separate evaluation of the power. At present it is held constant at about 6%.

RESCUE WINDING ASSEMBLIES

- Our reference design currently assumes that cars are pneumatically coupled, and that those cars which have stopped over unpowered pipeline sections can be moved along by the pneumatic coupling between cars, with some cars getting their initial motion by being stopped over powered sections of pipeline. Therefore, with the reference design, rescue windings are not used.

- Nevertheless, the cost model does allow for the presence of rescue windings. In this case where the main motor windings must occupy less than 100% of the length, and it is assumed that the remaining length is covered by a lesser winding whose function is only to act as a low speed stepping motor to advance cars stalled after a power outage to the motor sections.

Option A: The model assumes these windings cost 0.25 as much as the main winding per unit length. Scaling is assumed to be the same as for the main motor winding in 3.2.

Option B: The model assumes these rescue windings consist of one phase of the standard 3 phase motors. They are costed at one-third the cost of the standard motor windings.

BACK IRON

- We have an estimate for laminated iron for one winding section. Each piece is 18' long x 0.5" high by 0.0185" thick and costs $2.30. A total of 615 pieces are used for one section, for a total cost of $1,415/section. In this case, the section is 20' long. Unless explicitly listed, back iron costs are combined with the motor winding costs.

- Assume scaling proportional to pipe diameter. The spread sheet allows for less than 100% length coverage, and the fraction of back iron coverage is set equal to the fraction of motor winding coverage.

POWER UNITS

- System approach 1: Reference design: pneumatic coupling

  - Assume power scales as follows:

$$P = mv(n\_v) + Ldv^3/Re^{0.2}$$

    The term captures the wheel bearing friction relation in the product of the mass (m), velocity (v), and number of vehicles (n-v). In the second term, we assume that the nose of each vehicle is fitted with a plate to essentially seal the tube and effectively move all the air inside the entire length (L) of pipe, of diameter (d), at velocity (v) equal to the vehicle speed. The Reynolds number in the denominator raised to the 0.2 power provides the appropriate scaling for frictional losses inside smooth pipe for Reynolds numbers above about 2 x 10^4 (about 2 miles per hour in a 24" pipe).

  - Calculations (Mechanical_calcs 5-95.xls) show that for a 30-mile-long, 24-inch diameter pipe, with the air stream moving at 40 mph, the losses over the length of the pipeline are 1.08 MW. Makeup power for these losses should be applied at each block.

  - With this strategy, rescue windings are not required, as it is assumed that a vehicle over a powered section can use the air "cushion" between itself and successive unpowered vehicles to move those vehicles ahead.

- Assume $1/watt for power supplies.

  The purchase price of the power units used in the demonstration project was only $0.17/watt, but in keeping with the low duty cycle demonstration, are pulse-rated rather than steady-state rated. An allowance of two times is allowed for purchase of steady-state rating. The balance of the allowance is for installation and ac power infrastructure to supply the distributed motor controllers.

The motor controllers will be distributed along the length of the pipeline and need to be supplied with control signals and power. If no existing power infrastructure is readily accessible, ac power lines would need to parallel to pipeline. Since a trench is necessary for the pipe, it would be cost effective to also bury a 15kV armored cable in the same trench. Periodic substations would be supplied to in turn supply a sub-set of motor controllers. Sufficient allowance has made in the motor control unit cost to cover the necessary ac infrastructure.

- Assume the return power scales in the same manner as the outgoing power. There is also some possibility that the vehicles might be able to return with an alternate cargo, so the return payload, if any, is entered as a separate input.

BLOCK CONTROL UNITS

- Assume $2,500 allowance for each block for controls and connections.

  The development of the demonstration control cards proved more expensive than anticipated because of design iterations. The final control cards, however, can be reproduced for the unit cost used in the studies

- Scaling is with the number of blocks, which equals the number of vehicles times the fraction of drive motor coverage.

CENTRAL CONTROL

- Assume a fixed $500K.

PIPELINE

- Alternate 1 basis (MIT purchase costs + *Means Building and Construction Cost Data,* 52nd Annual Addition, 1994)

  - Means data--Means gives the following (1994) data for a 24” diameter pipeline:

Table A-2. Pipeline Costs.

Piping installation cost, including material, before markup	\$22.64/ft
Material cost before markup	\$15.40/ft
Excavation	\$1.53/yd^3
Backfill by dozer, no tamping	\$1.47/yd^3
Backfill by dozer with air tamping	\$7.10/yd^3

- MIT purchase price for pipe--MIT purchased pipe for \$47.50/ft, excluding joints, in late 1997. We will substitute our piping cost for the \$15.40/ft material cost given by Means, and escalate the remaining Means data to 1998.

  Option A: Purchase price \$47.50/ft

  Option B: Reduction of 30% in purchase price. The pipe was purchased in small quantity and shipped to Boston at \$47.50/ft. The vendor estimates that a substantial order would be approximately 30% less because the small order was impacted by setup charges and shipping. Once quantities are large enough so that setup is not a factor, there is no further price reduction with quantity.

- Escalation of Means data--The Bureau of Labor and Statistics (BLS) gives CPI escalation factor from 1994 to 1998 of 160.5/144.5 = 1.111

- Joint materials--We will add 15% on to the per foot cost of our pipe, as an estimate to include the cost of joint materials.

- Trench work--We will assume the pipeline trench will be wide enough to accommodate the two-way pipeline. We will further allow for 15" on either side of the side-by-side pipe for men to work in the trench. We will also allow for an OD-OD spacing of the outgoing and return lines of 6". Thus, our trench width will equal 2*(OD) + 36". We will assume the pipeline will lie with its OD three feet below the surface of the ground, resulting in a trench depth of OD + 36". Finally, we will include trench finish work with the air tamping.

We neglect the cost for any blasting work.

Note: The costs associated with burial of the pipe are about 10% of the two way pipeline cost. If the pipe were located above ground these costs could be eliminated, but some allowance for supports would need to be added.

- Summary for 24” diameter, 2-way pipeline (Option A)

Table A-3. Two Way Pipeline Cost.

MIT purchased-price for 24” pipe with 15% adder	($/ft)	109.25
Other installation costs (Means with esc and markup)	($/ft)	19.90
Escalation costs for 24” pipe (Means plus esc)	($/ft)	2.20
Back fill with air tamping (Means plus esc)	($/ft)	10.23
Total Pipeline cost	($/ft)	141.57
Total Pipeline Cost	($/mile)	747,510

- Finally, we will assume this baseline cost scales with the pipe OD^2 for other diameter pipes.

Note: This would result in a cost for the IMC-Canada line of $2.289M/mi x 8.3 = $19M for the line excluding the cost for vehicles, pumps, valves, etc., for which we have allowed below (see first paragraph under "Alternate 2 basis") a cost of $10M. De-escalating this cost to 1992 using BLS data, would give $16.6M vs. ($30M - $10M) = $20M assumed below, so we have a good “reasonableness check of the Alternate 1 basis, which may be applied to any configuration.

- Right-of Way Cost Assumptions

Option A: No costs are included for right-of-way.

Option B: Typical RR right-of-way costs are estimated on the basis of a 100 foot right-of-way with undeveloped land at $10K and developed land at $30K/acre. If we assume a 50 foot right of way as appropriate for the pipe line, and a hypothetical 30 mile route where 20% of the total required a right of way under developed land, and the remaining 80% required a right of way under undeveloped land, the total cost would be about $3M, approximately 5% of the typical total project cost. Right of way costs are not included for short haul cases as that is assumed to be on company property.

- Alternate 2 basis

- Assume costs are similar to the IMC-Canada pneumatic pipeline proposal (Sontech 1992). Their total for the 42 inch, two-way “pipeline system” was $30M for the 8.3 mile route, with no breakdown of costs given. We will assume $13M for pipe and assembly, $7M for civil work, and $10M for the balance (vehicles, pumps, valves etc). The 42 inch x 3/8 wall steel pipe (2-way) weighs about 13M lbs, or $1/lb under our assumed split. The total cost is, therefore, about $800K/mile of one way pipe, and

$1.6M/mile for the double pipe. (We do not differentiate between the cost of a steel or plastic pipeline for the present.)

- Assume the pipeline and assembly scale with the pipe diameter squared.

- Assume the civil work costs $7M for the trench and infrastructure to accommodate the dual 42 inch pipe. The total is again about $800K/mile for the 8.3 miles long trench.

- Assume that the civil work scales with the pipe diameter.

LOAD/UNLOAD STATIONS

- See Appendix B.

- On-the-Fly Unloading Option

The December Report cost model assumed that the cars are brought to a stop for both loading and unloading. This assumption does not greatly impact the pipeline and vehicle portion of the system costs because the only real difference between an on-the-fly system and a stationary system is in the number of parallel branches needed at the ends to match the vehicle arrival intervals with required load or unload intervals. There would be major differences in the cost of the end stations, however.

In an on-the-fly *unload* system, for example, the vehicles do not slow down and there is no need for parallel branches. That eliminates the need for switches and for acceleration and deceleration windings associated with the end stations. The on-the-fly station would need the required mechanisms and magnetic structures for manipulating the vehicles. We estimate that a single on-the-fly *unload* station would cost about 45% of that of a single stationary *unload* station. In addition, the cost of any parallel branches is eliminated.

In the long-haul, large-tonnage base case, there were seven parallel branches. Replacing the seven stationary *unload* stations with a single on-the fly station could therefore reduce the *unload* component cost by a large factor.

We recommend that the option of replacing the stationary *unload* cost assumption in the cost model with an on-the-fly assumption be retained. We have done a conceptual design for such an *unload* station, and are testing the necessary magnetic manipulation systems in the demonstration project.

We do not recommend, however, including the option of changing the stationary *load* station assumption. After reviewing several ideas and existing systems, for example the Japanese pneumatic load/unload mechanisms, we do not believe it is feasible to develop an on-the-fly load system.

- Stationary Load and Unload Options

Table A-4. Typical Stationary Load/Unload Cycle Parameters at 40 MPH.

	3M ton/year Station	30M ton/year Station
Launch interval	7.7 seconds	1.9 seconds
Main line deceleration length	340 feet	340 feet
Deceleration time in main line	12.6 seconds	12.6 seconds
Branch deceleration length	60 feet	60 feet
Deceleration time in branch	5.3 seconds	5.3 seconds
Load time in branch	2.0 seconds	2.0 seconds
Acceleration time in branch	5.3 seconds	5.3 seconds
Total time in branch	12.6 seconds	12.6 seconds
Number of branches	two	seven

A trade-off study comparing multiple parallel stationary unload stations with on-the fly operations demonstrates that unloading on the fly is desirable but not essential. Reduction in capital investment for the cases considered range from 5% to 15% if on-the-fly unloading is introduced.

The elements which go into the estimate of end stations are motor modules, power conversion, pipe and supports, assembly, switches, mechanisms and local control.

The number of modules required is based on how much deceleration is done in the main line, and how much in each parallel branch. In cases where the launch interval is relatively long, most of the deceleration is done in the main line. When the interval is short, more of the deceleration must take place in each of the parallel branches.

No real engineering studies have been done of a large capacity station, but we have compared our estimates against pneumatic tube load/unload station designs. A Sonotube design for an IMC Canadian mine priced the load and unload stations for an 8 megaton/year system at $6M each. Our economic model would cost comparable capacity electromagnetic stations at 80% of that cost. We would argue that the electromagnetic systems are simpler than the pneumatic end stations, and are thus comfortable with the comparison.

Note: The IMC-Canada (Sontech 1992) load and unload stations cost about $10M each for a capacity of 20,000 t/day.

COST PER TON

- For the cases where capital cost is shown to be recovered as a specific line item in the operating cost, simply amortize the capital cost over 5 years with at no interest.

- For the cases where we look at possible capital recovery rates based on comparisons with alternatives like the railroad, we calculate the break-even capital recovery rate, typically over 15 or 20 years, on a case by case basis.

- Depreciation

  Option A: No depreciation was used in Option A.

  Option B: IMC-Agrico depreciates capital investment in 8 years. They use a non-linear formula allowed by the government as shown below:

Table A-5. Depreciation Schedule.

Year	% of Original Investment
0	14.3
1	24.5
2	17.5
3	12.5
4	8.9
5	8.9
6	8.8
7	4.5
Total	100.0

- Taxes

  Option A: No taxes are included.

  Option B: A tax rate of 27% is used for the first 15 years, and 30% thereafter.

- Discount Rate

  Option A: No discount rate is used.

  Option B: IMC-Agrico typically uses 10%.

OPERATING COST

- Operating power

  Option A: Operating power is assumed to cost $0.12 per kW/hr

  Option B: In Florida, costs can be as low as $0.04 per kW/hr

- Maintenance costs

  Maintenance costs are assumed to be a fixed percentage (3%) of investment. Note: Dr. Kosugi from Sumitomo mentioned in his 4/2/98 visit that the hardware maintenance costs are typically 3% of capital costs, so we have applied that factor here.

- Labor Costs

  Option A: Assume that a minimum crew of a manager, a senior engineer, two junior engineers, and two technicians are required regardless of the size of the line and number of cars. In addition, assume a fixed percentage (0.5%) of vehicles need maintenance each day, and that one tech can repair a fixed number (2) of vehicles per day. Add additional technicians to repair vehicles based on system vehicle count.

  Operations Labor Assumptions: a fixed crew of a manager, senior engineer, two junior engineers and two techs are assumed independent of size of system. They would divide the shifts between themselves. The burdened rates used are consistent with current IMC-Agrico rates.

Table A-6. Labor Burdened Rates.

Category	Burdened Hourly rates assumed:
Manager	$100
Senior Engineer	$75
Junior Engineer	$50
Technician	$37

Maintenance Labor assumptions: Additional techs are supplied for maintenance by the formula: 0.25% of two-way cars repaired each day, 0.5 tech-day needed per vehicle. For a 2000 car system, 10 cars would be maintained each day, requiring a crew of 5 additional techs.

Option B: Short-Haul Cases only

Since the dominant operating cost for the short haul case is actually the labor cost for operations, a more careful examination of the assumptions is warranted. It would seem reasonable that the senior engineer could assume management responsibility, eliminating the need for an additional manager position. This alone reduces the labor cost by 30%. It is also assumed that the two technicians included in the operating crew could maintain the vehicles, thus not requiring any additional maintenance techs.

SAMPLE CALCULATIONS INVOLVING RATE OF RETURN

- Sample Calculation of Capital Recovery Cost at a fixed rate of return as used in Option A.

Option A cases calculate the "system cost per ton-mile" by combining the annual operating cost with the cost of capital recovery at an assumed rate of return. Calculations were on a before-tax basis.

Table A-7. Sample Calculation Using Fixed Rate of Return.

Distance (miles)	30
Tonnage (M tons/year)	10
Capital cost ($M)	61.20
Operating cost ($M)	6.70
Operating cost ($/ton-mile)	0.022
Capital recovery annual cost ($M) *	12.6
Capital recovery cost ($/ton-mile)	0.042
System cost ($/ton-mile)	0.064

* Using a fixed rate of return of 20% over 20 years

- Sample Calculation of before and after-tax rate of return using net present value and depreciation as used in Option B.

Table A-8. Sample Calculation Using Net Present Value.

Depreciation & Tax Impact		Yr 0	Yr 1	Yr 2	Yr 3 *	Notes
Capital cost	61.20					
Operating cost	6.70	6.70	6.70	6.70	6.70	
Pre-project operating cost	24.00	24.00	24.00	24.00	24.00	
Pretax savings	17.30	17.30	17.30	17.30	17.30	
Pretax cash flow		-43.90	17.30	17.30	17.30	
Present value	55.73					
Rate of return	37.11					
Depreciation (%)		14.3	24.5	17.5	12.5	
Depreciation		8.75	14.99	10.71	7.65	3
Taxable income		8.55	2.31	6.59	9.65	
Tax at 27%		2.31	0.62	1.78	2.61	1
After tax savings		14.99	16.68	15.52	14.69	
After tax cash flow		-46.21	16.68	15.52	14.69	
Present value	37.27					
Rate of return	29.14					
Payback period	2.95					
NPV of cash flow						
Discount Rate (%)	10					2
Discount multiplier		1	0.91	0.83	0.75	
NPV of pre-tax savings	0.00					
Discount Rate (%)	37.11					4
Discount multiplier		1	0.73	0.53	0.39	
NPV of after-tax savings	0.00					
Discount Rate (%)	29.14					4
Discount multiplier		1	0.77	0.60	0.46	
Cumulative after tax savings		14.99	31.67	47.19	61.88	
Increment to go		46.21	29.53	14.01	-0.68	
%				90.27	-4.66	
Payback period (yrs)	2.95					5

*The table shows the first four years of a typical ten year calculation.

Table A-8 Notes:

1) Taxes: A tax rate of 27% is used for the first 15 years, and 30% thereafter.

2) IMC-Agrico typically uses a 10% discount rate.

3) Depreciation

IMC-Agrico depreciates capital investment in 8 years. They use a non-linear formula allowed by the government as shown in Table A-5.

4) The rate of return is defined as equal to the discount rate that if applied, would result in a zero net present value.

5) The payback period is defined as the number of years beyond the zero year at which the accumulated savings equals the initial investment.

Appendix B

ORE CAR LOAD/UNLOAD STATION CONFIGURATION, DESIGN ALGORITHM AND COSTING

Appendix B

ORE CAR LOAD/UNLOAD STATION CONFIGURATION, DESIGN ALGORITHM AND COSTING

INTRODUCTION

This memo describes a possible load/unload station configuration for the ore car project, and develops the associated cost model. At present, the model assumes that payload is carried only one way, and that the cars return empty. Thus, at one end of the line there are only load stations, and at the other end of the line, there are only unload stations. This assumption can be modified later, if necessary. After describing the model, the algorithm and the cost inputs, a spreadsheet example is given.

ENGINEERING MODEL DESCRIPTION

The assumed configuration is shown in the attached figure for a two-end-station case. This will also be called a two-branch case, as the transit line branches into two unload stations. To simplify the explanation and calculations, the load and unload ends of the line are assumed to be perfectly symmetric, so the “unload” stations shown in the figure could just as well be “load” stations at the other end of the line.

Coupled car sets approach the unload station as a group along the lower transit line with a time separation given by the launch interval (p_launch, seconds), which is determined by the individual car payload (m_load, tons), the number of coupled cars (n_c), the required annual tonnage rate (Cyt, tons/y), and an assumed time utilization factor (f_time).

$$p\_launch = \left(\frac{(m\_load)(n\_c)(f\_time)}{\frac{(Cyt)}{60*60*24*365}} \right) \qquad (1)$$

Cars then enter the deceleration region, where their speed is reduced from the transit speed (v_t) to some initial end-station speed (v_esi), which is discussed further below. To avoid car bunching or collisions at the ends, the required number of end-stations (or number of branches, n_branch) is determined by the total time each coupled-car set spends in the end station (t_end) divided by the launch interval.

$$n\_branch = \frac{t\_end}{p\_launch} \qquad (2)$$

To simplify the calculations, the total end-station length (l_end) is assumed to be constant for each branch. Its minimum value is set as an input to the spread sheet. In the end-station length, coupled cars are switched as a group into their branches, decelerated from their initial end-station speed (v_esi) to a full stop at the unload station, and unloaded. Once unloaded, the cars are sent back through a second switch where they are simultaneously pre-accelerated to v_esi and routed to the return line accelerator, where they complete their accelerations to transit velocity, v_t, for the return trip. Several points may be made about this end-station configuration.

The end station time is comprised of three components:

1) deceleration time in the end-station length
2) unload (load) time (t_u)
3) acceleration time in the end-station length , which, with symmetry, is assumed to equal the deceleration time in the end-station length

The total deceleration from transit velocity is divided between the main deceleration region, and the end-station deceleration. With a fixed end-station length (l_end) and a maximum available deceleration (a_max), we can define the minimum deceleration time in the end station

$$td\_end = \sqrt{\frac{2(l\_end)}{a\_\max}} \qquad (3)$$

and the total time in the end-station (t_end) is then given by

$$t\_end = 2td\_end + t\_u \qquad (4)$$

Now, if we were to combine (1) - (4), we would have an explicit expression for the number of end-station branches required.

We note, however, that the deceleration regions may be split between the main line and the end stations, *provided the terminal velocity, v_esi, at the end of the main line deceleration region is not so slow as to place two coupled-car sets in the same motor section.* We can be sure that this is true by ensuring that this velocity is sufficiently large:

$$v\_esi \geq \frac{l\_s}{p\_launch} \qquad (5)$$

Note that there are two implications of equation (5):

1) If the determined p_launch from (1) is so small that v_esi is greater than our nominal design transit velocity, v_t (ft/s), for the given design section length, we must either decrease the section length, or increase the velocity if we are to have a feasible design. This condition, in reality, sets the limit capacity of the line. If v_esi just equals (l_s)/(p_launch), we are at the line limit capacity, *and there can be no deceleration in the main line; all deceleration must be in the branches.* We can write the equation for the line limit capacity, Cyt_lim, as

$$Cyt\_lim = \frac{(m\_load)(n\_c)(f\_time)(v\_t)}{l\_s}(3600)(24)(365) \qquad (6)$$

2) We are assuming in (3) that we can decelerate from v_esi to 0 in l_end at a maximum deceleration of a_max. This also implies

$$v\_esi \le (a\_\max)(td\_end) \qquad (7)$$

If the constraints on v_esi in (5) and (7) are incompatible, then we must increase l_end until we can be assured that we can decelerate from v_esi to 0 in the allotted distance. Equation (7) ensures this condition, and, together with equation (3), sets a lower limit on l_end:

$$v\_esi \le \sqrt{2(l\_end)(a\_\max)} \qquad (8)$$

or

$$l\_end \ge \frac{(v\_esi)^2}{2(a\_\max)} \qquad (9)$$

or, using (5) with (8):

$$l\_end \ge Max\left[\left(\frac{(l\_s)^2}{2(a\_\max)(p\_launch)^2}\right), l\_end\_\min\right] \qquad (10)$$

where *Max(a,b) is the maximum of a or b.*

Finally, we would like to calculate how many vehicles will be in the end stations, since the total number of vehicles required for the system will be the sum of those in transit plus those in the end stations. This calculation can be made simply if we observe that, in steady state, we are neither accumulating vehicles to nor depleting vehicles from the end stations. Therefore, the number of vehicles in each end station (nv_end) will be simply equal to the rate at which vehicles are entering the end station times the time in the end station.

$$nv\_end = \left(\frac{t\_end}{p\_launch}\right)n\_c \qquad (11)$$

ALGORITHM SUMMARY

Finally, our algorithm may be summarized as follows:

Table B-1. Algorithm Summary Inputs.

Parameter	Symbol	Units
Yearly tonnage requirement	Cyt	tons/yr
Payload per car	m_load	tons
Time utilization fraction	f_time	dimensionless
Minimum end station length	l_end_min	ft
Max accel or decel rate	a_max	ft/s^2
Design transit velocity	v_t	ft/s
Unload time	t_u	s
Motor section length	l_s	ft
Number cars coupled together	n_c	dimensionless

1) Solve for the launch interval, p_launch, using (1):

$$p\_launch = \left(\frac{(m\_load)(n\_c)(f\_time)}{\frac{(Cyt)}{60*60*24*365}} \right)$$

2) Solve for l_end using (10):

$$l\_end = Max\left[\left(\frac{(l\_s)^2}{2(a\_\max)(p\_launch)^2} \right), l\_end\_\min \right]$$

where *Max(a,b) is the maximum of a or b*

3) Solve for the deceleration (acceleration) time in the end station, td_end, using (3)

$$td\_end = \sqrt{\frac{2(l\_end)}{a\_\max}}$$

4) Solve for the time in the end station using (4)

$$t\_end = 2td\_end + t\_u$$

5) Solve for the number of end-station branches, n_branch, using (2):

$$n\_branch = \frac{t\_end}{p\_launch} \qquad \text{(next integer)}$$

6) Solve for the number of motor decelerator plus accelerator sections, n_motor_branch, in each branch

$$n\_motor\_branch = 2\left(\frac{l\_end}{l\_s}\right) - 1 \qquad \text{(next integer)}$$

Note that this equation assumes the last motor section is, in fact, at the unload station. It is used to bring the car to a stop, is turned off for unload, and then is re-powered to be the first stage of acceleration for the return trip.

7) Solve for the total deceleration distance, ld, at a_max.

$$ld = \frac{(v\_t)^2}{2(a\_\max)}$$

8) Solve for the deceleration distance, ld_main at the end of the main line.

$$ld\_main = ld - l\_end$$

9) Solve for the number of motor decelerator and accelerator sections at the ends of the main line.

$$n\_motor\_main = 2\frac{ld\_main}{l\_s}$$

10) Solve for the number of switches, n_switch, at each end station.

$$n\_switch = 2(n\_branch) \qquad \text{(next integer)}$$

11) Assume the number of pipe and support sections in the end stations equals the number of motor sections in the end stations.

12) Calculate the number of vehicles in each end station using (11).

$$nv\_end = \left(\frac{t\_end}{p\_launch}\right)n\_c$$

COST ASSUMPTIONS

The assumed unit costs are summarized in the following table.

Table B-2. Assumed Unit Costs.

Element	**Unit cost ($k)**	**Comment**
Windings (motors)	15	
Power conversion	10	per winding
Pipe and support	5	per winding
Assembly	2.5	per winding
Switch	25	
Load/unload mechanism	100	per branch
Local control	25	per end

EXAMPLE

The spreadsheet on the next page summarizes the end station (one end) costs for a specific case (3 Mt/yr, 40 mph, 6' car, 60' minimum end station length). Engineering inputs are given first, followed by engineering calculations, then the unit cost inputs are given (repeating the table in 3.0 above) and finally, the cost elements are given with a total cost.

Table B-3. Example of End Station Cost Calculation.

Engineering inputs in blue				
	Yearly tonnage requirement	Cyt	tons/yr	3.00E+06
	Hopper length	Lh	ft	4
	Hopper diameter	Dh	ft	1.67
	% full	f_fill		0.7
	Ore density	rho	lb/ft^3	100
	Time utilization	f_time		0.8
	Minimum end station length	l_end_min	ft	60
	Maximum accel or decel	a_max	ft/s^2	4.30
	Design transit velocity		mph	40
		v_t	ft/s	58.67
	Motor section length	l_s	ft	20.00
	Unload time	t_u	s	2.00
	Number of cars coupled together	n_c		3.00
Engineering calculations				
	Payload/vehicle	m_load	tons	0.31
1	Launch interval	p_launch	s	7.71
	Line limit capacity	Cyt_lim	tons/yr	6.78E+07
2	End station length	l_end	ft	60
3	Deceleration time in end station	td_end	s	5.28
4	Time in end station	t_end	s	12.56
5	Number of end-station branches	n_branch		2
6	Number of motor sections/branch	n_motor_branch		5
7	Total deceleration distance	ld	ft	400
8	Main line decel distance	ld_main	ft	340
9	Number of main line motor decel sections/end	n_motor_main		34
	Total number of motors/end	n_motor_end		44
10	Number of switches/end	n_switch		4
11	Number of pipe and support sections/end	n_ps		44
12	Number of vehicles in each end station	n_v		6
Cost inputs in green		Unit cost ($k)		
	Windings (motors)	15		
	Power conversion/motor	10		
	Pipe and support/motor	5		
	Assembly/motor	2.5		
	Switches	25		
	Load/unload mechanism/branch	100		
	Local control	25		
Calculated cost per end		Cost ($k)		
	Windings (motors)	660		
	Power conversion	440		
	Pipe and support	220		
	Assembly	110		
	Switches	100		
	Load/unload mechanism	200		
	Local control	25		
		1,755		

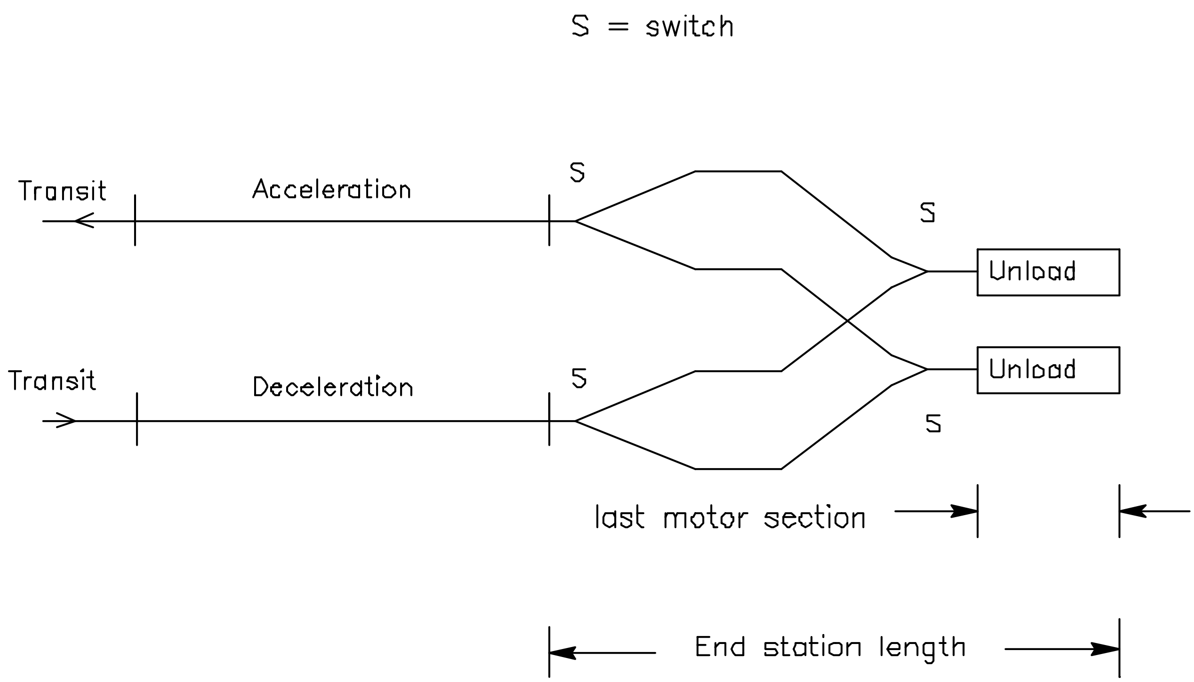

Figure B-1. End Station Configuration.